CANADA

DEPARTMENT OF MINES

HON. CHARLES STEWART, MINISTER; CHARLES CAMSELL, DEPUTY MINISTER

MINES BRANCH

JOHN McLEISH, DIRECTOR

INVESTIGATIONS IN ORE DRESSING AND METALLURGY

(Testing and Research Laboratories)

1928

OTTAWA
F. A. ACLAND
PRINTER TO THE KING'S MOST EXCELLENT MAJESTY
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Annual reports on Mines Branch investigations are now issued in four parts, as follows:—

- Investigations of Mineral Resources and the Mining Industry.
- Investigations in Ore Dressing and Metallurgy (Testing and Research Laboratories).
- Investigations of Fuels and Fuel Testing (Testing and Research Laboratories).
- Investigations in Ceramics and Road Materials (Testing and Research Laboratories).

Other reports on Special Investigations are issued as completed.

MINES BRANCH INVESTIGATIONS IN
ORE DRESSING AND METALLURGY, 1928

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm

Chief of Division

During the year 1928 there was a considerable increase over the previous year in the amount of experimental test and research work on ore treatment by the investigative staff of the Division in the Ore Testing Laboratories. The increased activity in the mining industry in the development of new and old mining properties, and the progress being made in metallurgical practice have resulted in many requests for test work. The investigative work has been increased to such an extent that the work in progress and on hand is sufficient to keep the present staff busily engaged for a year or more. To meet the demands of the mining public increased laboratory facilities and staff are required so that more investigative work can be undertaken at the same time.

Progress was made during the year on the erection of a new pyrometallurgical laboratory, primarily for work on iron and steel. This laboratory will be equipped with melting furnaces, mainly electric of the high frequency and arc types, electric heat-treating furnaces, and the standard laboratory physical testing and metallographic equipment, all of laboratory and semi-commercial scale, and will also include roasting, calcining, sintering and metallizing furnaces for ferrous and non-ferrous ores. With these facilities it is hoped to extend to the iron and steel industry, the same degree of co-operation that has proven so advantageous to the non-ferrous mining industry. T. W. Hardy, a ferrous metallurgist of high standing and repute, will have charge of the investigations in the iron and steel section.

The co-operative arrangements entered into with the Base Metals Extraction Company, Limited, and the Cassel Cyanide Company of Canada were still existent. Progress was made by the metallurgists and chemists of the Base Metals Extraction Company in the working out of the details of their process for the treatment of bulk concentrates from base-metal sulphide ores, such as the copper, lead, zinc, and iron admixtures. Special attention was paid to the treatment of the residues. The company is looking into the possibility of establishing, in eastern Canada, a plant for the treatment of such concentrates from the more complex ores which are difficult to concentrate by selective flotation methods. A synopsis of the investigative work is given in Section IV dealing with hydrometallurgical and electrochemical investigations. During the year the Cassel

Cyanide Company of Canada maintained a research fellow in the Ore Testing Laboratories on the use of cyanide for the selective flotation of base-metal sulphide ores. The investigative work was chiefly confined to the copper-zinc-iron admixtures such as Amulet, Abana, and Sherritt-Gordon ores. A pamphlet dealing with the milling and concentration of metalliferous ores will shortly be issued.

An investigation of Canadian graphite ores for the production of crucible flake was conducted on two carload shipments, one from the Buckingham district, Quebec, and the other from Guenette in the Mount Laurier district of Quebec. The best economic method of concentration, compatible with high recovery of coarse flake was worked out. The results showed that rod-milling was preferable to ball-milling and that a combination of table concentration and flotation was essential to the production of high-grade products with maximum recovery of coarse flake. The concentration results are given in Section III dealing with the dressing of non-metallics.

An investigation of the physical properties of calcined gypsum was undertaken during the year. Some samples of gypsum from different Canadian occurrences were examined. Calcining tests and physical tests of the calcined gypsum were made. A summary of the results is given in Section III dealing with non-metallics; and the detailed results are given in Mines Branch Rept. No. 714, "The Gypsum Industry of Canada."

Large-scale continuous tests were made on the metallization of the iron content in ilmenite ore to determine the suitability of a rotary kiln type of furnace for the preparation of the ore for the manufacture of titanium pigment. The results of small-scale laboratory tests, given in the Reports of Investigations for 1925 and 1926, indicated the commercial feasibility of utilizing Canadian ilmenites by metallizing the iron content and, either treating the metallized ore with waste sulphuric liquors from the titanium pigment process, and thus obtaining a high-grade titanic oxide, or dissolving the metallized iron with ferric chloride and electrolysing the iron. The results which show 92 to 95 per cent metallization of the iron content are given in Section IV.

