CANADA
DEPARTMENT OF MINES
HON. CHARLES STEWART, MINISTER; CHARLES CAMSELL, DEPUTY MINISTER

MINES BRANCH
JOHN McLEISH, DIRECTOR

INVESTIGATIONS IN 1921

ORE DRESSING AND METALLURGY

(Testing and Research Laboratories)

(*a*) General review of investigations: by W. B. Timm.
(*b*) List of ores and metallurgical products on which experimental test and research work was conducted: by W. B. Timm.
(*c*) Reports of the investigations conducted: by R. K. Carnochan, C. S. Parsons, R. J. Traill, and others.
(*d*) Other test work of the Division: by W. B. Timm.
(*e*) Additional equipment for the laboratories: by W. B. Timm.
(*f*) Work and investigations of the Chemical Laboratory: by H. C. Mabee.

(Annual Summary Report of the Mines Branch, pp. 78–204.)

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ORE DRESSING AND METALLURGICAL DIVISION

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm

Chief of Division

The Ore Dressing and Metallurgical Division conducted, during the calendar year 1921, a considerable number of investigations involving extensive experimental and research work on Canadian ores and metallurgical products. Some of these investigations were completed and reported upon, and on others the work is being carried into 1922.

An idea of the extent and scope of the work of the testing and research laboratories can be obtained from a review of the list of ores and metallurgical products received for experimental purposes, which is given in tabular form following, under their respective test numbers. These may also be classified under the following investigations:—

1. THE LOW-GRADE COPPER ORES OF HIDDEN CREEK, ANYOX, B.C.

This investigation is described under test No. 140 of the Ore Dressing and Metallurgical Laboratories, and consisted of a series of flotation tests on a small scale, using the laboratory type apparatus, and also of larger tests, using the commercial size machinery. Various reagents were used, such as the Rex flotation reagents, oils, with and without other reagents, and the Alphabetical reagents. The larger scale tests were conducted on both the mechanical agitation and pneumatic types of flotation machines. A few of the larger scale tests remain to be conducted, but the results of the work to date show that the ore is amenable to concentration by flotation. A high recovery of the copper values was obtained in a good grade of concentrate.

2. THE FLIN FLON COMPLEX SULPHIDE ORE

This investigation is described fully under test No. 142 of the Ore Dressing and Metallurgical Laboratories, and consisted of a large number of flotation tests, using the laboratory type apparatus on the differential flotation of the minerals in this ore. The Flin Flon ore-body consists of three distinct mineralized zones—the hanging wall zone of disseminated ore, the centre zone of practically solid sulphides of iron, copper and zinc, and a footwall zone. This investigation covers the experimental and research work on the centre sulphide ore, and the object was to determine if the minerals could be separated into products suitable for reduction, or metallurgical treatment, into the metallic state. The shipments received consisted of drill cores crushed to about one-half inch size, from the exploration work on this property.

A microscopic examination of some of the larger pieces showed an intimate mixture of the three sulphides, with very little gangue mineral. Grinding to 200 mesh freed the greater part of the mineral constituents, and it was at this mesh that the most of the test work was conducted. The results of the test work showed that by differential flotation a copper concentrate, a zinc concentrate, and an iron tailing can be produced, but whether this can be accomplished on a large scale in practice has yet to be proven. It was also shown that the precious metal values in the ore are evenly distributed in all the sulphides, and that only 40 per cent of these values report in the copper concentrate. Besides the microscopic examinations made by the Mines Branch, examinations were made by Professor Wandke of Cambridge University, on the ore and concentration products, and his findings are given in the description of the test work under test No. 142.

3. THE CONCENTRATION OF GRAPHITE FROM THE NORTH AMERICAN MINE

This investigation is described fully under test No. 143 of the Ore Dressing and Metallurgical Laboratories, and consisted of a number of experimental tests, run under different conditions and various methods of procedure, to determine which would be the most desirable and practical for the recovery of the greatest proportion of large flake of a commercial grade, consistent with a high recovery of the graphite in the ore. A method of concentration was proven, in which these points were determined, adaptable to this particular ore.

4. THE CONCENTRATION AND SEPARATION OF THE MINERALS IN THE LEAD-ZINC-SILVER ORES OF KIMBERLEY, B.C.

This investigation is described under tests Nos. 144 and 145 of the Ore Dressing and Metallurgical Laboratories, and consists of the experimental work on two shipments of ore—one that is known as carbonate ore, and the other as sulphide ore. These ores are typical of the district, the carbonate ore being found in the upper portions of the ore-body, and the sulphide ore at greater depth. The carbonate ore consisted chiefly of lead carbonate, coated and locked up in a mass of iron oxides, formed by the oxidation of the iron sulphides. The sulphide ore was an intimate mixture of lead, zinc, and iron sulphides, the gangue minerals representing less than 10 per cent of the ore. On the carbonate ore experimental test work was conducted by sulphidizing previous to flotation and table concentration of the flotation tailing, and also by table concentration followed by sulphidizing and flotation of the table tailing. Volatilization tests were also conducted to determine whether the ore was amenable to this process.

By these methods a recovery of between 70 per cent and 80 per cent of the lead values, and 40 per cent of the silver values, is obtained.

On the sulphide ore, a series of tests were conducted, using various flotation reagents, to obtain a differential flotation of the lead, zinc, and iron sulphides. A lead concentrate was first obtained, and by changing conditions of the pulp, a zinc concentrate was next obtained, leaving a tailing of iron sulphides and gangue mineral. The lead and zinc products were of commercial grade, with good recoveries of the metals in these products.

5. SOME COMPARATIVE TESTS ON THE COBALT SILVER SAND TAILINGS

In continuation of some extensive experiments made in 1917 on the use of Canadian wood oils for the concentration of the values in the Cobalt slime tailing, a few experiments were conducted on the use of the Rex, Alphabetical and other reagents, on a shipment of sand tailing from one of the Cobalt mines. The results of this work are described under test No. 146 of the Ore Dressing and Metallurgical Laboratories.

6. THE MILLING OF ASBESTOS ROCK

This investigation is described under tests Nos. 147, 149, and 152 of the Ore Dressing and Metallurgical Laboratories, and consisted of a number of milling tests on the "slip fibre" variety of asbestos rock from the Boston mine, East Broughton, Que., and on the "cross fibre" variety from Black Lake, Que. The investigations carried on during the calendar year were confined to the use of the conical ball mill for the crushing of the rock and liberation of the fibre. The use of Cyclone and Jumbo fiberizers in the present practice in the asbestos districts is found to be costly as to power consumption, maintenance, and repair. The violent action in these machines as operated for the crushing of the rock and liberation of the fibre is found to be more or less destructive of the long fibre. It was found that the conical ball mill, operated under favourable conditions and equipped with suction take-off for the removal of the fibre when liberated, could be used in the milling of the rock, giving very satisfactory results. In conducting these milling tests, other points of interest were brought out, which are noted in the description of the investigations.

As one of the chief items of cost in the milling of asbestos rock is that of drying the ore for the present practice, and as this cost is increased greatly during the winter and spring seasons, it is the intention to continue the investigations on the wet grinding of the rock. If the dry milling practice could be dispensed with, and a wet process evolved to treat the rock successfully, it would eliminate the dust, resulting in much more pleasant conditions for the workmen and for the inhabitants of the districts in the vicinity of the mills.

7. THE CONCENTRATION OF THE LOWER GRADE COPPER-NICKEL ORES

This investigation is described under tests Nos. 148, 155, and 160 of the Ore Dressing and Metallurgical Laboratories, and consists of a number of table concentration and flotation tests for the elimination of the gangue and the concentration of the copper-nickel minerals in a higher grade product for further metallurgical treatment. In the case of the experimental work conducted under test No. 148, it was desired that the gangue minerals be eliminated and the metallic minerals of copper, nickel, and iron concentrated into a product suitable for the refining process of the Burrows Refining Co., who are conducting investigations on the manufacture of certain alloy steels. In the case of the experimental work conducted under test Nos. 155 and 160, the test work was confined to the flotation of the lower grade ores to produce a high-grade copper-nickel product with a high recovery of these values. It was found that the ores are amenable to concentration by flotation, and a copper-nickel product can be obtained with a base metal value the equivalent of the blast furnace matte, and with a good recovery of the copper-nickel values in the ores. In connection with the investigation, a point which has not been proven is whether the precious metal values are contained in the concentrate or the tailing, and further work is being conducted to determine where these values report.

8. THE CONCENTRATION OF GARNET FROM ITS ROCK

This investigation is described in detail under tests Nos. 150 and 158 of the Ore Dressing and Metallurgical Laboratories, and consists of experimental work on samples from the vicinity of Bancroft, Ont., and from Depot Harbour, Ont., for the determination of garnet content, percentage of recoverable garnet, and methods of concentration and separation of the garnet from the gangue. In the case of the deposits at Depot Harbour, considerable attention was given to the work on the samples, which were taken systematically, and represented a large deposit situated favourably from a shipping standpoint for the disposal of the concentration products, and on account of the high percentage of recoverable garnet. It was found that the garnet could be concentrated and separated from the gangue, and recovered in marketable products, suitable for the trade, with high recoveries of the garnet content in the rock. Methods for its recovery were worked out, with a possible saving of certain mica products as a by-product from the operations.

9. THE RECOVERY OF BRASS FROM FOUNDRY REFUSE

The experimental work on this investigation is described under test No. 151 of the Ore Dressing and Metallurgical Laboratories, and consists of tests made for the concentration and separation of the brass from the slag, sweepings, and other refuse collected around brass foundries. It was found that it would be possible for many of the brass foundries to install a small unit, consisting of a ball mill, classifier, and table, to recover the brass from their refuse, and use the concentrated product in subsequent remelts.

10. THE METALLURGICAL TREATMENT OF THE GOLD ORES FROM THE RICE LAKE DISTRICT, MANITOBA

The experimental work on this investigation is described under tests Nos. 141 and 153 of the Ore Dressing and Metallurgical Laboratories. It consists of a series of tests to determine methods applicable to this particular type of gold ore, which contains copper minerals. It was found that the ore could be treated for the recovery of gold, with a high extraction of the precious metals, and also for the recovery of the copper, as a by-product of the milling operations. This report, with previous investigations on the ore from Little Rice lake, which is described under test No. 135 of the Summary Report for 1920, constitutes the experimental test work undertaken by the Department on the gold ores from the south-eastern section of Manitoba. The Report of the Ore Dressing and Metallurgical Laboratories, test No. 132, describes the experimental test work on the gold ore from the Herb Lake district, in the northwestern section of the province. The ore from this district contains chalcopyrite, arsenopyrite, pyrite, sphalerite, and galena, and in this respect is somewhat different from the ordinary gold ores and requries a different method of treatment. It was found that the ore was amenable to treatment, described under test No. 132 in the Summary Report for 1920.

11. CONCENTRATION TESTS ON SOME MOLYBDENITE ORES

The concentration of the molybenite ores of Canada has been fairly well covered by the experimental work carried on in the Ore Dressing and Metallurgical Laboratories during the war years, when there was a brisk demand for the metal for the manufacture of tool-steel and for other munition purposes. Since that time the industry has been more or less dormant, the production in Canada falling off to practically nothing. Interest during the year has been revived somewhat, due to the research conducted to find new uses for industrial purposes. It has been determined that the use of fractional percentages of the metal, that is, quantities up to one per cent, is beneficial to most steels. With a revival of the steel industry it is logical to assume that there will be a revival of the molybdenite industry of Canada. The experimental work conducted during the year was on ores representing two of the most promising deposits in Ontario and Quebec, and it was found that these ores were amenable to concentration, with a high recovery of the molybdenite values in a high grade concentrate. The test work is described under tests Nos. 154 and 157 of the Ore Dressing and Metallurgical Laboratories.

12. THE RECOVERY OF THE VALUES IN A SILVER LEAD ORE FROM ARROWHEAD, B.C.

The experimental work on this ore is described under test No. 156 of the Ore Dressing and Metallurgical Laboratories. The ore is a lead carbonate carrying silver values. During the year some concentration tests were made by tabling and flotation. Further work is being conducted and it will be reported on during the new year.

13. THE CONCENTRATION OF THE RADIO-ACTIVE MINERAL EUXENITE

The occurrence of this mineral in the feldspar quarry of lot 13, concession V, of the township of South Sherbrooke, county of Lanark, in the province of Ontario, has been reported on by Miller and Knight, who visited the property in June, 1917. The report was published in the American Journal of Science, Volume XLIV, September, 1917, and contains a description of the occurrence as seen at that date. The mineral was identified as euxenite by the Imperial Institute, London, England, who made the analysis which is given in the report. In the operation of the quarry since that date more of the mineral has been exposed, and shipments have been made to the Department for experimental work on the concentration of the mineral from the feldspar. It was found that due to its specific gravity, being almost 5, and that of feldspar between 2·5 and 2·75, there was no particular difficulty in making a separation of the minerals. The experimental work is described under test No. 159 of the Ore Dressing and Metallurgical Laboratories.

14. THE USES OF FLOTATION REAGENTS MANUFACTURED IN CANADA

During 1917, the Division of Ore Dressing and Metallurgy, in co-operation with the Forest Products Laboratories of Canada, Forestry Branch, Montreal, started an investigation on the use of Canadian wood oils for ore flotation. This investigation was undertaken at the request of the Cobalt mining interests, and was chiefly confined to the cobalt-silver ores. At that particular time some difficulty was being experienced by the mining companies using the flotation process for the recovery of valuable minerals in their ores, in obtaining an adequate supply of imported pine oil. The investigation followed along two lines, namely, the production in Canada of a satisfactory grade of pine oil in sufficient quantities from Canadian resinous wood waste, or the possibility of finding a substitute for pine oil. The investigation showed that it was impossible to produce at a profit a satisfactory grade of pine oil, on account of the low yield from Canadian resinous wood waste, so that this line of investigation was eliminated, and attention paid to finding a substitute. The most likely substitute was the hardwood oils obtained from the destructive distillation of Canadian hardwoods, which could be produced as a by-product. It was found that as far as the Cobalt ores are concerned, after numerous experiments on a small scale in the Ore Dressing and Metallurgical Laboratories, and on a large scale of several days run in one of the Cobalt mills, that results equally good as those obtained by the use of pine oil could be obtained.

This investigation was discontinued on account of more important work in connection with the war minerals, but is now being continued on other ores where flotation is used to recover the valuable minerals. The investigation will not be confined to the hardwood oils, or the distillation products from hardwoods, but will cover the use of the mineral oils and other reagents manufactured in Canada, or which could be produced at a profit. A questionnaire has been submitted to the mining companies asking for their co-operation in stimulating the use of Canadian-made products, and for data with respect to the quantity of oils and reagents used, costs, etc. As soon as this information is collected, and samples of the various flotation feeds are obtained, the experimental work will be conducted, to determine if Canadian products cannot be substituted in many cases for those imported.

15. THE RECOVERY OF THE METALS OF THE PLATINUM GROUP FROM PLACER CONCENTRATES

An investigation was commenced in 1919, on the recovery of the platinum group metals from placer concentrates, but due to the limited supply of material submitted for test purposes, and the small quantity of these metals in the sample, little progress was made. The test work conducted was described in the Summary Report for 1919, under test No. 109 of the Ore Dressing and Metallurgical Laboratories. A larger sample has been received, and the investigation is being continued.

Some experimental work was conducted by making fusion tests on the placer concentrates, by fluxing them with limestone, silica, and charcoal, and adding sufficient galena to the charge to collect the precious metals in a lead button. The results of the work are given under the report of the work of the Chemical Laboratories of the division.

II

LIST OF ORES AND METALLURGICAL PRODUCTS ON WHICH EXPERIMENTAL TEST AND RESEARCH WORK WAS CONDUCTED

In the following table is given a list of the ores and metallurgical products received, with the test number, class of ore or metallurgical product, source of shipment, shipper, and weight of shipment, on which experimental test and research work was conducted.

Test No.	Ore, or metallurgical product	Source of shipment	Shipper and address	Weight of shipment
				lbs.
140	Copper.........	Hidden creek, No. 2 mine.	Granby Consolidated Mining, Smelting & Power Co., Ltd., Anyox, B.C.	50,560
141	Gold...........	Rice Lake, Manitoba.	Pan Extension Gold Mines, Ltd., Winnipeg, Man.	10
142	Copper.........	Flin Flon mine, Manitoba.	Mining Corporation of Canada, Ltd., Toronto, Ont.	250
143	Graphite.......	North American mine, Buckingham, Que.	H. P. H. Brumell, Buckingham, Que.	50
144	Lead-silver.....	North Star mine, Kimberley, B.C.	Thompson & McKinney, Kimberley, B.C.	370
145	Lead-zinc-silver	North Star mine, Kimberley, B.C.	Thompson & McKinney, Kimberley, B.C.	350
146	Silver..........	Coniagas mine, Cobalt, Ont.	Coniagas Mines, Ltd., Cobalt, Ont..	2,000
147	Asbestos.......	Boston mine, E. Broughton, Que.	Asbestos Mines, Limited, Montreal, Que.	500
148	Copper-nickel...	Strathcona mine, Sudbury, Ont.	Burrows Refining Co., Ltd., Ottawa, Ont.	48,000
149	Asbestos........	Boston mine, E. Broughton, Que.	Asbestos Mines, Limited, Montreal, Que.	26,000
150	Garnet.........	Lot 7, con. XI, Faraday tp., Hastings Co., Ont.	Robt. A. Bryce, Toronto, Ont.......	100
150	Garnet.........	Lots 29, 30, 31 and 32, con. XIV, XV, Ashby tp., Lennox and Addington co., Ont.	Robt. A. Bryce, Toronto, Ont.......	100
151	Brass foundry refuse.	Ontario Specialties, Ltd., Ottawa, Ont.	Ontario Specialties, Ltd., Ottawa, Ont.	16,520
152	Asbestos........	Black Lake, Que......	Black Lake Asbestos & Chrome Co., Black lake, Que.	20,000
153	Gold...........	Rice Lake, Man.......	Pan Extension Gold Mines, Ltd., Winnipeg, Man.	2,000
154	Molybdenite....	Lots 3 and 4, con. XX, Cardiff tp., Haliburton co., Wilberforce, Ont.	W. E. Joiner, Toronto, Ont..........	800
155	Copper-nickel...	No. 3 mine (Frood)...	International Nickel Co., Copper Cliff, Ont.	100
156	Silver-lead......	Arrowhead, B.C......	Mansfield Mining Co., Mansfield, Washington, U.S.A.	220
157	Molybdenite....	Lots 53 and 54, range X, Masham tp., Ottawa co., Indian lake, Masham, Que.	John Bain, Ottawa, Ont............	100
158	Garnet.........	Depot Harbour, Ont..	Robt. A. Bryce, Toronto, Ont.......	13,500
159	Euxenite........	S. Sherbrooke tp., Maberley, Ont.	Orser-Kraft Feldspar, Ltd., Perth, Ont.	1,600
160	Copper-nickel...	Murray mine, Nickelton, Ont.	British America Nickel Corporation, Nickelton, Ont.	200

A description of the experimental test and research work conducted is given under the heading of the investigation, with the corresponding test numbers, as a sub-heading, so that the test numbers will not be in order as denoted above, in the tabular form. In some cases an investigation will cover three test numbers.

III

REPORTS ON THE INVESTIGATIONS CONDUCTED

The reports on the investigations conducted by officers of the Division cover a wide range of metallurgical problems. The investigations were assigned to the technical officers who, by experience, training, and knowledge of the subject, were most fitted to carry out the particular investigation. Those investigations entailing experimental test and research work along flotation lines were assigned to C. S. Parsons. Others, in which test and research work on milling and other ore dressing and metallurgical problems was required, were assigned to R. K. Carnochan. Investigations with respect to the recovery of the precious metals, including those of the platinum group, were assigned to H. C. Mabee. These technical officers were assisted by R. J. Traill and B. P. Coyne in carrying out certain phases of the work which they were specially capable of, due to their knowledge of physical and chemical reactions. All investigations were under the direction of the Chief of Division, who, in consultation with the technical officers of the division, outlined the procedure to be followed in conducting the test and research work.

(1)

THE LOW GRADE COPPER ORES OF HIDDEN CREEK, ANYOX, B.C.

R. K. Carnochan and C. S. Parsons

Test No. 140

A carload of copper ore, shipping weight 50,760 pounds, was received at the Ore Dressing and Metallurgical Laboratories, September 4, 1920, from the Granby Consolidated Mining, Smelting and Power Company, Anyox, B.C. The ore was a representative shipment of the lower grade from Hidden Creek No. 2 mine, and consisted of chalcopyrite and iron sulphides in a greenstone gangue.

The purpose of the experimental work on this ore was to determine its adaptability to concentration by flotation methods; to obtain some comparative data on the use of oils, the Rex, Alphabetical, and other flotation reagents, and also on the use of the pneumatic and mechanical agitation types of flotation machines.

A number of small laboratory tests had been made on similar ore by R. E. Peterson, using certain oils and the addition of lime, giving very satisfactory results. The results of two of these tests are given below, as representative when making low and high grade concentrate:—

Test No. 913:

Amount tested: 500 grams.
Apparatus used: Janney test machine.
Reagents used: ¾ pound per ton of 50 per cent mixture of coal tar creosote and hardwood creosotes and tars; 2 pounds of lime per ton.

Product	Weight grams	Analysis per cent				Per cent of metals in products			
		Cu.	Fe.	S.	Insol.	Cu.	Fe.	S.	Insol.
Heads.............	500	2·30	25·3	19·0	34·8	100·0	100·0	100·0	100·0
Concentrate........	133	7·61	36·7	37·4	8·6	88·1	38·4	52·0	6·7
Middling..........	85	1·04	27·2	20·7	28·8	7·7	18·4	18·4	14·5
Tailing............	282	0·17	19·5	10·0	47·4	4·2	43·2	29·6	78·8

This test indicates a copper recovery of 93·3 per cent on retreatment of the middlings, with a concentration ratio of 3·54 to 1.

Test No. 916:

Amount tested: 500 grams.
Apparatus used: Janney test machine.
Reagents used: ¾ pound per ton hardwood creosotes and tars; 8 pounds per ton of lime.

Product	Weight grams	Analysis per cent				Per cent of metals in products			
		Cu.	Fe.	S.	Insol.	Cu.	Fe.	S.	Insol.
Heads.............	500	2·30	25·3	19·0	34·8	100·0	100·0	100·0	100·0
Concentrate........	69	12·66	27·0	23·8	17·4	78·3	14·8	17·4	7·1
Middling..........	53	2·93	24·7	18·3	27·2	13·9	10·4	10·3	8·5
Tailing............	378	0·23	24·8	18·0	38·0	7·8	74·8	72·3	84·4

This test indicates a copper recovery of 91·5 per cent on re-treatment of the middling, with a concentration ratio of 6·02 to 1.

These tests show a very satisfactory separation of the copper minerals from the iron pyrites and the gangue, and that this ore can be successfully treated by the flotation process, by the use of oils, with the addition of lime, yielding a good recovery and an economical ratio of concentration.

SMALL SCALE LABORATORY TESTS USING THE REX REAGENTS

A series of small scale tests were made on the laboratory type flotation machines, using the Rex flotation reagents. The results of these tests are given in the following table:—

Test No.	Heads per cent Cu.	Concentrates per cent Cu.	Middlings per cent Cu.	Tails per cent Cu.	Concentration ratio	Recovery
27..................	1·97	9·15	....	0·27	1 : 5·5	88·6
28..................	1·97	17·30	....	1·15	1 : 20·0	44·0
29..................	1·97	13·62	....	0·58	1 : 8·8	74·0
30..................	1·97	11·67	....	0·57	1 : 7·7	75·4
31..................	1·97	10·45	....	0·35	1 : 6·1	85·4
32..................	1·97	15·80	....	0·72	1 : 11·9	66·7
33..................	1·97	12·30	5·00	0·43	1 : 8·3	79·0
34..................	1·97	14·45	7·75	0·75	1 : 12·6	62·9
37..................	1·97	11·10 7·44	1·96	0·25	1 : 6·1	87·7
45..................	1·75	14·30	5·75	1·20	1 : 28·8	31·9
47..................	1·75	10·05	....	0·52	1 : 7·9	73·8
49..................	1·75	17·20	....	0·90	1 : 19·6	50·7
58..................	0·90 (tails No.49)	6·76	1·89	0·40	1 : 15·8	53·6 77·1 =total rec'y No. 49 & 58.
70..................	1·08	7·95	2·15	0·45	1 : 13·4	59·7
71..................	1·08	13·00	1·50	0·15	1 : 19·4	84·6
72..................	1·08	7·55	1·45	0·25	1 : 9·1	77·9
2*a*..................	2·25	7·20	....	0·40	1 : 3·7	87·0
3*a*..................	2·25	7·60	....	0·42	1 : 4·2	87·6
4*a*..................	2·25	10·63	3·10	0·70	1 : 6·4	73·5
5*a*..................	2 25	10·35	2·50	0·25	1 : 5·1	90·7
6*a*..................	2·25	8·55	1·15	0·15	1 : 4·0	95·0
8*a*..................	2·25	13·50	2·60	0·12	1 : 6·2	95·5
14*a*..................	2 46	18·15	1·28	0·31	1 : 6·2	88·4
15*a*..................	2·25	15·06 19·64	2·97	0·39	1 : 11·4	78·9

The above tests were run under the following conditions, except where differences are noted:—

Janney laboratory flotation machine used, 500 grams of Anyox ore ground to 100 mesh, mixed in pebble mill 5 minutes, agitated in machine 5 minutes, and floated for 8 minutes.

Differences in procedure in running tests :—

Test No.	——
30	No mixing.
31	200 mesh.
32	200 mesh.
33 34	Floated 8 minutes for concentrates and 8 minutes more for middlings.
37	Ran two 500-gram charges, floating 15 minutes; re-ran combined concentrates, agitating 5 minutes and floating 8 minutes for first concentrates and 8 minutes more for second concentrates.
45	Ore, −60. Floated 8 minutes for concentrates and 5 minutes more for middlings.
47	Ruth machine. Three charges of 1,000 grams, −60 each. Floated 5 minutes—products mixed to form one concentrate and one tailing.
49	Ore, −60. Three charges of 500 grams run—products mixed to form one concentrate and one tailing.
58	500 grams tailings test No. 49, −100. Floated three minutes for concentrates and three minutes more for middlings.
70	Callow machine. 500 grams ore, −10, ground 2 hours in ball mill, floated for 13 minutes.
71	Callow machine. 500 grams ore, −10, ground 1 hour in ball mill, floated for 12 minutes.
72	Callow machine. 500 grams ore, −10, ground one-half hour in ball mill, floated for 12 minutes.
2*a*	No mixing, agitated 3 minutes, floated 20 minutes.
3*a*	“ “ 3 “ 20 “ 200 mesh.
4*a*	“ “ 3 “ 15 “ 200 “
5*a*	“ “ “ 20 “ 200 “
6*a*	Size of ore not stated, no mixing, agitated, floated for 20 minutes.
8*a*	Size of ore not stated, no mixing, agitated 3 minutes, floated for 15 minutes.
14*a*	No mixing, agitated 3 minutes, floated 5 minutes for 1st concentrate, then floated for rougher concentrate. Rougher concentrate re-cleaned giving 2nd concentrate and middling.
15*a*	Ore 250 mesh, mixed for 15 minutes, times of agitating and floating not stated, rougher concentrate made and this re-cleaned, giving concentrate and middling.

Small Scale Laboratory Tests.

Test	HEADS			CONCENTRATE			MIDDLING			TAILING			Total Extn. 75% recvy. mids.	Reagents added to ball mill				Reagents added to machines							Machine used	Pulp acid or alkaline
	Charge grams	Anal. % Cu	Mesh	Wt. gms.	Anal. % Cu.	Recvy. %	Wt. gms.	Anal. % Cu.	% of Metal	Wt. gms.	Anal. % Cu.	Loss %		Lime	$NaCO_3$	NH_4OH	Neutral Alkaline	Lime	$NaCO_3$	NH_4OH	X.Y.	Rex.	Other Oils	Other reagents		
11	990	2·20	−100	278	7·15	91·1	121	0·91	5·17	591	0·14	3·80	....	0	0	0	N	0	0	0	1·0	....	....	....	R	N
15	1,000	2·20	−100	197	9·26	81·4	105	2·31	10·8	698	0·25	7·80	....	½	..	....	N	..	..	..	0·5	....	....	....	R	N
14	1,000	2·20	−100	120	13·5	73·4	160	2·8	20·3	718	0·14	6·30	87·6	1	..	....	N	..	..	..	0·5	....	....	....	R	N
12	992	2·22	−100	249	8·13	91·0	125	1·13	5·10	618	0·10	2·80	....	2	..	....	N	..	..	..	1·0	....	....	....	R	N
16	1,000	2·25	−100	87	21·5	83·4	68	3·08	9·06	855	0·20	7·60	....	4	..	....	A	..	..	..	0·5	....	....	....	R	A
17	1,000	2·20	−100	104	19·4	87·7	59	2·40	6·10	837	0·17	6·2	....	4	..	....	A	..	..	..	1·0	....	....	....	R	A
19	1,000	2·20	−100	212	8·8	85·0	84	2·56	9·4	708	0·18	5·6	....	.	4	....	A	..	..	..	1·0	....	....	....	R	A
20	1,000	2·20	−100	85	20·8	81·0	69	3·44	10·9	849	0·22	8·2	....	1	..	....	N	3	..	..	1·0	....	....	....	R	A
21	1,000	2·67	−100	42	15·78	24·8	104	3·45	18·1	816	1·88	57·1	....	4	..	....	A	..	..	..	....	1·0	....	....	R	A
22	1,000	3·0	−100	46	14·21	21·6	100	9·72	32·0	854	1·65	46·4	....	1	..	....	N	..	..	..	....	1·0	....	....	R	N
24	1,000	2·6	−100	92	7·92	27·9	207	2·87	22·8	693	1·84	49·3	....	2	..	....	N	..	..	..	....	1·0	....	....	R	N
25	1,000	2·4	−100	274	7·83	89·0	173	0·70	5·0	542	0·27	6·0	....	..	4	....	A	..	..	..	....	1·0	....	....	R	A
26	1,000	2·6	−100	196	11·0	83·8	195	1·55	11·6	607	0·20	4·6	....	..	2	1·0	A	..	..	..	....	1·0	....	....	R	A
27	1,000	2·7	−100	315	7·70	90·8	178	0·71	4·6	508	0·26	4·6	....	..	2	0·5	N	..	2	..	....	0·5	....	....	R	A
28	1,000	2·7	−100	340	7·60	93·0	81	0·70	2·0	573	0·18	4·76	....	..	2	0·5	N	..	2	..	0·5	....	5c.c. Pine oil.		R	A
13	1,000	2·2	−100	239	7·93	84·2	110	1·60	......	650	0·30	......	....	..	..	....	N	..	..	..	....	....	1·0 Fuel oil.		R	N
29	1,000	2·2	−100	192	12·5	88·6	113	1·68	7·0	704	0·17	4·4	....	4	..	....	A	..	..	..	....	 1	·25 P.O. ·50F.O.		R	A
30	1,000	2·7	......	180	13·68	90·5	55	1·87	1·03	764	0·20	1·53	....	4	..	....	A	..	..	..	1·0	....	....	....	R	A
31	1,000	2·7	......	223	11·0	89·6	94	1·04	0·97	684	0·28	1·91	....	..	2	1	A	..	..	..	....	1·0	....	....	R	A
32	1,000	2·65	......	243	8·42	76·85	269	1·88	5·06	472	0·20	0·94	....	..	2	1	A	..	..	..	....	1·0	....	....	R	A
34	1,000	2·62	......	252	9·70	93·1	93	0·58	2·2	645	0·18	4·4	....	..	..	....	A	4	..	..	1·0	....	....	....	C	A
35	1,000	2·76	......	595	8·27	91·6	154	2·22	6·3	1,206	0·09	2·0	....	..	..	....	A	4	..	..	1·0	....	....	....	C	A
36	1,000	......	......	208	5·2	....	168	1·5	......	597	0·30	......	....	4	..	....	A	..	..	..	....	....	1·0 ketone residue.		R	A
37	1,000	......	......	112	11·5	86·4	47	1·8	5·0	821	0·15	8·05	....	4	..	....	A	..	..	..	·75	....	....	....	R	A
39	1,000	1·60	......	152	4·49	42·8	153	1·38	13·2	683	1·02	44·0	....		2	1	A	..	..	..	....	1·0	....	....	R	A
38	....	......	......	299	4·6	85·6	....	......	......	......	......	......	....	..	..	....	....	..	..	..	....	....	....	....	....	....

Test No. 38:

Amount tested: 1,000 grams.
Apparatus used: Ruth test machine.
Degree of crushing: 60 mesh.
Reagents used: 2·5 c.c. Rex reagent dissolved in carbon bi-sulphide.
Results: The first concentrate came up in a very dirty looking and sticky froth. The addition of soda ash with the addition of more Rex reagent improved the results.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate No. 1	107	7·20	7·70	48·1
Concentrate No. 2	192	3·12	5·99	37·4
Tailing	701	0·33	2·31	14·5
Heads	1,000	1·60	16·00	100·0

Test No. 40:

Amount tested: 1,000 grams.
Apparatus used: Ruth test machine.
Degree of crushing: 100 mesh.
Reagents used: 8 pounds lime per ton added to ball mill; a few c.c. of the acid soluble portion of resin pitch in H_2SO_4 was added to cell.
Results: A froth consisting of watery looking bubbles heavy with copper mineral formed.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	138	9·12	12·58	81·5
Middling	154	1·25	1·93	12·5
Tailing	708	0·13	0·92	6·0
Heads	1,000	1·54	15·43	100·0

Test No. 41:

Amount tested: 1,000 grams.
Apparatus used: Ruth machine.
Degree of crushing: 60 mesh.
Reagents used: The insoluble portion of resin pitch after dissolving in caustic soda was taken up with ammonia.
Results: None.

Test No. 42:

Amount tested: 1,000 grams.

Apparatus used: Ruth test machine.

Degree of crushing: 60 mesh.

Reagents used: 1 pound per ton of NH_4OH and 4 pounds per ton of soda ash added to ball mill; resin pitch dissolved in H_2SO_4 and the acid soluble portion was distilled to remove any oil.

Results: Good, but it required a lot of this reagent to form a stable froth.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	164	6·22	10·20	66·4
Middling	145	2·85	4·13	26·9
Tailing	691	0·15	1·04	6·7
Heads	1,000	1·54	15·37	100·0

Test No. 43:

Amount tested: 1,000 grams.

Apparatus used: Ruth test machine.

Degree of crushing: 60 mesh.

Reagents used: 1 pound per ton of NH_4OH and 4 pounds per ton of soda ash added to ball mill; 1 pound per ton of double strength solution of Rex reagent from which the acid soluble portion was extracted.

Result: The action was similar to Rex reagent.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	156	8·05	12·56	78·3
Middling	171	1·45	2·48	15·4
Tailing	673	0·15	1·01	6·3
Heads	1,000	1·60	16·05	100·0

Test No. 44:

Amount tested: 1,000 grams.

Apparatus used: Ruth test machine.

Degree of crushing: 60 mesh.

Reagents used: 1 pound per ton NH_4OH and 4 pounds per ton soda ash added to ball mill; Rex reagent No. 12 was separated with ether and alcohol, the water soluble portion of this was used.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrat	138	8·00	11·04	72·0
Middling	165	1·63	2·69	17·6
Tailing	697	0·23	1·60	10·4
Heads	1,000	1·54	15·33	100·0

Test No. 45:

Amount tested: 1,000 grams.
Apparatus used: Ruth test machine.
Degree of crushing: 60 mesh.
Reagents used: Same as test No. 44, but the ether soluble portion of reagent used.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	88	12·77	11·24	74·8
Middling	84	3·03	2·55	17·0
Tailing	828	0·15	1·24	8·2
Heads	1,000	1·50	15·03	100·0

COMPARATIVE SMALL SCALE TESTS WITH ALPHABETICAL, REX, AND OIL REAGENTS

Three tests were made on the Callow pneumatic testing machine, in which two lots of 1000 grams each, ground to 60 mesh, were used. The machine was operated on a continuous feed, and continuous discharge, as in practice. The concentrate was recleaned on the Ruth testing machine.