A large part of the investigative work of the division was on the selective flotation of base-metal sulphide ores and the treatment of refractory gold ores. The Ore Testing Laboratories are well equipped for this work with various types of flotation machines for making small batch tests—continuous grinding and flotation tests on a basis of 100 pounds an hour—and continuous grinding and flotation tests on a basis of 1,000 pounds an hour. The continuous units consist of three sets of flotation machines for the selective float of three minerals. For cyanide work, the equipment consists of the regular small bottle and mechanical agitators, and a continuous grinding and cyanide unit on a basis of 10 to 15 pounds per hour.

The investigations were carried out under the direction of W. B. Timm, Chief Engineer, Division of Ore Dressing and Metallurgy.

Section II of the Summary Report contains the results of investigations on the milling and concentration of metallic ores conducted by C. S. Parsons, A. K. Anderson, and J. S. Godard.

Section III contains the results of the investigations on non-metallics conducted by R. K. Carnochan and R. A. Rogers. The work on the concentration of flake graphite was conducted by C. S. Parsons and R. K. Carnochan.

Section IV contains the results of the investigations by R. J. Traill, W. R. McClelland, and J. D. Johnston, and a summary of the results of investigations conducted by W. E. Harris of the Base Metals Extraction Co., Limited.

Section V contains the work and investigations in the chemical laboratories, conducted by B. P. Coyne, R. A. Rogers, H. L. Beer, and L. Lutes, under the direction and supervision of H. C. Mabee, Chief Chemist of the Division.

Section VI contains a paper on the proposed work of the iron and steel section of the division giving the equipment for the new pyrometallurgical laboratory.

The operation and maintenance of the laboratories was performed by a staff of mill operators, laboratory assistants, and mechanics, under the immediate supervision of B. M. Derry, mill foreman.

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 products received at the Ore Testing Laboratories for experimental test and research. The tabulated statement includes the report numbers, class of ore or product, source of shipment, name of shipper, and weight of shipment.

Report No.	Page	Ore or product	Source of shipment	Shipper and address	Weight
					lb.
291	9	Lead..........	Ogema mine, Dorion, Ont.	North American Lead Refining Company, Dorion, Ont.	2,500
292	6	Copper-gold.....	Horne mine, Noranda, Que.	H. C. Cooke, Geological Survey, Ottawa.	120
293	8	Copper-zinc.....	Abana mine, Dupuy, Que.	Abana Mines, Ltd., Dupuy, Que.	2,186
294	10	Gold..........	Grace mine, Michipicoten, Ont.	Power and Mines Corporation, Ltd., Hawk Junction, Ont.	200
295	15	Gold..........	Malartic mine, Fournière tp., Que.	Malartic Gold Mines, Ltd., Amos, Que.	5,000
296	24	Copper-molybdenum.	Golconda mine, Keremeos, B.C.	D. McEachern, Keremeos, B.C.	100
297	26	Copper-zinc.....	Sherritt-Gordon mine, Cold Lake, Man.	Sherritt-Gordon Mines, Ltd., Toronto, Ont.	46,000
298	37	Copper-zinc.....	Amulet mine, Rouyn, Que.	Amulet Mines, Ltd., Rouyn, Que.	43,000
299	48	Lead-zinc.......	Larchwood, Ont....	Sudbury Concentrating and Mining Co., Ltd.	365
300	58	Gold..........	Second Relief mine, Erie, B.C.	R. O. Oscarson, Erie, B.C......	50
301	62	Copper-gold.....	Oxford lake, northern Manitoba.	Ventures, Ltd., Toronto, Ont...	20
302	65	Copper-zinc-pyrite.	Aldermac mine, Aldermac, Que.	Aldermac Mines, Ltd., Aldermac, Que.	2,350
303	69	Silver-lead-barite.	Homestake mine, Adams Lake, B.C.	Wentworth F. Woods, Kamloops, B.C.	150