Test No. 47:

Reagents used: 6 pounds lime per ton; 1 pound X-Y reagent per ton.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	390	6·45	25·16	80·9
Middling	150	1·90	2·85	9·2
Tailing	1,404	0·10	1·40	4·5
Clean up	56	0·30	1·68	5·4
Heads	2,000	1·55	31·09	100·0

Test No. 48:

Reagents used: Rex No. 12 reagent with soda ash and NH_4OH.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	158	5·55	8·77	28·2
Middling	114	3·82	4·35	14·0
Tailing	1,656	0·97	16·06	51·6
Clean up	72	2·67	1·92	6·2
Heads	2,000	1·56	31·10	100·0

Test No. 49:

Reagents used: Fuel oil, 1 pound per ton; pine oil and 4 pounds of lime per ton.

Product	Weight grams	Analysis per cent Cu.	Content pounds Cu.	Percentage of metal
Concentrate	258	9·15	23·61	81·1
Middling	52	2·90	1·51	5·2
Tailing	1,581	0·12	1·90	6·5
Clean up	109	1·93	2·10	7·2
Heads	2,000	1·46	29·12	100·0

LARGE SCALE TESTS USING REX FLOTATION REAGENTS

A number of large scale tests were made, using about one ton of the ore for each test, as follows:—

Test No. 1:

Amount tested: 1,932·5 pounds.
Apparatus used: Ruth four cell flotation machine (standard size).
Degree of crushing: 60 mesh.
Dilution: High.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	36·0	15·10	5·44	18·7
" No. 2	57·0	11·90	6·78	23·3
" No. 3	42·0	10·55	4·43	15·3
" No. 4	54·0	8·45	4·56	15·7
Tailing	1,743·5	0·45	7·85	27·0
Heads	1,932·5	1·55	29·95	100·0

Recovery 73·0 per cent. Ratio of concentration 10·2 to 1.

Test No. 2:

Amount tested: 1,915 pounds.
Apparatus used: Ruth four cell flotation machine (standard size).
Degree of crushing: 80 mesh.
Dilution: High.
Concentrate taken off first three cells: middling off fourth, and returned.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	44·0	17·85	7·85	26·9
" No. 2	46·0	15·70	7·22	24·7
" No. 3	33·0	12·90	4·26	14·6
Tailing	1,792·0	0·55	9·86	33·8
Heads	1,915·0	1·70	32·55	100·0

Recovery 66·2 per cent. Ratio of concentration 15·6 to 1.

Test No. 5:

Amount tested: 1,941·5 pounds.
Apparatus used: Ruth four cell flotation machine (standard size).
Degree of crushing: 80 mesh.
Dilution: High.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	46·5	13·05	6·07	20·9
" No. 2	61·0	11·80	7·20	24·8
" No. 3	39·0	9·25	3·61	12·4
" No. 4	49·0	6·95	3·41	11·2
Tailing	1,746·0	0·50	8·73	30·1
Heads	1,941·5	1·60	31·06	100·0

Recovery 69·9 per cent. Ratio of concentration 9·93 to 1.

Re-run of tailing from test No. 5:

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	0·50	4·50	0·02	0·1
" No. 2	0·75	5·30	0·04	0·1
" No. 3	1·50	7·00	0·10	0·3
" No. 4	1·25	7·20	0·09	0·3
Tailing	1,725·75	0·50	8·65	29·3
Feed	1,729·75	0·50	8·65	30·1

Total recovery 70·7 per cent. Ratio of concentration 9·73 to 1.

Test No. 6:

Amount used: 1,947·5 pounds.
Apparatus used: Ruth four cell flotation machine.
Degree of crushing: 80 mesh.
Dilution: High.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	38·0	16·90	6·42	21·7
" No. 2	43·0	14·00	6·02	20·4
" No. 3	31·5	11·40	3·59	12·1
" No. 4	31·0	8·75	2·71	9·2
Tailing	1,804·0	0·60	10·82	36·6
Heads	1,947·5	1·55	30·19	100·0

Recovery 63·4 per cent. Ratio of concentration 13·6 to 1.

Test No. 8:

Amount tested: 1,834·5 pounds.
Apparatus used: Ruth four cell flotation machine.
Degree of crushing: 80 mesh.
Dilution: one of solids to six of water.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	113·5	10·25	11·63	43·9
" No. 2	96·5	5·55	5·36	20·3
" No. 3	90·0	4·60	4·14	15·6
" No. 4	62·5	4·05	2·53	9·6
Tailing	1,472·0	0·19	2·80	10·6
Heads	1,834·5	1·65	30·27	100·0

Recovery 89·4 per cent. Ratio of concentration 5·1 to 1.

Test No. 9:

Amount tested: 1,750·5 pounds.
Apparatus used: Ruth four cell standard machine.
Degree of crushing: 80 mesh.
Dilution: One of solids to six of water.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	47·5	13·95	6·63	27·4
" No. 2	54·0	9·90	5·35	22·1
" No. 3	41·0	6·75	2·77	11·5
" No. 4	89·5	6·15	5·50	22·7
Tailing	1,518·5	0·26	3·95	16·3
Heads	1,750·5	1·55	27·13	100·0

Recovery 83·7 per cent. Ratio of concentration 7·5 to 1.

Test No. 10:

Amount tested: 1,591 pounds.
Apparatus used: Ruth four cell standard machine.
Degree of crushing: 80 mesh.
Dilution: One of solids to five of water.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	49·5	12·07	5·97	24·2
" No. 2	35·0	14·70	5·15	20·8
" No. 3	43·0	10·38	4·46	18·0
" No. 4	47·0	10·72	5·04	20·4
Tailing	1,416·5	0·29	4·11	16·6
Heads	1,591·0	1·35	21·48	100·0

Recovery 83·4 per cent. Ratio of concentration 9·1 to 1.

Test No. 33:

Amount tested: 1,696 pounds.
Apparatus used: Ruth four cell standard machine.
Degree of crushing: 80 mesh.
Dilution: High.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	84·0	10·52	8·84	33·1
" No. 2	61·5	9·35	5·75	21·6
" No. 3	54·0	7·27	3·93	14·7
" No. 4	52·0	7·47	3·88	14·5
Middling	130·5	1·37	1·79	6·7
Tailing	1,314·0	0·19	2·50	9·4
Heads	1,696·0	1·60	27·14	100·0

Recovery 83·9 per cent. Ratio of concentration 6·75 to 1.

Test No. 46:

Amount tested: 1,878 pounds.
Apparatus used: Ruth four cell standard machine.
Degree of crushing: 80 mesh.
Dilution: One of solids to fourteen of water.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate No. 1	53·5	8·62	4·61	17·6
" No. 2	39·5	10·17	4·02	15·4
" No. 3	22·5	10·60	2·38	9·1
" No. 4	59·5	9·10	5·41	20·7
Tailing	1,703·0	0·57	9·71	37·2
Heads	1,878·0	1·65	30·99	100·0

Recovery 62·8 per cent. Ratio of concentration 10·7 to 1.

LARGE SCALE TESTS USING X-Y REAGENTS

Two tests were conducted, one on the Ruth machine, and one on the Callow, on about one ton of ore using the Alphabetical reagent, X-Y mixture.

Test No. 18:

Amount tested: 1,652·5 pounds.
Apparatus used: Ruth four cell standard machine.
Degree of crushing: 80 mesh.
Dilution: One of solids to six of water.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate	225·0	6·70	15·07	68·9
Middling	86·5	4·76	4·12	18·8
Tailing	1,341·0	0·20	2·68	12·3
Heads	1,652·5	1·72	28·42	100·0

Recovery 80·2 per cent, assuming 60 per cent recovery of values in middling.
Ratio of concentration 6 to 1.

Test No. 23:

Amount tested: 1,661·5 pounds.
Apparatus used: Callow standard machine.
Degree of crushing: 80 mesh.
Dilution: One of solids to five of water.

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate	157·5	8·50	13·39	61·4
Middling	91·0	4·90	4·46	20·5
Tailing	1,413·0	0·28	3·96	18·1
Heads	1,661·5	1·49	24·76	100·0

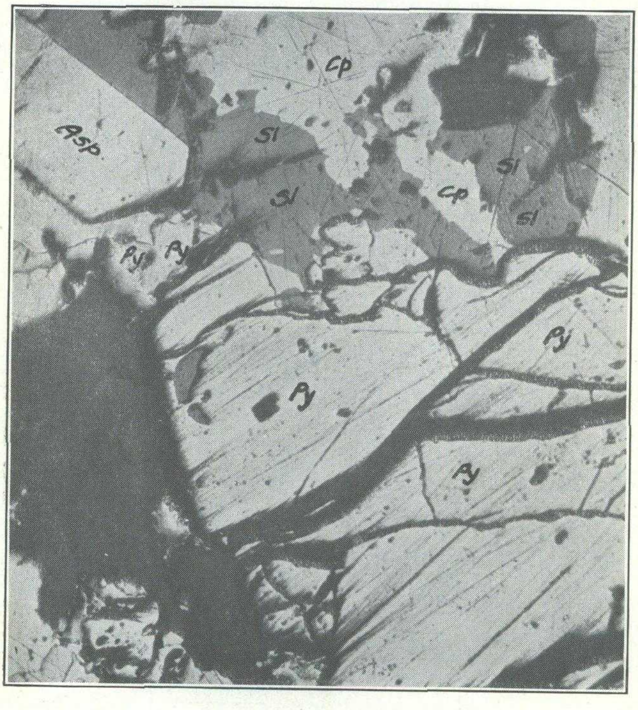
Recovery 73·7 per cent, assuming 60 per cent recovery of values in middling.
Ratio of concentration 8 to 1.

PLATE I

Hidden Creek ore, x 100

200 mesh

Face page 98.



Hidden Creek ore, x 100
Py = pyrite, Cp = chalcopyrite
Sl = sphalerite, Asp = arsenopyrite

200 mesh

PLATE III

Thin section of Hidden Creek ore, x 100
Black is sulphide


200 mesh

Large Scale Test using Fuel Oil, Pine Oil, and Lime

Test No. 52:

Two tons of ore were ground in the ball mill to the following degree of fineness:—

+100 mesh	16·1 per cent
−100+150	11·9 "
−150+200	34·2 "
−200	37·8 "

TEST MADE ON CALLOW STANDARD MACHINE

Product	Weight pounds	Analysis per cent Cu.	Content pounds Cu.	Per cent of Cu. values
Concentrate	217·0	15·75	34·18	71·2
Tailing	3,453·0	0·40	13·81	28·8
Heads	3,770·0	1·35	51·00	100·0

Recovery 71·2 per cent. Ratio of concentration 17·4 to 1.

SUMMARY AND CONCLUSIONS

The experimental work conducted on this ore shows that it is amenable to concentration by flotation methods. A very satisfactory separation of the copper minerals from the iron pyrite and gangue can be obtained, with a good recovery of the copper values, and an economical ratio of concentration.

The small scale tests show that such reagents as oil, with the addition of lime, Rex reagents, or the Alphabetical reagents give good results. However, in the tests run for comparative results under the same conditions and procedure, oil and the Alphabetical reagents give much better results than the Rex reagents.

While the small scale tests give good results with the use of Rex reagents, these reagents have never been applied in practice on a large scale, and until this is done, the feasibility of their use cannot be considered as practicable.

The results from the large scale experimental work are not as satisfactory as those from the smaller scale tests. This is due to the fineness of crushing. In these tests the ore was reduced to 80 mesh, while finer grinding is necessary for better recoveries and grade of concentrates. Since these tests were run a Dorr classifier with bowl attachment has been installed in closed circuit with the ball mill. With this arrangement any degree of crushing can be obtained to 200 mesh, and a number of tests will be conducted on a large scale, and this report supplemented with the results of these tests.

A number of tests will also be made with the use of other reagents, and the results of this work will be contained in the supplement to this report.

(2)

THE FLIN FLON COMPLEX SULPHIDE ORE

C. S. Parsons and R. J. Traill

Test No. 142

Location.—The Flin Flon ore-body is situated at the southeast end of Flin Flon lake, which drains through a series of lakes into the Saskatchewan river. The ore-body lies in township 67, ranges XXIX and XXX, west, approximate latitude 54° 45′ and approximate longitude 101° 55′—four miles east of the second meridian, west. It lies practically on the boundary of the provinces of Manitoba and Saskatchewan, in the southwest corner formed by the right angle connection of this boundary line. It is 68 miles by air line NNW. from The Pas, on the Canadian National railway, the distributing centre for the district.

Extent of Ore-body.—The ore-body which skirts the southeast shore of the lake strikes with the country rock, and dips approximately 70° east. From the records of the diamond drill holes at the south end of the deposit, it would appear that it pitches at a low angle to the south. Horses of unmineralized greenstone separate it into more or less independent lenses. The deposit has a known length of 2,600 feet, and has been proven to a depth of 900 feet, over a length of 1,000 feet. Its greatest width is 400 feet. From the results of diamond drilling and underground development work, the total tonnage has been calculated to be sixteen million, exclusive of horses of greenstone. This estimate makes no allowance for possible ore below the 900 foot level, or at depth in the line of pitch, at the south end of the ore-body.

Mineralogical Composition.—The chief metallic minerals are pyrite, sphalerite, and chalcopyrite. Galena has been found in vugs in the otherwise unmineralized rock, but does not occur in quantity in the ore-body. Small amounts of pyrrhotite and arsenopyrite, are present, and the analysis of the ore shows the presence of traces of bismuth, which would be present as bismuthinite. Native copper is found in leaf form as a secondary product in the upper sulphide zone. Gold and silver are present. It has been determined that the precious metals are evenly distributed through all the sulphides. This has been demonstrated by the analysis of products from the differential flotation tests on the sulphide ore. The principal gangue is some mineral of the chlorite group.

Characteristics of the Ore-body.—The ore-body consists of three zones and two types of ore—a hanging wall zone of disseminated ore, a centre zone of solid sulphides, and a footwall zone of disseminated ore. In some places the sulphide zone is in direct contact with the hanging wall, but as a rule, a selvage of disseminated ore separates it from the hanging wall. On the footwall there is found invariably a considerable width of disseminated ore. Contact with the walls, and as a general rule between the two types of ore, is well defined. The richest copper ore is found in the disseminated footwall zone, which gives values of from three to five per

PLATE IV

Flin Flon ore, x 100

200 mesh

Face page 100.

PLATE V

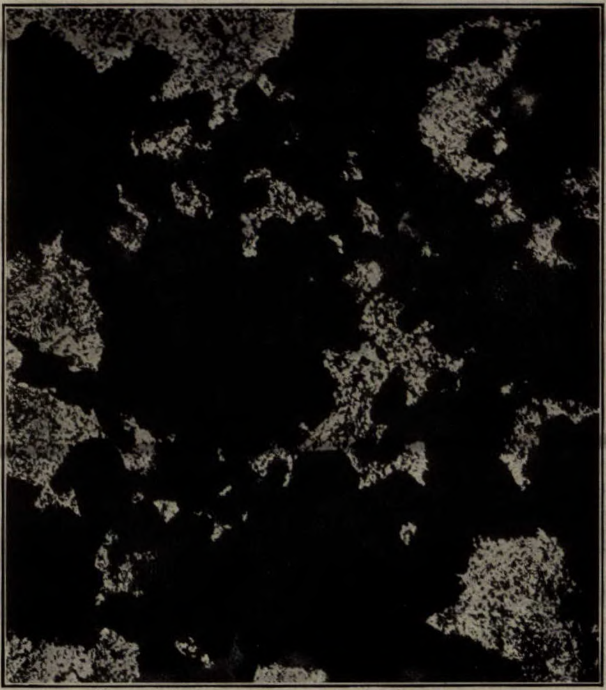
Flin Flon ore, x 100

200 mesh

Flin Flon ore, x 100
Py = pyrite, Cp = chalcopyrite
Sl = sphalerite

□ 200 mesh

PLATE VII



Thin section of massive sulphide ore like that of Plates IV, V, and VI, x 100
Black is sulphide, light is carbonate

200 mesh

cent copper. The sulphide ore carries about 1·65 per cent copper, the average for the whole ore-body being approximately 1·9 per cent. Zinc, which averages about 3·8 per cent for the whole ore-body, shows higher values in the sulphide ore.

Exploration and Development.—The ore-body was discovered in the summer of 1915. During the winter of 1915-16, work was confined to surface sampling and cross trenching. During 1916, 6,000 feet of drilling was done by New York and Boston interests. During 1917 and 1918 forty-four holes were drilled by Toronto interests, representing a total lineal footage of 25,664 feet. In March, 1920, an option was taken by New York and Canadian interests, and since that date two shafts were sunk over 500 feet apart. One shaft reached the 200 foot level where 210 feet of cross-cutting was done to traverse the ore-body, and approximately 100 feet of drift was driven southward. The No. 2 shaft was sunk to the 100-foot level and a cross-cut approximately 165 feet long driven at this level.

Shipments Received for Experimental Test Purposes.—In June, 1920, application was made by the Mining Corporation of Canada, who hold the option on the property, to the Department of Mines for assistance in solving the metallurgical problems in connection with the treatment of the sulphide ore. Later this was extended to cover the disseminated ore. On June 15, 1920, there was received at the Ore Dressing and Metallurgical Laboratories, a shipment of 100 pounds of sulphide ore, and on April 22, 1921, a further shipment of 129 pounds of similar ore was received. Both these shipments were representative, being portions of the drill cores, crushed to about one-half inch, from diamond drilling operations. A shipment of the disseminated ore has been received, the experimental work on this ore will be conducted during 1922.

Possible Lines of Investigation for the Metallurgical Treatment of the Sulphide Ore.—On account of the nature of this ore, which consists of practically solid sulphides of iron, zinc, and copper, the gangue minerals being less than ten per cent, it presents a most intricate metallurgical problem to determine the most economical method of treatment. Three of the most promising lines of investigation were decided on as follows:—

1. The concentration of the copper mineral by selective flotation, and the separation by differential flotation of the chalcopyrite, sphalerite, and pyrite, resulting in copper and zinc products, suitable for subsequent reduction to the metals.

2. The hydro-metallurgical treatment of the ore, or of concentration products, by roasting, leaching, and electrical deposition of the metals.

3. Pyritic smelting followed by reduction of the matte in converters to blister copper.

The experimental work conducted in the Ore Dressing and Metallurgical Laboratories was confined to the first line of investigation. A few small laboratory tests were made on the roasting and leaching of the ore, but the laboratories were not equipped to carry out extensive tests from which reliable conclusions could be drawn. The third is a matter for study of conditions at metallurgical plants operating on a similar class of ore, and will require the installation of a small unit at the property to demonstrate the feasibility of this method of treatment.

Microscopic Examination of the Sulphide Ore.—A number of polished sections were made and these examined under the microscope to determine the special characteristics of the minerals and their relation to each other. The conclusions arrived at were practically the same as those of Prof. Alfred Wandke, of Cambridge, Mass., who examined three type specimens of the ore. Prof. Wandke's findings were as follows:—

> The gangue is essentially carbonate, either calcite or magnesium, or iron bearing, with very minor amounts of sericite, quartz, and rutile. It is but a small proportion of the ore, and being a carbonate, would not interfere with the fine grinding of the rock. The quartz is sparely developed and moulds about the pyrite, sometimes a little chlorite occurs on the edges of some of the pyrite crystals, and chalcopyrite seems intimate with the chlorite.
>
> The minerals present are pyrite, sphalerite, chalcopyrite, arsenopyrite, a very little bornite, and chalcocite, and a few specks resembling galena. The pyrite occurs both as granular aggregate and as distinct crystals measuring up to ·2 mm. in section. Pyrite probably encloses both sphalerite and chalcopyrite. The chalcopyrite measures frequently under ·02 mm. (less than 400 mesh). The sphalerite cements the pyrite, often occurs in bands, and although usually free from chalcopyrite, it may nevertheless enclose minute ·1 mm. (150 mesh) grains of chalcopyrite. The chalcopyrite is the chief copper mineral, the only one present, for the bornite and chalcocite occur in insignificant amounts. The mineral cements and is intergrown with grains of pyrite and sphalerite. While the grains of chalcopyrite may attain a diameter of ·5 mm. in section, it rarely runs above ·2 mm. (65 mesh) and as a rule the grains measure less than ·1 mm. (150 mesh). Hence very fine grinding (to slime) would be necessary to liberate the chalcopyrite. Sometimes the hand specimens and the polished section show tarnishing, suggesting surface enriching solutions, but the enrichment is always sub-microscopic. The arsenopyrite occurs in clean-cut crystals enclosed by pyrite. It is almost always surrounded by pyrite and makes up but a fraction of one per cent of the ore.

He made a special report on the occurrence of the gold, but found no mineral which would give a clue as to its association. Since it is possible when the gold is coarse to detect the native metal in ore that runs less than $2 per ton, it seems certain that in the Flin Flon ore, the gold is extremely finely divided and so avoids detection. He states that from evidence furnished by other districts, it is suggested that the gold may go with the sphalerite and chalcopyrite. This does not seem to be the case, however, as the results of the differential flotation tests show that the gold is evenly distributed in the ore, occurring with all three minerals, the pyrite, sphalerite, and chalcopyrite.

He also mentions that the extreme fineness of the grains of chalcopyrite have indicated to him that the flotation process might not be advisable. However, the results of the flotation tests show that on grinding to 200 mesh, to which fineness the ore reduces readily, a high grade copper concentrate is obtained, with good recoveries, and a zinc product is made, with a recovery of from 55 per cent to 60 per cent of the zinc. As far as the copper mineral is concerned, the ore is amenable to the flotation process.

It was also determined that the gangue mineral was principally carbonate, and that magnesia was greatly in excess of calcium. It is possible that the mineral which Prof. Wandke takes for galena, is bismuthinite, as bismuth has been detected in the electrolytic deposition of the copper.

The ore has been found to contain 0·5 per cent arsenic. This is roughly equivalent to 1 per cent arsenical pyrites.

Tests on the Concentration and Separation of the Minerals.—Small shipments of the ore were sent by the Mining Corporation of Canada to the General Engineering Company, Salt Lake City, Utah, to the company's Cobalt works, and to the Ore Dressing and Metallurgical Laboratories, Mines Branch, Department of Mines, for experimental work on the concentration and separation of the minerals.

The results of the tests conducted by the General Engineering Company are summarized below:—

Test No. 3:

Amount tested: 1,000 grams.
Apparatus used: Callow testing machine.
Degree of crushing: 200 mesh.
Object of test: Concentration of the copper values.
Reagents used: 11 pounds lime per ton; 1 pound X-Y mixture per ton.

Products	Analysis						Recov. Per cent	Conc. Ratio
	Cu. per cent	Zn. per cent	Fe. per cent	Insol. per cent	Au.-oz.	Ag.-oz.		
Copper concentrate........	13·70	5·3	33·0	5·2	....	....	86·6	10·6 : 1
Tailing....................	0·22	5·4	....	....	0·06	1·00	....	....

Test No. 4:

Amount tested: 1,000 grams.
Apparatus used: Callow testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals, producing a copper concentrate, a zinc concentrate, and an iron tailing.
Reagents used: 14 pounds lime per ton; 0·1 pound X-cake; 4 pounds Wattsons 1-B (equivalent to 0·6 pound oil); producing a copper concentrate.

Refloated, using 6 pounds H_2SO_4; 6·5 pounds residue Wattsons 1-B; 1 pound G. E. Co. No. 209; 0·1 pound G. E. Co. No. 56, producing a zinc concentrate.

Products	Analysis						Recov. Per cent	Conc. Ratio
	Cu. per cent	Zn. per cent	Fe. per cent	Insol. per cent	Au.-oz.	Ag.-oz.		
Copper concentrate........	8·00	5·6	33·8	5·4	0·26	4·50	83·1	6·2 : 1
Zinc concentrate...........	1·25	37·6	16·7	....	0·08	2·12	60·3	1·3 : 1
Tailing....................	0·22	1·3	....	....	0·07	0·90	....	....

Test No. 14:

Amount tested: 1,000 grams.
Apparatus used: Callow testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals, producing a copper concentrate, a zinc concentrate, and an iron tailing.
Reagents used: 14 pounds lime; 0·15 pound of X-cake; 6 pounds Wattsons 1-B (equivalent to 0·9 pound oil), producing a copper concentrate.
Refloated, using 1 pound of G. E. Co.'s No. 209, 0·75 pound Wattsons residue 1-B; 0·10 pound G. E. Co.'s No. 56, producing a zinc concentrate.

Products	Analysis						Recov. per cent	Conc. ratio
	Cu. per cent	Zn. per cent	Fe. per cent	Insol. per cent	Au.-oz.	Ag.-oz.		
Copper concentrate	6·70	7·8	32·8	5·6	0·22	3·90	83·4	5·3 : 1
Zinc concentrate	1·20	38·5	18·2	2·0	0·08	1·75	54·8	18·3 : 1
Tailing	0·25	0·24	....	....	0·07	0·80	....	....

The results of the tests conducted by the Ore Dressing and Metallurgical Division of the Mines Branch, Department of Mines, are summarized below—

Test No. 13:

Amount tested: 1,000 pounds.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals, producing a copper concentrate, a zinc concentrate, and an iron tailing.
Reagents used:
  Flotation of copper: 40 pounds lime per ton added to ball mill. 4 pounds X-Y per ton added to cells.
  Flotation of zinc: 14-B wood tar and Wattsons residue. This was unsuccessful.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Au.-oz.	Ag.-oz.	Cu.	Zn.	Au.	Ag.
Copper conc.	86	13·0	4·8	2·02		41·4	6·5	....	....
Copper midd.	152	6·3	5·0	0·12	1·72	35·5	12·0	....	....
Tailing	779	0·8	6·6	0·08	0·74	23·1	81·5	89·0	51·0

Test No. 14:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals producing a copper concentrate, a zinc concentrate, and an iron tailing.
Reagents used:
Flotation of copper: 40 pounds per ton lime added to ball mill; 5 pounds per ton X-Y reagent added to cell.
Flotation of zinc: 10 pounds per ton of soda ash.
1 pound per ton No. 34 gravity fuel oil.

Products	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Au.-oz.	Ag.-oz.	Cu.	Zn.	Au.	Ag.
Copper conc	146	14·2	4·4	0·10	1·38	78·5	10·2	....	....
" midd	159	2·3	5·9	0·14	1·27	13·8	14·9	....	....
Zinc conc	96	0·9	29·0	0·16	0·90	3·3	44·1	....	....
" midd	65	0·5	6·3	0·10	1·10	1·2	6·5	....	....
Tailing	568	0·15	2·7	....	....	3·2	24·3	....	....

Test No. 15:

Amount used: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals, producing a copper concentrate, a zinc concentrate, and an iron tailing.
Reagents used:
Flotation of copper: 40 pounds per ton lime added to ball mill; 5 pounds per ton X-Y reagent added to cell.
Flotation of zinc: Dewatered copper tailing; 8 pounds per ton soda ash to cell; 1 pound per ton No. 34 gravity fuel oil.

Product	Weight grams	Analysis			Per cent of values in heads			
		Cu. per cent	Zn. per cent	Au. and Ag.-ozs.	Cu.	Zn.	Au.	Ag.
Copper conc	161	12·5	2·7	1·54	77·9	7·7	....	....
" midd	110	1·4	7·0	0·90	6·0	13·6	....	....
Zinc conc	90	2·4	36·5	1·70	8·4	58·3	....	....
" midd	100	0·8	5·5	0·40	3·1	9·8	....	....
Tailing	542	0·22	1·1	0·71	4·6	10·6	....	....

Test No. 16:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals.
Reagents used:
Flotation of copper: 24 pounds per ton lime added to ball mill; 4 pounds per ton X-Y reagent to cell.
Flotation of zinc: Dewatered copper tailing: 1 pound per ton P.T.T. Co. No. 1580 to cell. This was unsuccessful.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Au.-oz.	Ag.-oz.	Cu.	Zn.	Au.	Ag.
Copper conc........	106	17·8	3·6	0·10	2·14	69·2	6·0	15·8	21·4
Copper midd.......	148	3·6	7·2	trace	2·00	19·6	16·8	4·2	28·0
Tailing............	766	0·4	6·4	0·07	0·70	11·2	77·2	80·0	50·6

Test No. 18:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Concentration of the copper mineral.
Reagents used: 30 pounds per ton lime to ball mill; 5 pounds per ton X-Y reagent to cell.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Fe. per cent	Au.-oz.	Ag.-oz.	Cu.	Zn.	Au.	Ag.
Copper conc........	158	14·9	28·7	0·12	....	86·0	....	23·7	....

Test No. 19:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals.
Reagents used: Flotation of copper:
  24 pounds per ton lime added to ball mill; 5 pounds per ton X-Y reagent to cell.
  Flotation of zinc:
    Dewatered copper tailing.
    8 pounds per ton soda ash added to cell.
    3 pounds per ton Wattsons residue 1-B.
    1 pound per ton copper sulphate.
    1 pound per ton No. 34 gravity fuel oil.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Fe. per cent	Au.-oz.	Cu.	Zn.	Fe.	Au.
Copper conc.......	145	16·7	4·4	24·3	0·10	87·94	10·56	....	18·2
" midd......	29	1·8	8·7	28·3	0·10	1·89	4·17	....	3·6
Zinc conc..........	78	0·45	40·7	17·9	0·11	1·27	52·52	....	10·8
" midd.........	74	0·5	8·9	33·7	0·07	1·34	10·90	....	6·6
Tailing...........	695	0·3	1·9	35·4	0·07	7·56	21·84	....	60·8

Test No. 20:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Concentration of the copper mineral.
Reagents used: 20 pounds per ton lime added to ball mill.
  3 pounds per ton X-Y mixture added to cell.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Fe. per cent	Au.-oz.	Cu.	Zn.	Fe.	Au.
Copper conc........	93	20·8	....	....	0·13	70·4	....	....	....
Copper midd.......	39	7·4	....	....	....	10·6	....	....	....

Test No. 21:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals.
Reagents used:
- Flotation of copper:
  - 16 pounds per ton lime added to ball mill.
  - 0·3 pound per ton X-Y reagent added to cell.
- Flotation of zinc:
  - Dewatered copper tailing.
  - 8 pounds soda ash per ton.
  - 1 pound fuel oil per ton.
- Flotation of zinc was not successful.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Au.-oz.	Ag.-oz.	Cu.	Zn.	Au.	Ag.
Copper conc........	138	17·4	3·2	0·08	2·64	87·4	7·37	13·8	....

Test No. 22:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals.
Reagents used:
- Flotation of copper:
  - 20 pounds per ton lime added to ball mill.
  - 0·3 pound per ton X-Y reagent to cells.
- Flotation of zinc:
  - Dewatered copper tailing.
  - 5 pounds per ton lime.
  - 8 pounds per ton soda ash.
  - 1 pound per ton fuel oil and pine oil.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Au.-oz.	Ag.-oz.	Cu.	Zn.	Au.	Ag.
Copper conc.......	129	15·7	2·7	0·12	2·58	73·7	....	....	....
" midd......	46	2·6	6·8	0·12	1·84	....	....	....	....
Zinc conc..........	61	....	44·5	....	....	....	45·2	....	....
" midd.........	116	0·58	6·7	0·08	1·18	....	....	....	....
Tailing............	....	....	2·8	....	....	....	....	....	....

Test No. 23:

Same procedure as in test No. 22.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Fe. per cent	Au.-oz.	Cu.	Zn.	Fe.	Au.
Copper conc.	130	18·0	3·7	26·4	0·16	85·0			
Zinc conc.	90		39·5	15·4	0·18		59·0		
Tailing	594	0·2	0·8	36·6	0·11				

Test No. 24:

Same procedure as in tests Nos. 22 and 23.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Fe. per cent	Au.-oz.	Cu.	Zn.	Fe.	Au.
Copper conc.	222	10·9	4·5	27·7	0·10	88·0	17·0		
Zinc conc.	61		42·5	14·2	0·20		44·0		
" midd.	150	0·6	8·6	31·7	0·14		21·9		
Tailing	590	0·1	1·7	37·1	0·06		17·0		

Test No. 32:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Differential flotation of the minerals.
Reagents used:
  Flotation of copper:
    20 pounds lime per ton to ball mill.
    0·3 pound X-Y reagent to cells.
  Flotation of zinc:
    Dewatered copper tailing.
    10 pounds per ton caustic soda.
    8 pounds per ton lime.
    1 pound per ton fuel oil.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Fe. per cent	Au.-oz.	Cu.	Zn.	Fe.	Au.
Copper conc.	200	8·80	2·4			67·8	7·9		
" midd.	96	2·35	4·3			8·7	6·8		
Zinc conc.	100	3·42	26·3			13·2	43·5		
" midd.	140	0·70	7·3			3·8	16·9		
Tailing	486	0·35	3·1			6·5	24·8		

Test No. 34:

Amount tested: 1,000 grams.
Apparatus used: Ruth testing machine.
Degree of crushing: 200 mesh.
Object of test: Preferential flotation of the minerals.
Reagents used:
  Flotation of copper:
    40 pounds per ton lime added to ball mill.
    0·2 pound X-Y reagent added to cells.
  Flotation of zinc:
    6 pounds per ton soda ash.
    1 pound per ton gravity fuel oil.

Product	Weight grams	Analysis				Per cent of values in heads			
		Cu. per cent	Zn. per cent	Fe. per cent	Au.-oz.	Cu.	Zn.	Fe.	Au.
Copper conc.	194	10·95	3·6	29·7	0·10	65·5	11·6	....	....
" midd.	89	7·75	4·9	31·6	0·08	21·2	7·2	....	....
Zinc conc.	93	2·30	35·0	21·2	0·15	6·6	54·1	....	....
" midd.	92	1·05	7·8	34·1	0·06	3·0	11·9	....	....
Tailing	608	0·20	1·5	....	....	3·7	15·2	....	....

Cyanidation of Flotation Tailing.—The tailing from test No. 34 was agitated in cyanide solution for the recovery of the gold and silver. The tailing after being treated in this manner gave traces of gold and silver, showing that the precious metal values could be recovered in this way. However, the cost of treatment would very probably prohibit the use of this method.

Microscopic Examination of Flotation Products.—Samples of flotation products from tests Nos. 15, 19, and 24, were sent to the Mining Corporation of Canada, who submitted same to Prof. Alfred Wandke, for microscopic examination. A copy of his discussion and conclusions is given below:—

DISCUSSION OF RESULTS.—The microscopic examination of the flotation products of Flin Flon ore was made on the following types of products:—

Copper concentrates, copper middlings, zinc concentrates, zinc middlings, and tails.

The examination was made with the following objectives in view:—

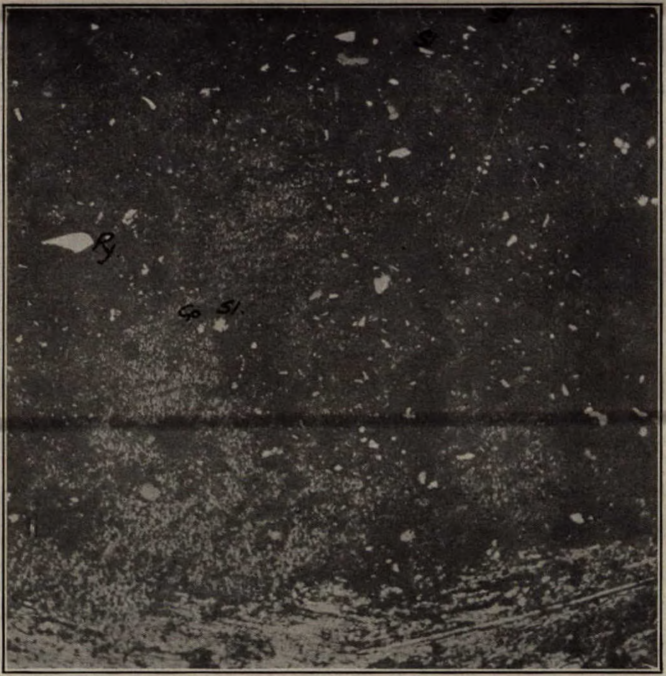
1. To study the degree of liberation of the various minerals of these products after being ground and floated.
2. To determine whether even finer grinding would be necessary to further liberate the various minerals, and so effect an even higher concentration of both copper and zinc.
3. To see whether any mineral was present in the various products that might account for the gold and silver content of the ore, and whether finer grinding might aid in recovering the precious metals.