Report No.	Page	Ore or product	Source of shipment	Shipper and address	Weight
					lb.
304	75	Arsenical gold	Cameron island, lake of the Woods, Ont.	J. G. Cross, Port Arthur, Ont.	125
305	79	Copper	Hollinger-Kamiskotia mine, Ont.	G. J. MacKay, Kingston, Ont.	200
306	82	Iron	Bristol iron mine, Bristol Mines, Que.	Blackburn Bros., Ottawa, Ont.	130,000
307	88	Ilmenite	Ivry mine, Ivry, Que.	McArthur-Irwin, Ltd., Montreal, Que.	20,000
308	95	Asbestos	Sproat mountain, Arrowhead, B.C.	Lardeau Mines Exploration Co., Vancouver, B.C.	300
309	96	Limestone	L'Etang, N.B.	Canadian Calcite Mines, Ltd., St. George, N.B.	1,320
310	98	Talc	Robertsonville, Quebec.	Robertsonville Soapstone Quarries, Leeds Station, Que.	3,600
311	99	Calcite	Frontenac lead mine Perth Road, Ont.	Industrial Processes Development Co., Ltd., Kingston, Ont.	10,000
312	102	Oyster shells	Orangedale, C.B.	Wm. Mombourquette, Orangedale, C.B.	115
313	103	China clay	St. Rémi, Que	R. A. Bryce, Toronto, Ont.	60,000
314	110	Graphite	Buckingham, Que.	Crucible Graphite Co., Buckingham, Que.	37,115
315	118	Graphite	Guenette, Que	Canadian Graphite Corporation, Guenette, Que.	28,000
316	124	Garnet	Labelle, Que	Labelle Nickel and Garnet Co., Ltd., Montreal, Que.	10,363

II

REPORTS OF INVESTIGATIONS: ORE DRESSING AND METALLURGICAL LABORATORIES

Report No. 291

CONCENTRATION OF A LEAD ORE FROM THE OGEMA MINE, DORION, ONTARIO

C. S. Parsons

Shipment.—A shipment of 22 bags of ore from the Ogema mine was received December 24, 1927, from the North American Lead Refining Co., Dorion, Ont.

Characteristics and Analysis.—The ore contains lead in the form of galena. The gangue is principally calcite, and only traces of gold and silver are present. Analysis showed lead—11·80 per cent.

Purpose of Tests.—The sample was submitted for the purpose of determining the best method of concentrating the ore.

EXPERIMENTAL TESTS

A part of the galena was found to be disseminated throughout the gangue in relatively fine particles, necessitating crushing to −40 mesh to free them. For this reason concentration by flotation was considered the best method of treatment. The flotation tests were made as follows: One small-scale test on 2,000 grammes of the ore, and one large-scale test on approximately 1,200 pounds of ore.

Test No. 1

A lot of 2,000 grammes of ore crushed to pass 14 mesh was ground in a small ball mill to pass 48 mesh and until 60 per cent was through 200 mesh.

Reagents.—

Soda ash 0·5 lb/ton added to ball mill.
Xanthate 0·2 lb./ton; pine oil 0·2 lb./ton added to flotation cell.

Product	Weight, per cent	Assay				Per cent of values			
		Pb, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Pb	Zn	Au	Ag
Concentrate......	24·3	65·45	2·84	0·02	0·66	98·8	64·4	99·0	99·0
Tailing..........	75·7	0·16	0·05	....	....	1·2	35·6	....	....

Test No. 2

This test was run in a small continuous grinding and flotation unit which treated 150 pounds of ore an hour. The same reagents were used as in Test No. 1.

Product	Assay			Recovery, per cent
	Pb, per cent	Au, oz./ton	Ag, oz./ton	Pb
Head	11·80	trace	trace	
Lead concentrate	59·4	0·02	0·66	99·0
Tailing	0·07	none	none	1·0

CONCLUSIONS

These tests show that the sample of ore submitted can be readily concentrated by flotation yielding a high-grade lead product containing practically all of the lead in the ore. The sample submitted does not contain commercial quantities of zinc.

Report No. 292

DETERMINING THE ASSOCIATION OF THE GOLD VALUES IN ORE SAMPLES FROM THE HORNE MINE, NORANDA, QUEBEC

C. S. Parsons

Shipment.—Two samples of ore, weight 120 pounds, were received from H. C. Cooke of the Geological Survey on November 16, 1927. Both samples were from the Horne Mine, Noranda, Quebec.

Characteristics and Analysis.—Lot No. 1 consisted of 60 pounds of chlorite schist mineralized with iron sulphides. No native gold was visible.

Lot No. 2 consisted of 60 pounds of rhyolite veined with chalcopyrite and pyrite. This lot contained samples some of which carried principally veins of pyrite, and others principally veins of chalcopyrite. These were separated into two lots, A and B. Lot A consisted of pieces containing veins of chalcopyrite, and Lot B the pieces containing veins of pyrite.

—	Lot 1	Lot 2 (A)	Lot 2 (B)
Gold	3·55 oz./ton	0·02 oz./ton	0·02 oz./ton
Copper	0·06 per cent	2·57 per cent	0·11 per cent
Sulphur	1·79 "	3·95 "	1·786 "

Purpose of Experimental Tests.—The work was requested for the purpose of determining, if possible, with what mineral or minerals the gold is associated.

EXPERIMENTAL TESTS

LOT No. 1

A test was made to determine if any gold were present in a form which could be amalgamated. A sample of the ore ground to −14 mesh was taken and ground with 10 per cent by weight of mercury in a small ball mill. After 35 minutes of grinding the ore was reduced to approximately all −48 and 60 per cent −200 mesh, the mercury was added and the grinding continued for 15 minutes. At the end of this time the ore was ground to all −65 mesh and 80 per cent −200 mesh. The mercury was then separated from the sample and the tailings assayed for gold.

Results.—

Head	3·55 oz./ton
Tailing	2·78 "
Per cent of gold amalgamated	2·7 per cent

Tests Nos. 2 and 3

Two flotation tests were made to concentrate the sulphides:—

Test No.	Product	Weight, per cent	Assay		Per cent of values	
			Sulphur, per cent	Au, oz./ton	Sulphur	Au
2	Flotation concentrate	14·6	12·15	17·38	96·3	72·5
	Flotation tailing	85·4	....	1·13	3·7	27·5
3	Flotation concentrate	9·0	19·5	26·00	96·0	67·0
	Flotation tailing	91·0	0·08	1·28	4·0	33·0

It will be observed that in Test No. 2, by flotation of 14·6 per cent of the ore, 96·3 per cent of the sulphur was removed in the form of metallic sulphides and only 72·5 per cent of the gold. In Test No. 3, in floating 9 per cent of the ore, 96 per cent of the sulphur was removed but only 67 per cent of the gold. In order to determine if any of the gold remaining in the flotation tailing were free the tailing was amalgamated. No gold was removed by amalgamation, showing that the free gold in the ore had been removed with the sulphides. The sulphide concentrate was tested for tellurides, after it was found that the gold in the flotation test tailing could not be readily extracted by cyanidation and, roughly, 0·25 per cent of tellurium was found.

LOT No. 2

(A) Chalcopyrite Portion.—A sample of the material was concentrated by flotation. The object of the test was to make as high a copper product as possible in order to determine how much of the gold reported with the copper product.

Product	Weight, per cent	Assay			Per cent of values		
		Cu, per cent	Sulphur per cent	Au, oz./ton	Cu	Sulphur	Au
Copper concentrate	7·9	31·0	....	0·16	94·6	....	51·2
Copper middlings	3·1	1·13	....	0·03	1·4	....	3·8
Iron concentrate	5·5	0·05	0·56	0·01	1·6	....	33·8

These results show that 51·2 per cent of the gold reported with the copper concentrate. The copper concentrate was very high grade, approximating nearly pure chalcopyrite. It is evident that only part (about one-half) of the gold is associated with the copper. It is also probable that any gold set free during crushing also reported with the copper concentrate.

(B) Pyrite Portion.—No work, other than analysis, was performed on this portion of Lot No. 2. It was found that the amount of gold present was the same as in the chalcopyrite portion. This in itself proves conclusively that the gold is not entirely associated with the copper.

CONCLUSIONS

The results indicate that the gold is associated with both the pyrite and the chalcopyrite. It may also be of interest to state that from work done on Horne F ore-body, there is an indication that two classes of pyrite in addition to pyrrhotite are present, and that only one of these pyrites contains the gold. Also that only 45 to 50 per cent of the gold would be recovered with a clean chalcopyrite concentrate. The pyrrhotite has been removed by magnetic separation and found not to contain gold.

Report No. 293

THE SELECTIVE FLOTATION OF COPPER-ZINC ORE FROM THE ABANA MINE, DUPUY, QUEBEC

A. K. Anderson and J. S. Godard

Shipment.—Twenty-four bags of ore weighing 2,186 pounds were received on March 22, 1928, from the Abana Mines, Limited, Dupuy, Quebec.