Plate VIII

Copper concentrate from Flin Flon sulphide ore, x 100

200 mesh

PLATE IX



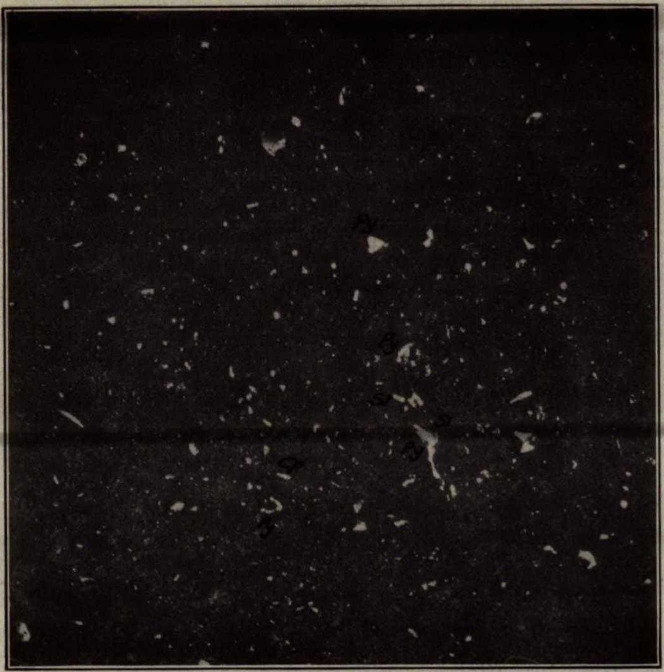
Copper middling from Flin Flon sulphide ore, x 100

200 mesh

PLATE X

Zinc middling from Flin Flon sulphide ore, x 100

☐ 200 mesh



Zinc concentrate from Flin Flon sulphide ore, x 100

200 mesh

PLATE XII

Tailing from Flin Flon sulphide ore, x 100
Py = pyrite, Cp = chalcopyrite, Sl = sphalerite

200 mesh

Degree of Liberation.—The examination showed that each of the three chief sulphide minerals of the ore—pyrite, chalcopyrite and sphalerite—occurs for all purposes practically free from any other sulphide or gangue minerals. In examining the copper concentrates fully 90 to 95 per cent of the particles present occur uncontaminated. The remaining 5 to 10 per cent occur intergrown with either pyrite and sphalerite in such fine division (minus 400 mesh) that further grinding would be impracticable.

The zinc concentrates showed the sphalerite to be practically entirely liberated, the copper and iron present being largely accounted for by grains of chalcopyrite and pyrite.

In the middlings from both the copper and zinc concentrating experiments the contaminating sulphides, whether chalcopyrite, sphalerite, or pyrite are present in free grains. The middling products thus show an imperfect separation. Regrinding of the middlings would be unnecessary to cause further liberation of the chalcopyrite or sphalerite as the case may be.

In the tails the chalcopyrite or sphalerite are both present largely as well liberated grains, only a minor portion of either chalcopyrite or sphalerite occurring locked up in the pyrite. Further grinding, it seems, would hardly liberate this locked up material for it rarely equals 400 mesh in size.

Would Finer Grinding Help the Degree of Liberation.—As stated above, the various sulphide minerals occur remarkably well liberated in the various products. Much of the material already ranges well below minus 200 mesh in size, and it would hardly seem feasible to even consider finer grinding because of the prohibitive cost.

Occurrence of the Gold and Silver.—Each of the products was scrutinized with great care to see if any clue could be obtained as to the occurrence of the gold and silver. No mineral was detected that would indicate the presence of gold and silver. It is concluded, therefore, that these precious minerals occur in a state of extreme subdivision and will probably defy concentration. They apparently occur associated with either the chalcopyrite, the sphalerite, or the pyrite. Thus the microscopic examination substantiates what the assays had already indicated regarding the occurrence of gold and silver in these ores.

CONCLUSIONS

1. The study of ore specimens had already indicated that this Flin Flon ore would have to be very finely ground in order to selectively concentrate either the chalcopyrite or the sphalerite. The flotation products show that the requisite degree of fine grinding was carried out for but a very minor amount of composite grains are present.

2. In the ore specimens no clue was found which would suggest the possible occurrence of the gold and silver. In the products the gold and silver still defy detection.

3. Since the gold and silver form necessary recoverable values in this ore, it seems hardly likely that any scheme of preferential flotation will permit the recovery of these precious metals. The reasons for this statement are as follows:—

(*a*) As judged from their appearance under the microscope the various sulphide grains seem to be at least 60 per cent through 200 mesh in size. This crushing has liberated the various sulphides to a remarkable degree. The composite grains show the enclosed sulphide particles to be well under 200 mesh in size, and hence of a size that wil defy liberation by grinding.

(*b*) The assays seem to indicate that the gold and silver occur more or less indifferently with either the chalcopyrite, the sphalerite, or the pyrite. Hence flotation merely separates the chief sulphide minerals of the ore and leaves the gold and silver almost equally disseminated in each.

4. It would appear, therefore, that the treatment of this ore must be accomplished either by direct smelting or by a roasting and leaching process.

ROASTING AND LEACHING TESTS

Roast No. 1

Roasting.—A sample of 100 grams of ore, 200 mesh, was roasted in an electric muffle furnace (multiple unit type) under the following conditions and with the following results:—

The 100 grams was spread over an area of 8″ × 3″, doors were kept closed during roast, except when charge was hand rabbled, and only inlet for air was ⅞″ circular hole in front door, and possibly to a slight extent, space around the door.

A suction was applied to rear door to draw off excess SO_2 gas, and also to create a slight in-draught of air to assist in slow oxidation of sulphur.

Temperature observations:—

10 a.m. (15 mins. after placing charge)	300° C.
10.30 a.m.	350° C.
11.00 a.m.	400° C.
11.30 a.m.	440° C.
12.00 noon	450° C.

This temperature maintained until 3 p.m., and then increased gradually until 4.30 p.m., when temperature reached 650° C.

Charge withdrawn at 4.30 p.m.

Time of roast, 6½ hours.

Charge showed a loss in weight of 16·5 per cent.

Temperature range, minimum 300°, maximum 650° C.

Rabbling was performed every half hour.

Leaching.—Two gram samples were leached at a point below boiling temperature for 1 hour. Head samples run as in standard analysis:

Results of coarse material:

	%		%
Head sample Cu	2·35		
Water soluble Cu	0·58	—extraction	24·7
H_2SO_4 soluble Cu	2·19	—extraction	93·2
Head sample Zn	6·75		
Water soluble Zn	4·36	—extraction	64·6
H_2SO_4 soluble Zn	6·12	—extraction	90·7

Insoluble residue from 4 p.c. acid leach, 65%
Heads SO_2 in sample, 10·50%

Dry ground, about 200 mesh:

	%		%
Water soluble Cu	none		
H_2SO_4 soluble Cu	2·10	—extraction	89·3
Water soluble Zn	4·10	—extraction	60·7
H_2SO_4 soluble Zn	6·4	—extraction	94·8

Insoluble residue from 4% acid leach, 59·60%—Fe. 55·25%
Sulphide S, 0·4%—Total S. 4·60%

Roast No. 2

Roasting.—Sample of 100 grams 20 mesh ore roasted in E. M. furnace under following conditions and with the following results:—

General conditions similar to test No. 1, with exception that front door of furnace was left open slightly (½″) to allow a current of warm air to pass over material. Suction applied at rear end of furnace to an extent that no sulphur fumes could be detected coming out of front door. Rabbled every half hour.

Temperature observations:—

Starting temperature, 9.30 a.m.	450° C.
10.00 a.m.	475
10.30 a.m.	480
11.00 a.m.	500
11.30 a.m.	550
12.00 noon	585
12.30 p.m.	610

Roast withdrawn at 12.30. Time of roast, 3 hours.
Temperature range, min. 450°, max. 610° C.
Charge showed a loss in weight of 17·5 per cent.
Leaching.—Method same as in test No. 1.

Results on coarse material:

	%		%
Heads—Cu	2·35		
Water soluble Cu	0·36	—extraction	15·3
4% H_2SO_4 soluble Cu	2·18	—extraction	93·0
Heads—Zn	6·77		
Water soluble Zn	3·95	—extraction	58·34
4% H_2SO_4 soluble Zn	5·50	—extraction	81·24
Total S	4·94		
Sulphide S	1·15		
SO_3 in sample	9·50		
Fine dry ground:			
4% H_2SO_4 soluble Cu	2·15	—extraction	91·5
4% H_2SO_4 soluble Zn	5·20	—extraction	76·8

A sample drawn at 11 a.m. gave the following analysis on leaching:—

	%		%
Water soluble Cu	none		
4% H_2SO_4 soluble Cu	1·25	— extraction	53·60

	%		%
Water soluble Zn	none		
4% H_2SO_4 soluble Zn	2·65	—extraction	39·14

SUMMARY AND CONCLUSIONS

1. Concentration of the Ore.—The results of the experimental work show no difficulty in the concentration of the chalcopyrite by preferential flotation after grinding to 200 mesh. A copper concentrate is obtained as high as 15 per cent copper with a recovery of 85 per cent of the copper values in the ore. There is no reason to doubt, as far as the copper is concerned, that these results could be obtained in practice, and it is reasonable to assume that better results would be obtained with experience and manipulation after operations have progressed for some time.

The results also show that a separation has been made of the chalcopyrite from the sphalerite, pyrite, and gangue, and of the sphalerite from the pyrite and gangue. A zinc concentrate, 40 per cent zinc, with a recovery of 50 per cent of the zinc values, has been obtained. Before conclusions can be drawn on this separation as to its feasibility in practice, the installation of a pilot plant at the mine would be necessary.

When consideration is given to the market conditions with respect to zinc; to the low zinc content in the ore; to the more or less difficult metallurgical treatment; to the cost of production; and to the location of the property, it is doubtful if the recovery of zinc from this ore would be a commercial possibility for some years to come.

The results of the flotation tests show that the gold and silver values are evenly distributed in all the sulphides. If these values were confined to the chalcopyrite, there would be no doubt as to the adaptability of concentration by flotation to this ore, and this process would be the solution for the treatment, followed by subsequent smelting of the flotation concentrate. As the ore contains precious metal values of $2 per ton, and as only a small proportion (about 20 per cent) is recoverable in the copper concentrate by flotation methods, the recovery of these values. or the greater proportion of them, is necessary for a satisfactory metallurgical treatment.

Cyanidation of the flotation tailing has shown that the precious metal values can be extracted, but at a cost that would prohibit the use of this process.

2. The Hydro-Metallurgical Treatment of the Ore.—The results of the small laboratory roasting and leaching tests show good extractions of the copper and zinc. These small tests are, however, not conclusive with regard to the feasibility of this process as applied to the ore. To obtain reliable data, tests should be conducted in a small plant over a period of time of at least two weeks, so as to obtain conditions approaching practice.

This process would also have to be supplemented by an additional one for the recovery of the precious metal values, as they would report in the residues from leaching operations, and as before stated, any satisfactory metallurgical treatment must include the recovery of these values, it is extremely doubtful if a wet metallurgical treatment could be successfully applied.

3. Pyritic Smelting, with Subsequent Reduction to Blister Copper.—This seems to be the only satisfactory treatment for the sulphide ore. Some trouble may be experienced, on account of the zinc content in the ore, in obtaining a fluid slag. However, this difficulty can be overcome by proper mixing of the ores so as to keep the zinc content as low as possible, around 3 per cent. This method of treatment will result in a high recovery of the copper, with practically a total recovery of the precious metal values, and appears to be the process most adaptable, from a commercial standpoint, for the successful treatment of the Flin Flon sulphide ore.

(3)

THE CONCENTRATION OF GRAPHITE FROM THE NORTH AMERICAN MINE

C. S. Parsons

Considerable experimental work has been conducted in the Ore Dressing and Metallurgical Laboratories on graphite ores from different points in the Dominion. The introduction of the oil flotation process has made radical changes in the treatment of graphite ores. The old methods have been for the most part abandoned, for this newer process simplifies operations, with the production of higher grade products, and with a much higher recovery of the graphite in the ores.

In the Summary Report of the Mines Branch for 1919, under Test No. 126 of the Ore Dressing and Metallurgical Laboratories, a description is given of the experimental work conducted on a large shipment from the Quebec Graphite Company, Buckingham, Que. In this case a process was worked out which was later put into practice by the installation of a new mill by the company. This mill was very successful in treating their ore following the methods outlined by the Department. The test work conducted on graphite ores can be found under Tests Nos. 85, 102, and 103 of the Summary Report for 1918, Tests Nos. 113 and 126 of the Summary Report for 1919, and Tests Nos. 129 and 139 of the Summary Report for 1920. This work has been largely instrumental in the introduction of more advanced methods of treating graphite ores.

Test No. 143

A 41·5 pound sample of graphite ore was received January 3, 1921, at the Ore Dressing and Metallurgical Laboratories from Mr. H. P. H. Brumell, of Buckingham, Quebec. The ore was from the North American Graphite mine, Buckingham, and consisted of graphite flake of a fair size in a gangue mainly of calcite and pyroxene, with small amounts of mica and iron pyrites.

A few specimens were selected from the sample and the balance was crushed down gradually, with frequent screening, in a small jaw crusher and a small set of rolls, until it all passed through 10 mesh. A head sample was cut out of this −10 mesh material, which gave upon analysis, 13·40 per cent carbon.

Concentration tests were desired to determine the possibility of recovering a high percentage of the contained carbon in the form of a high grade coarse flake. Tests were conducted, using flotation, and in some cases concentration on tables was used in conjunction with flotation.

Test No. 1.—Five hundred grams of ore, −10 mesh, were floated in a small Ruth flotation machine, making a concentrate and tailing. The tailing was dewatered, ground 20 minutes in a small pebble jar, and floated again in the Ruth machine. The concentrate from the first flotation was tabled on a small Wilfley table, making a concentrate and a tailing. The following table shows the products and the results obtained:

Product	Weight grams	Per cent C.	Grams C.	Per cent C. values
Table conc	130	43·65	56·74	86·9
Table tails	69	8·20	5·66	8·6
Flot. conc	20	9·40	1·88	2·9
Flot. tails	275	0·38	1·04	1·6
	494		65·32	
Heads	500	13·40	67·00	100·0

Test No. 2.—Five hundred grams of ore, −14 mesh were floated in a small Ruth flotation machine. The concentrate from this flotation was tabled, making a concentrate and a tailing. The table and flotation tailings were mixed, dewatered, ground in a pebble jar for 20 minutes, and floated in a Ruth machine. The concentrate from this operation was tabled. The following table shows the products and results obtained:—

Product	Weight grams	Per cent C.	Grams C.	Per cent C. values
1st table conc	126	45·70	57·58	85·8
2nd table conc	27	29·05	7·84	11·7
Table tails	41	2·60	1·07	1·6
Flot. tails	284	0·22	0·62	0·9
	478		67·11	
Heads	500	13·40	67·00	100·0

Test No. 3.—Five hundred grams of ore, −20 mesh, were treated in the same way as the ore used in test No. 2. The following table shows the products and results obtained:—

Product	Weight grams	Per cent C.	Grams C.	Per cent C. values
1st table conc	143	43·25	61·85	88·5
2nd table conc	20	30·25	6·05	8·7
Table tails	22	2·86	0·63	0·9
Flot. tails	293	0·46	1·35	1·9
	478		69·88	
Heads	500	13·40	67·00	100·0

Conclusions derived from Tests Nos. 1, 2 and 3.—The results of these tests show that while recoveries are good, and the tailings low in carbon content, the grade of concentrates produced is very low, due no doubt to the fact that the graphite flake is not entirely freed from the gangue material. In order to raise the grade, further grinding is necessary, and test No. 4 was conducted along this line.

Test No. 4.—One thousand grams of ore, −10 mesh, were floated in a small Ruth flotation machine, the tailing was ground in a pebble jar for 20 minutes and floated again. The concentrates from both operations were mixed and ground in a pebble jar for 20 minutes, and then floated. The flotation concentrate was tabled on a small Wilfley table, making a concentrate and a tailing. The table concentrate was dried and screened on 80 and 150 mesh screens (Tyler). The following table shows the results obtained:—

Product	Weight grams	Per cent C.	Grams C.	Per cent C. values
Table conc. +80	140	73·40	102·76	73·5
Table conc. +150	40	39·80	15·92	11·4
Table conc. −150	24	44·05	10·57	7·5
Table tails	55	11·00	6·05	4·3
Flot. midds	181	1·90	3·44	2·5
Flot. tails	547	0·20	1·09	0·8
	987		139·83	
Heads	1,000	13·40	134·00	100·0

Conclusions derived from Test No. 4.—A high percentage of carbon values is recovered in the form of coarse flake, but the grade is not as high as could be desired. More regrinding is necessary. A further test was made, the details of which are given under test No. 5.

Test No. 5.—One thousand grams of the ore, which had previously been crushed to 10 mesh, were ground in a small ball mill until practically all the gangue would pass a 20 mesh screen.

Flotation was employed for the separation of the graphite from the gangue. The oils used were kerosene, which was added to the ball mill, equivalent in amount to one pound per ton of ore, and pine oil, which was added to the flotation machines as required to form a desirable froth. Further pine oil and coal oil was added from time to time in recleaning the concentrates. The following method of procedure was carried out:—

The ore was floated, making a rougher concentrate and low tailing. The rougher concentrate was dewatered and ground in a pebble mill, refloated, making a second concentrate and tailing. The second concentrate was dewatered and reground in a pebble mill, refloated, making a cleaner concentrate and tailing. The cleaner concentrate was screened on 65 and 150 mesh. All tailing products and the final concentrate were dried, weighed, sampled, and analysed. The results of this test are given in the following tables:—

Products	Weight grams.	Analysis per cent C.	Content grams C.	Percentage of C. values
Cleaner concentrate	145	87·90	127·46	92·7
Rougher tailing	674	0·35	2·36	1·7
Second tailing	120	2·53	3·04	2·2
Cleaner tailing	49	9·54	4·67	3·4
Heads	1,000	13·75	137·50	100·0

NOTE.—The second tailing could be discarded or returned to the circuit. The cleaner tailing would be returned to the circuit, or treated separately, and a percentage of the carbon values recovered.

SCREENING OF CLEANER CONCENTRATE

Products	Weight grams	Percentage of weight	Analysis per cent C.	Content grams C.	Percentage of C. values
+ 65 mesh	68	46·9	94·40	64·19	46·7
−65 +150 mesh	41	28·3	90·25	37·00	26·9
−150 mesh	36	24·8	73·10	26·32	19·1
Totals	145	100·0	87·90	127·46	92·7

Summary of Results.—One ton of ore contains and yields the following products:—

Content: 2,000 pounds at 13·75% C. = 275 pounds of carbon.
Products: 290 pounds of concentrates at 87·9% C.
of which 136 pounds is +65 mesh material at 94·40% C.
82 pounds is −65 +150 mesh material at 90·25% C.
and 72 pounds is −150 mesh material at 73·10% C.

CONCLUSIONS

The above test shows that a high grade graphite concentrate can be obtained without difficulty with a high recovery of the contained carbon values; that the screened products are of excellent grade; that a good percentage is recovered as coarse flake; that in order to obtain these results regrinding is essential to free the flake from attached particles of gangue.

These results have been obtained from a small test under the most favourable conditions. Whether similar results can be obtained in practice is problematical. The grinding action in large mills may have the tendency to destroy the coarseness of the flake much more than in the treatment of the small lot of this test.

(4)

THE CONCENTRATION AND SEPARATION OF THE MINERALS IN THE LEAD-ZINC-SILVER ORES OF KIMBERLEY, B.C.

C. S. Parsons and R. J. Traill

Methods and processes for the concentration and separation of the minerals in the ores of the Kimberley district, B.C., have been the subject of investigation and research for a number of years by the Consolidated Mining and Smelting Company of Canada, Trail, B.C.

Improvements have steadily advanced to a stage where the difficult ores of the Sullivan mine can be successfully treated by differential flotation, resulting in marked recoveries of the lead and zinc in these ores.

The following tests were made on ores from the North Star mine, on the carbonate ore from the upper portion of the ore-body, and on the sulphide ore found at greater depth. The carbonate ore consisted chiefly of lead carbonate, coated and locked up in a mass of iron oxides, formed by the oxidation of iron sulphides. The sulphide ore was an intimate mixture of lead, zinc, and iron sulphides, the gangue minerals representing less than 10 per cent by weight of the ore.

Test No. 144

A shipment of 370 pounds of lead carbonate ore was received at the Ore Dressing and Metallurgical Laboratories, January 26, 1921, from the North Star mine, Kimberley, B.C.

The ore consisted chiefly of lead carbonate, coated and locked up in a mass of iron oxides, formed by the oxidization of the iron sulphides, and gave an analysis as follows:—

Lead (Pb)..	10·85 %	
Iron (Fe)	14·30 "	Fe_2O_3 20·45%
Alumina (Al_2O_3)	4·55 "	
Silver (Ag)	6·55 oz. per ton	
Moisture (H_2O)	3·60%	

The object of the experimental work was to determine a method of concentration for the ore. A number of small scale tests were made by sulphidizing the lead carbonates and oxides for flotation, using gravity concentration on tables, and by the chloride volatilization process.

A series of tests were first conducted using a combination of flotation and table concentration. In these tests the table tailings were sulphidized with sodium sulphide before flotation. It was thought that a long contact with the sodium sulphide was necessary, but later it was found that much better results could be obtained by allowing the ore only a short contact with the sulphidizing reagent before flotation. The results of two of the first series of tests are given below:—

Test No. 4.—One thousand grams of ore crushed to 60 mesh was agitated for 30 minutes with 20 pounds per ton of sodium sulphide salt, Na_2S. $9H_2O$. It was floated and the flotation tailing tabled. The following other reagents were added:—

Soda ash	1	pound per ton
Coal tar creosote	4	"
Pine oil (No. 5 G.N.S)	0·4	"

Product	Weight grams	Pb. per cent	Pb. grams	Recovery of Pb.	Remarks
Flotation conc.	149	31·35	46·71	45·4	74·4 per cent recovery assuming that 70 per cent of the lead in the middling would report in the concentrate.
" midd.	191	7·80	14·90	14·4	
Table conc.	44	44·27	19·48	19·0	
" tailing	360	3·15	11·34	11·0	
" slime	179	5·86	10·49	10·2	

Test No. 9.—One thousand grams of ore crushed to 100 mesh was agitated $1\frac{1}{4}$ hours with 10 pounds per ton of sodium sulphide salt Na_2S. $9H_2O$. It was floated and the flotation tailing tabled:—

Product	Weight grams	Pb. per cent	Pb. grams.	Recovery of Pb.	Remarks
Flotation conc.	155	40·35	62·5	57·6	79 per cent recovery assuming that 70 per cent of the values in the middling would report in the concentrate.
" midd.	171	7·25	12·4	11·4	
Table conc.	43	34·33	14·8	13·6	
" tailing	631	2·98	18·8	17·4	

A second series of tests were conducted, varying the time of contact with the sulphidizing reagent, and the strength of this reagent used. The strength of solution used is given in terms of Na_2S. (Note.—The commercial salt contains varying quantities of water.) The results of these tests, and reagents used, are given in the tables following.

The procedure followed was to grind the ore wet in a ball mill to 200 mesh, dewater, and pass through a filter press. The solution of sodium sulphide made up to strength required was then added to the cake, and the two agitated together for a certain length of time. It was found that the sulphidization of the lead particles, which were free and exposed to the action of the reagent, took place almost instantly, and were easily recovered by flotation. The recovery of the silver values was much more difficult, a maximum of 40 per cent was the best that could be obtained.

In test No. 30 a strong solution of sodium sulphide was left in contact with the ore for 15 minutes. The solution on being tested showed no free Na_2S. The recovery of the silver values was no greater than when the weaker solution was used for shorter periods of contact. It was also found that the lowest tailing carried 3 per cent lead. The difficulty in obtaining high recoveries of the lead and silver values seems to be due to the physical characteristics of the ore. The particles of lead carbonate and the silver values, in whatever form they occur, seem to be locked up in a mass of iron oxides, which completely film them. Wet grinding to 200 mesh is necessary to remove as much of this coating as possible, and even at this fine state of division the minerals are not entirely freed, but remain enclosed in the film of iron oxide. This was demonstrated by taking 1,000 grams of the ore, grinding and tabling successively by stages until a fineness of 200 mesh was reached. It was found at this stage of crushing that no more lead was freed to report as concentrate on the table, and the tailing at this stage gave an assay of 3·5 per cent lead.

A recovery of between 70 per cent and 80 per cent of the lead values, and 40 per cent of the silver values, seems to be the maximum that could be expected by the combined method of sulphidizing for flotation and tabling the flotation tailing. The control of the sulphidizing condition seems to be simple, gives uniform results, and the above recoveries should be obtained quite readily from the ore as submitted for test purposes.

CHLORIDE VOLATILIZATION TESTS

Two small tests were made to determine whether this process was applicable to the ore. The results obtained, as to recovery of the lead and silver values, were practically the same as those obtained by flotation and tabling.

Test No. 1:

Assay of ore	Silver, 6·55 oz	Lead, 10·85 %
Assay of tailing	Silver, 4·61 oz	Lead, 3·41 %
Volatilized	Silver, 35·30 %	Lead, 70·4 %

Test No. 2:

Assay of ore	Silver, 6·55 oz	Lead, 10·85 %
Assay of tailing	Silver, 5·06 oz	Lead, 4·80 %
Volatilized	Silver, 29·00 %	Lead, 58·70 %

Reagents Used

Test No.	Strength of solution in dry Na_2S salt	Time	Oils used	Remarks
25	2·20 grams in 500 c. c.	15 mins.	10% C.T.; 80% C.T.C.; 10% pine oil—1 c.c.	Too much frothing oil.
26	2·20 " 1,000 "	15 "	Same as test No. 25—1 c.c.	
27	4·40 " 2,000 "	15 "	10% 1-B; 80% C.T.C.; 10% C.T.; and enough crude turps—1 c.c.	
28	2·20 " 2,000 "	15 "	Same as test No. 27—1 c.c.	
29	4·40 " 4,000 "	15 "	Same as test No. 28—1·5 c.c.	
30	8·80 " 3,000 "	15 "	Same as test No. 29.	
31	2·20 " 3,000 "	3 "	Same as test No. 29.	Added solution directly in cells.
32	2·20 " 3,000 "	3 "	10% B-14; 20% C.T.; 70% C.T.C.	Solution added directly to cells. Very high grade looking froth.
33	4·40 " 3,000 "	5 "	Same as test No. 32.	Na_2S solution added directly in cells. Very high grade looking froth.
34	4·40 " 3,000 "	5 "	X-Y reagent—0·25 c.c.	Na_2S solution added directly in cells. Very watery looking froth and voluminous.
35	4·40 " 3,000 "	5 "	10% B-14; 20% C.T.; 70% C.T.C., plus crude turpentine.	Quite a heavy good froth. This sample was only crushed for 15 minutes.
36	4·40 " 3,000 "	5 "	10% B-14; 20% C.T.; 70% C.T.C—1 c.c.	This test ground very fine for two hours. Fine looking black froth.

Results of Tests

Test No.	CONCENTRATE						MIDDLING						TAILING					
	Weight grams	Assay		Content		% of Pb. values	Weight grams	Assay		Content		% of Pb. values	Weight grams	Assay		Content		% of Pb. values
		Pb. %	Ag. ozs.	Pb. gms.	Ag. gm.-ozs.			Pb. %	Ag. ozs.	Pb. gms.	Ag. gm.-ozs.			Pb. %	Ag. ozs.	Pb. gms.	Ag. gm.-ozs.	
25..........	99	39·75	......	39·4	....	37·4	118	13·5	......	......	....	....	....	......	......	......	....	....
26..........	189	31·7	......	......	....	55·0	....	......	......	......	....	....	....	......	......	......	....	....
27..........	119	53·35	11·06	63·5	....	58·9	132	8·96	......	11·8	....	10·0	712	4·70	......	33·5	....	31·1
28..........	96	50·7	......	48·7	....	44·8	162	10·0	......	16·2	....	14·9	....	......	......	......	....	....
29..........	131	46·5	......	60·9	....	56·2	202	8·5	......	17·17	....	15·8	....	......	......	......	....	....
30..........	134	49·6	10·36	66·5	1,393·6	61·2	86	13·2	......	11·4	....	10·3	780	3·97	......	31·0	....	28·5
31..........	148	51·8	......	76·7	....	70·5	173	6·4	......	11·1	....	10·2	679	3·10	......	21·0	....	19·3
32..........	142	53·5	......	76·0	....	70·0	129	7·0	......	9·0	....	8·3	761	3·20	......	24·4	....	21·6
33..........	125	59·8	......	74·8	....	68·9	86	7·9	......	6·8	....	6·3	816	3·3	......	26·9	....	24·8
34..........	91	56·8	10·4	51·7	946·4	47·6	110	13·3	8·96	14·6	....	13·5	815	5·18	5·4	42·2	4,401	38·9
35..........	113	61·5	19·6	69·5	2,214·8	63·8	59	7·5	6·7	4·4	....	4·0	856	4·1	4·0	35·1	3,430	32·2
36..........	134	50·3	16·4	67·4	2,197·6	61·7	168	8·9	7·4	14·9	....	13·5	732	3·7	4·28	27·1	3,130	24·8

Test No. 145

A shipment of 351 pounds of lead-zinc sulphide ore was received on January 29, 1921, from the North Star mine, Kimberley, B.C.

The ore consisted of lead, zinc, and iron sulphides, and a siliceous gangue. The sulphide minerals represent about 90 per cent by weight, and the gangue about 10 per cent. The analysis of the ore was as follows:—

Lead (Pb)	18·00 %
Zinc (Zn)	30·45 "
Iron (Fe)	15·60 "
Silica (SiO_2)	7·50 "
Silver (Ag)	13·65 ozs.

The object of the experimental work was the separation of the sulphide minerals by preferential flotation, producing a marketable lead and zinc product.

A number of small scale tests were conducted, using oils, and the Callow Alphabetical flotation reagents. The results of the experimental tests, and the reagents used, are given in the following tables:—

REAGENTS USED

Test No. 1:

Flotation of lead—

2 pounds per ton Selecto added to ball mill.
5 pounds per ton lime added to ball mill.
Small quantity of steam distilled pine oil, sufficient to produce a good froth, added to cells.

Flotation of zinc—

Lead tailings were dewatered.
5 pounds soda ash added to cells.
2 pounds No. 34 gravity fuel oil (Great Western Oil Co., Cleveland, O.).
Pine oil to froth.

Test No. 6:

Flotation of lead—

2 pounds per ton Selecto added to ball mill.
6 pounds per ton lime added to ball mill.
Small quantity of pine oil, sufficient to produce a good froth.

Flotation of zinc—

Lead tailings were dewatered.
6 pounds soda ash added to cells.
1 pound copper sulphate.
Pine oil to froth.

Test No. 7 :

Flotation of lead—

2 pounds per ton Selecto added to ball mill.
6 pounds per ton lime added to ball mill.
0·25 pound per ton Z-cake added to ball mill.
Small quantity of Aldol used as frother.

Flotation of zinc—
- Lead tailings were dewatered.
- 6 pounds soda ash added to cells.
- 1 pound copper sulphate added to cells.
- 0·3 pound X-Y mixture.

Test No. 8:

Flotation of lead—
- 3 pounds per ton Selecto added to ball mill.
- 8 pounds per ton lime added to ball mill.
- 4 pounds per ton Na_2S (dry) added to flotation cells.
- 0·3 pound per ton pine oil.

Flotation of zinc—
- Lead tailings were dewatered.
- 6 pounds per ton soda ash added to cells.
- 2 pounds per ton copper sulphate added to cells.
- 2 pounds per ton No. 34 gravity fuel oil.

Test No. 9:

Flotation of lead—
- 4 pounds per ton Selecto added to ball mill.
- 8 pounds per ton lime added to ball mill.
- 0·25 pound per ton Z-cake added to ball mill.
- 8 pounds per ton Na_2S added to ball mill.

Flotation of zinc—
- Lead tailings were dewatered.
- 5 pounds per ton soda ash added to cells.
- 1 pound per ton copper sulphate added to cells.
- 0·3 pound per ton X-Y mixture added to cells.

Test No. 10:

Flotation of lead—
- 2 pounds per ton Selecto added to ball mill.
- 5 pounds per ton lime added to ball mill.
- 0·3 pound per ton No. 5 pine oil added to cells.

Flotation of zinc—
- Lead tailings were dewatered.
- 5 pounds per ton soda ash added to cells.
- 2 pounds per ton copper sulphate added to cells.
- 1 pound per ton No. 34 gravity fuel oil.
- 0·1 pound per ton No. 5 pine oil.

Results of Tests

Test No.	Flotation products	Weight grams	Analysis			Content			Percentage of values in heads		
			Pb. %	Zn. %	Ag. ozs.	Pb. gms.	Zn. gms.	Ag. gm.-ozs.	Pb.	Zn.	Ag.
1	Lead concentrate	223	60·3	8·5	42·00	134·47	18·95	9,366	76·2	6·5	69·6
	Zinc concentrate	545	7·0	45·9	6·60	38·15	250·15	3,597	21·6	86·6	26·7
	Tailing	232	1·6	8·5	2·10	3·71	19·72	487	2·1	6·8	3·6
	Heads	1,000	17·63	28·88	13·45	176·33	288·82	13,450	99·9	99·9	99·9
6	Lead concentrate	342	38·0	22·5	....	129·96	76·95	....	74·3	25·7	....
	Zinc concentrate	459	8·3	44·0	....	38·10	201·96	....	21·8	67·4	....
	Tailing	208	3·2	10·0	....	6·65	20·80	....	3·8	6·9	....
	Heads	1,009	17·47	29·97	....	174·71	299·71	....	99·9	100·0	....
7	Lead concentrate	379	42·2	22·8	....	159·94	86·41	....	86·8	29·4	....
	Zinc concentrate	430	4·8	44·7	....	20·64	192·21	....	11·2	65·4	....
	Tailing	195	1·9	7·9	....	3·70	15·40	....	2·0	5·2	....
	Heads	1,004	18·35	29·28	....	184·28	294·02	....	100·0	100·0	....
8	Lead concentrate	245	53·73	13·52	....	131·64	33·12	....	72·9	11·2	....
	Zinc concentrate	554	8·14	45·67	....	45·10	253·04	....	25·0	85·5	....
	Tailing	205	1·83	4·73	....	3·75	9·70	....	2·1	3·3	....
	Heads	1,004	17·98	29·47	....	180·49	295·86	....	100·0	100·0	....
9	Lead concentrate	301	43·65	16·59	....	131·39	49·94	....	80·4	16·7	....
	Zinc concentrate	452	5·76	48·51	....	26·04	219·27	....	15·9	73·4	....
	Tailing	258	2·30	11·47	....	5·93	29·59	....	3·6	9·9	....
	Heads	1,011	16·16	29·55	....	163·36	298·80	....	99·9	100·0	....
10	Lead concentrate	255	59·83	14·15	....	152·57	36·08	....	81·2	11·9	....
	Zinc concentrate	477	5·60	49·68	....	26·71	236·97	....	14·2	78·5	....
	Tailing	275	3·16	10·49	....	8·69	28·85	....	4·6	9·6	....
	Heads	1,007	18·66	29·98	....	187·97	301·90	....	100·0	100·0	....

NOTE: Iron content in zinc concentrate test No. 1, was 12·3%.

SUMMARY AND CONCLUSIONS

From the foregoing table of results it has been determined that the sulphide mineral constituents of the ore can be separated by preferential flotation, and satisfactory lead and zinc products produced with a good recovery of the values in the ore. No difficulty was experienced in obtaining the separation, provided that fresh ore was used in conducting the tests. Fresh ore seems to be essential, as is shown by the examination of the results of tests Nos. 6, 7, 8, and 9. The lump ore received was crushed to ¼ inch, and sampled. A portion was cut out, ground to 20 mesh for test purposes, and used in tests Nos. 1 to 9 inclusive. Tests Nos. 1 to 5 inclusive were run immediately, and gave uniform results similar to those shown under test No. 1 in the table. After a month's time, further work was resumed on the portion ground to 20 mesh, for test purposes, and it was found that the results of the preceding tests could not be duplicated under the same conditions. Examples of this are given in the results of tests Nos. 6 and 7.

The conclusion was reached that the lead had become slightly oxidized. A little sodium sulphide was added, which improved the results, as shown in tests Nos. 8 and 9. In order to verify this conclusion and check results of tests Nos. 1 to 5, a fresh sample of ore was prepared from the ¼ inch material, and a test conducted under the same conditions. The results are given under test No. 10, and check closely with those of test No. 1.

The use of the Alphabetical reagents seems to give a better recovery of the lead, while the oil gives the higher recovery of the zinc. In the tests conducted with the Alphabetical reagents, the use of the reagent Selecto governs the separation of the lead and zinc. The lime prevents the iron, and to some extent, the zinc, from floating. After the flotation of the lead the pulp must be dewatered to eliminate the effect of the Selecto reagent on the subsequent flotation of the zinc. It was found that to obtain a high recovery of the zinc, it was necessary to use soda ash or caustic soda. The use of copper sulphate was found to be beneficial, but not essential. To obtain the best results, the lime should be added in sufficient quantities when the ore is being ground, before the flotation of the lead, otherwise it will be found that the iron sulphides will have a greater tendency to float with the zinc sulphides.

In the tests conducted, the ore was ground wet in a ball mill, to pass a 150 mesh screen. It is doubtful if good results could be obtained on coarser material.

(5)

COMPARATIVE TESTS ON COBALT SILVER SLIMES

C. S. Parsons and R. K. Carnochan

Test No. 146

In continuation of the experiments made in 1917 on the use of Canadian wood oils for the concentration of the values in the Cobalt slime tailing, a number of experimental tests were conducted on the use of the Rex, Alphabetical, and other reagents, on a shipment of 2,000 pounds of sand tailings, received at the Ore Dressing and Metallurgical Laboratories, February 9, 1921, from the Coniagas Mines, Limited, Cobalt, Ont. The results of these tests are given in the following tables:—

TABLE I

Results of Tests using Rex Flotation Reagents

Test No.	Heads ozs. Ag.	Conc. ozs. Ag.	Midds. ozs. Ag.	Tails ozs. Ag.	Conc. ratio	Recovery
38..................	8·08	66·60	....	5·88	1 : 26·3	30·8
39..................	8·08	88·37	....	6·38	1 : 44·3	24·3
41..................	8·08	74·72	....	4·90	1 : 23·8	40·2
42..................	8·08	82·12	....	4·44	1 : 22·7	46·1
43..................	8·08	72·44	....	3·90	1 : 17·0	53·8
73..................	3·70	328·20	20·46	2·24	1 : 144·2	53·9
74..................	3·70	251·75	18·00	2·42	1 : 104·4	52·8

The above tests were run under the following conditions, except where differences are noted.

Janney flotation machine and Rex flotation reagents used. Five hundred grams of silver slimes ground to 100 mesh, mixed in pebble mill 5 minutes, agitated in machine 5 minutes, and floated for 8 minutes.

Test No.
41 Ruth machine, 1,000 grams.
42 Ruth machine, 1,000 grams.
43 Ruth machine, 1,000 grams.
73 Four charges run, and concentrates combined and re-run to clean them up. In re-running, mixed 5 minutes, agitated 5 minutes, and floated for 5 minutes.
74 Same as 73, except that concentrates were floated 10 minutes when re-cleaning.

TABLE II

Results of Tests using Alphabetical and Rex Flotation Reagents

Head sample of shipment assayed 3·48 ounces of silver per ton.

If not otherwise stated, the ore in each of the following tests was crushed dry to pass 60 mesh, and then 1,000 grams charged to a small ball mill, ground with one part of water to two parts of ore, for 30 minutes.

Test No.	Re-agents used, etc.	Concentrate		Middling		Tailing	
		Wt. gram	Assay oz. Ag.	Wt. gram	Assay oz. Ag.	Wt. gram	Assay oz. Ag.
1	1 pound per ton X. Y.................	19	114·3	96	5·02	893	0·80
2	2 " soda ash 1 " X. Y.................	46	52·70	128	1·72	823	0·98
3	4 " soda ash 1 " X. Y.................	20	122·4	61	4·50	913	0·88
4	6 " soda ash 1 " X. Y.................	18	78·51	91	3·10	877	1·24
5	6 " lime 1 " X. Y.................	32	60·64	113	2·40	858	1·70
6	2 " lime 1 " X. Y.................	54	42·30	175	1·30	771	1·28
7	2 " soda ash 2 " Rex No. 12...........	40	19·08	182	4·14	756	2·76
8	3 " Rex No. 12...........	26	23·22	134	4·14	840	3·04
9	4 " soda ash 2 " Rex No. 12...........	36	17·76	158	4·14	794	2·86
10	4 " sodium sulphide 2 " Rex No. 12...........	50	8·32	....	....	923	3·30
11	4 " soda ash 2 " Rex No. 12 1 c. c. ammonia......................	27	28·0	118	3·22	839	2·53
12	4 pound per ton soda ash 2 " Rex No. 12 1 c. c. ammonia......................	45	13·40	126	4·04	790	3·34
13	0·2 pounds per ton pine oil....... 1·6 " C.T. creosote...... 0·2 " coal oil............	70	33·04	143	3·34	764	0·70
14	4 pounds per ton soda ash 1 " NH_4OH 2 " Rex No. 12........... ground sands for 45 minutes...........	39	17·08	190	3·38	751	2·78
15	10 pounds per ton soda ash 2 " Rex No. 12 1 pound per ton soda silicate for recleaning concentrate....................	49	15·58	163	3·64	759	2·96
16	3 pounds per ton sod. hydroxide 2 " Rex No. 12...........	68	13·18	274	3·02	633	2·82

(6)

THE MILLING OF ASBESTOS ROCK

R. K. Carnochan

The general milling practice for the recovery of the asbestos fibre from the serpentine rock has not changed to any extent since its innovation. It is true that marked improvements have been made, resulting in more efficient recovery of the fibre, but production of the fibre in marketable grades has claimed the attention of the operators, and little attention has been given to experimental work on new processes for the recovery of the fibre from the rock. While metallurgical practice has radically changed in connection with the treatment of the metallic ores, no changes of such a nature have taken place in the milling of asbestos rock.

The general practice for the recovery of the fibre is a dry process. After as much crude fibre as is economically possible is picked out from the broken rock in the pits and the dry rock discarded, the remainder, the milling rock, is delivered to the mill for the separation and recovery of the fibre. The rock is broken in primary breakers, such as jaw and gyratory crushers, to about three inch size. It is then dried in rotary or vertical dryers. From this stage the process is one of breaking the rock to release the fibre, and at the same time fiberizing or fluffing the fibre, and the removal of the fibre by suction as the crushed material passes over shaking tables. This process is repeated until the rock is reduced to such a size that it contains only very short fibre, the recovery of which is not economically feasible. The breaking and fiberizing machines used are hammer crushers, rolls, Simons disc crushers, cyclone and jumbo fiberizers. The hammer crushers, cyclone and jumbo fiberizers were developed in the district as being adapted to the breaking and fiberizing of asbestos rock. The fibre is collected in fibre collectors, passed through dusters to remove dust and fine particles of rock adhering to it. It is then passed through graders, which grade it into the various commercial products. A standard asbestos testing machine was developed in the district, which is used by all operators, on which the grades are tested for length of fibre. Shipments are made conforming to certain tests on this machine.

The investigations carried on in the Ore Dressing and Metallurgical Laboratories were confined to the use of the conical ball mill for the cracking of the rock and liberation of the fibre after primary crushing. The use of the cyclone and jumbo fiberizers in the present practice is found to be costly as to power consumption and to maintenance and repairs. The violent action in these machines is also more or less destructive of the long fibre. Shipments were received of both the "slip fibre" variety of rock from East Broughton, Que., and the "cross fibre" variety from Black Lake, Que. The object of the experimental work was to determine if some other types of crushing machinery could not be used for the cracking of the rock and liberation of the fibre, with less destruction of the fibre than by the use of cyclone and jumbo fiberizers. In conducting this work many points of interest were brought out, which are noted under the description of the investigation.

It is proposed to carry the investigation further along the lines of wet crushing of the rock. As one of the chief items of cost, especially in certain seasons of the year, is that of drying the rock for the present practice, a considerable saving could be made if a wet process could be applied, with an improvement of the working conditions in the mills of the district.

Test No. 147

A shipment of 500 pounds of tailings was received on February 9, 1921, at the Ore Dressing and Metallurgical Laboratories, from the Asbestos Mines, Limited, East Broughton, Que.

This material represented the discard of their milling operations, and contained a considerable quantity of fine fibre not freed from the rock.

The purpose of the test work on this shipment was to determine if any products suitable for trade purposes could be obtained from the rock, what further treatment was necessary, and what use could be made of such products.

No experimental work was done on this shipment, as other work was more pressing, and it was found that the mills of the district were making a large quantity of fine fibre, for which there was not a brisk demand. It was decided advisable to await a more opportune time for conducting the work.

Test No. 149

Two shipments of asbestos rock were received at the Ore Dressing and Metallurgical Laboratories, from the Asbestos Mines, Ltd., East Broughton, Quebec. The rock was from the Boston mine, was received crushed to about three-inch size, the product of their secondary crushers, and represented the feed to the Jumbo and Cyclone crushers or fiberizers.

The rock from the Boston mine is of the "slip fibre" type, representative of the East Broughton deposits, which contains no crude, but is a straight milling rock. In this respect it is dissimilar to the Thetford mines, and Black Lake deposits. The rock is also much softer than that from the other districts.

The first shipment of three tons was received on March 30, 1921, and the second shipment of ten tons on May 16, 1921.

In the company's milling operations, the Jumbo and Cyclone crushers or fiberizers are used to separate the fibre from the rock. These machines, while being good fiberizers, are costly to operate, as to power consumed and repairs required. Moreover, the action is so violent that the fibre is more or less broken up, resulting in an excessive amount of fines. The object of the experimental work was to determine what results could be obtained from the Hardinge mill in comparison with the Jumbo and Cyclone crushers or fiberizers, as to amount and grade of fibre produced, and also as to power consumption and repairs. As no data were available or could be secured as to the efficiency of the crushers and fiberizers in use, the only course to pursue was to perform the tests and leave the operators to draw their own conclusions and comparisons of results.

The first shipment of three tons was used up in making adjustments to the Hardinge mill, to suit the crushing of this type of rock. Tests were run using pebbles and balls as the grinding or breaking media, increasing the speed of the mill, increasing the inclination of the mill, and the removal of the fibre from the mill, on being freed from the rock, by suction.

In the crushing of asbestos rock, the main point to consider is the separation of the fibre from the rock to obtain the maximum amount of long fibre. The objective is, therefore, to crush or break the rock in such a manner as to remove the fibre without grinding, and with as little damage to it as possible, and to remove it as soon as it is free, so that it will not be damaged by further crushing or breaking action. Up to the present, on account of its universal use as a grinding mill, it was considered that the Hardinge mill would not be adaptable to this class of work, in fact it was claimed that asbestos rock, due to its nature, would grind quite freely for a time and then pack on the lining until crushing action ceased almost entirely. Our tests on this first shipment proved this not to be the case. By elevating the feed end of the mill to obtain quick discharge; by using large balls to obtain coarse grinding; by increasing the speed of the mill to obtain a breaking and not a grinding action; and by connecting the discharge end of the mill with a suction to remove the fibre as it was freed, favourable results could be obtained. It was also determined that the rate of feed to the 4′ 6″ × 13″ mill should be about three tons per hour.

Having obtained these data, a new fibre collector was built, with connections to the discharge end of the mill and to a shaking screen, and also to an exhauster which exhausted into another collector and to the air. With this arrangement there was practically no loss of fibre. The ball mill was adjusted to what had been proven from the runs on the first shipment as the most desirable setting. It was also decided not to feed three-inch material to the mill, but to crush the rock in a jaw crusher and rolls to one-inch size before feeding to the ball mill, as this was the practice that was contemplated in the revised layout of the Boston mill. By crushing to one-inch size before feeding to the ball mill a large percentage of the fibre would be freed, and removed by suction.

Run No. 1.—Nine tons of the rock from shipment No. 2 were weighed, crushed in a jaw crusher to 1½″, and reduced to 1″ in rolls. This operation gave the following:—

Product	Weight pounds	Per cent of heads
– 1″	17,874·5	99·30
Dust loss	125·5	0·70
Total	18,000·0	100·00

Run No. 2.—The rock reduced to 1″ was sampled by an automatic Vezin sampler, the main flow passing over a shaking screen, fitted with a blank screen for the first half and with a $\frac{1}{16}$″ slot screen for the second half at the discharge end. Two suction pipes into a main header to the collector were placed over the screen for the removal of the fibre. An exhauster was placed between the first collector and discharged into a second collector from which a second exhauster discharged into the air. Any fibre not collected in the first collector was deposited in the second one. With this arrangement there was practically no loss of fibre. This operation gave the following:—

Product	Weight Pounds	Per cent of heads	Test
$+\frac{1}{16}''$	12,150·0	68·24	
$-\frac{1}{16}''$	3,312·0	18·60	
Fibre—1st collector	2,070·0	11·65	0—0·5 —4·7 —10·8
Fibre—2nd collector	21·0	0·12	(Large per cent rock)
Dust loss	123·5	0·69	
Sample	194·0		
Feed	17,874·5	99·30	

Run No. 3—First ball mill run.—The $+\frac{1}{16}''$ material from Run No. 2 was sampled by an automatic Vezin sampler and fed to a Hardinge mill at the rate of 2·75 tons per hour. The 4′ 6″ x 13″ mill was run at 36 r.p.m. and carried a load of 2,900 pounds of 4″ steel balls. The feed end of the mill was raised three inches from the horizontal. The discharge end was fitted with a suction pipe to the first collector, with a T pipe close to the discharge, permitting of the regular discharge and also of the fibre being sucked over into the collectors. This first ball mill run gave the following:—

Product	Weight pounds	Per cent of heads
Fibre—1st collector	6,163	35·04
Dust—2nd collector	113	0·64
Rock, discharged	5,540	31·49
Rock, in mill	188	1·07
Sample	146	
Feed	12,150	68·24

Run No. 4—Second ball mill run.—The rock discharged from the first run of the ball mill was sampled by an automatic Vezin sampler and fed back to the mill, which contained some rock from the preceding operation. The mill was run under the same conditions. After the completion of the run the mill was dumped. This operation gave the following:—

Product	Weight pounds	Per cent of heads
Fibre—1st collector	1,377·0	8·04
Dust—2nd collector	38·5	0·22
Tailing, discharged	3,476·0	20·30
Tailing, from mill	553·0	3·23
Dust loss	131·0	0·77
Sample	137·5	
Feed	5,713·0	32·56

Run No. 5—Fibre from first ball mill run.—After sampling the fibre from Run No. 3, or the first ball mill run, it was run over the shaking screen to remove the rock and dust. This gave the following products:—

Product	Weight pounds	Per cent of heads	Test
Fibre—1st collector	940	5·41	0—0 —3·2 —12·8 (fairly clean)
Dust—2nd collector	14	0·08	
+$\frac{1}{16}$″ rock	2,678	15·40	
−$\frac{1}{16}$″	2,360	13·57	
Dust loss	101	0·58	
Sample	70		
Feed	6,163	35·04	

The +$\frac{1}{16}$″ rock contained considerable unfiberized material and should have been returned to the circuit.

The −$\frac{1}{16}$″ material contains some fibre that would produce a marketable grade as shown in a following table, under Run No. 12.

Runs Nos. 6, 7 and 8—Fibre from second ball mill run.—The fibre from Run No. 4, or the second ball mill run, was used in these three runs, an attempt being made to remove the rock and bring the fibre up to a suitable grade. In Run No. 6, the fibre from Run No. 4 was run over the shaking screen, using a 12×12 mesh screen, and with the suction at the end of the screen near the discharge, so as to allow the fines to screen out from the fibre before it was sucked up into the collectors. In Run No. 7, the fibre from Run No. 6 was re-run, and in Run No. 8 the fibre from Run No. 7 was re-run. The second collector was not cleaned out until the finish of Run No. 8. The following are the products obtained:—

Product	Weight pounds	Per cent of heads	Test
Fibre—1st collector	180·0	1·05	0 —0 —2·8 —13·2
Dust—2nd collector	100·0	0·58	
+12, run 6	443·5	2·60	
+12, run 7	25·0	0·15	
+12, run 8	29·0	0·17	
−12, run 6	230·0	1·35	
−12, run 7	109·0	0·64	
−12, run 8	194·0	1·14	
Dust loss	61·5	0·36	
Feed	1,372·0	8·04	

The +12 product from Run No. 8 contained considerable unfiberized material. A test was made on this product, the results of which are shown in the following table of fibre recovered:—

Run No. 9—On fibre from initial crushing in jaw crusher and rolls to one inch.—The fibre from Run No. 2 was run over the shaking screen fitted with a 12×12 mesh screen, as in Run No. 8. The object of this run was to improve the grade. The following products were made:—

Product	Weight pounds	Per cent of heads	Test
Fibre, 1st collector	900	5·09	0—0·5—4·5—11·0 (fairly clean).
Dust, 2nd collector	5	0·03	
+12	376	2·12	
−12	746	4·22	
Dust loss	34	0·19	
Feed	2,061	11·65	0—0·5—4·7—10·8 (large per cent rock).

Run No. 10—On $-\frac{1}{16}''$ material from initial crushing.—The $-\frac{1}{16}''$ material from Run No. 2 was re-run over the shaking screen fitted with a 12×12 mesh screen to recover any fibre which passed through this screen. This run would not have been necessary if the square mesh screen had been used instead of the $\frac{1}{16}''$ slotted screen. The results were as follows:—

Product	Weight pounds	Per cent of heads	Test
Fibre, 1st collector	204	1·15	0—0—1·9—14·1 (clean).
Dust, 2nd collector	2	0·02	
+12	1,098	6·17	
−12	1,957	10·98	
Dust loss	49	0·28	
Feed	3,312	18·60	

Run No. 11—Re-run of fibre from first ball mill run.—The fibre from Run No. 5 was re-run over the shaking screen to remove any further rock and dust from it. The 12×12 mesh screen was used. The products from this run were as follows:—

Product	Weight pounds	Per cent of heads	Test
Fibre, 1st collector	475	2·75	0—0—4·1—11·9 (clean).
Dust, 2nd collector	6	0·03	
+12	21	0·12	
−12	417	2·41	
Dust loss	17	0·10	
Feed	936	5·41	0—0—3·2—12·8 (fairly clean).

Run No. 12—On $-\frac{1}{16}''$ material from first ball mill run.—The $-\frac{1}{16}''$ material from Run No. 5, was run over the shaking screen fitted with a 12 × 12 mesh screen to recover any coarse fibre which passed through the $-\frac{1}{16}''$ slotted screen. Had the square mesh screen been used in Run No. 5, this run would not have been necessary. The following products were obtained:—

Product	Weight pounds	Per cent of heads	Test
Fibre, 1st collector	134	0·77	0·—0—1·5—14·5 (clean).
Dust, 2nd collector	4	0·02	
+12	492	2·81	
−12	1,737	9·96	
Dust loss	1	0·01	
Feed	2,368	13·57	

Runs Nos. 13 and 14—Re-run of fibre from initial crushing.—These two runs were made on the fibre from Run No. 9, to remove any dust and rock remaining in it. Both runs were made over the shaking screen, using a 12 × 12 mesh screen. Run No. 13 did not remove sufficient of the rock, so the fibre was re-run over the screen a second time. The products were as follows:—

Product	Weight pounds	Per cent of heads	Test
Fibre, 1st collector run No. 14	366	2·07	0—1·8—5·6—8·6 (clean).
Dust, 2nd collector runs Nos. 13 and 14	11	0·06	
+12, run No. 13	114	0·64	
+12, run No. 14	21	0·12	
−12, run No. 13	257	1·45	
−12, run No. 14	114	0·65	
Dust loss, runs Nos. 13 and 14	17	0·10	
Feed	900	5·09	0—0·5—4·5—11·0 (fairly clean).
Fibre, 1st coll. run No. 13	509	....	0—0·9—4·9—10·2 (fairly clean).

The +12 material from Runs Nos. 13 and 14 contained a considerable amount of unfiberized material. These products should have been returned to milling operations. Test runs were made on these, the results of which are given in the table of fibre recovered.

Runs Nos. 15 and 16—On fibre from Run No. 14.—These two short runs were made on a portion of the fibre from Run No. 14, to determine if it would be possible to bring some of the fibre up to a high grade, and also to determine if it was possible to figure out fairly accurately what grades of fibre could be produced from a given lot of fibre.

The two runs were made by passing the fibre over a ¼″ shaking screen, the screen being used as a grader. Run No. 16 is on the +¼″ fibre from Run No. 15.

Products Run No. 15	Weight pounds	Per cent of heads	Test
+¼-in. fibre	6·00	0·36	0·5+5·0+5·2+5·3 (clean).
−¼-in. fibre, 2nd hopper	13·25	0·79	0·0+1·0+7·1+7·9 (clean).
−¼-in. fibre, 1st hopper	15·50	0·92	0·0+0·3+6·9+9·7 (clean).
Feed, run No. 15	34·75	2·07	0·0+1·8+5·6+8·6 (clean).

Products Run No. 16	Weight pounds	Per cent of heads	Test
	lbs.		
+¼-in. fibre	1·60	0·10	2·2+7·6+2·8+3·4 (clean).
−¼-in. fibre	4·40	0·26	0·0+3·8+5·6+6·6 (clean).
Feed, run No. 16	6·00	0·36	0·5+5·0+5·2+5·3 (clean).

It is here demonstrated that it is possible to produce a high grade fibre by grading. If in each of these runs the figures given in the fibre test on the products are multiplied by the weight of the product, and corresponding terms added, and the sum divided by the weight of the feed, it will be found to give very nearly the fibre test of the feed. Hence, we may conclude that it is possible to figure fairly accurately what grades can be produced from a certain fibre, and also that it is correct in figuring recovery, to sum up from the different fibre products the percentage of heads on the same testing screen.

The following table shows the different fibres recovered from all the different runs, etc. In the list will be noticed some +12 products. These are really middlings which would be returned to the circuit in regular mill work. Small tests have been conducted on these products, and whatever fibre was found possible to recover from them has been tabulated.

Product	Per cent of heads	Per cent fibre	Per cent re-covered	Testing				Per cent heads recov'd.			
				+2	+4	+10	−10	+2	+4	+10	−10
Fibre run 8......	....	....	1·05	......	......	2·8	13·2	......	......	0·184	0·866*
“ 10......	....	....	1·15	......	......	1·9	14·1	......	......	0·137	1·013
“ 11......	....	....	2·75	......	......	4·1	11·9	......	......	0·705	2·045*
“ 12......	....	....	0·77	......	......	1·5	14·5	......	......	0·072	0·698*
“ 14......	....	....	2·07	......	1·8	5·6	8·6	......	0·233	0·725	1·112
+12 run 8......	0·17	17·5	0·03	......	0·1	2·2	13·7	......	......	0·004	0·026*
“ 9......	2·12	12·9	0·27	......	0·3	3·6	12·1	......	0·005	0·061	0·204
“ 11......	0·12	37·9	0·05	......	0·2	5·8	10·0	......	0·001	0·018	0·031*
“ 13......	0·64	60·0	0·38	......	0·2	5·8	10·0	......	0·005	0·138	0·237
“ 14......	0·12	80·0	0·10	......	0·2	5·8	10·0	......	0·001	0·036	0·063
Total fibre.......	....	....	8·62	......	0·5	3·8	11·7	......	0·245	2·080	6·295

Note.—Products marked * are from ball milling.

A small sample of the feed to the first ball mill run, Run No. 3, was separated by screening into a number of different sizes. As much fibre as possible was picked out of the larger sizes, and sucked out of the smaller ones. The sizes were all mixed and ground in a small ball mill, and the product screened, picked, and sucked, as before. This milling, screening, picking, and sucking was continued until the rock was clean of fibre. The fibre obtained was tested in the usual manner, and the following table shows the results of this small test compared with what was obtained from the large runs:—

Product	Per cent of heads	Per cent fibre	Per cent re-covered	Testing				Per cent heads recover'd			
				+2	+4	+10	−10	+2	+4	+10	−10
Feed run 3........	68·24	12·87	8·78	......	0·1	0·9	15·0	......	0·055	0·494	8·231
Fibre from mill... (large run).	....	....	4·65	......	......	3·4	12·6	......	0·983	0·001	3·666

In this table of comparison, it is noticed that the small test recovered much more +4 than the large runs. This was to be expected, as in hand picking in the small test, a knife was used to cut off the fibre from the rock, and much unfiberized asbestos was put with the picked material.

It will also be noticed that the large runs give the largest amount of total fibre, +10. This would indicate that the Hardinge ball mill, with suction, is a very effective machine for the work upon which it was employed in the larger runs.

We are unable to make a comparison of the efficiency of the Hardinge ball mill with the Cyclone or Jumbo fiberizers, as no data are available on the work of the latter machines.

From the results given above, however, the mill operator may be able to draw some conclusions on the relative efficiency of the two types of mills.

Test No. 152

A shipment of ten tons of asbestos rock was received at the Ore Dressing and Metallurgical Laboratories, May 27, 1921, from the Black Lake Asbestos and Chrome Co., Ltd., Black Lake, Quebec.

The rock was of the cross fibre variety, representative of the Black Lake type of rock, and much harder than the slip fibre rock of the East Broughton deposits. As received it was composed of about three-inch material, with the fines removed, and as the fines from the primary crushing carry much more fibre than the coarse when reduced to this size, the percentage of fibre was, therefore, lower than the average mill feed, and the rock would also contain shorter fibre, the longer being freed by the primary crushing, and entering the fines.

The rock received represented the coarser material fed to the Cyclone and Jumbo fiberizers. It was desired that test work be conducted on this rock for the recovery of the fibre, and to obtain some data with respect to the crushing and liberation of the fibre in a Hardinge mill, as compared with the operations of the fiberizers in use for this purpose.

It is a fairly well established fact that the rock should be reduced finer than three inch before feeding to the fiberizers, and that this reduction can be accomplished in such crushing devices as jaw crushers, gyratories, rolls, hammer crushers, etc., without great injury to the fibre. The three-inch material received was, therefore, reduced to $1\frac{1}{2}''$ and $1''$ size before feeding to the Hardinge mill.

As no data were available as to the efficiency of the Cyclone and Jumbo fiberizers, no comparative results could be noted. The following tests, however, give the results obtained by the use of the Hardinge mill on material finer than $1\frac{1}{2}''$.

Test No. 1

Run No. 1.—Five tons of the asbestos rock was weighed, crushed in a jaw crusher to $1\frac{1}{2}''$, and sampled by an automatic Vezin sampler, the main flow passing over a Ferraris shaking screen fitted with a blank screen for the first half, and with a 12 mesh square screen for the second half, at the discharge end. One suction pipe into a main header to a collector was placed over the screen at the discharge end of the screen for the removal of the fibre. An exhauster was placed between the first collector and discharged into a second collector, from which a second exhauster discharged into the air. Any fibre not collected in the first collector was deposited in the second one. With this arrangement there was practically no loss of fibre. Any fibre deposited in the second collector was extremely fine, and of no value. The fibre deposited in the first collector was re-run over the shaking screen twice, to remove fines and pieces of rock, and unfiberized asbestos. These operations gave the following results:—

Product	Weight pounds	Per cent of heads	Test
Fibre	18	0·182	1·1—5·6—6·5—2·8
—12 (fines from screen)	666	6·729	
+12 (tailings from screen)	9,114	92·089	
Dust (2nd collector and loss)	99	1·000	
Feed	9,897	100·000	

Run No. 2—First ball mill run.—The +12 mesh material from run No. 1 was sampled by an automatic Vezin sampler and fed to a Hardinge mill at the rate of 2·5 tons per hour. The 4′ 6″ × 13″ mill was run at 35 r.p.m. and carried a load of 3,000 pounds of 4″ steel balls. The feed end of the mill was raised three inches from the horizontal. The discharge end was fitted with a suction pipe to the first collector, with a T pipe close to the discharge, permitting of the regular discharge, and also of the fibre being sucked over into the collectors. The fibre from the first collector was run over the shaking screen twice to remove the rock and dust. The following are the products obtained:—

Product	Weight pounds	Per cent of heads	Test
Fibre	29·0	0·296	0·0—2·4—4·4—9·2
—12 (from screen)	704·5	7·195	
+12 (tailings from mill and screen)	8,261·0	84·364	
Dust (2nd collector and loss)	23·0	0·234	
Feed	9,017·5	92·089	

Run No. 3—Second ball mill run.—The +12 mesh material from run No. 2 was sampled by an automatic Vezin sampler and fed back to the mill, which contained some rock from the preceding operation. The mill was run under the same conditions. The fibre from the first collector was run over the shaking screen twice to remove the rock and dust. In this mill run, the mill tailings were found to contain quite a quantity of fibre which had not been removed by the suction, so they were run over the shaking screen to get this fibre out, and the fibre itself was cleaned by passing it over the shaking screen once. At this stage of the test work it was observed that the bags that had contained the asbestos rock were covered inside with quite a quantity of long fibre. The bags were all turned inside out and the fibre removed. This fibre from the bags was cleaned up by passing it over the shaking screen twice. The result of all these operations follows:—

Product	Weight pounds	Per cent of heads	Test
Fibre (from mill suction)..................	20	0·206	0·0—2·9—3·2—9·9
Fibre (from mill tails).....................	24	0·248	1·4—5·5—6·2—2·9
Fibre (from bags)........................	4	0·041	5·1—3·3—3·5—4·1
—12 (from screen)........................	3,217	33·199	
+12 (from screen).........................	4,889	50·453	
Dust (2nd collector and loss)..............	21	0·217	
Feed..................................	8,175	84·364	

Run. No. 4—Third ball mill run.—The +12 material from run No. 3 was sampled by an automatic Vezin sampler and fed back to the mill, which contained some rock from the preceding operation. The mill was run under the same conditions. The fibre from the first collector was run over the shaking screen twice to remove the rock and dust. The tailings from the mill contained some fibre, and this fibre was removed by passing the tailings and the cleanout of the mill over the shaking screen. The fibre obtained in this way was cleaned up by passing it over the shaking screen twice. On the completion of this run the second collector was cleaned out along with the pipes leading to it, and 61·5 pounds of dust obtained. The dust produced in the work up to this point was 339 pounds. The third ball mill run gave the following:—

Product	Weight pounds	Per cent of heads	Test
Fibre (from mill suction)..................	4·5	0·047	0·0—2·6—4·1—9·3
Fibre (from mill tails and clean out).......	6·5	0·069	0·3—4·3—7·4—4·0
—12 (from screen)........................	2,113·0	22·354	
+12 (from screen)........................	2,449·0	25·909	
Dust (2nd collector and loss)..............	196·0	2·074	
Feed..................................	4,769·0	50·453	

The following table shows the amount of fibre obtained in all the runs of Test No. 1:—

FIBRE OBTAINED—TEST No. 1

Run No.	Per cent of heads	Test	Per cent of heads			
			+2	+4	+10	—10
Run No. 1..............................	·182	1·1—5·6—6·5—2·8	·012	·064	·074	·032
Run No. 2..............................	·296	0·0—2·4—4·4—9·2	·0	·044	·082	·170
Run No. 3 (mill suction)...............	·206	0·0—2·9—3·2—9·9	·0	·307	·041	·128
Run No. 3 (mill tails).................	·248	1·4—5·5—6·2—2·9	·022	·085	·096	·045
Run No. 3 (from bags)..................	·041	5·1—3·3—3·5—4·1	·013	·008	·009	·011
Run No. 4 (mill suction)...............	·047	0·1—2·6—4·1—9·3	·0	·008	·012	·027
Run No. 4 (mill tails).................	·069	0·3—4·3—7·4—4·0	·001	·019	·032	·017
Total fibre....................	1·089	0·7—3·9—5·1—6·3	·048	·265	·346	·430

Test No. 2

Run No. 1.—The balance of the shipment of asbestos rock (10,149 pounds) was crushed in a breaker to pass ¾″, and sampled by an automatic Vezin sampler, the main flow passing to the shaking screen, where the fibre was sucked out of it. This fibre was then cleaned by passing it over the shaking screen once. The following are the products:—

Product	Weight pounds	Per cent of heads	Test
Fibre	28·5	0·284	0·5—4·6—6·4—4·5
−12 (from screen)	1,213·0	12·104	
+12 (tailings from screen)	8,552·0	85·342	
Dust (2nd collector and loss)	227·5	2·270	
Feed	10,021	100·000	

Run No. 2—Ball mill run.—The +12 material from run No. 1 was split into two portions by shovelling. One of these portions was approximately four-fifths of the whole, and this was used for run No. 2. The larger portion was fed to the ball mill under the same conditions as the previous ball mill runs, except that no suction was used on the mill. The discharge from the mill was run over the shaking screen to remove the freed fibre. After the whole portion had been fed to the mill and the discharge from the mill put over the screen, the +12 from the screen was fed back to the mill and the mill tailings treated as before. This series of operations was repeated five times, when it was found that the +12 from the screen contained no appreciable amount of fibre. The fibre from the first collector was then cleaned up by passing it over the shaking screen once. This work gave the following results:—

Product	Weight pounds	Per cent of heads	Test
Fibre	204·5	2·558	
−12 (from screen)	2,316·0	28·971	
+12 (tailings from screen)	4,147·5	51·880	
Dust (2nd collector and loss)	154·5	1·933	
Feed	6,822·5	85·342	

The following is a summary of the fibre obtained in Test No. 2:—

FIBRE OBTAINED—TEST No. 2

Run No.	Per cent of heads	Test	Per cent of heads			
			+2	+4	+10	−10
Run No. 1	·284	0·5 4·6 6·4 4·5	·009	·082	·113	·080
Run No. 2	2·558	0·0 0·3 8·4 7·3	·0	·048	1·343	1·167
Total fibre	2·842	0·1 0·7 8·2 7·0	·009	·130	1·456	1·247

Test No. 3

This test was made on approximately one-fifth of the +12 from test No. 2, run No. 1. In calculating this test, it is necessary to consider that a run has already been made on the material used, so we must call the first actual run, run No. 2, and for run No. 1, take the results obtained in run No. 1, test No. 2.

Run No. 2.—The +12 material was crushed, sampled by an automatic Vezin sampler, and fed to the shaking Ferraris screen, the first half of the table being fitted with a 12 mesh square screen, and the second half with a ⅛″ slotted screen. The suction pipe to remove the fibre was placed over the end of the 12 mesh screen where it joined the ⅛″ screen. The +⅛″ product was crushed and fed back to the screen until everything passed ⅛″. The fibre produced was cleaned by passing it over the screen once. These operations gave:—

Product	Weight pounds	Per cent of heads	Test
Fibre	9	0·469	0·7—5·5—4·7—5·1
−⅛″+12 (from screen)	878	45·745	
−12 (from screen)	730	38·034	
Dust (2nd collector and loss)	21	1·094	
Feed	1,638	85·342	

Run No. 3.—This run was made on approximately one-eighth of the +12 material from run No. 2. The +12 was crushed and fed to the shaking screen fitted up as in run No. 2, the +12 being re-crushed and fed back until all passed through the 12 mesh screen. The fibre obtained was cleaned by passing over the screen once. The following are the products:—

Product	Weight pounds	Per cent of heads	Test
Fibre	0·25	0·102	0·0—0·4—4·0—11·6
−12 (from screen)	112·00	45·542	
Dust (2nd collector and loss)	0·25	0·101	
Feed	112·50	45·745	

The following is a summary of the fibre obtained from the different runs of test No. 3:—

FIBRE OBTAINED—TEST No. 3

Run No.	Per cent of heads	Test	Per cent of heads			
			+2	+4	+10	−10
Run No. 1	·284	0·5—4·6—6·4— 4·5	·009	·082	·113	·080
Run No. 2	·469	0·7—5·5—4·7— 5·1	·021	·161	·138	·149
Run No. 3	·102	0·0—0·4—4·0—11·6	....	·003	·025	·074
Total fibre	·855	0·6—4·6—5·1— 5·7	·030	·246	·276	·303

Test No. 4

This is a test on 51 pounds of the head sample of test No. 1, which was made to see how the heads of test No. 1 compared with the heads of test Nos. 2 and 3. The test was made in the same manner as test No. 3, the rock being crushed to ¾″, then to ⅛″, and then to 12 mesh, the fibre being removed during the crushing, and cleaning up after each stage of crushing had been completed.

Product	Weight pounds	Per cent of heads	Test
Fibre from crushing to ¾″	0·114	0·224	2·5—4·2—5·1— 4·2
" " ⅛″	0·178	0·349	3·4—5·3—3·2— 4·1
" " 12 mesh	0·024	0·047	0·0—0·0—5·8—10·2
—12 from crushing to ¾″	5·500	10·784	
" " ⅛″	23·500	46·079	
" " 12 mesh	19·750	38·725	
Dust (2nd collector and loss)	1·934	3·792	
Feed	51·000	100·000	

FIBRE OBTAINED—TEST No. 4

Product	Per cent of heads	Test	Per cent of heads			
			+2	+4	+10	—10
Fibre, crushing to ¾″	·224	2·5—4·2—5·1— 4·2	·035	·059	·071	·059
" " ⅛″	·349	3·4—5·3—3·2— 4·1	·074	·116	·070	·089
" " 12 mesh	·047	0·0—0·0—5·8—10·2	....	....	·018	·030
Total fibre	·620	....	·109	·175	·158	·178

After finishing test No. 4, the second collector and pipes leading to it were cleaned out and 209 pounds of dust obtained. This dust was from tests Nos. 2, 3 and 4. In these tests the total dust produced was 405 pounds.

CONCLUSIONS

An examination of the fibre produced in tests Nos. 3 and 4, shows that although the amounts of the different sizes recovered show a great difference, still the +2 and +4 added together in each test give practically the same figures, and the +2, +4 and +10 added in each test, are not a great deal apart.

Making a suitable allowance for test No. 4, being on such a small amount of rock, one can conclude that the feeds to tests Nos. 1, 2 and 3 are the same.

Test No. 3 was carried on in such a way as to give practically all the long fibre that it would be possible to extract from the rock, and the results of this test can be used as a standard to compare with tests Nos. 1 and 2.

On comparing test No. 1 with test No. 3, we see that test No. 1 recovered more fibre in all the sizes, and from this can conclude that the Hardinge mill operated with suction is a very suitable machine to treat the kind of asbestos rock submitted.

Comparing the results of test No. 2 with those of test No. 3, we find that the +2 and +4 recovered are both less in test No. 2 than in test No. 3, and that the +10 and −10 are both more. From this we can conclude that the Hardinge mill with suction is superior to the same mill without suction, and that without suction the result would seem to be to grind up the long fibre into shorter material.

The results of the test work show that the Hardinge mill operated under favourable conditions can be used for the crushing of asbestos rock of the "cross fibre" variety. It is not possible to give a comparison of efficiency over the fiberizers in use, as no data are available as to results from these particular machines.

The results also show that in order to obtain the greatest percentage of long fibre, stage crushing is necessary, with the removal of the freed fibre after each stage; that instead of taking off clean fiberized material by the regulation of the suction used, it is better to employ a strong suction that will lift free unfiberized material, re-clean, and if necessary, fiberize the suction product. In this manner the fibre is not broken up by the grinding action of coarse particles of rock.

(7)

THE CONCENTRATION OF THE LOWER GRADE COPPER-NICKEL ORES

C. S. Parsons

The present practice of smelting the copper-nickel ores of the Sudbury district, direct in the blast furnace or by mixing the green ore with a certain portion which had been heap roasted, has given very favourable results, in the past. This practice was especially adaptable to the higher grade ores when consideration is given to the favourable market conditions that have prevailed to within the last year or so.

In the treatment of the lower grade ores of the district, it is a question whether it would not be more economical to apply certain methods of concentration to make a product with a fairly high copper-nickel content prior to smelting operations. By the elimination of the gangue rock and a large portion of the pyrrhotite it was hoped to obtain a product with a copper-nickel content the equal of the blast furnace matte. Such methods of operation would probably eliminate the blast furnace practice in the treatment of the lower grade ores.

The investigations conducted on the lower grade ores were confined to the concentration of the ores from the property of the Burrows Refining Co., to obtain a product containing the metallic minerals, copper, nickel and iron, suited to the refining process of their company, who were carrying on investigations in the manufacture of certain alloy steels; and also, to the concentration of the lower grade ores from other mines of the district by preferential flotation, to determine if the copper-nickel values could not be concentrated into a high grade product, with a good recovery of the values.

The results of the experimental work are given under the following test numbers:—

Test No. 148

(Preliminary Report.)

Concentration tests made on nickel ore from the mines of the Burrows Refining Company, in the Sudbury district, Ontario.

Lots No. 1 and No. 2.—The ore which had been crushed to pass 20 mesh before being received at the laboratory, was badly oxidized, and had to be passed through rolls to break up the lumps of cemented particles.

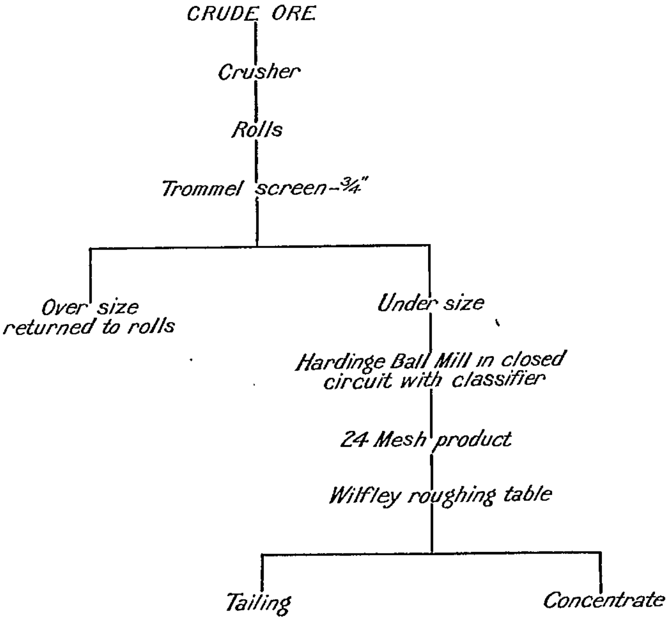


FIG. 3. Flow sheet, concentration test on nickel ore, Test 148, No. 1.

Test No. 1

(See flow sheet, Fig. 3)

Lots 1 and 2 were sampled and fed to a Wilfley table.

Lot 1—	
Weight	1,819 pounds
Analysis	Ni. 2·70 % : Cu. 0·25 %
Nickel content	49·1 pounds
Copper content	4·55 pounds
Lot 2—	
Weight	1,290 pounds
Analysis	Ni. 2·62 %
Nickel content	33·8 pounds
Lots 1 and 2 total—	
Weight	3,109 pounds
Analysis	Ni. 2·67 %
Nickel content	83·01 pounds

The ore was fed over a 24 mesh Callow screen, the −24 mesh material going direct to a Wilfley table without further sizing, the +24 mesh material to a storage tank for separate treatment later over the same table.

Results of tabling—

−24 mesh product—	
Table concentrate	−24 mesh
Weight	1,712 pounds
Analysis	3·60% Ni.
Content	61·63 pounds Ni.
Percentage of total nickel values	74·2
Table tailing	−24 mesh
Weight	499 pounds
Analysis	1·15% Ni.
Content	5·74 pounds Ni.
Percentage of total nickel values	6·9
+24 mesh product—	
Table concentrate	+24 mesh
Weight	150 pounds
Analysis	2·0% Ni.
Content	3·0 pounds Ni.
Percentage of total nickel values	3·6
Table tailing	+24 mesh
Weight	393·25 pounds.
Analysis	0·6% Ni.
Content	2·13 pounds Ni.
Percentage of total nickel values	2·6

Summary of the above results—

Product	Weight pounds	Analysis per cent Ni.	Content pounds Ni.	Percentage of total Ni. value
Concentrate	1,862	3·47	64·63	77·8
Tailing	853·75	0·92	7·87	9·5
Loss	395·25	2·67	10·51	12·7
Heads	3,109·00	2·67	83·01	100·0

This large loss was due to the heavy sulphide lying in the pumps and pipe lines, and also to slime loss from tailing. There was also approximately 0·25 per cent soluble nickel in the ore, a part of which went into solution in the mill water. The recovery of 77·8 per cent is very conservative, and it is probable that the actual recovery was over 80 per cent.

The ore tabled very well considering that the feed to the table was not sized. The capacity of the table was, however, low, the rate of feed being 241·5 pounds per hour. It is probable that a different system of riffling would increase the capacity considerably.

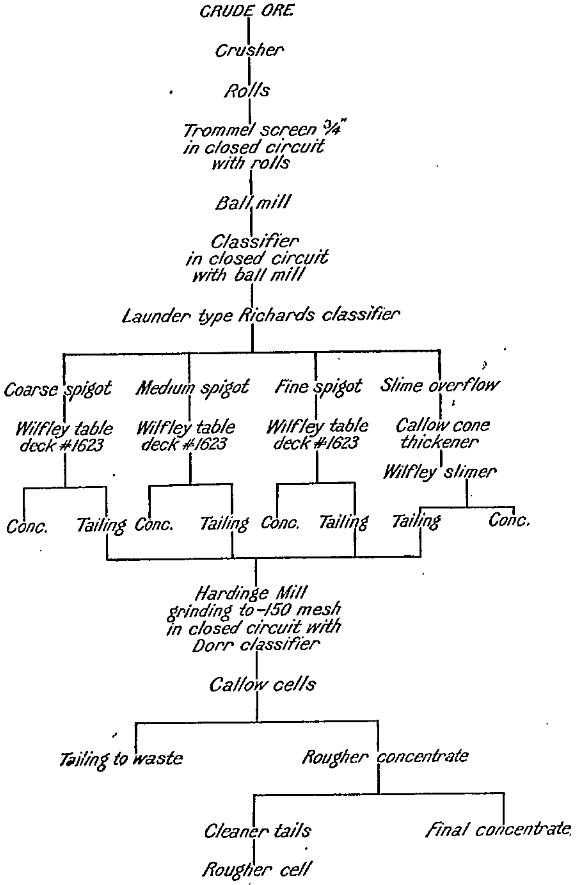


FIG. 4. Flow sheet, concentration test on nickel ore, Test 148, No. 2.

Test No. 2

(See flow sheet, Fig. 4.)

Lot No. 4.—This lot of ore was received in lump form. It was crushed in a jaw crusher to ¾", and then ground in a Hardinge ball mill to pass a 24 mesh Callow belt screen, the oversize being returned to the mill. The undersize from the screen was led to two Richards hydraulic launder type classifiers, where three products were made, a coarse spigot product, a fine spigot product, and an overflow slime. Each of these products was fed separately to a Wilfley table.

Weight of ore fed to mill	11,912 pounds
Analysis	2·50% Ni.
Content	297·8 pounds Ni.
Loss of weight during test	1,060 pounds
Analysis calculated of material lost	2·8% Ni.

The loss of weight was due chiefly to the heavy sulphide remaining in the ball mill and to a loss of slimes from the collecting tank, into which the overflow from the Richards classifier was caught and settled.

Screen Analyses of Feeds to Classifier and Table:

Mesh	Feed to classifier		Feed to table from first spigot		Feed to table from second spigot	
	Weight gms.	%	Weight gms.	%	Weight gms.	%
− 24 + 28	5	0·30	93	3·73	0·4	0·02
− 28 + 35	51	2·20	681	27·31	4·4	0·26
− 35 + 48	53	3·80	685	27·44	133·0	7·74
− 48 + 65	124	0·05	504	20·20	275·0	16·00
− 65 +100	165	12·00	312	12·50	427·0	24·85
−100 +150	134	9·70	95	3·81	266·0	15·47
−150 +200	437	31·90	65	2·60	261·0	15·18
−200	428	31·30	60	2·40	352·0	20·48

Concentration of coarse spigot product.—The product was fed to a Wilfley table with a No. 1623 deck. The table was fed at a rate which produced 0·21 tons of concentrate per hour. An approximate increase of 25 per cent in this capacity could be safely figured, which would give an average capacity of 0·26 tons of concentrate per hour.

Concentrate	
Weight	1031·0 pounds
Analysis	2·20% Ni.
Content	22·7 pounds Ni.
Tailing	
Weight	Not known
Analysis	0·55% Ni.

Concentration of fine spigot product.—This product was fed to a Wilfley table with a No. 1623 deck. The table was fed at a rate which produced 0·13 tons of concentrate per hour. The table, however, was operated at approximately 50 per cent of its capacity, and an estimated maximum capacity would be about 0·26 tons per hour of this material.

Concentrate	
Weight	1,326·5 pounds
Analysis	2·50% Ni.
Content	33·15 pounds Ni.
Tailing	
Weight	Not taken
Analysis	0·45% Ni.

Concentration of the overflow, or slime product from the classifier.—This product was fed to a Wilfley table with a No. 1623 deck, at a rate which produced 0·173 tons, or 346 pounds of concentrate per hour. A 10 per cent increase could be figured on this capacity, giving 0·187 tons of concentrate per hour, but this would be about the maximum capacity of the table.

Concentrate	
Weight	3,809·0 pounds
Analysis	3·50 % Ni.
Content	133·50 pounds Ni.
Tailing	
Weight	Not known
Analysis	1·50% Ni.

Tailings re-ground in pebble mill.—The whole table tailing was reground in a Hardinge mill to pass an 80 mesh Callow belt screen, and treated by flotation in Callow cells. The reagents used were soda ash and X-Y mixture.

Concentrate	
Weight	223 pounds
Analysis	15·5% Ni.
Content	33·45 pounds Ni.
Tailing	
Weight	Not known
Analysis	0·80% Ni.

SUMMARY OF PRODUCTS

Product	Weight pounds	Analysis % Ni.	Content pounds	Percentage of total nickel	
				Direct %	Accum. %
Concentrate No. 1	1,031·0	2·20	22·7	8·75	8·75
" " 2	1,326·5	2·50	33·2	12·80	21·55
" " 3	3,809·0	3·50	133·5	51·45	73·00
Callow Con. No. 4	223·0	15·50	34·5	13·30	86·30
Total Conc.	6,389·5	3·51	223·96	86·30	86·30
Tailings	4,462·5	0·80	35·70	13·70	....

This gives a recovery of 86·30 per cent of the nickel with a production of 0·589 tons of concentrate per ton of ore.

CONCLUSIONS

Lot 4, Test 2.—It will be noted that only 73 per cent of the nickel was saved by table concentration, and that this low recovery was due to the large loss in the tails from the tabling of the overflow product from the classifier. This overflow product contained over 60 per cent of the total nickel values, and the tailing from it analysed 1·5 per cent nickel, while the tailing from the spigot products contained only 0·55 per cent and 0·45 per cent nickel respectively. There is no doubt that a better saving could have been effected by operating the table on a lighter feed, but it is very evident that a further classification of the overflow product should be made before attempting to table it. In the proposed flow sheet, Fig. 4, a third classifier has been shown, which would divide this overflow product into a fine sand product, and a slime overflow product, for treatment on separate tables.

A better recovery was made in test No. 1, where the ore was tabled direct without either sizing or classification, but this result was effected by the slow rate of feed to the table. The table treated only 241·5 pounds per hour, or 0·120 tons, a rate of feed so slow as to be practically prohibitive.

Flow sheet, Fig. 5, gives the results that could be expected if classification was dispensed with.

It is also evident that flotation can be applied very successfully to the table tailing after re-grinding, and we have found by more recent tests that the recovery by flotation can be greatly increased.

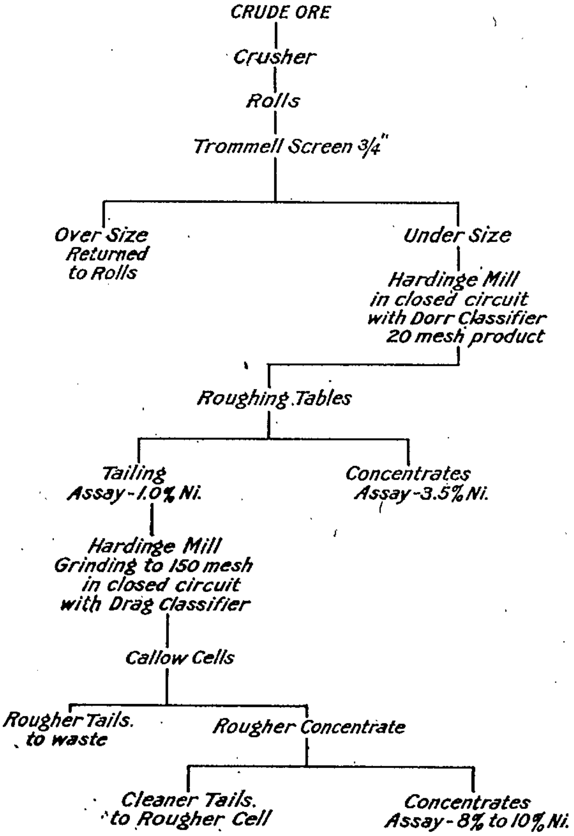


FIG. 5. Flow sheet, concentration test on nickel ore, Test 148, No. 3

Test No. 3A

(See flow sheet, Fig. 6.)

Lot No. 5.—The ore was received in lump form. It was crushed in a jaw crusher to ¾", and ground in a Hardinge mill to pass an 80 mesh Callow belt screen. The oversize was returned to the ball mill.

Head sample...................... 2·60 per cent Ni.

Part of this ore was floated in Callow cells, using soda ash and X-Y reagents.

Flotation concentrate	
Weight	359 pounds
Analysis	11·3% Ni.
Recovery	65·6%
Flotation tailing	
Weight	Not taken
Analysis	1·0% Ni.

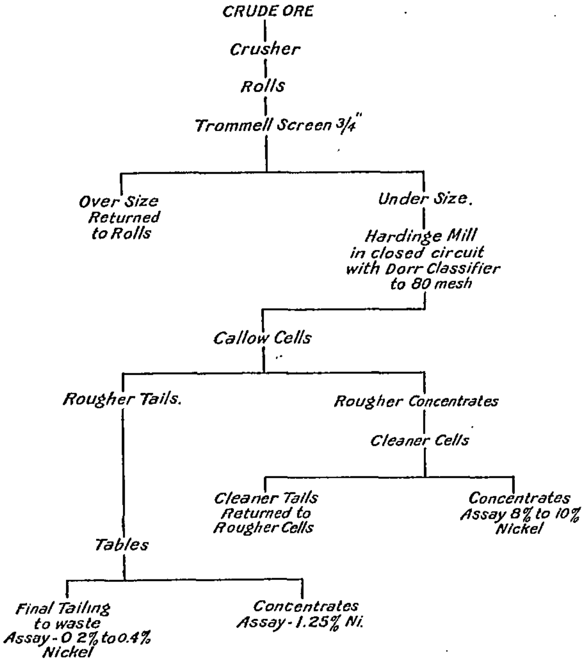


FIG. 6. Flow sheet, concentration test on nickel ore, Test 148, Nos. 3A and 3B.

The flotation tailing was retreated without classification on a Wilfley table, with a No. 1623 deck.

Table concentrate

Weight	1,198·5 pounds
Analysis	1·25% Ni.

Table tailing

Weight	Not taken
Analysis	0·20% Ni.

SUMMARY

Product	Weight pounds	% of total weight	Analysis % Ni.	Recovery %
Concentrate No. 1	359·0	....	11·30	....
" " 2	1,198·5	....	1·25	....
Total concentrate	1,557·7	....	3·50	98·0
Total tailing	....	....	0·20	2·0

Recovery of nickel in flotation concentrate	65·0%
Recovery of nickel in flotation and table concentrate	98·9%
Analysis of nickel in total concentrate	3·50%

Test No. 3B

(See flow sheet, Fig. 6)

A further portion of this ore was treated by flotation, using oil, instead of the X-Y reagents, and lime instead of soda ash. Reagents used: lime, and a gravity fuel oil, with a paraffine base.

Flotation concentrate	
Weight	710 pounds
Analysis	7·35% Ni.
Recovery	74·5% Ni. content
Flotation tailing	
Weight	Not taken
Analysis	0·90% Ni.

This tailing was tabled with the following results:—

Table concentrate	
Weight	950 pounds
Analysis	1·70% Ni.
Table tailing	
Weight	Not taken
Analysis	0·53% Ni.

SUMMARY

Product	Weight pounds	% of total weight	Analysis % Ni.	Recovery %
Concentrate No. 1	710	....	7·35	....
" 2	950	....	1·75	....
Total concentrate	1,660	....	4·12	91·3
Total tailing	....	....	0·53	8·7

Recovery of nickel in total concentrate	91·3%
Analysis of nickel in total concentrate	4·12%
Tons of concentrate produced per ton crude	0·58

CONCLUSIONS

The flotation of this ore in these first two tests was not very successful due to the difficulty in maintaining a froth on the cells. A series of small scale tests have since been made, and this difficulty was found to be due to the ore not being ground fine enough. It was found necessary to grind the ore fine for two reasons; first, to free the nickel-bearing mineral, and second, to produce enough rock and sulphide slimes to stabilize and toughen the froth in the flotation cells, so that it could be brought over.

The method of treatment, outlined in flow sheet, Fig. 6, and described in test No. 3A, seems to be the most practical method of producing the product required by the Burrows Refining Company.

It is evident that by the above process of treatment (see flow sheet Fig. 6), namely that of treating the crude ore by flotation, and then tabling the flotation tailing, great flexibility can be obtained. It would be possible to produce a concentrate which would contain any desired ratio of nickel to iron between the limits of 3·5 per cent nickel and 8·0 per cent nickel, and still average a fair recovery. The copper content of this concentrate would vary with the nickel content, and would range between 0·4 per cent cu. to 1·25 per cent cu.

Some Small Scale Tests by Flotation

The object of these tests was to produce a high grade nickel-copper concentrate by differential flotation.

Test No. 5

One thousand grams of ore was crushed in a small ball mill until 65 per cent passed a 200 mesh screen. The reagents used were 5 pounds soda ash per ton and 1 pound X-Y reagent per ton.

Product	Weight grams	Analysis	Recovery Ni	—
Concentrate	307	6·60%	83·5%	Total recovery, assuming that 60% of the Ni. content in middling could be recovered in the concentrate, 87·7%.
Middling	142	1·30%	....	
Tailing	538	0·45%	....	
Total	987			

Test No. 6

One thousand grams of ore was crushed in a small ball mill until 68 per cent passed a 200 mesh screen. The reagents used were 3 pounds of soda ash and 0·6 pounds of X-Y reagent per ton.

Product	Weight grams	Analysis Ni.	Recovery	—
Concentrate	151	11·60%	68·7%	Total recovery, assuming that 60% of the Ni. content in middling could be recovered in the concentrate, 80%.
Middling	201	2·20%	17·5%	
Tailing	660	0·53%	13·7%	
Total	1,012			

CONCLUSIONS

A second series of these tests will be run at a later date, to determine if it is possible by finer crushing to produce a higher grade nickel concentrate with a higher recovery.

These results are very encouraging, and show fairly conclusively that the Sudbury ores can be concentrated by flotation.

Test No. 155

A shipment of 100 pounds of copper-nickel ore was received at the Ore Dressing and Metallurgical Laboratories, October 10, 1921, from the International Nickel Company, Copper Cliff, Ont. The ore was from their No. 3 mine.

The object of the experimental work on this ore was to determine if it could be concentrated by flotation.

A number of small scale tests were made using the Alphabetical reagents of the General Engineering Company. The results of this work are contained in the two following tables.

Table No. I gives the results of tests Nos. 1, 2, 4 and 5. Recoveries indicated include the actual recovery in the concentrate made, together with 50 per cent of the copper-nickel values in the middling, which would be recovered in practice by returning them to the rougher cells. Table No. II gives the results of tests Nos. 6 and 8, in which the middlings were re-run without further grinding. With re-grinding of the middling product, there is no doubt that higher recoveries could be expected.

TABLE I

Test No.	Concentration Products	Weight grams	Analysis			Content		Percentage of values		Total Recoveries		
			Cu. per cent	Ni. per cent	Cu. + Ni. per cent	Cu. gms.	Ni. gms.	Cu.	Ni.	Cu. per cent	Ni. per cent	Cu. + Ni. per cent
1	Concentrate....	106	12·50	9·60	22·10	13·25	10·18	93·2	76·2	95·3	82·4	89·1
	Middling.......	147	0·40	1·13	......	0·59	1·66	4·1	12·4			
	Tailing.........	758	0·05	0·20	......	0·38	1·52	2·7	11·3			
2	Concentrate....	75	16·50	10·20	26·70	12·37	7·65	88·3	58·7	91·9	68·8	80·8
	Middling.......	155	0·65	1·70	.....	1·01	2·63	7·2	20·2			
	Tailing........	785	0·08	0·35	.....	0·63	2·75	4·5	21·1			
4	Concentrate....	95	13·66	9·75	23·41	12·98	9·26	93·6	69·8	95·4	78·3	87·1
	Middling.......	146	0·35	1·55	......	0·51	2·26	3·7	17·0			
	Tailing.........	758	0·05	0·23	......	0·38	1·74	2·7	13·1			
5	Concentrate....	170	7·40	6·20	13·60	12·58	10·54	92·0	80·0	93·9	84·5	89·2
	Middling.......	94	0·55	1·27	......	0·52	1·19	3·8	9·0			
	Tailing.........	730	0·08	0·20	......	0·58	1·46	4·2	11·0			

Reagents used—

Test No. 1—6 lbs. soda ash, 0·24 lb. X-Y per ton.
" 2—5 lbs. lime, 0·24 lb. X-Y per ton.
" 4—2 lbs. lime, 3 lbs. soda ash, 0·24 lb. X-Y per ton.
" 5—5 lb. soda ash, 0·24 lb. X-Y per ton.

TABLE II

Test No.	Concentration Products	Weight grams.	Analysis			Content		Percentage of values		
			Cu. per cent	Ni. per cent	Cu. + Ni. per cent	Cu. gms.	Ni. gms.	Cu.	Ni.	Cu. + Ni.
6	Concentrate.....	376	7·60	5·70	13·30	28·58	21·43	91·4	86·5	89·2
	Tailing..........	1,594	0·17	0·21	0·38	2·71	3·35	8·6	13·5	10·8
	Ore.............	....	1·56	1·24	2·80	31·29	24·78	100·0	100·0	100·0
8	Concentrate.....	292	9·10	7·50	16·60	26·57	21·90	96·7	83·4	90·2
	Tailing..........	1,675	0·055	0·26	0·31	0·92	4·35	3·3	16·6	9·8
	Ore.............	....	1·37	1·31	2·68	27·49	26·25	100·0	100·0	100·0

Reagents used—

Test No. 6—5 lbs. soda ash, 0·2 lb. X-Y per ton
Test No. 8—5 lbs. soda ash, 0·2 lb. X-Y per ton.

NOTE.—T.T. Reagent gives similar results.

Observations and procedure followed in conducting tests.—There is no difficulty in making a fairly high grade copper-nickel concentrate with a good recovery of the copper-nickel values in the ore. The Alphabetical reagents were used, being the most suitable for this class of ore. Further work will be done using other reagents, to determine whether similar, or better results can be obtained.

In using the Alphabetical reagents, the essential point is to carry an alkaline pulp, obtained by the addition of lime, soda ash, or caustic soda. Lime has a tendency to prevent some of the nickel from floating, but gives a higher grade product. Soda ash gives the best recoveries, but a lower grade product. A combination of the two seems to give the best results. The chalcopyrite floats with remarkable ease. With soda ash alone, without any other flotation reagent, a large proportion of it is floated.

No test work was done to determine the degree of fineness to which the ore should be ground to obtain the best results. For the above tests it was ground so that approximately 95 per cent passed a 200 mesh screen. The procedure was as follows: Ore crushed to 20 mesh; 1,000 grams were ground wet in a small ball mill, with the lime or soda ash required, for about 40 minutes; a rougher concentrate and tailing were first made; the rougher concentrate was re-cleaned, producing a concentrate and a middling. In tests Nos. 6 and 8, two 1,000 gram lots were used following the above procedure, in order to obtain sufficient middling for re-floating. The concentrate from re-floating the middling was put with the final concentrate, and the tailing with the rougher tailing, so as to make only two products, concentrate and tailing.

Test No. 160

A shipment of 100 pounds of copper-nickel ore was received at the Ore Dressing and Metallurgical Laboratories, October 20, 1921, and a second shipment, November 15, 1921, from the British America Nickel Corporation, Ltd., Nickelton, Ont.

The object of the experimental work on this ore was to determine if it could be concentrated by flotation.

A number of small scale tests were made using the Alphabetical reagents of the General Engineering Company.

The first shipment of ore received consisted of fines, and was not amenable to concentration by flotation methods. After a number of experiments had been tried, the cause of the failure of the nickel to float was found to be due to oxidation. It was found from previous tests, on other nickel ores, that it is necessary to have freshly broken unoxidized ore to obtain a good recovery of the nickel values by flotation. The nickel-bearing mineral oxidizes very rapidly, and when slightly oxidized does not float readily.

The second shipment consisted of coarse lump ore, recently mined, and was not subject to oxidation to the extent that was the case in the first shipment of fines. It was on this second shipment that the results given in the following table were obtained. This ore gave an analysis as follows:—

Copper	(Cu.)	0·47%
Nickel	(Ni.)	1·17%
Iron	(Fe.)	20·90%
Sulphur	(S.)	11·19%

The table following gives the results of tests Nos. 1 to 8. The recoveries indicated include the actual recovery in concentrate made, together with the assumption that 50 per cent of the copper-nickel values in the middling would be recovered in practice, by returning this product to the rougher cells. This assumption has been checked in former tests by retreating the middlings, and found to be a conservative estimate. With regrinding of the middling product, there is no doubt that higher recoveries could be expected.

For the first seven tests, the ore was ground so that approximately 90 per cent passed 200 mesh screen. In test No. 8, it was crushed to pass 100 mesh screen, and so that 67 per cent passed 200 mesh. A screen test on the feed of test No. 8 gave the following:—

Mesh	Weight grams	Per cent
+100	2·0	0·8
+150	14·0	6·0
+200	62·0	26·5
−200	156·0	66·7
Total	234·0	100·0

REAGENTS USED

Test No. 1—5 pounds per ton soda ash..............................Ratio of dilution
0·24 pound per ton X-Y mixture...................... 1 : 4

Test No. 2—5 pounds per ton soda ash..............................Ratio of dilution
0·2 pound per ton Thio Fizzan........................ 1 : 4

Test No. 3—5 pounds per ton soda ash..............................Ratio of dilution
0·24 pound per ton T.T. mixture...................... 1 : 4

Test No. 4—5 pounds per ton soda ash..............................Ratio of dilution
1 pound per ton No. 34 gravity fuel oil................. 1 : 4
0·2 pound per ton No. 5 pine oil........................

Test No. 5—5 pounds per ton soda ash..............................Ratio of dilution
0·24 pound per ton X-Y mixture...................... 1 : 8

Test No. 6—4 pounds per ton soda ash
1 pound per ton lime
0·2 pound per ton X-Y mixture

Test No. 8—5 pounds per ton soda ash
1 pound per ton No. 34 gravity fuel oil (G.W. Oil Co.)
and No. 5 pino oil (G.N.S.)

Results of Tests

Test No.	Concentration Products	Weight grams	Analysis			Content		Percentage of values			Total recoveries		
			Cu. %	Ni. %	Cu. & Ni. %	Cu. gms.	Ni. gms.	Cu.	Ni.	Cu. & Ni.	Cu. %	Ni. %	Cu. & Ni. %
1	Concentrate.........	155	2·6	6·2	8·8	4·03	9·61	87·4	82·6	84·0	90·0	86·8	87·8
	Middling............	159	0·15	0·62	......	0·24	0·98	5·2	8·4				
	Tailing..............	694	0·05	0·15		0·34	1·04	7·4	8·9				
2	Concentrate.........	175	2·30	5·75	8·05	4·03	10·06	89·3	84·7	86·0	90·7	87·0	88·0
	Middling............	118	0·10	0·45		0·12	0·53	2·7	4·5				
	Tailing..............	715	0·05	0·18		0·36	1·29	8·0	10·8				
3	Concentrate.........	194	2·15	5·26	7·41	4·17	10·20	89·7	87·1	82·4	90·9	81·6	84·1
	Middling............	82	0·15	0·57		0·12	0·47	2·6	3·6				
	Tailing..............	730	0·05	0·29		0·36	2·12	7·7	10·6				
4	Concentrate.........	220	1·90	4·80	6·70	4·18	10·56	91·5	88·5	89·3	92·3	90·3	90·8
	Middling............	144	0·05	0·30		0·07	0·43	1·5	3·6				
	Tailing..............	630	0·05	0·15		0·32	0·94	7·0	7·9				
5	Concentrate.........	64	3·05	7·12	10·17	1·95	4·56	83·7	76·6	78·6	86·3	80·0	81·8
	Middling............	69	0·17	0·56		0·12	0·39	5·1	6·5				
	Tailing..............	369	0·07	0·27		0·26	1·00	11·2	16·8				
6	Concentrate.........	66	6·65	13·36	20·01	4·39	8·82	82·1	73·0	75·7	84·9	79·6	81·2
	Middling............	122	0·25	1·33		0·30	1·62	5·6	13·4				
	Tailing..............	825	0·08	0·20		0·66	1·65	12·3	13·6				
8	Concentrate.........	168	2·70	5·83	8·53	4·53	9·97	93·2	80·4	84·0	94·6	85·5	88·1
	Middling............	205	0·07	0·61		0·14	1·25	2·9	10·3				
	Tailing..............	632	0·03	0·18		0·19	1·14	3·9	9·3				

SUMMARY

There is no apparent difficulty experienced in making a fairly high grade copper-nickel concentrate, with a good recovery of the copper-nickel values in the ore. The Alphabetical reagents were used being the most suitable for this class of ore.

In using these reagents, the essential point is to carry an alkaline pulp, obtained by the addition of lime, soda ash, or caustic soda. Lime has a tendency to prevent some of the nickel from floating, but gives the higher grade product. Soda ash gives the best recoveries, but a lower grade product. A combination of the two seems to give the best results. The chalcopyrite floats with remarkable ease. A large proportion of it is floated with soda ash alone, without any other flotation reagent.

(8)

THE CONCENTRATION OF GARNET FROM ITS ROCK

R. K. Carnochan

There are considerable areas in Canada in which the rock formation is favourable to the presence of garnet and other non-metallic minerals.

Under favourable conditions where deposits of these minerals are close to rail or water transportation and where they are of suitable grade and extent, they could be produced at a profit. Up to the present there has been no production of garnet, but within the last year there has been more or less activity in the way of exploration work on several deposits to determine their extent, grade, and the commercial possibilities of establishing an industry in this country.

Several shipments were received at the Ore Dressing and Metallurgical Laboratories. Two were from the vicinity of Bancroft, Ontario, and one from Depot Harbour, Ontario. The experimental work consisted of determining the garnet content, percentage of recoverable garnet, and methods of concentration and separation of the garnet from the gangue. On account of its favourable situation from a shipping standpoint for the disposal of the concentration products, the work of the shipment from Depot Harbour was more exhaustive, covering more detail than in the case of the other shipments.

Test No. 150 (Part 1)

A shipment of garnet rock weighing 95 pounds was received at the Ore Dressing and Metallurgical Laboratories, April 11, 1921, from Mr. Robt. A. Bryce, Toronto, Ont.

The rock came from the vicinity of Bancroft, Ont., and consisted of garnets of fair size in mica schist.

It was desired that tests be conducted on this rock to produce a high grade garnet concentrate, on which the owner wished to conduct abrasive tests.

A specimen was selected from the sample, and the balance was then crushed in a small jaw crusher to pass 20 mesh, and screened on 40 and 100 mesh screens, with the following results:—

	lbs.	p.c.
−20+ 40	28	30·4
−40+100	37·5	40·8
−100	26·25	28·5
Loss	0·25	0·3
Heads	92	100·0

A small portion was cut out of each of the three sizes and tabled on a small Wilfley table, with the following results:—

−20 +40	Concentrates	159 gms.	3·2% by wt. of heads
	Middlings	306 "	6·1% " "
	Tails	1,052 "	20·9% " "
	Loss	13 "	0·2% " "
	Feed	1,530 "	30·4% " "
−40 +100	Concentrates	127 gms.	2·8% by wt. of heads
	Middlings	97 "	2·2% " "
	Tails	1,590 "	35·3% " "
	Loss	21 "	0·5% " "
	Feed	1,835 "	40·8% " "
−100	Concentrates	82 gms.	1·7% by wt. of heads
	Middlings	90 "	1·8% " "
	Tails	1,147 "	23·0% " "
	Loss	100 "	2·0% " "
	Feed	1,419 "	28·5% " "

A larger portion was then cut out of each of the three sizes and concentrated on a large Wilfley table, with the following results:—

−20 +40	Concentrates	2 lbs.	2 ozs.	3·1%	by wt. of heads
	Tails	16 "	— "	23·4%	" "
	Loss	2 "	10 "	3·9%	" "
	Feed	20 "	12 "	30·4%	" "
−40 +100	Concentrates	2 lbs.	5 ozs.	3·3%	by wt. of heads
	Tails	25 "	8 "	36·5%	" "
	Loss		11 "	1·0%	" "
	Feed	28 "	8 "	40·8%	" "
−100	Concentrates	1 lbs.	— ozs.	1·5%	by wt. of heads
	Tails	12 "	— "	17·5%	" "
	Loss	6 "	9 "	9·5%	" "
	Feed	19 "	9 "	28·5%	" "

CONCLUSIONS

A clean separation has been made of the garnet from the gangue. The results of the above tests show that 7·9 per cent of the weight of the rock is recoverable as garnet concentrate, that is, for every 100 tons of rock treated 7·9 tons of garnet concentrates would be produced.

Tests were run on the magnetic separator and on the electrostatic separator, but a poor separation in each case was obtained. The results are not worthy of recording.

From the results obtained, a satisfactory method of dressing this rock would be, crushing wet to 10 mesh, or possibly finer, classification, and tabling of the classified material.

Test No. 150 (Part 2)

A shipment of garnet rock weighing 95·5 pounds was received at the Ore Dressing and Metallurgical Laboratories, May 7, 1921, from Mr. Robt. A. Bryce, Toronto, Ont.

The rock was from the vicinity of Bancroft, Ont., and consisted of garnets of fair size in mica schist.

It was desired that tests be conducted on this rock to produce a high grade garnet concentrate, on which the owner wished to conduct abrasive tests.

A specimen was selected from the sample, and the balance was then crushed in a small jaw crusher to pass 20 mesh, and screened on 40 and 100 mesh screens, with the following results:—

	lbs.	p.c.
−20+ 40	48	52·0
−40+100	29·5	32·0
−100	14·5	15·7
Loss	0·25	0·3
Heads	92·25	100·0

The −20 + 40, and the −40 +100, sizes were tabled separately on a large Wilfley table, and the −100 material on a small Wilfley table, with the following results:—

Size	Product	Weight pounds	Percentage by weight of heads
−20 +40	Concentrate	11·80	12·8
	Tails	33·00	35·7
	Loss	3·20	3·5
	Feed	48·00	52·0
−40 +100	Concentrate	7·74	8·4
	Tails	20·00	21·7
	Loss	1·76	1·9
	Feed	29·50	32·0
−100	Concentrate	2·64	2·9
	Tails	10·25	11·1
	Loss	1·61	1·7
	Feed	14·50	15·7

CONCLUSIONS

A clean separation has been made of the garnet from the gangue. The results of the above tests show that 24·1 per cent of the weight of the rock is recoverable as garnet concentrates, that is, for every 100 tons of rock treated, 24·1 tons of garnet concentrates would be produced.

From the results obtained, a satisfactory method of dressing this rock would be, crushing wet to 10 mesh, or possibly finer, classification, and tabling of the classified material.

Test No. 158

Source of shipment.—A shipment of 127 bags of garnet rock, weight 13,528 pounds, was received at the Ore Dressing and Metallurgical Laboratories, November 22, 1921, from a garnet deposit at Depot Harbour, Ont. The deposit is situated on railway ground on the harbour, with excellent shipping facilities for disposal of the products. The shipment was made by Robt. A. Bryce, Toronto.

Exploration work.—The deposit was stripped at certain intervals and trenched. Moiled samples were cut from the bottom of the trenches. Some twelve trenches were sampled in this manner, and each bag represented a sample over a certain width along the trench. The trenches sampled were marked 1 to 12, and starting at one end of a trench with sample A for a certain length, followed by sample B, and so on in succession along the trench and across the deposit until the end of the trench was reached. The stripping and trenching showed the width and extent of the deposit, and determined to some extent the workable area from which the rock could be extracted for the recovery of the garnet content.

Purpose of the shipment.—The samples as submitted were for the purpose of determining the garnet content of the deposit as far as exploration work was conducted in the way of stripping and trenching over the area; to determine the average content of recoverable garnet; and also for determining methods of recovering the garnet from the rock in grades suitable for the market.

Characteristics of the deposits.—The deposits cover considerable area, as shown on the plan, in places over 200 feet wide, separated by bodies of gneiss. They occur in the gneiss and are composed of garnets up to one-half inch in size in a gneissic mica schist. The rock showed that it contained approximately 15 per cent garnet, 55 per cent mica, and 30 per cent silica and other gangue minerals. From tests made for the owners, the garnet had high abrasive qualities, and was suitable for the trade.

Plan of deposits.—The accompanying plan shows the location of the deposits; their extent as far as exploration work has proven them; the stripping and trenches; those trenches which have been sampled for the determination of garnet content and for material for experimental work on methods of treatment for the recovery of the garnet; and also tables showing the percentage of recoverable garnet. (See Fig. 7.)

Examination of garnet used for the manufacture of garnet paper.—The greatest portion of the garnet consumption is used for this purpose. It was, therefore, necessary to obtain some idea as to the size of the particles used in the manufacture of the various grades of paper. Samples were secured which gave the following screen analyses:—

Grade No.	Per cent on mesh—Tyler standard screen Sc.						
	+20	+28	+35	+48	+65	+100	−100
2½	0·0	12·7	86·6	0·3	0·4	....	....
2	0·0	0·4	98·3	0·8	0·4	0·2	....
1½	0·0	0·0	52·2	46·1	1·4	0·3	....
1	0·0	0·0	0·0	83·5	16·3	0·1	0·1
½	0·0	0·0	0·0	3·0	87·5	9·1	0·4

This test showed that the largest percentage of garnet used for the manufacture of garnet paper was all through 20 mesh (Tyler Standard) and practically all on 100 mesh, and as far as this particular trade was concerned there was no advantage in providing a product coarser than 20 mesh, especially when a cleaner product could be obtained around this mesh, as any attached or included gangue in the garnet crystal would be released, and treatment on jigs could be eliminated, provided that the reduction of the rock to this mesh did not result in an excessive amount of fine garnet −100 mesh.

From data collected it is found that the grades of garnet produced are known to the trade as Nos. 5, 4½, 4, 3½, 3, 2½, 2, 1½, 1, ½, 0, 2/0, 3/0 and 7/0. This means a considerably wider range than that given in the above table, and probably from 3 mm. size to 200 mesh. The greatest consumption is no doubt between 20 and 200 mesh. An examination of the garnet crystals from this deposit shows that the crystals are more or less fractured, especially on the outside, and in order to produce individual particles containing no fractures, reduction to 20 mesh is necessary. There is, therefore, no particular advantage in making a jig product, as this product would probably have to be crushed in order to obtain particles free from fractures.

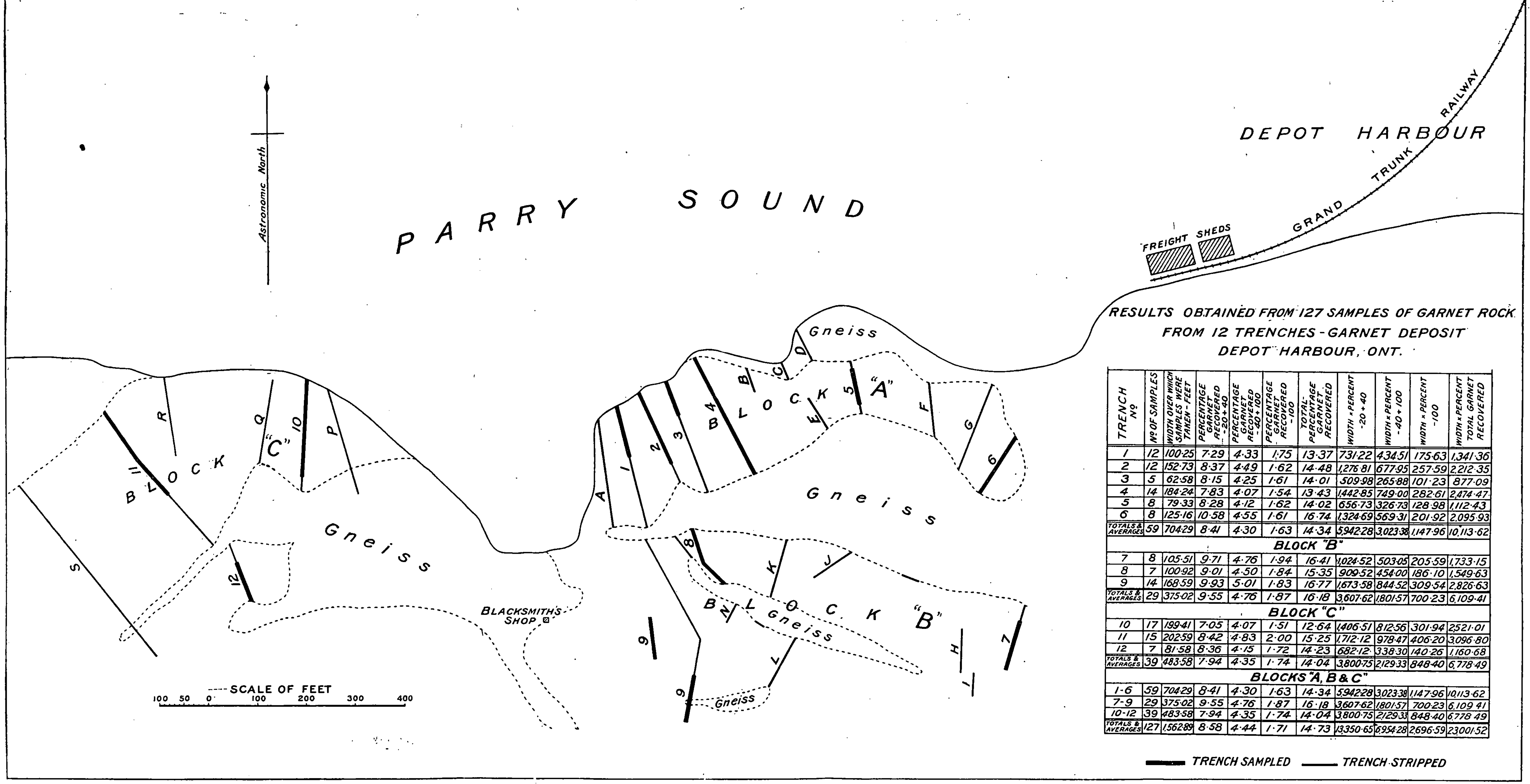
Examination of garnet shipped from the trade.—Two samples of garnet were secured, one of coarse material and the other of finer material. Screen analyses were made on both these products. Tyler standard screens were used.

Mesh	Coarse sample	Fine sample
	%	%
+6	0·0	....
+8	7·1	....
+10	39·3	....
+14	30·7	....
+20	14·6	....
+28	4·8	....
+35	2·8	....
+48	0·4	0·0
+65	0·1	1·6
+100	0·1	35·3
+150	0·1	32·2
+200	....	20·7
−200	....	10·2

The screen tests on these two products show that coarser material than 20 mesh is shipped for the trade, and material finer than 100 mesh is used. It may be that the excess of coarse garnet is ground to the required mesh after being concentrated as coarse as possible on jigs. An examination of the coarser particles showed them to contain included gangue in the fractures. The finer material may be used for the manufacture of the finer grades of garnet paper. Whether it will be profitable to save the fine garnet is a matter that will be determined in the milling practice. The table duty for the recovery of the fines is low, so that it may be more profitable to increase the rock tonnage to the mill and recover coarse garnet, eliminating the fines with the tailings.

Procedure followed for determining percentage of recoverable garnet.—Each bag containing the sample was weighed and the contents crushed in a small jaw crusher to about one-half inch, and a sample cut out by repeated passes through a Jones riffled sampler until a sample of from five to ten pounds was obtained for the small test. The same number of passes was given each sample, so that the remaining large portion would bear the same relation to the whole as before the samples were taken. This small sample was then reduced to 20 mesh by stage crushing in a small set of rolls, so as to make as little fines as possible.

The sample through 20 mesh was sized on 40 and 100 mesh, giving three sizes, −20+40, −40 +100 and −100. Each of the three sizes was run separately over a small laboratory type Wilfley concentrator, making a concentrate, a middling, and a tailing. The middling product was re-run in each case, making a concentrate and tailing. The concentrate was



RESULTS OBTAINED FROM 127 SAMPLES OF GARNET ROCK FROM 12 TRENCHES - GARNET DEPOSIT DEPOT HARBOUR, ONT.

TRENCH Nº	Nº OF SAMPLES	WIDTH OVER WHICH SAMPLES WERE TAKEN - FEET	PERCENTAGE GARNET RECOVERED -20+40	PERCENTAGE GARNET RECOVERED -40+100	PERCENTAGE GARNET RECOVERED -100	TOTAL-PERCENTAGE GARNET RECOVERED	WIDTH × PERCENT -20+40	WIDTH × PERCENT -40+100	WIDTH × PERCENT -100	WIDTH × PERCENT TOTAL GARNET RECOVERED
1	12	100·25	7·29	4·33	1·75	13·37	731·22	434·51	175·63	1,341·36
2	12	152·73	8·37	4·49	1·62	14·48	1,276·81	677·95	257·59	2,212·35
3	5	62·58	8·15	4·25	1·61	14·01	509·98	265·88	101·23	877·09
4	14	184·24	7·83	4·07	1·54	13·43	1,442·85	749·00	282·61	2,474·47
5	8	79·33	8·28	4·12	1·62	14·02	656·73	326·73	128·98	1,112·43
6	8	125·16	10·58	4·55	1·61	16·74	1,324·69	569·31	201·92	2,095·93
TOTALS & AVERAGES	59	704·29	8·41	4·30	1·63	14·34	5,942·28	3,023·38	1,147·96	10,113·62
BLOCK "B"										
7	8	105·51	9·71	4·76	1·94	16·41	1,024·52	503·05	205·59	1,733·15
8	7	100·92	9·01	4·50	1·84	15·35	909·52	454·00	186·10	1,549·63
9	14	168·59	9·93	5·01	1·83	16·77	1,673·58	844·52	309·54	2,826·63
TOTALS & AVERAGES	29	375·02	9·55	4·76	1·87	16·18	3,607·62	1,801·57	700·23	6,109·41
BLOCK "C"										
10	17	199·41	7·05	4·07	1·51	12·64	1,406·51	812·56	301·94	2,521·01
11	15	202·59	8·42	4·83	2·00	15·25	1,712·12	978·47	406·20	3,096·80
12	7	81·58	8·36	4·15	1·72	14·23	682·12	338·30	140·26	1,160·68
TOTALS & AVERAGES	39	483·58	7·94	4·35	1·74	14·04	3,800·75	2,129·33	848·40	6,778·49
BLOCKS "A, B & C"										
1-6	59	704·29	8·41	4·30	1·63	14·34	5,942·28	3,023·38	1,147·96	10,113·62
7-9	29	375·02	9·55	4·76	1·87	16·18	3,607·62	1,801·57	700·23	6,109·41
10-12	39	483·58	7·94	4·35	1·74	14·04	3,800·75	2,129·33	848·40	6,778·49
TOTALS & AVERAGES	127	1,562·89	8·58	4·44	1·71	14·73	13,350·65	6,954·28	2696·59	23001·52

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Fig. 7. Plan of Garnet deposit, Parry Island, Parry Sound district, Ont.

added to the concentrate from the first pass over the table, and the tailing to the tailing from the first pass, so that only a final concentrate and tailing resulted from the operations. In this manner the percentage of recoverable garnet from each size was obtained, as well as the total percentage. The results are given in tables following under the heading "Results of small scale test work on individual samples."

After completion of the above small scale work, each concentrate— −20+40, −40+100 and −100—from every sample was cut once in a small Jones riffled sampler, one half was put back in the respective sample containers, and the other half was combined, making up three concentrate products, −20+40, −40+100 and −100, each representing a sample of one of the three sizes from all the samples submitted. Each of these three sizes was run over a magnetic separator to determine the percentage of gangue still remaining in the various sizes so that a correction could be made on the results of the laboratory table work. This is given under the heading "Magnetic Separation Tests on laboratory concentrator products."

Results of Small Scale Test Work on Individual Samples:—

Results obtained from 127 moiled samples of Garnet Rock from 12 trenches: Garnet Deposit, Depot Harbour

Trench No.	Sample No.	Width of sample, feet	Percentage garnet recovered				Percentage × width			
			−20+40	−40+100	−100	Total	−20+40	−40+100	−100	Total
1.........	A	14·00	6·95	5·29	2·22	14·46	97·30	74·06	31·08	202 44
	B	8·67	7·59	4·03	1·57	13·19	65·81	34·94	13·61	114·36
	C	6·00	7·60	4·06	1·60	13·26	45·60	24·36	9·60	79·56
	D	5·33	7·32	4·40	1·48	13·20	39·02	23·45	7·89	70·36
	E	8·42	8·76	4·66	1·97	15·39	73·76	39·24	16·58	129·58
	F	5·00	6·73	3·43	1·43	11·59	33·65	17·15	7·15	57·95
	G	11·00	6·86	4·59	1·46	12·91	75·46	50·49	16·06	142·01
	H	8·00	6·79	3·61	1·31	11·71	54·32	28·88	10·48	93·68
	I	8·58	7·86	3·33	1·65	12·84	67·44	28·57	14·16	110·17
	J	9·25	6·44	4·39	1·58	12·41	59·58	40·59	14·62	114·79
	K	6·00	6·78	4·68	2·10	13·56	40·68	28·08	12·60	81·36
	L	10·00	7·86	4·47	2·18	14·51	78·60	44·70	21·80	145·10
Totals and averages	12	100·25	7·29	4·33	1·75	13·37	731·22	434·51	175·63	1,341·36
2.........	A	14·00	8·18	4·21	1·62	14·01	114·52	58·94	22·68	196·14
	B	19·00	9·06	3·99	1·66	14·71	172·14	75·81	31·54	279·49
	C	12·33	7·20	3·79	1·41	12·40	88·78	46·73	17·38	152·89
	D	12·75	7·37	4·67	1·38	13·42	93·96	59·54	17·60	171·10
	E	13·33	7·95	3·70	1·39	13·04	105·97	49·32	18·53	173·82
	F	12·33	6·52	4·51	1·21	12·24	80·39	55·61	14·92	150·92
	G	12·50	9·82	3·67	1·71	15·20	122·75	45·87	21·38	190·00
	H	12·67	6·68	3·75	1·50	11·93	84·64	47·51	19·00	151·15
	I	12·33	11·34	4·42	2·01	17·77	139·82	54·50	24·78	219·10
	J	12·83	8·22	5·76	1·88	15·86	105·46	73·90	24·12	203·48
	K	12·83	8·98	5·76	2·00	16·74	115·21	73·90	25·66	214·77
	L	5·83	9·12	62·3	3·43	18·78	53·17	36·32	20·00	109·49
Totals and averages	12	152·73	8·37	4·49	1·62	14·48	1,276·81	677·95	257·59	2,212·35
3.........	A	8·01	8·01	7·28	2·34	17·63	64·08	58·24	18·72	141·04
	B	9·00	9·98	5·91	2·24	18·13	89·82	53·19	20·16	163·17
	C	14·00	10·13	4·29	1·39	15·81	141·82	60·06	19·46	221·34
	D	15·58	7·56	3·06	1·51	12·13	117·78	47·67	23·53	188·98
	E	16·00	6·03	2·92	1·21	10·16	96·48	46·72	19·36	162·56
Totals and averages	5	62·58	8·15	4·25	1·61	14·01	509·88	265·88	101·23	877·09
4.........	A	10·42	7·03	4·26	1·47	12·76	73·25	44·39	15·32	132·96
	B	15·92	8·24	3·33	1·45	13·02	131·18	53·01	23·08	207·28
	C	15·83	7·30	4·75	1·85	13·90	115·56	75·19	29·29	220·04
	D	17·00	10·79	4·47	1·68	16·94	183·43	75·99	28·56	287·98
	E	15·00	7·06	4·07	1·16	12·29	105·90	61·05	17·40	184·35
	F	15·50	6·19	3·34	1·42	10·95	95·95	51·77	22·01	169·73
	G	15·83	6·29	3·25	1·58	11·12	99·57	51·45	25·01	176·03
	H	14·00	7·42	4·26	1·73	13·41	103·88	59·64	24·22	187·74
	I	12·58	5·95	5·16	1·46	12·57	74·85	64·91	18·37	158·18
	J	12·83	8·33	4·81	1·68	14·82	106·87	61·71	21·55	190·14
	K	11·83	7·93	3·85	1·30	13·08	93·81	45·55	15·38	154·73
	L	8·50	9·67	3·78	1·34	14·79	82·20	32·12	11·39	125·71
	M	13·50	9·60	3·94	1·65	15·19	129·59	53·19	22·28	205·07
	N	5·50	8·51	3·46	1·59	13·56	46·81	19·03	8·75	74·58
Totals and averages	14	184·24	7·83	4·07	1·54	13·43	1,442·85	749·00	282·61	2,474·47

Results obtained from 127 moiled samples of Garnet Rock from 12 trenches: Garnet Deposit, Depot Harbour—Continued.

Trench No.	Sample No.	Width of sample, feet	Percentage garnet recovered				Percentage × width			
			−20+40	−40+100	−100	Total	−20+40	−40+100	−100	Total
5.........	A	11·00	7·10	2·55	1·28	10·93	78·10	28·05	14·08	120·23
	B	9·67	10·84	4·05	1·61	16·50	104·82	39·16	15·57	159·55
	C	7·92	7·61	4·27	1·80	13·68	60·27	33·82	14·26	108·35
	D	10·25	9·44	5·13	1·84	16·41	96·76	52·58	18·86	168·20
	E	9·83	8·49	3·72	1·40	13·61	83·46	36·57	13·76	133·79
	F	10·83	8·39	3·49	1·64	13·52	90·86	37·80	17·76	146·42
	G	13·50	7·21	5·27	1·73	14·21	97·33	71·15	23·36	191·84
	H	6·33	7·13	4·36	1·79	13·28	45·13	27·60	11·33	84·06
Totals and averages	8	79·33	8·28	4·12	1·62	14·02	656·73	326·73	128·98	1,112·43
6.........	A	15·25	8·32	4·24	1·90	14·46	126·86	64·65	28·98	220·51
	B	13·83	8·43	8·55	0·98	17·96	116·59	118·25	13·55	248·39
	C	15·50	6·20	3·52	1·24	10·96	96·10	54·56	19·22	169·88
	D	15·00	7·62	2·98	1·17	11·77	114·30	44·70	17·55	176·55
	E	15·25	6·18	3·13	1·23	10·54	94·24	47·73	18·76	160·74
	F	17·83	12·09	4·70	1·08	17·87	215·56	83·80	19·26	318·62
	G	17·00	17·11	2·47	2·77	22·35	290·87	42·00	47·09	379·95
	H	15·50	17·43	7·33	2·42	27·18	270·17	113·62	37·51	421·29
Totals and averages	8	125·16	10·58	4·55	1·61	16·74	1,324·69	569·31	201·92	2,095·93
7.........	A	12·42	10·24	4·76	1·85	16·85	127·18	59·12	22·98	209·28
	B	10·00	10·50	4·82	2·12	17·44	105·00	48·20	21·20	174·40
	C	11·42	11·44	5·29	1·59	18·32	130·64	60·41	18·16	209·21
	D	13·00	10·12	4·29	1·61	16·02	131·56	55·77	20·93	208·26
	E	13·50	9·97	4·64	1·98	16·59	134·60	62·64	26·73	223·96
	F	12·50	8·62	4·70	2·30	15·62	107·75	58·75	28·75	195·25
	G	15·67	8·96	5·45	2·28	16·69	140·40	85·40	35·73	261·53
	H	17·00	8·67	4·28	1·83	14·78	147·39	72·76	31·11	251·26
Totals and averages	8	105·51	9·71	4·76	1·94	16·41	1,024·52	503·05	205·59	1,733·15
8.........	A	16·42	9·13	4·10	1·51	14·74	149·90	67·32	24·79	242·02
	B	13·00	8·89	5·29	1·95	16·13	115·56	68·77	25·35	209·69
	C	13·67	8·54	4·03	1·68	14·25	116·74	55·09	22·97	194·80
	D	13·17	11·18	4·69	2·18	18·05	137·24	71·77	28·71	237·72
	E	15·58	8·25	4·10	1·83	14·18	128·54	63·88	28·51	220·92
	F	16·33	7·76	3·67	1·64	13·07	126·72	59·93	26·79	213·43
	G	15·75	8·56	4·27	1·84	14·67	134·82	67·24	28·98	231·05
Totals and averages	7	100·92	9·01	4·50	1·84	15·35	909·56	454·00	186·10	1,549·63
9.........	A	13·00	11·59	5·62	1·52	18·73	150·67	73·06	19·76	243·49
	B	15·83	11·05	6·07	1·89	19·01	174·92	96·09	29·92	300·93
	C	13·17	9·77	5·19	1·98	16·94	128·67	68·35	26·08	223·10
	D	15·42	8·93	4·37	1·91	15·21	137·70	67·39	29·45	234·54
	E	14·00	8·51	5·22	1·68	15·41	119·14	73·08	23·52	215·74
	F	10·50	9·17	4·71	1·99	15·87	96·29	49·46	20·90	166·64
	G	10·67	9·13	4·70	2·04	15·87	97·42	50·15	21·77	169·33
	H	10·42	11·44	4·94	2·23	18·61	119·20	51·47	23·24	193·92
	I	9·33	8·68	4·96	2·00	15·64	80·98	46·28	18·66	145·92
	J	12·25	10·40	3·58	1·62	15·60	127·40	43·86	19·85	191·10
	K	11·00	10·05	4·85	1·91	16·81	110·55	53·35	21·01	184·91
	L	11·58	8·60	5·09	1·77	15·46	99·59	58·94	20·50	179·03
	M	9·67	10·48	5·08	1·68	17·24	101·34	49·12	16·25	166·71
	N	11·75	11·04	5·44	1·50	17·98	129·71	63·92	17·63	211·26
Totals and averages	14	168·59	9·93	5·01	1·83	16·77	1,673·58	844·52	308·54	2,866·63

Results obtained from 127 moiled samples of Garnet Rock from 12 trenches: Garnet Deposit, Depot Harbour—Continued.

Trench No.	Sample No.	Width of sample, feet	Percentage garnet recovered				Percentage × width			
			−20+40	−40+100	−100	Total	−20+40	−40+100	−100	Total
10........	A	7·00	8·16	3·91	1·83	13·90	57·11	27·37	12·81	97·30
	B	11·00	7·18	3·54	1·39	12·11	78·98	38·94	15·29	133·21
	C	9·50	6·67	3·16	1·35	11·18	63·37	30·02	12·83	106·21
	D	12·00	5·38	3·82	1·13	10·33	64·56	45·84	13·56	123·96
	E	11·83	6·78	3·67	1·12	11·57	80·21	43·42	13·25	136·87
	F	10·00	6·46	4·24	1·55	12·25	64·60	42·40	15·50	122·50
	G	10·33	7·75	4·18	1·53	13·46	80·06	43·18	15·80	139·04
	H	9·33	6·68	3·88	1·60	12·16	62·32	36·20	14·93	113·45
	I	11·00	4·87	5·16	1·88	11·91	53·57	56·76	20·68	132·11
	J	10·00	5·54	4·24	1·52	11·30	55·40	42·40	15·20	113·00
	K	12·67	9·42	4·92	1·58	15·92	119·35	62·34	20·02	201·70
	L	14·42	7·15	3·74	1·27	12·16	102·89	53·93	18·31	175·35
	M	12·00	8·40	4·56	1·58	14·54	100·80	54·72	18·96	174·48
	N	13·58	7·70	4·68	2·23	14·61	104·57	63·55	30·28	198·40
	O	14·00	5·88	3·69	1·55	11·12	82·32	51·66	21·70	155·68
	P	14·00	7·35	3·26	1·11	11·72	102·90	45·63	15·54	164·08
	Q	16·75	7·97	4·43	1·55	13·95	133·50	74·20	25·96	233·66
Totals and averages	17	199·41	7·05	4·07	1·51	12·64	1,406·51	812·56	301·94	2,521·01
11........	A	10·17	8·15	5·05	1·70	14·90	82·89	51·35	17·28	151·53
	B	13·00	9·25	4·55	1·94	15·74	120·25	59·15	25·22	204·62
	C	14·00	8·33	4·28	1·94	14·55	116·62	59·92	27·16	203·70
	D	13·42	8·44	5·19	2·23	15·86	113·26	69·65	29·93	212·84
	E	14·00	9·70	6·26	2·13	18·09	135·80	87·64	29·82	253·26
	F	15·00	10·13	6·75	3·15	20·03	151·95	101·25	47·25	300·45
	G	15·50	8·21	5·88	2·20	16·29	127·26	91·14	34·10	252·50
	H	13·83	8·00	4·45	2·44	14·89	110·64	61·54	33·75	205·93
	I	13·42	8·06	4·24	1·86	14·16	108·17	56·90	24·96	190·03
	J	13·25	7·08	4·78	2·17	14·03	93·81	63·34	28·75	185·90
	K	13·00	9·12	4·36	1·62	15·10	118·56	56·68	21·06	196·30
	L	12·67	8·68	4·29	1·46	14·43	109·98	54·35	18·50	182·83
	M	13·00	7·69	4·38	1·82	13·89	99·97	56·94	23·66	180·57
	N	12·75	8·53	4·67	1·69	14·89	108·76	59·54	21·55	189·85
	O	15·58	7·33	3·15	1·49	11·97	114·20	49·08	23·21	186·49
Totals and averages	15	202·59	8·42	4·83	2·00	15·25	1,712·12	978·47	406·20	3,096·80
12.....	A	14·00	9·68	4·35	1·49	15·52	135·52	60·90	20·86	217·28
	B	12·00	10·41	4·92	1·69	17·02	124·92	59·04	20·28	204·24
	C	9·00	5·44	4·31	2·25	12·00	48·96	38·79	20·25	108·00
	D	10·00	7·34	3·55	1·08	11·97	73·40	35·50	10·80	119·70
	E	12·00	7·46	4·00	1·48	12·94	89·52	48·00	17·76	155·28
	F	11·58	6·88	3·66	1·83	12·37	79·67	42·38	21·19	143·24
	G	13·00	10·01	4·13	2·24	16·38	130·13	53·59	29·12	212·94
Totals and averages	7	81·58	8·36	4·15	1·72	14·23	682·12	338·30	140·26	1,160·68

Magnetic separation tests on laboratory concentration products.—Three samples of the garnet concentrates from the small tests, representing the three sizes of concentrates produced, were obtained by cutting each sample into two portions, one of which was returned to the sample envelope from which it was taken, and the other used to make up the sample for the magnetic separation tests. In this manner a sample of 32 pounds representing −20+40 concentrate; a sample of 16 pounds representing −40 +100 concentrate; and a sample of 6·5 pounds representing the −100 concentrate was obtained. Each of these sizes was run through a magnetic separator, making a finished garnet product and a tailing. The results are given in the following table:—

Material	Weight		Magnetic product		Non-magnetic product		Per cent by weight final concentrate
	Pounds	Per cent	Pounds	Per cent	Pounds	Per cent	
−20 +40	32·0	58·7	26·0	81·3	6·0	18·7	59·1
−40 +100	16·0	29·4	13·0	81·3	3·0	18·7	29·5
−100	6·5	11·9	5·0	76·9	1·5	23·1	11·4
Totals and averages	54·5	100·0	44·0	80·7	10·5	19·3	100·0

This test showed that the concentrate produced on the small table still contained some gangue, and that this gangue could be eliminated by the use of magnetic separators, removing the garnet from the gangue. The gangue material remaining in the table concentrate consists of material approaching the specific gravity of garnet. In the −20+40 concentrate, 81·3 per cent is clean garnet concentrate; in the −40+100 concentrate, also 81·3 per cent; and in the −100 concentrate, 76·9 per cent. The percentage of garnet recovered as recorded in the small table tests, will have to be corrected to give the actual recovery as finished concentrate. From the above table it will be seen that the finished concentrate consists of 59·1 per cent −20+40 mesh, 29·5 per cent −40+100 mesh, 11·4 per cent −100 mesh. It is also seen that table concentration gives as clean a product on the −40+100 material as on the −20+40 material.

The following table gives the actual recovery of garnet in finished concentrate, as determined by the magnetic concentration tests on the table concentrate. In each case the recoveries indicated in former tables are multiplied by 81·3 per cent in the −20+40; by 81·3 per cent in the −40+100; and by 76·9 per cent in the −100. This is done for the twelve trenches.

Table giving Correction Results of Small Scale Work

BLOCK "A"

Trench No.	Number of samples	Width of sample, feet	Percentage—finished concentrate			
			−20+40 factor 81·3	−40 +100 factor 81·3	−100 factor 76·9	Total factor 80·7
1	12	100·25	5·93	3·52	1·35	10·79
2	12	152·73	6·80	3·65	1·25	11·69
3	5	62·58	6·65	3·46	1·24	11·31
4	14	184·24	6·37	3·31	1·18	10·84
5	8	79·33	6·73	3·35	1·25	11·31
6	8	125·16	8·60	3·70	1·24	13·51
Totals and averages	59	704·29	6·83	3·49	1·25	11·57

BLOCK "B"

7	8	105·51	7·89	3·87	1·49	13·24
8	7	100·92	7·33	3·66	1·41	12·39
9	14	168·59	8·07	4·07	1·41	13·53
Totals and averages	29	375·02	7·76	3·87	1·43	13·06

BLOCK "C"

10	17	199·41	5·73	3·31	1·16	10·20
11	15	202·59	6·85	3·93	1·54	12·31
12	7	81·58	6·80	3·37	1·32	11·48
Totals and averages	39	483·58	6·45	3·54	1·34	11·33

BLOCKS "A", "B" AND "C"

1–6	59	704·29	6·84	3·50	1·25	11·57
7–9	29	375·02	7·76	3·87	1·44	13·06
10–12	39	483·58	6·46	3·54	1·34	11·33
Totals and averages	127	1,562·89	6·97	3·61	1·31	11·89

Percentages obtained from Weights of Samples

Trench No.	Number of samples	Weight of samples, less small samples	Percentage garnet recovered				Weight percentage of garnet recovered
			Total	−20+40	−40+100	−100	
1	12	1,368	13·27	7·23	4·30	1·74	18,249·95
2	12	1,132	14·47	8·37	4·48	1·62	16,383·72
3	5	511	14·65	8·52	4·44	1·69	7,486·78
4	14	1,313	13·44	7·83	4·06	1·54	17,642·54
5	8	691	14·04	8·29	4·13	1·62	9,703·04
6	8	695	16·96	10·72	4·61	1·63	11,784·81
7	8	777	16·53	9·78	4·50	1·95	12,847·34
8	7	683	15·05	8·83	4·41	1·81	10,277·52
9	14	1,415	16·67	9·87	4·98	1·82	23,594·67
10	17	1,466	12·55	7·00	4·03	1·51	18,405·18
11	15	1,311	15·28	8·44	4·84	2·00	20,026·06
12	7	617	13·96	8·20	4·07	1·69	8,611·32
Totals and averages	127	11,979	14·74	8·59	4·41	1·71	174,994·93
			Factor 80·7	Factor 81·3	Factor 81·3	Factor 76·9	
Correction for finished concentrate			11·89	6·98	3·59	1·32	

Procedure followed for determining methods of treatment.—After each of the samples had been crushed to one-half inch and the small sample cut out for the small scale laboratory tests to determine the garnet content, the remaining portion of the samples from each trench were mixed together, making twelve large samples, each representing a trench. These lots were used for the larger tests to determine methods of treatment, and as the tests made were not all similar in methods of procedure, each test is described separately below in the order in which they were run.

Trench No. 10.—After the reduction of the samples to 12 mesh, some mica was removed by a light suction. One-half of the 12 mesh material was classified into three products and a classified product fed to the concentrating table. The other half was sized and a sized product fed to the table. This was done to compare the work of the table on a classified and sized feed.

Trench No. 11.—After reduction to 12 mesh and the removal of some mica by a light suction, the material was screened wet on 34 mesh, and 80 mesh Callow belt screens, and each size tabled.

Trench No. 9.—Procedure similar to Trench No. 11.

Trench No. 4.—After reduction to 12 mesh, the mica was removed by medium suction. The −12 mesh material was sized wet on Callow belt screens, 34 and 80 mesh, and each size run separately over a Wilfley concentrator. A sample of the mica product was screened and each size tabled to determine the garnet content in the mica product.

Trench No. 2.—After reduction to 12 mesh, the mica was removed by a strong suction. As very little fines remained in the −12 mesh material, it was screened wet on a 34 mesh Callow belt screen only, and the two sizes made were run separately over a Wilfley concentrator. A sample of the mica product was sized and each size tabled to determine the garnet content of this product.

Trench No. 1.—Same procedure followed as on the samples from Trench No. 2. Suction used was not quite as strong.

Trench No. 5.—Same procedure followed as on the samples of Trench No. 1.

Trench No. 6.—Same procedure as on the samples of Trench No. 1.

Trench No. 7.—Same procedure as for samples of Trench No. 1.

Trench No. 8.—Same procedure as for samples of Trench No. 1.

Trench No. 12.—After reduction to 12 mesh and the removal of a mica product by suction, the material was classified to remove any slime, sized wet on a 34 mesh Callow belt screen, and each size run separately over a Wilfley concentrator, making a concentrate, middling, and tailing. In the case of the coarser size, the concentrate was screened on a 24 mesh Callow belt screen, making a finished concentrate and a middling. The middling from the first run over the table was re-run, making a concentrate and tailing. This concentrate was screened on a 24 mesh Callow belt screen, making a finished concentrate and middling. The middlings, or throughs, from the screening were added together and re-tabled, making a concentrate and tailing, the concentrate from which was screened on a 34 mesh Callow belt screen, making a finished concentrate and middling. The middling from this operation was added to the −34 mesh material to be tabled. A similar procedure was followed with the −34 mesh size, except that screening was done on a 44 mesh Callow belt screen.

Trench No. 3.—Same procedure followed as for samples of Trench No. 12, with the exception that no mica product was removed.

Procedure followed and results of test work on samples from Trench No. 10.—The samples from this trench were crushed in rolls set at $\frac{1}{8}''$, and screened on a Ferraris shaking screen fitted with 12 mesh screen, and with suction take off at the discharge end, for the removal of the mica. The oversize was returned to the rolls, until all passed through the screen or was removed by the suction. The suction used was low so that only a small amount of mica was removed in this manner.

Product	Weight pounds	Per cent of feed
Suction	93	6·46
−12 mesh	1,288	89·44
Dust loss	59	4·10
Feed	1,440	100·00

The −12 mesh material was cut into two equal portions. One was classified into three products in a two-spigot launder classifier, and each run separately over the Wilfley concentrator, giving the following results:—

Product	Weight pounds	Per cent of heads
Coarse concentrate	45·5	6·32
Medium concentrate	44·5	6·18
Fine concentrate	6·5	0·90
Total concentrates	96·5	13·40

The other portion was screened wet on 34 mesh and 80 mesh Callow belt screens, and each size fed separately to the table, giving the following results:—

Product	Weight pounds	Per cent of feed
−12+34 concentrate	62·0	8·61
−34+80 concentrate	24·0	3·33
−80 concentrate	11·5	1·60
Total concentrates	97·5	13·54

A comparison of the concentrates produced in the above tests on two equal portions of the −12 mesh material shows that the larger amount and better grade of concentrate was obtained on a sized feed. It was found that the table worked better on a sized feed than on a classified feed, so that a sized feed was used in all subsequent tests.

Procedure followed and results of test work on samples from Trench No. 11. —The samples from this trench were reduced to 12 mesh and mica removed as in the preceding tests, with the exception that very low suction was used, taking off less mica product.

Product	Weight pounds	Per cent of feed
Suction	11·5	0·88
−12 mesh	1,226·5	93·55
Dust loss	73·0	5·57
Feed	1,311·0	100·00

The 12 mesh material was screened wet on 34 mesh and 80 mesh Callow belt screens, and each size fed separately to the table, with the following results:—

Product	Weight pounds	Per cent of heads
−12+34 concentrate	128·25	9·78
−34+80 concentrate	57·00	4·35
−80 concentrate	24·75	1·89
Total concentrates	210·00	16·02

Procedure followed and results of test work on samples of Trench No. 9.—The same procedure was followed as with the samples from Trench No. 11. The suction used was similar to that employed on the samples from Trench No. 10.

Product	Weight pounds	Per cent of heads
Suction	80·00	5·66
−12 mesh	1,276·75	90·29
Dust loss	57·25	4·05
Feed	1,414·00	100·00

The 12 mesh material was screened wet on 34 mesh and 80 mesh Callow belt screens, and each size tabled separately:—

Product	Weight pounds	Per cent of heads
−12+34 concentrate	155·5	11·00
−34+80 concentrate	58·5	4·14
−80 concentrate	25·5	1·80
Total concentrates	239·5	16·94

Procedure followed and results of test work on samples of Trench No. 4.—The samples from this trench were reduced to 12 mesh in a similar manner to the preceding ones, with the exception that a medium suction was used at the feed end of the screen, and some oversize, consisting of mica, was not fed back to the rolls.

Product	Weight pounds	Per cent of feed
Suction	213·0	16·22
−12 mesh	1,034·0	78·75
+12 mesh (mica)	7·5	0·57
Dust loss	58·5	4·46
Feed	1,313·0	100·00

The 12 mesh material was sized wet on 34 mesh and 80 mesh Callow belt screens and each size tabled:—

Product	Weight pounds	Per cent of heads
−12+34 concentrate	117·00	8·91
−34+80 concentrate	37·25	2·84
−80 concentrate	12·50	0·95
Total concentrates	166·75	12·70

As the suction was applied at the feed end of the shaking screen, a certain amount of fine garnet was drawn up with the mica into the collecting tank. A sample of the suction product was screened, and the various sizes tabled to determine the percentage of garnet in this mica product. It was found to contain 0·68 per cent of the weight of the sample of Trench No. 4. This added to the 12·70 per cent recovered in the table concentrate would give as value for the trench, 13·38 per cent garnet concentrate. The suction product contained 4·21 per cent garnet, 0·09 per cent coarser than 35 mesh, 4·12 finer than 35 mesh. Any garnet lost in the suction product was very fine, the greater proportion of which would be lost in subsequent tabling operations if left in the material fed to the table.

Procedure followed and results of test work on samples from Trench No. 2.—The samples from this trench were crushed to 12 mesh in a similar manner to those of Trench No. 10. A strong suction was used to remove the mica at the feed end of the shaking screen:—

Product	Weight pounds	Per cent of feed
Suction	563·5	49·15
−12 mesh	519·0	45·27
Dust loss	64·0	5·58
Feed	1,146·5	100·00

The strong suction used removed almost 50 per cent of the feed to the screen, and delivered to the collector the mica and fines. It was, therefore, not necessary to make three sized products for the table. The −12 mesh material was only sized wet on a 35 mesh Callow belt screen.

Product	Weight pounds	Per cent of heads
−12+34 concentrate	101·75	9·02
−34 concentrate	21·25	1·88
Total concentrates	123·00	10·90

A small scale test was made on a sample of the suction product. It was found to contain garnet representing 3·70 per cent of the weight of the samples of Trench No. 2. This added to the 10·90 per cent recovered in table concentrate would give a value for the trench of 14·60 per cent garnet concentrate. The suction product contained 7·53 per cent garnet, 1·94 per cent +35 mesh, and 3·59 per cent −35 mesh. These results show that too strong a suction was used, removing too much garnet with the mica.

Procedure followed and results of test work on samples from Trench No. 1.—The samples from this trench were crushed to 12 mesh in a similar manner to those of Trench No. 10. A strong suction was used to remove the mica at the feed end of the shaking screen. The suction was not quite as strong as that used in the preceding test on the samples of Trench No. 2.

Product	Weight pounds	Per cent of feed
Suction	609·0	44·57
−12 mesh	687·0	50·27
Dust loss	70·5	5·16
Feed	1,366·5	100·00

The −12 mesh material was sampled and the balance, 681·5 pounds, was screened wet on a 34 mesh Callow belt screen, and the two sizes fed separately to a Wilfley concentrator.

Product	Weight pounds	Per cent of heads
−12+34 concentrate	116·0	8·55
−34 concentrate	31·0	2·29
Total concentrates	147·0	10·84

A small scale test was made on a sample of the suction product. It was found to contain garnet representing 2·25 per cent of the weight of the samples of Trench No. 1. This percentage added to the 10·84 per cent recovered in table concentrate would give a value for the trench of 13·09 per cent garnet concentrate. The suction product contained 5·06 per cent garnet, 0·67 per cent +35 mesh, and 4·39 per cent −35 mesh.

Procedure followed and results of test work on samples from Trench No. 5.—Same procedure followed as on samples from Trench No. 1.

Screen product	Weight pounds	Per cent of heads
Suction	292·0	42·35
−12 mesh	349·0	50·62
Dust loss	48·5	7·03
Feed	689·5	100·00

Table product	Weight pounds	Per cent of heads
−12+34 concentrate	60·0	8·85
−34 concentrate	14·5	2·14
Total concentrates	74·5	10·99

Suction product.. 0·75% +35 mesh
5·14% −35 mesh

Total................ 5·89% garnet,
representing 2·49 per cent of heads, making a total value for trench —13·48 per cent.

Procedure followed and results of test work on samples from Trench No. 6.—Same procedure followed as on the samples from Trench No. 1.

Screen product	Weight pounds	Per cent of heads
Suction	296·0	42·68
−12 mesh	358·5	41·70
Dust loss	39·0	5·62
Feed	693·5	100·00

Table product	Weight pounds	Per cent of heads
−12+34 concentrate	81·00	11·88
−34 concentrate	14·75	2·16
Total concentrates	95·75	14·04

Suction product.. 1·55% +35 mesh
5·49% −35 mesh

Total.............. 7·04% garnet,
representing 3·00 per cent of heads, making a total value for trench, 17·04 per cent.

Procedure followed and results of test work on samples from Trench No. 7.—Same procedure followed as on the samples from Trench No. 1.

Screen product	Weight pounds	Per cent of heads
Suction	381·0	48·78
−12 mesh	378·0	48·40
Dust loss	22·0	2·82
Feed	781·0	100·00

Table product	Weight pounds	Per cent of heads
−12+34 concentrate	87·5	11·37
−34 concentrate	18·5	2·40
Total concentrates	106·0	13·77

Suction product.. 1·25% +35 mesh
5·66% −35 mesh

Total.............. 6·91% garnet,
representing 3·37 per cent of heads, making a total value for trench, 17·14 per cent.

Procedure followed and results of test work on samples from Trench No. 8.—Same procedure as on the samples from Trench No. 1.

Screen product	Weight pounds	Per cent of heads
Suction	311·0	45·17
−12 mesh	357·0	51·85
Dust loss	20·5	2·98
Feed	688·5	100·00

Table product	Weight pounds	Per cent of heads
−12+34 concentrate	68·50	10·10
−34 concentrate	17·75	2·62
Total concentrates	86·25	12·72

Suction product.. 0·98% +35 mesh
6·20% −35 mesh

Total.............. 7·18% garnet,

representing 3·24 per cent of heads, making a total value for trench, 15·96 per cent

Procedure followed and results of test work on samples from Trench No. 12—The samples from this trench were crushed in rolls set at $\frac{1}{8}''$ and screened on a Ferraris shaking screen fitted with a 12 mesh screen and with suction take off at the feed end. The oversize was returned to the rolls until all passed through the screen.

Product	Weight pounds	Per cent of feed
Suction	299·5	48·50
−12 mesh	301·0	48·75
Dust loss	17·0	2·75
Feed	617·5	100·00

The −12 mesh material was fed to a Dorr drag classifier to remove any slimes, and the sand screened on a 34 mesh Callow belt screen, making two sizes, which were fed separately to a Wilfley concentrator. The subsequent procedure can best be illustrated by the following flow sheet:—

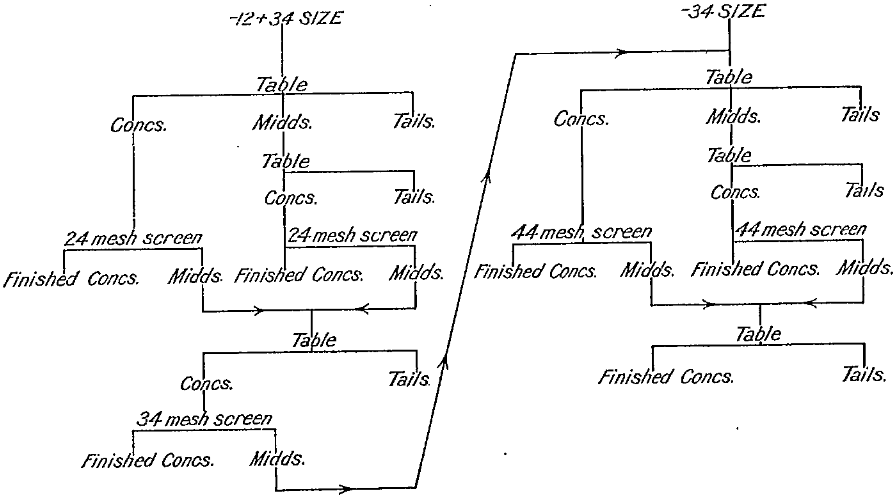

FIG. 8. Flow sheet showing procedure in testing samples of garnet rock from Trench No. 12, Depot Harbor, Ont.

294·5 pounds of −12 mesh material was fed to the classifier. There were 8·5 pounds of slime overflow, and 286 pounds of sands for concentration. 1½ pound of material was cleaned up after the test.

Product	Weight pounds	Per cent of heads
+24 concentrate	19·5	3·23
+24 concentrate from middling	15·5	2·57
+34 concentrate	7·5	1·24
+44 concentrate	2·5	0·41
+44 concentrate from middling	4·0	0·66
−44 concentrate	4·5	0·75
Total concentrates	53·5	8·86

Suction product.. 0·71% +35 mesh
5·84% −35 mesh
Total.............. 6·55% garnet,
representing 3·18 per cent of heads, making a total value for trench, 12·04 per cent

Procedure followed and results of test work on samples from Trench No. 3.—The samples from this trench were crushed in rolls set at ⅛″ and screened on a Ferraris shaking screen fitted with 12 mesh screen. No suction was used to remove the mica. The oversize was returned to the rolls until all passed through the screen.

Product	Weight pounds	Per cent of heads
−12 mesh	483·0	92·26
Dust loss	40·5	7·74
Heads	523·5	100·00

The −12 mesh material was fed to a Dorr drag classifier to remove the slimes, and the sands screened on a 34 mesh Callow belt screen, making two sizes, which were fed separately to a Wilfley concentrator. The subsequent procedure is the same as that given in the flow sheet under trench No. 12. 474·5 pounds of −12 mesh material was fed to the classifier. There were 15 pounds of slime overflow, and 459·5 pounds of sands for concentration. 0·5 pound of material was cleaned up after the test.

Product	Weight pounds	Per cent of heads
+24 concentrate	17·5	3·41
+24 concentrate from middling	12·5	2·43
+34 concentrate	7·5	1·46
+44 concentrate	7·0	1·36
+44 concentrate from middling	3·0	0·58
−44 concentrate	16·0	3·11
Total concentrates	63·5	12·35

This test and the preceding one were made to determine whether a finished concentrate could not be produced by screening the table concentrate and re-tabling the throughs. It was found that the screened product was very clean, but that it was difficult to obtain a clean concentrate by re-tabling the throughs or middlings, as the gangue material contained in them was of almost the same specific gravity as the garnet. These points were determined by magnetic separation tests made on the concentration products.

SUMMARY OF RESULTS FROM LARGE SCALE TESTS ON TRENCH SAMPLES

Trench No.	Per cent concentrates				
	−12+34	−34+80	−80	In suction product	Total
10	8·61	3·33	1·60	....	13·54
11	9·78	4·35	1·89	....	16·02
9	11·00	4·14	1·80	....	16·94
4	8·91	2·84	0·95	0·68	13·38
2	9·02	1·88	....	3·70	14·60
1	8·55	2·29	....	2·25	13·09
5	8·85	2·14	....	2·49	13·48
6	11·88	2·16	....	3·00	17·04
7	11·37	2·40	....	3·37	17·14
8	10·10	2·62	....	3·24	15·96
12	7·04	1·82	....	3·18	12·04
3	7·30	5·05	....	....	12·35

NOTE.—The percentages given under the column—34+80 for Trench Nos. 2, 1, 5, 6, 8, 12, and 3, are for −34 mesh concentrates.

On examination of the figures for total percentages, it will be found that these correspond very closely with the results obtained from the small scale tests, showing that the small laboratory results are practically the same as that obtained from actual milling operations.

Magnetic separation tests on concentration products.—These tests were performed to eliminate the non-magnetic gangue remaining in the table concentrate, and to determine the percentage of finished concentrate in the table products. As shown in preceding tests, the table concentrate contained some gangue which was close to the specific gravity of the garnet and reported on the table with the garnet in the concentrate.

Test No. 1.—This test was run on three samples representing the +34, −34+80, and −80 concentrates from all the trench samples, except those of Trenches Nos. 12 and 3. The samples were run dry through an Ullrich magnetic separator.

Product	+34		−34+80		−80	
	Weight		Weight		Weight	
	lbs.	%	lbs.	%	lbs.	%
Concentrate	70·25	81·2	70·50	82·0	15·50	77·5
Tailing	16·25	18·8	15·50	18·0	4·50	22·5
Feed	86·50	100·0	86·00	100·0	20·00	100·0

It will be noted that these results correspond very closely to those obtained from the concentrate of the small scale tests. For comparison the two results are again given:—

Small scale tests		Large scale tests	
−20+40	81·3 per cent F.C.	−12+34	81·2 per cent F.C.
−40+100	81·3 per cent F.C.	−34+80	82·0 per cent F.C.
−100	76·9 per cent F.C.	−80	77·5 per cent F.C.

Test No. 2.—This test was run on the products obtained by screening the +34 mesh concentrate on 28 mesh to determine whether a finished concentrate could be obtained by screening.

Product	+28		−28	
	Weight		Weight	
	lbs.	%	lbs.	%
Concentrate	65·5	99·3	9·5	50·0
Tailing	0·5	0·7	9·5	50·0
Feed	66·0	100·0	19·0	100·0

The results of this test show that a clean finished product is made by screening the 34 mesh table concentrates on 28 mesh, that practically all the gangue passes through the screen and this product is 50 per cent gangue, 50 per cent garnet. The −28 mesh material would require re-cleaning by the use of a magnetic separator.

Test No. 3.—This test was run on the products obtained by screening the −34+80 concentrates on 48 mesh to determine whether a finished concentrate could be obtained by screening.

Product	+48		−48	
	Weight		Weight	
	lbs.	%	lbs.	%
Concentrate	49·5	95·2	20·5	62·1
Tailing	2·5	4·8	12·5	37·9
Feed	52·0	100·0	33·0	100·0

The results of this test show that a clean finished product is made by screening the −34+80 table concentrates on 48 mesh; that practically all the gangue passes through the screen, and this product is 37·9 per cent gangue, 62·1 per cent garnet. The −48 mesh material would require re-cleaning by the use of a magnetic separator.

Test No. 4.—This test was run on the products obtained by screening the −80 table concentrates on 100 mesh to determine whether a finished concentrate could be obtained by screening.

Product	+100		−100	
	Weight		Weight	
	lbs.	%	lbs.	%
Concentrate	8·5	77·3	6·5	68·4
Tailing	2·5	22·7	3·0	31·6
Feed	11·0	100·0	9·5	100·0

The results of this test show that a finished product cannot be made by screening the −80 mesh table concentrate on 100 mesh. The +100 mesh contains 22·7 per cent gangue, and the −100 31·6 per cent gangue. The −80 table concentrate would require re-cleaning by the use of a magnetic separator.

Test No. 5.—A test was run on samples of the table concentrates +34, −34+80 and −80 on the Ullrich magnetic separator, using a wet feed instead of a dry one. A stronger magnetic field was used. The results obtained were practically the same as those from the dry feed, possibly a slightly cleaner concentrate was made, but the tailing seemed to contain a little more garnet. If the magnetic field had been slightly stronger, the results would have been very similar.

The Possibilities of Recovering Mica Products Suitable for the Trade.—In the majority of the tests made on the trench samples, a mica product was obtained by suction, similarly to the practice for the removal of asbestos fibre from the crushed rock. In some cases this mica product represented as high as 50 per cent of the feed when a strong suction was used, but contained in such cases a considerable amount of fine garnet. However, as the saleable mica product would be sizes coarser than 35 mesh, the fines could be returned to the classifier with the rest of the feed and the fine garnet saved. When a moderate suction was employed very little garnet entered this product. The possibility of recovering a mica product would depend on the rock being dry. When the rock contained much moisture this system could not be used. However, it would be well to make provision for the recovery of a mica product in the construction of a mill, as during certain seasons of the year the rock would be in a condition favourable for its recovery, and certain grades could be produced, with a possibility of their use for certain purposes. Samples of the suction product from several of the tests were sized, and the sizes, coarser than 35 mesh, gave a clean product that might be used profitably, and might constitute a valuable by-product.

From the results of the above tests for the determination of methods of treatment for the recovery of the garnet from the rock, in products suitable for the trade, the following flow sheet, Fig. 9, is suggested:—

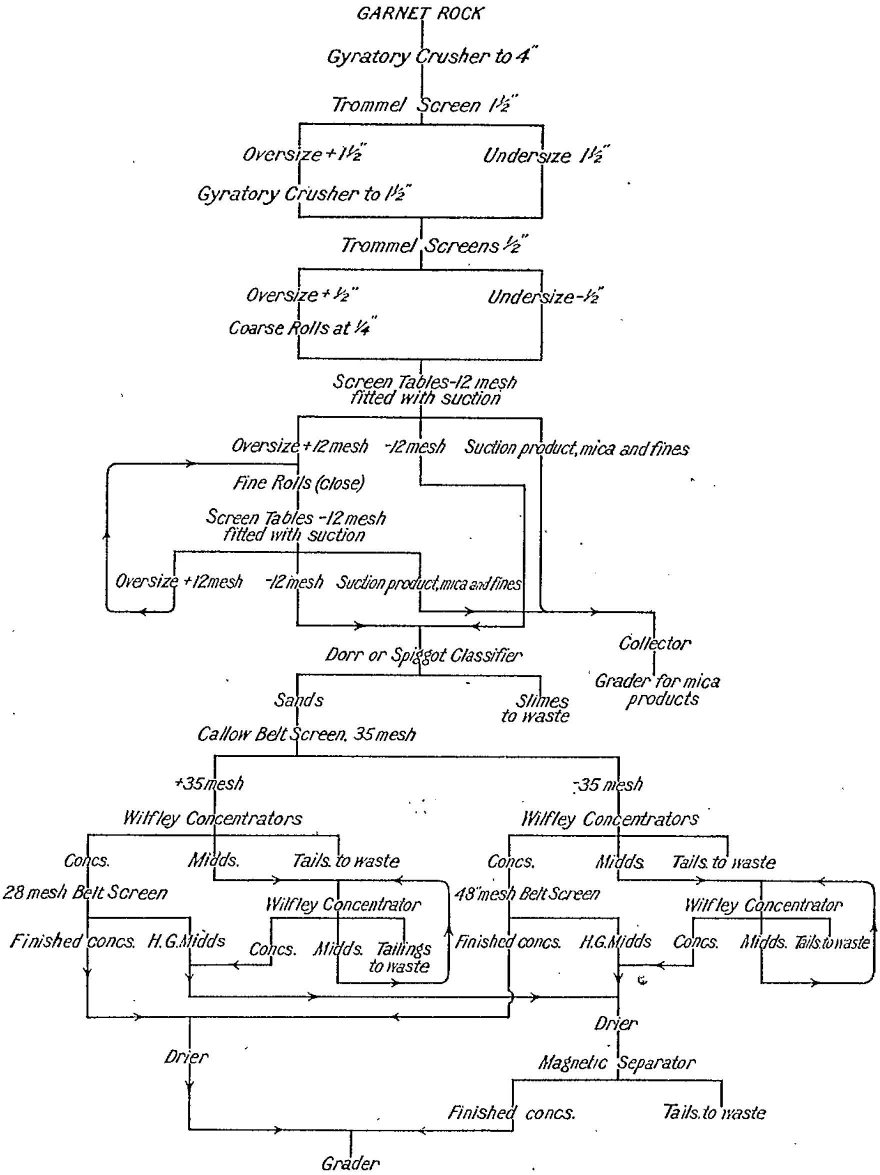

FIG. 9. Flow sheet of garnet concentrator, showing methods of treatment for recovery of garnet from the rock.

If it is found more economical to treat all table concentrates on a magnetic separator, the following flow sheet can be substituted after classification for the elimination of slimes.

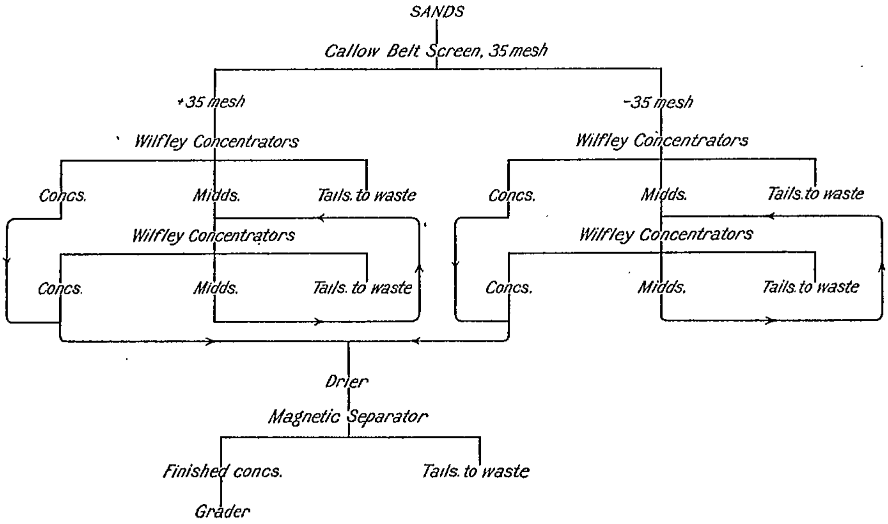

FIG. 10. Flow sheet of garnet concentrator, showing treatment of table concentrates on a magnetic separator.

If, on the other hand, after careful study of the requirements for the trade, it is found necessary to make a finished product coarser than 1 mm., and as coarse as 3 mm., jigging would have to be resorted to. The following is given as a suggested improved flow sheet of a garnet concentrator based on the practice in the United States. (See Fig. 11.)

SUMMARY AND CONCLUSIONS

The samples submitted for test purposes, being moiled from the bottom of the trenches, should be as representative as it is possible to obtain them, and should give an accurate value of the deposit as to garnet content. The samples may be slightly low in garnet, as there was the possibility of some garnet crystals dropping free from their matrix, and unless great care was taken, some of these crystals would be lost, giving a lower value to the sample.

Table concentration of the 127 samples submitted gave an average of 14·73 per cent garnet concentrates, made up as follows:—

	%
−20+ 40 concentrate	8·58
−40+100 "	4·44
−100 "	1·71

The results of the large scale tests on the trench samples gave practically the same results.

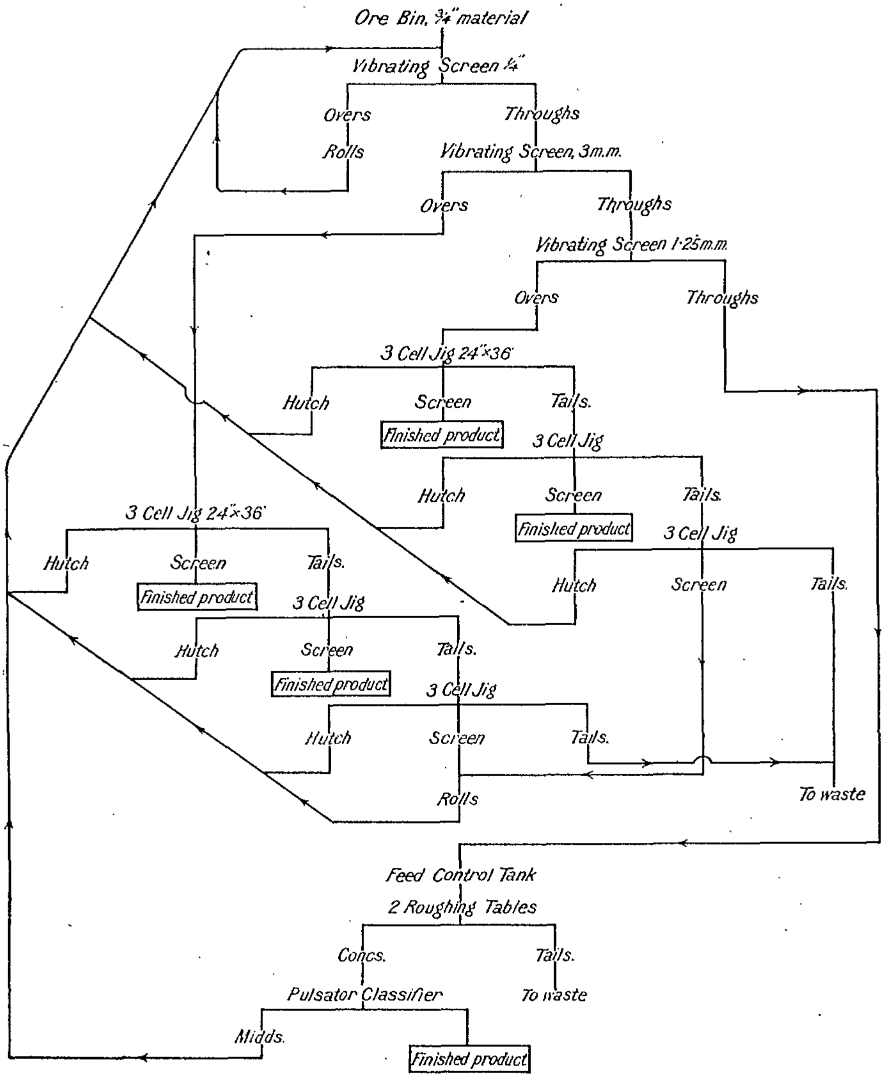

FIG. 11. Suggested improved flow sheet of garnet concentrator based on practice in the United States.

The table concentrates contain some gangue, difficult to remove by table concentration, on account of its specific gravity being very close to that of the garnet. This can be eliminated by screening, and by the use of the magnetic separator, giving a finished product. The finished concentrates show an average of 11·89 per cent, made up as follows:—

	%
−20+ 40 concentrate	6·97
−40+100 "	3·61
−100 "	1·31

The results of the large scale tests on the trench samples gave practically the same results.

The tests made on the trench samples to determine methods of treatment for the recovery of the garnet, show that this can be accomplished by table concentration, screening of the table products, and magnetic separation for the middlings, or by magnetic separation for the table concentrates, resulting in finished products, with a high recovery of the garnet content of the rock.

On account of the nature and size of the garnet crystals, it does not seem possible to apply jigging methods to this rock for the recovery of the garnet. It is quite possible that at some depth below the surface the rock will be harder, and the garnet crystals more perfect and free from fracture, in which case jigging would be feasible for the coarser sizes, and table concentration for the fines.

The garnet rock contains small amounts of iron pyrites and magnetite. This was noticeable in some of the samples. These minerals can be easily eliminated on the tables as they report above the garnet. Any magnetic iron not eliminated in this manner can be removed in the subsequent operations by the magnetic separator.

As the rock contains a considerable quantity of mica, there is the possibility of obtaining from it, in the milling practice, mica products suitable for the trade. The tests show that the mica can be separated by suction, and on being graded, gives clean products on the coarse sizes. Samples of the various grades were taken for the purpose of securing information as to possible markets.

(9)

THE RECOVERY OF BRASS FROM FOUNDRY REFUSE

C. S. Parsons

There are many brass foundries in the cities and larger towns of the Dominion, in which a considerable quantity of refuse is collected at intervals, and sold to United States points, where it is treated for the recovery of the brass; or the slag is broken up and the larger pieces hand sorted out, the remainder being discarded as useless. The shipment of this material to points outside the country brings little return after freight and treatment charges are deducted. In many cases the refuse could more profitably be treated at the foundry by the installation of a small plant. It was for this reason that the following test work was conducted.

Test No. 151

A shipment of 16,520 pounds of foundry slag and sweepings was received at the Ore Dressing and Metallurgical Laboratories on May 4, 1921, from Mr. S. R. Francis, of the Ontario Specialties, Limited, Ottawa, Ont.

Test work was desired on this shipment to obtain a separation of the metal from the slag in as coarse a form as possible, and to determine a simple flow sheet for the recovery of the metal from the slag and sweepings.

The slag was fed directly to a 4′ 6″ Hardinge mill, containing a ball charge of 2,000 pounds. It was crushed wet, and the pulp from the mill fed to a standard size Wilfley table equipped with a No. 1623 deck, without being sized or classified. Only the very fine brass was discharged, the coarse metal remaining in the mill. After running for some time with an unclassified feed to the table, the discharge of the mill was run into a Richards launder classifier, and two products were made and fed separately to the table. The classified feed gave the better results, a lower tailing being produced, and the concentrate was cleaner.

At the end of the test the Hardinge mill was cleaned out and it was found that the major portion of the brass, consisting of the coarse pieces, remained in the mill. The clean up of brass from the mill, weighing 1,766 pounds, was dried and passed over a magnetic cobber, removing 100 pounds of iron, and leaving 1,666 pounds of coarse brass metal.

The table concentrates were dried, weighed, and passed over a magnetic cobber, removing 356·5 pounds of magnetic material, leaving 887 pounds of fine brass.

The total amount of brass recovered was 2,553 pounds, obtained from 16,520 pounds of slag, or a recovery of 309 pounds of metal per ton of slag.

The following table gives the weights and analyses of the products from the shipment:—

Products	Weight pounds	Analysis			
		Cu. %	Pb. %	Zn. %	Sn. %
Coarse metal from clean up of ball mill...	1,666·0	79·0	7·8	2·0	11·0
Iron removed from clean up of ball mill..	100·0	....	....	....	....
Fine metal from table concentrates.......	887·0	82·05	4·5	6·13	6·2
Magnetic product from table concentrates	356·5	22·85	....	....	....
*Tailings from table....................	13,510·5	3·20	....	....	....
Total......................	16,520·0	15·5	....	....	....

*Note.—Tailings from unclassified feed gave an analysis 3·80 per cent Cu.
Tailings from classified feed gave an analysis 2·70 per cent Cu.

CONCLUSIONS

No particular difficulty was experienced in the separation of the metal from the slag. The metal obtained was a high grade product, suitable for re-melting.

For the small brass foundry, a simple equipment of a small size Hardinge mill, and a half-size Wilfley table, would be sufficient to handle the slag and sweepings.

(10)

THE METALLURGICAL TREATMENT OF THE GOLD ORES OF RICE LAKE DISTRICT, MANITOBA

R. K. Carnochan

In the Summary Report of the Mines Branch for 1920, there is described under test No. 135, the experimental work conducted on a shipment of gold ore from Little Rice lake, Manitoba. That report, together with the investigation conducted on the ores from Big Rice lake given below, covers the metallurgical treatment of the gold ores from Rice Lake district, in the southeastern section of the province, as far as exploration work has proven them to date. These ores, containing copper minerals and iron sulphides, are somewhat different from the ordinary gold ores covered by metallurgical practice, and require a different treatment to recover the valuable minerals.

In the Summary Reports of the Mines Branch for 1919 and 1920, there is described under tests Nos. 127 and 132 the experimental work conducted on the gold ores from the Herb Lake district in the northwestern section of the province. The ores from this district contain chalcopyrite, arsenopyrite, pyrite, sphalerite and galena, requiring special treatment for the recovery of the valuable minerals.

Test No. 141

Two small samples of gold ore were received January 18, 1921, at the Ore Dressing and Metallurgical Laboratories, from the Pan Extension Gold Mines, Ltd., Winnipeg, Man. The samples were from the company's property at Big Rice lake, and consisted of quartz and greenstone, carrying a small amount of sulphides.

Amalgamation and concentration tests were desired to ascertain whether the ore was amenable to these methods of treatment. Cyanide tests were also conducted.

Assays of the samples gave the following:—

Sample No. 2	Au	0·64 oz. per ton
Sample No. 3	Au	1·65 oz. per ton

Sample No. 2.—One thousand one hundred and twenty-four grams of the sample, crushed to pass 40 mesh, were mixed in a pebble mill for four hours with 100 grams of mercury and 400 grams of water. The pulp was panned from the mercury, and the pulp dried, weighed and sampled. The results were as follows:—

Amalgamation tailings	0·60 oz. per ton
Recovered by amalgamation	6·3 %
Remaining in tailings	93·7 %

One thousand and twenty-nine grams of tailings from amalgamation were run over a small Wilfley table, with the following results:—

Concentrates:		
Weight	56	grams
Assay, Au	3·24	oz. per ton
Content, Au	181·4	gram/ozs.
Percentage Au. values	24·4	
Tailings:		
Weight	954	grams
Assay, Au	0·54	oz. per ton
Content, Au	515·2	gram/ozs.
Percentage Au. values	69·3	

Nine hundred and fifteen grams of table tailings were ground to 200 mesh and agitated for eight hours in 6,000 c.c. of water, to which 15 grams of sodium cyanide and 10 grams of lime were added. After agitation, the tailings were filtered, washed, dried and sampled, with the following results:—

Cyanide tailings	0·08	oz. per ton
Recovered by cyanidation	59·0	%
Remaining in tailings	10·3	%

Sample No. 3.—One thousand and twenty-seven grams of the sample, crushed to 40 mesh, were mixed in a pebble mill for four hours, with 100 grams of mercury and 400 grams of water. The pulp was panned from the mercury, dried, weighed and sampled. The results were as follows:—

Amalgamation tailings	1·52	oz. per ton
Recovered by amalgamation	7·9	%
Remaining in tailings	92·1	%

Nine hundred and fifty-two grams of tailings from amalgamation were run over a small Wilfley table, with the following results:—

Concentrates:		
Weight	77	grams
Assay, Au	1·20	oz. per ton
Content, Au	92·4	gram/ozs.
Percentage Au values	6·0	
Tailings:		
Weight	856	grams
Assay, Au	1·56	oz. per ton
Content, Au	1335·4	gram/ozs.
Percentage Au values	86·1	

Eight hundred and six grams of table tailings were ground to 200 mesh and agitated for eight hours in 6,000 c.c. of water, to which 15 grams of sodium cyanide and 10 grams of lime were added. After agitation the tailings were filtered, washed, dried and sampled, with the following results:—

Cyanide tailings	0·10	oz. per ton
Recovered by cyanidation	80·6	%
Remaining in tailings	5·5	%

SUMMARY

The following tables show the percentages of the gold values recovered:—

Sample No. 2:	p.c.
By amalgamation	6·3
By concentration	24·4
By cyanidation	59·0
Remaining in tailings	10·3

Sample No. 3:	p.c.
By amalgamation	7·9
By concentration	6·0
By cyanidation	80·6
Remaining in tailings	5·5

CONCLUSIONS

The results of the amalgamation tests on both these samples show poor recoveries of the gold values. This may be due to the samples being oxidized, and the gold not being in the best condition for amalgamation. The samples were too small to allow further test work being conducted to prove definitely whether or not better results could be obtained.

The concentration tests showed poor recoveries of the gold values in the concentrates from the Wilfley table, and if the samples are representative, tabling could be eliminated from the milling practice.

The results of the cyanidation tests show that the ore is readily amenable to cyanidation, and that good recoveries can be obtained by this process of treatment.

Test No. 153

A shipment of gold ore was received May 27, 1921, at the Ore Dressing and Metallurgical Laboratories, from the Pan Extension Gold Mines, Limited, Winnipeg, Man. The ore was from their property at Big Rice lake, Man., and consisted of quartz and greenstone, with a small amount of copper and iron sulphides. Free gold was visible, especially in the higher grade sample. The shipping weight was 2,100 pounds, in 27 sacks, one of which was high grade and the remainder lower grade ore.

Test work was desired on this shipment to determine a method of treatment. Amalgamation, concentration, and cyanidation tests were conducted, the results of which are given below.

Samples cut out from the two lots gave the following assays:—

Lower grade lot (26 sacks):	Copper, 0·60%	Gold, 2·39 oz. per ton
Higher grade lot (1 sack)	Copper, 0·70%	Gold, 15·01 oz. per ton

LOWER GRADE LOT

This lot was crushed to 40 mesh, and a sample cut out for assay. From this crushing some metallics were obtained. It was then put through an amalgamator and over plates, and the amalgamation tailings concentrated on a Wilfley table, making a concentrate and tailing. The results of this work are contained in the following table:—

Product	Weight pounds	Assay		Content gold oz.	Per cent gold values
		Gold oz.	Cu. per cent		
Bullion from metallics				0·037	1·9
Bullion from amalgam				1·212	62·4
Table concentrate	43·0	9·70	13·65	0·209	10·7
Table tailing	1,939·5	0·50		0·485	25·0
Heads	1,982·5	2·39		1·943 2·369	100·0

A 1,000 gram sample of the table tailing, ground to 100 mesh, was agitated for 8 hours, in 6,000 c.c. of water with 15 grams of sodium cyanide and 10 grams of lime. After agitation the tailing was filtered, washed, dried, sampled, and assayed. The results were as follows:—

Cyanide head	0·50 oz. per ton
Cyanide tailing	0·08 oz. per ton
Recovery by cyanidation	84 per cent, or 21 per cent of the gold values in ore
Remaining in tailing	16 per cent, or 4 per cent of the gold values in ore

Another 1,000 gram sample of the table tailing, ground to 100 mesh, was concentrated in a flotation machine. The following table shows the results obtained:—

Product	Weight, grams	Assay gold, oz.	Content gold, gm.-oz.	Per cent gold value in table tail	Per cent gold value in ore
Concentrate	48	9·24	443·5	80·2	20·1
Middling	162	0·48	77·8	14·1	3·5
Tailing	790	0·04	31·6	5·7	1·4
Table tailing	1,000	0·50	552·4 550·0	100·0	25·0

SUMMARY OF RESULTS

The following tables show the percentage of gold values recovered by the different processes:—

By amalgamation	64·3 p.c.	96·0 %
By table concentration	10·7 p.c.	
By cyanidation	21·0 p.c.	
Remaining in tailing	4·0 p.c.	

By amalgamation	64·3 p.c.	98·6 %
By table concentration	10·7 p.c.	
By flotation	23·6 p.c.	
Remaining in tailing	1·4 p.c.	

HIGHER GRADE LOT

This lot was crushed to 20 mesh. Some metallics were obtained. After sampling, the ore was ground with mercury in a small ball mill, and the mill discharged into an amalgamator, the pulp running over plates to settling boxes. The amalgamation tailing was concentrated on a Wilfley table, making a concentrate and tailing. The results of this work are contained in the following table:—

Product	Weight pounds	Assay		Content gold oz.	Per cent gold values
		Gold oz.	Cu. per cent		
Bullion from metallics				0·035	6·8
Bullion from amalgam				0·410	80·1
Table concentrate	5·0	18·70	3·50	0·047	9·2
Table tailing	36·5	1·12		0·020	3·9
Loss in handling	25·5				
Heads	67·0	15·01		0·512 0·503	100·0

A 1,000 gram sample of the table tailing, ground to 100 mesh, was agitated for 8 hours in 6,000 c.c. of water, with 15 grams of sodium cyanide, and 10 grams of lime. After agitation the tailing was filtered, washed, dried, sampled and assayed. The results were as follows:—

Cyanide heads	1·12 oz. per ton
Cyanide tailing	0·06 "
Recovery by cyanidation	94·6 % or 3·7 % of the gold values in ore.
Remaining in tailing	5·4 % or 0·2 % of the gold values in ore.

SUMMARY OF RESULTS

The following table shows the percentage of gold values recovered:—

By amalgamation	86·9	99·8 %
By concentration	9·2	
By cyanidation	3·7	
Remaining in tailing	0·2	

CONCLUSIONS

Drawing our conclusions from the results of the test work on the lower grade lot, as the other was probably a picked sample, or selected from a rich portion of the vein material, we are of the opinion that the ore as shipped to our laboratories is amenable to treatment by amalgamation, table concentration, and cyanidation of the table tailings, or by amalgamation, table concentration, and flotation of the table tailings.

The method of treatment adopted will depend on local conditions. The gold values remaining after amalgamation seem to be associated with the sulphide minerals, and can be recovered in a marketable concentrate by tabling and flotation. This concentrate, besides containing the greater portion of the gold values after amalgamation, will also contain the greater portion of the copper values in the ore.

A simple flow sheet for such an ore would consist of:—

Primary rock crusher, rolls, wet ball mill or stamp battery, amalgamation plates, concentration tables, regrinding ball or tube mill, flotation unit or cyanide plant.

It must be remembered, however, that these conclusions are drawn from the results obtained from the shipment of ore submitted to our laboratories, and are only correct and reliable in so far as the shipment is a representative one of the milling ore from the property.

(11)

CONCENTRATION TESTS ON MOLYBDENITE ORES

C. S. Parsons

Concentration methods and processes for the recovery of molybdenite from its ores, have been fairly well covered by the extensive experimental work carried on in the Ore Dressing and Metallurgical Laboratories during the war years, when there was a brisk demand for the metal for the manufacture of tool-steel and for other munition purposes. Ores from all parts of the Dominion were received and experimental work conducted for the recovery of the molybdenite values. Since that time the industry has been more or less dormant, the production in Canada falling off to practically nothing.

Interest during the year has been revived somewhat, due to the research conducted to find new uses for molybdenite This research has disclosed the fact that when quantities of the metal less than one per cent are used, a beneficial effect is given to most steels.

The experimental work conducted during the year was on ores representing two of the most promising deposits in Ontario and Quebec. It was found that these ores were amenable to concentration, with a high recovery of the molybdenite values in a high grade concentrate. The experimental work is described under the test numbers.

Test No. 154

A shipment of four lots of molybdenite ore, of 200 pounds each, was received at the Ore Dressing and Metallurgical Laboratories, September 15, 1921, from W. E. Joiner, Toronto, Ontario.

These lots represented four types of ore from different points on lots 3 and 4, concession XX, in the township of Cardiff, county of Haliburton, in the vicinity of Wilberforce, Ontario.

These deposits, while being rather low in molybdenite content, have, from the exploration work that has been done on them to date, been reported on as being more or less extensive, and give favourable promise of developing into a source of production of the mineral.

The object of the experimental work was to determine the grade of concentrate and recovery of molybdenite content that could be expected from the ores, and whether they were amenable to concentration by the general methods applied to molybdenite ores.

A sample cut out from each of the four lots gave the following analysis:—

Lot No. 1	MoO_3—0·027 %	MoS_2—0·68 %
2	0·027 %	0·32 %
3	0·025	0·38 %
4		0·12 %

In lot No. 1, the molybdenite flake was of the coarse variety, the ore containing more iron sulphide than the other lots. In lots Nos. 2 and 3 the molybdenite flake was much smaller, being more evenly disseminated through the rock, in which very little iron sulphide was noticeable. In lot No. 4, the rock contained besides molybdenite, an appreciable amount of graphite.

Flotation tests were made on lots Nos. 1, 2, and 3. Lot No. 4 was too low grade to be commercially workable for molybdenite, and for this reason no experimental work was conducted on it.

Table No. I gives the results of the flotation tests. The recoveries indicated include the actual recovery in each product, together with 80 per cent of the molybdenite in the middlings, and the −80 mesh product, which would be recovered in practice by returning them to the rougher cells in closed circuit. From our experience we believe this to be a fair method for the calculation of recoveries.

The tests were conducted in the Callow laboratory flotation cells. Kerosene was used as the collecting oil, and pine oil as the frothing oil. A little lime was added. The cleaner concentrate was screened over an 80 mesh screen, where the floculating effect of the coal oil is taken advantage of to produce a high grade concentrate.

This ore floats readily and gives a high grade concentrate with a good recovery.

TABLE No. I

Test No.	Flotation product	Weight	MoS_2 per cent.	MoS_2 grams	Recovery per cent.	Remarks
1	Concentrate					
	+80 mesh.......	7	93·31	6·53	81·9	Total recovery 91·0 % assuming that 80 % of MoS_2 in middling could be recovered.
	−80 mesh.......	31	2·20	0·68	8·5	
	Middling............	57	0·38	0·22	2·8	
	Tailing..............	902	0·06	0·54	6·8	
2	Concentrate					
	+80 mesh.......	3	80·25	2·41	75·6	Total recovery 84·1 %
	−80 mesh.......	33	0·58	0·19	5·9	
	Middling............	86	0·18	0·15	4·7	
	Tailing.........	891	0·05	0·44	13·8	
3	Concentrate					
	+80 mesh.......	4	87·25	3·49	75·4	Total recovery 83·0 %
	−80 mesh.......	39	0·64	0·25	5·4	
	Middling............	96	0·20	0·19	4·1	
	Tailing..............	872	0·08	0·70	15·1	

Screen test on tailing test No. 1			Screen test on tailing test No. 2		
Mesh	Grams	Per cent	Mesh	Grams	Per cent
+ 35..................................	3	0·36	+ 65	2	0·25
− 35 + 48..................................	11	1·32	− 65+100	111	13·74
− 48 + 65..................................	85	10·22	−100+150	199	24·63
− 65 +100..................................	250	30·05	−150+200	240	29·70
−100 +150..................................	147	17·67	−200	256	31·68
−150 +200..................................	139	16·70			
−200..................................	197	23·68			

Screen test on tailing test No. 3

Mesh	Grams	Per cent
+ 65	1	0·13
− 65 +100	176	22·00
−100 +150	225	28·12
−150 +200	120	15·00
−200	278	34·75

Test No. 157

A 100-pound shipment of molybdenite ore was received at the Ore Dressing and Metallurgical Laboratories, July 6, 1921, from Mr. John Bain, Ottawa, Ont.

The ore was from his property at Indian lake, Masham, Que., and consisted of flake molybdenite and heavy iron sulphides in a pyroxenite gangue.

The object of the test work was to determine the grade of concentrate and recovery of the molybdenite content that could be expected, and whether the ore was amenable to treatment by the ordinary methods employed in the concentration of molybdenite ores.

The ore was crushed to pass a 20 mesh screen, and during the process of crushing some flake molybdenite was recovered by screening on a 10 mesh screen.

Material	Weight per cent	Assay MoS_2 per cent.	Contents. Assay × per cent of weight	Per cent of total contents of ore
+10 flake MoS_2	2	64·55	129·10	35·25
−10 mesh	98	2·42	237·16	64·70
Ore	100	3·66	366·26	100·00

The −10 mesh material, assaying 2·42 per cent MoS_2, was ground to pass a 20 mesh screen, and a sample of 1,000 grams was taken, ground in a small ball mill for 15 minutes with 2 c.c. coal oil, 5 drops pine oil, 1 gram lime, and 500 c.c. water. It was then run in a laboratory Callow pneumatic flotation machine, where 5 drops more of pine oil were used, and the necessary water added. The following products were made, the concentrate being screened on an 80 mesh screen:—

Product	Weight per cent	Assay MoS_2 per cent	Contents. Assay × per cent of weight	Per cent of total contents of ore	Per cent of total contents of flotn. feed
−80 concentrate	1·77	89·25	157·97	42·12	65·05
−80 concentrate	3·72	17·00	63·24	16·86	26·04
Cleaner tails	3·33	1·94	6·46	1·72	2·65
Rougher tails	89·18	0·17	15·16	4·05	6·26
	98·00	....	242·83	64·75	100·00

A screen test of the flotation tailing gave as follows:—

Mesh	Per cent by weight	Cumulative per cent
+ 48	1·2	1·2
− 48 + 65	6·1	7·3
− 65 +100	33·8	41·1
−100 +150	15·5	56·6
−150 +200	21·1	77·7
−200	22·3	100·0
	100·0	....

In practice the rougher tailing is the only waste product. The other products, namely the −80 concentrate and cleaner tailing, are returned to the circuit and the molybdenite content ultimately recovered from them. The actual recovery expected is represented by the molybdenite content of all products, excepting the rougher tailing. The recovery in the flotation test is, therefore, 93·74 per cent, and if the flake screened out in crushing the ore to 20 mesh is considered, the total recovery of molybdenite in the ore is 95·95 per cent.

The test shows that a concentrate of 90 per cent MoS_2 grade, and a recovery of 95 per cent of the molybdenite content, can be expected from this ore.

(12)

THE RECOVERY OF THE VALUES IN A SILVER-LEAD ORE FROM ARROWHEAD, B.C.

R. K. Carnochan and C. S. Parsons

Test No. 156

A shipment consisting of two lots, one of 20 pounds, representing what was designated as black sands, and the other of 200 pounds, representing what was designated as yellow sands, was received at the Ore Dressing and Metallurgical Laboratories on November 11, 1921.

A sample cut out from each of these lots gave the following analysis:—

Black sands	Gold,	nil
	Silver,	trace
	Lead,	nil
Yellow sands	Gold,	0·52 oz. per ton
	Silver,	4·92 "
	Lead,	4·40%

The analysis of the black sand lot, which consisted of graphitic shale, showed practically no values, so that it was not necessary to conduct any further work on this lot.

The lot designated as yellow sands, which contained lead carbonate and carried values in gold and silver, could be classed as an ore, providing that the cost of mining and metallurgical treatment was below that of the value of the products which could be recovered from it. As this material showed values of about $16 per ton it had commercial possibilities, and an investigation was undertaken to determine metallurgical methods for the recovery of these values.

Experimental test work was conducted by table concentration to determine what percentage of recovery could be obtained in a concentrate that could be shipped as a smelter product. A concentrate, assaying 5 ozs. gold, 20 ozs. silver, and 40 per cent lead was the best that could be obtained, with a recovery of the gold values of 50 per cent, of the silver values, 20 per cent, of the lead values, 45 per cent. It was found that this treatment must be supplemented by other methods to recover more of the values in the ore.

Test work was conducted by table concentration, producing a concentrate, followed by sulphidizing the table tailing, and flotation for the recovery of more of the values. By this means, a total recovery was obtained of 60 per cent of the gold values, 24 per cent of the silver values, and 38 per cent of the lead values.

A further series of tests was conducted by table concentration, followed by cyanidation of the table tailings. This method of procedure gave the best recoveries. A concentrate was obtained similar to that given above, which with the recovery made by cyanidation, gave a total recovery of the gold values of 95 per cent, of the silver values, 63 per cent, and of the lead values, 45 per cent. This latter process of treatment seems to be the most adaptable to the ore.

Other test work is being conducted, and a report on the investigation will be contained in the Summary Report for next year.

(13)

THE CONCENTRATION OF THE RADIO-ACTIVE MINERAL EUXENITE

R. K. Carnochan

Test No. 159

The occurrence of the radio-active mineral euxenite in the feldspar quarry of lot 13, concession V, of the township of South Sherbrooke, county of Lanark, in the province of Ontario, has been described by Miller and Knight, who visited the property in June, 1917. A report was published in the American Journal of Science, Volume XLIV, September, 1917, and contains a description of the occurrence as seen on that date. The mineral was identified as euxenite by the Imperial Institute, London, England, who gave the analysis of the Canadian occurrence as follows:—

	%
Ta_2O_5	13·89
Nb_2O_5	12·73
TiO_2	27·70
ThO_2	1·34
Ce_2O_3, La_2O_3, etc.	0·62
Y_2O_3, etc.	25·64
Fe_2O_3	2·63
FeO	0·51

MnO	trace
PbO	0·20
U_3O_8	10·50
CaO	0·09
MgO	0·12
SiO_2	0·74
H_2O, etc	3·00
Total	99·71
Specific gravity	4·99

In the operation of the quarry since that date, more of the mineral has been exposed and shipments have been made to the Department of Mines for experimental work on the concentration of the mineral from the feldspar.

It was found that due to its specific gravity being almost 5, and that of feldspar between 2·5 and 2·75, there was no particular difficulty in making a separation by ordinary gravity means of concentration.

A shipment weighing 1,593 pounds net was received on November 29, 1921, from the Orser-Kraft Feldspar, Limited, Box 266, Perth, Ont. The ore came from the company's property at Maberley, Ont., and consisted of fair sized crystals of black euxenite in red orthoclase feldspar.

It was desired that the euxenite be separated from the feldspar, and that both be recovered in the form of clean products.

A number of specimens were picked out of the ore and the remainder was reduced in a jaw crusher to ¾″.

Product	Weight pounds	Per cent of heads
—¾-inch	1,582·00	99·80
Loss	3·25	0·20
Heads	1,585·25	100·00

The —¾″ material was crushed in rolls and then screened on 12 mesh by a shaking Ferraris screen, the oversize being fed back to the rolls until it all passed through the screen.

Product	Weight pounds	Per cent of heads
—12 mesh	1,472	92·86
Loss	110	6·94
—¾-inch	1,582	99·80

The −12 mesh material was sized on a Keedy sizer into four sizes: −8+20; −20+42; −42+86; and −86.

Product	Weight pounds	Per cent of heads
− 8 +20	394·0	24·86
−20 +42	535·5	33·78
−42 +86	260·5	16·43
−86	267·0	16·84
Loss	15·0	0·95
−12 mesh	1,472·0	92·86

Each of the four sizes from the Keedy sizer was tabled separately on a large Wilfley table, so as to make a euxenite concentrate and a feldspar tailing of each size. These products were collected as they came from the table in long settling boxes. The water from the tailing box was pumped to a Callow cone during the tabling of all the different sizes, and by this means a certain amount of slime was recovered. After tabling, all products were dried and weighed. The four concentrates produced were very good, being nearly all euxenite, with only a small percentage of feldspar. The tailings were composed of clean feldspar. The slime from the Callow tank was very fine, and drab in colour, and under the microscope was seen to be made up mostly of feldspar, with a little euxenite.

Product	Weight pounds	Per cent of heads	
Concentrate − 8+20	14·00	0·89	3·67
" −20+42	23·50	1·48	
" −42+86	10·50	0·66	
" −86	10·12	0·64	
Tailing − 8+20	370·00	23·34	83·45
" −20+42	504·00	31·79	
" −42+86	244·00	15·39	
" −86	205·00	12·93	
Slimes	42·00	2·65	
Loss	33·88	2·14	
Feed to table	1,457·00	91·91	

The concentrates, after sampling, and a sample representing in proper proportions the combined four sizes of tailings, were shipped to the Orser-Kraft Feldspar, Limited, Perth, Ont.

SUMMARY AND CONCLUSIONS

The euxenite concentrate produced equals 3·67 per cent of the ore treated.

The feldspar tailing product equals 83·45 per cent of the ore treated.

The slimes produced equal 2·65 per cent of the ore treated.

The loss in treatment equals 10·23 per cent of the ore treated.

A good separation of the euxenite from the feldspar can be made. Good clean products can be produced.

Dry crushing and tabling is a very suitable method of treating the ore, as submitted.

IV

OTHER TEST WORK OF THE DIVISION

W. B. Timm

Not included under the heading of investigations, test work was conducted, in some cases partaking of the nature of experimental work, on several shipments, as follows:—

A shipment of 100 pounds, on May 30, of what was supposed to be gold ore from the Grace Mining Co., Ltd., Vermilion Bay, Eagle lake, Ont.

A two-pound sample of screened asbestic sand, from the Windsor Asbestos Co., Ltd., Coleraine, Que., on July 19.

A shipment of 200 pounds of dolomite, on September 19, from the Grasselli Chemical Co., Flower Station, Ont.

A shipment of 2,200 pounds of diatomaceous earth on September 29, from E. A. D. Morgan, K.C., Rawdon, Que.

A shipment of 4,000 pounds of calcite, on November 23, from the Grasselli Chemical Co., Flower Station, Ont.

Supposed Gold Ore from the Grace Mining Co., Vermilion Bay.—The Grace Mining Company, Ltd., made application for the treatment of a few tons of gold bearing ore in the experimental laboratories of the Department of Mines. They were advised of the conditions under which experimental test work was conducted. Conforming to these conditions and advising that they had already a five-stamp battery and other equipment on the property, in which they had made test runs, but due to the varied information given them by a number of their representatives, they shipped 100 pounds of ore for test and check purposes. The shipment as received showed no gold, and only a trace of silver, so that test work as to methods of treatment was not necessary.

Screened Asbestic Sand from the Windsor Asbestos Co., Coleraine.—This sample was received for experimental purposes, to ascertain the amount of total fibre contained, and a report was submitted as follows:—

Weight of sample—900 grams, all used in test. Sample screened into different sizes and each size hand picked, or put over small suction table to remove the fibre. All rock plus twenty mesh was ground in ball mill for five minutes with twenty pounds of balls, screening and suction on sized material was repeated. This operation was repeated on any rock plus twenty mesh remaining, and the fibre removed after each grinding. The following results were obtained:—

Fibre+10 mesh	1 gram, representing	0·11%	by weight
Fibre−10 mesh	84 grams, "	9·33	"
Total fibre	85 " "	9·44	"
Rock	783 " "	87·00	"
Loss in handling	32 " "	3·56	"
Total	900 " "	100·00	"

Dolomite, from the Grasselli Chemical Co., Flower Station, Ont.—The object of the work on this shipment was to determine if this dolomite, which was of a pure white variety, could be ground to 250 mesh, and so meet the requirements of the paint and rubber trade. The whole shipment of 200 pounds, with the exception of a few pieces retained for specimens, was crushed in a jaw crusher and rolls and ground in small ball mills. The resultant product gave on screen analysis, the following:—

On 250 mesh	1%
Through 250 mesh	99%

The ground product was shipped back to the company to determine if it met the requirements of the trade.

Diatomaceous or Infusorial Earth from Rawdon, Que.—The work on this shipment was done for the Testing Laboratories of the Department of Public Works. The shipment as received consisted of 2,200 pounds in the natural state and required drying before grinding. The dried material was ground in a conical ball mill equipped with suction, to remove the fine material when disintegrated. The finished product was collected in a collector placed between the suction fan and the mill, and showed on screen analysis that all passed a 100 mesh screen, which was the fineness required. It was shipped to the Testing Laboratories of the Public Works Department.

Calcite from the Grasselli Chemical Co., Flower Station, Ont.—This shipment was from a deposit of pure white calcite in the vicinity of Flower Station. The object of the shipment was to determine the possibilities of grinding to 250 mesh to obtain a product suitable for the paint and rubber trade, and also to determine methods of grinding this material, as it was thought that the pyrite mill of the company could be used for the preliminary crushing, and with certain changes, for fine grinding. In this way the preparation of the material for the market could be accomplished with small additional cost in the way of new equipment.

A series of tests was made on the dry and wet grinding of the calcite. Our equipment, however, was not suitable for dry grinding. Reduction could easily be made to 100 mesh, but the reduction to 250 mesh was impossible on a large scale, with the present equipment. A few hundred pounds was obtained from the test operations, 95 per cent through 250 mesh, and this was shipped to the company to determine if it would meet the requirements of the trade.

A test was made on a ton of the calcite, by wet grinding in a conical mill, charged with pebbles, and in closed circuit with a bowl classifier. No trouble was experienced in grinding the material in this manner. A good tonnage was obtained from the mill, and the overflow of the bowl classifier gave a product practically all through 250 mesh. This product on being dried was slightly off colour, but, no doubt, could be used for certain purposes in the trade.

Note.—These tests were conducted by R. K. Carnochan, or under his immediate supervision.

V

ADDITIONAL EQUIPMENT FOR THE LABORATORIES

W. B. Timm

It is of the utmost importance that testing laboratories, where research and experimental work is conducted, be equipped with the latest and most improved type of machinery and apparatus for the metallurgical treatment of ores. To maintain an efficient laboratory in which test work can be carried on, the equipment must be up to date, and for research work it must be in advance of metallurgical practice. It is also of importance that the laboratory be equipped with special machinery and apparatus to cover the wide range of metallurgical problems that are undertaken. It is a simple matter to equip a laboratory to conduct test and research work on one or two particular classes of ore, but it is another problem to meet the needs of the whole industry. For these reasons it has been the policy of the Department to keep the laboratories well equipped, to meet, as far as possible, the requirements of the whole industry.

The equipment of the laboratories consists of two types of machinery and apparatus, namely, the small scale laboratory type for conducting preliminary investigation tests, and complete metallurgical tests on a small scale, and the large scale type of the smallest commercial size machinery, for conducting tonnage check tests, in which the procedure conforms very closely to actual metallurgical practice. On the laboratory type, series of tests are conducted under varying conditions and methods of procedure to determine the procedure most applicable, and when this is determined, a tonnage test is conducted to check results. Small scale tests are conducted on shipments of a few pounds up to 1,000 pounds of ore. Tonnage tests are conducted on shipments from 1,000 pounds up to a carload lot. A chemical laboratory consisting of an analytical and assay laboratory, equipped with modern appliances and apparatus for accurate and rapid determination of analyses and assays of ores and their metallurgical products, is maintained in connection with the testing and research laboratories.

Additional equipment purchased for the laboratories during the year was as follows:—

Standard Asbestos Testing Machine,
Dorr Simplex Classifier,
Magnetic Log Washer,
Roasting Furnace (muffle type).

In connection with the investigation on the milling of asbestos rock, it was necessary to purchase a standard asbestos testing machine, to test the fibre produced from test operations. This is the standard machine used in the asbestos districts, and the fibre produced from the mills is graded to conform to certain tests on this machine.

In connection with the fine grinding of metalliferous ores for flotation work, which in some cases require grinding to 200 mesh, and also for other classification problems, a Dorr simplex classifier equipped with bowl attachment was purchased. This machine was built specially for the testing laboratories, and can be used as an ordinary classifier, or where very fine grinding is necessary, with the bowl attachment. It was placed in closed circuit with a conical ball mill.

A magnetic log washer was purchased and installed for experimental work on the low grade iron ores. This is the latest type of magnetic machine used for the concentration of the low grade magnetic iron ores, having certain advantages for this purpose over the older types of magnetic separators. The addition of this machine to the laboratories makes the equipment fairly complete in the way of conducting experimental work on the beneficiation of the low grade iron ores. This equipment now consists of the necessary crushing and grinding machinery, magnetic cobber, Gröndal magnetic separator, magnetic log washer, laboratory sintering machine, and roasting devices.

A muffle furnace, consisting of four muffles, one above the other, in which the ore to be roasted can be fed to the top muffle and rabbled down to the bottom one where it is discharged, was built for roasting purposes in connection with the experimental work of the laboratories.

During the year certain changes were made in the laboratory equipment to provide for more efficient operation. A Roots blower operated in connection with the pneumatic flotation unit was transferred from the roaster building to the main laboratories. The pipe line leading from the blower to the flotation cells gave considerable trouble, due to vibration. This has now been overcome. An elevator was extended ten feet to elevate the pulp from the discharge of the conical ball mill to the Dorr classifier, and connections made from the classifier for sand oversize to be returned to the mill, and slime overflow to thickening tanks.

A 15 horse-power motor was purchased to drive the machinery in the upper portion of the laboratory and relieve the 25 horse-power motor on the ground floor, which was being overloaded.

A 40 horse-power motor was purchased and installed in the roaster building to operate the machinery and take the place of a locomotive type boiler and engine, which was formerly used for that purpose.

On account of the limited space in the laboratories for new equipment, several pieces of machinery and apparatus were crated and stored. When problems arise which require the use of this machinery, provision is made so that it can be set up again on its old foundation. This is not good policy, but it is the best that can be done under the present conditions, and until more accommodation is provided.

A fitting and repair shop was fitted up with machines and fittings obtained from the Department of Soldiers' Civil Re-Establishment, providing an up to date shop for repairs to the machinery and apparatus of the laboratories.

VI

THE WORK AND INVESTIGATIONS OF THE CHEMICAL LABORATORY

H. C. Mabee

Chemist in Charge

At the beginning of the year the chemical work of the Division was being done in the Mines Branch Chemical Laboratories, Sussex street. The restoration of the temporary frame building containing the chemical laboratories and offices of the Division on Booth street, which were destroyed by fire in February, 1920, was still in progress, and was not completed and the building ready for occupation until the middle of February, when work was resumed in our own laboratories.

Considerably more work was accomplished than in the preceding year, due to the increase in staff by the temporary appointment in July of B. P. Coyne, as associate chemist. A temporary technical officer was appointed as laboratory assistant, and his services were utilized in the chemical laboratories for a portion of the time.

As in past years, the laboratories were employed by the chemical staff in the determination of analyses and assays of ores and experimental test products from the investigations being carried on by the engineering staff of the Division. Assistance was given in the solution of the metallurgical problems covered by the investigations.

The total number of samples analysed and reported on during the year was 838, involving 3,000 chemical determinations. Of this number the largest proportion was from the experimental and research work on the investigations. These samples represented ores and minerals from localities throughout the Dominion, as the following list will indicate:—

Brass slag—8 samples—Refuse product from brass foundry. Determined, tin, antimony, lead, copper, and zinc.

Copper ores—371 samples—Ores from northern Manitoba, Anyox, B.C., and northern Ontario. Determined, copper, copper and nickel and zinc, gold and silver.

Ferro-molybdenum—2 samples—from Mines Branch. Determined, molybdenum, iron, carbon, sulphur, and copper.

Graphite ores—13 samples from Quebec. Determined, carbon.

Graphite shale—4 samples from New Brunswick. Determined, graphitic carbon, volatile and combustible.

Gold ores—33 samples—Ores from Trout lake, B.C., northern Ontario, northern Manitoba, and Quebec. Determined, gold, silver and lead.

Lead-zinc ore—139 samples—from Kimberley, B.C. Determined, lead, zinc, and silver.

Magnesite—23 samples—from Calumet, Que. Determined, lime and magnesia.

Molybdenite—29 samples—from Ontario and Quebec. Determined, molybdenite, molybdic oxide, copper, arsenic, iron, and lime.

Nickel—83 samples—from northern Ontario. Determined, nickel.

Nickel-copper—71 samples—ores from Sudbury, Ont. Determined, nickel and copper.

Platinum—8 samples—from the Cariboo district, B.C. Determined, platinum and gold.

Silver—54 samples—slimes from Cobalt ores and ores from Cape Breton. Determined, silver.

Besides the regular assays and chemical analyses performed on the numerous ores and minerals as indicated above, special investigation was also carried on in the case of some of the more complex ores, with the view of ascertaining the presence of any rare elements or precious metals.

THE RECOVERY OF THE METALS OF THE PLATINUM GROUP FROM PLACER CONCENTRATES

Part I

In 1919 an investigation was undertaken on a small shipment of placer concentrates for the recovery of the metals of the platinum group. The experimental work consisted of trying a new method for the amalgamation of the platinum group metals, as follows:—

The dressed plates were sprinkled with finely powdered zinc amalgam, made by pouring molten zinc into mercury, and grinding the amalgam thus produced to powder. The platinum sands were ground to 150 mesh and fed over the plates in the form of a wet pulp containing a weak solution of copper sulphate and sulphuric acid; an electro-chemical action takes place, forming a permanent coating of copper on the platinum, which in passing over the plates is held by the mercury. Due to the limited supply of material submitted and the small quantity of platinum in the sample, the results obtained were doubtful as to accuracy. The results were as follows:—

Pulp before amalgamation	Au. 0·10 oz.;	Pt. 0·003 oz.
Pulp after amalgamation	Au. 0·06 oz.;	Pt. 0·001 oz.
Extraction	Au. 40 p.c.	Pt. 66 p.c.

A larger sample of 450 pounds of concentrates from the same locality was received, and in the preparation of a sample cut out from the shipment for assay, it was found that all the metals of the platinum group were caught on a 100 mesh screen. The assay of this sample was —Au. 0·41 oz; Pt. 0·032 oz. —100 mesh material after removal of metallics —Au. 0·39 oz.; Pt. trace.

The results of this assay showed that the metals of the platinum group in these particular concentrates were free, and occurred in the form of fine flake-like particles coarser than 100 mesh. This finding was substantiated by further work. In order to obtain some material high enough in platinum content so that the above method could be tried for the recovery of the platinum group metals, the remainder of the shipment was ground in ball mills for two hours and screened on a 100 mesh screen. This gave the following products:—

Product	Weight lbs.	Au. oz. per ton	Pt. oz. per ton
+100 mesh	107	0·90	0·105
−100 mesh	278	0·10	trace

In the assay of the + 100 mesh material from a sample of 1,518 grams ground to pass 100 mesh, 10 grams of metallics were caught on the screen, which lot was assayed separately from the material passing through the screen. This assay gave when figured to the ton of head sample, of +100 mesh product:—

Metallics	Au. 0·48 oz.	Pt. 0·101 oz.
−100 mesh	Au. 0·42 oz.	Pt. 0·005 oz.
Total	Au. 0·90 oz.	Pt. 0·106 oz.

These results check the former ones, and prove that the values of the platinum group metals in this concentrate can be recovered as metallics by grinding to 100 mesh.

The +100 mesh product which shows values in gold of 0·90 oz. per ton, and in platinum group metals of 0·105 oz. per ton, will be used for experimental work in recovering the values by the method described above. This work will be conducted next year.

Part II

Experimental work was conducted by making fusion tests on the placer concentrates, which consist of black sands or iron oxides, chiefly the magnetic oxide containing gold and metals of the platinum group. In some cases the precious metals are locked up in the metallic oxides, and the usual methods of amalgamation have failed, even on fine grinding, to make a satisfactory recovery of these metals.

A series of experiments was undertaken along similar lines to those employed in the regular crucible assay, namely, of fusing and slagging off the iron content and gangue, and collecting the precious metals in a lead button by the addition of galena to the charge. Several mixtures were employed, using a quantity of limestone, silica, and charcoal as fluxing materials, a sufficient quantity of galena containing about 60 per cent metallic lead, and about 50 ozs. of silver. It was hoped to melt this mixture at a fairly high temperature to form a liquid slag, and produce metallic lead for the collection of the precious metal values. The fusions were made in a Case muffle furnace, but it was found that a much higher temperature was required than it was possible to obtain in this furnace, to produce a liquid slag. Further experimental work along this line will be conducted in other types of furnaces.