Characteristics and Analysis of the Ore.—The material consisted of mixed sulphides of copper, zinc, and iron intimately associated. The shipment was crushed to pass 8 mesh by passing through jaw crusher and rolls. A representative sample was cut out, which showed on analysis:

Copper..................	5·61 per cent	Gold....................	0·05 oz./ton
Zinc.....................	5·71 "	Silver...................	4·21 "
Iron.....................	26·50 "		

Purpose of Test.—The object of the tests was to determine whether the ore was amenable to treatment by selective flotation of the copper and zinc sulphides.

EXPERIMENTAL TESTS

The −8-mesh product was fed at the rate of 100 pounds per hour from a belt feeder to a rod mill in closed circuit with an Akins classifier. The grinding circuit was adjusted to yield a classifier overflow containing 1 part of ore to 3 parts of water, approximately 85 per cent of the material passing through 150 mesh. Soda ash, thiocarbanilide, and zinc sulphate were added to the ball mill while grinding. Cyanide was added to the classifier overflow which passed to a conditioning tank where the pulp was kept in suspension and in contact with the cyanide for approximately

20 minutes. The frothing oil, cresylic acid, was added to the overflow of this tank passing to an 8-cell, M.S. type flotation machine where a rougher copper concentrate was taken off. This concentrate was cleaned in a 3-cell, Denver Sub-A machine, the middling being returned to the head of the rougher cell. The tailing from the copper flotation cell was passed to a second conditioning tank where an additional amount of cyanide was added. Copper sulphate and potassium xanthate were added to the pulp overflowing from this tank and passed to a 6-cell, Denver Sub-A machine. This produced a rougher concentrate which was cleaned in a Callow pneumatic cell; the middling was returned to the head of the rougher cell.

Two tests were made on this ore, each for a period of six hours, and samples taken at regular intervals. The tests in detail follow:

Test No. 1

Reagents.—

To ball mill—		
Soda ash	5·1	lb./ton
Zinc sulphate	1·0	"
Thiocarbanilide	0·09	"
To copper conditioning tank—		
Cyanide	0·42	"
To copper flotation machine—		
Cresylic acid	0·42	"
To zinc conditioning tank—		
Cyanide	0·29	"
To zinc flotation machine—		
Copper sulphate	0·89	"
Xanthate	0·04	"

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate	25·03	21·86	4·28	0·13	14·55	97·50	18·74	78·95	85·05
Zinc concentrate...	8·40	0·52	51·60	0·02	1·73	0·71	75·83	2·63	3·27
Copper tailing......	....	0·20	6·47	0·02	0·96	....	....	....	....
Zinc tailing........	66·57	0·16	0·46	0·01	0·75	1·79	5·43	18·42	11·68
Head..............	....	5·61	5·71	0·05	4·21	....	....	....	....

Test No. 2

This test was carried out under conditions much the same as those of Test No. 1, the main difference being in the addition of zinc sulphate.

Reagents.—

To ball mill—		
Soda ash	3·8	lb./ton
Thiocarbanilide	0·07	"
To copper conditioning tank—		
Cyanide	0·31	"
To copper flotation cell—		
Cresylic acid	0·28	"
To zinc conditioning tank—		
Cyanide	0·30	"
To zinc flotation cell—		
Copper sulphate	0·72	"
Xanthate	0·04	"

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate	24·62	22·18	4·41	0·16	13·08	97·33	19·09	76·92	77·97
Zinc concentrate...	7·85	0·87	50·30	0·025	2·80	1·25	69·18	3·85	5·33
Copper tailing......	....	0·18	6·06	0·02	1·18	....	....	....	....
Zinc tailing........	67·53	0·12	1·00	0·02	1·02	1·42	11·73	19·23	16·70
Head..............	....	5·61	5·71	0·05	4·21	....	....	....	....

CONCLUSIONS

As shown in the tabulated results, both tests show a satisfactory recovery of both copper and zinc in concentrates of good grade. The precious metal values in the ore are largely recovered with the copper concentrate.

The separation of the zinc from the copper in this ore presents no great difficulties as it responds quite rapidly to treatment by flotation under conditions indicated in these tests. Any variations in the results secured in the two tests are due more to manipulation of the machines than to changes in reagent additions. Care must be exercised, however, to closely control the amounts of thiocarbanilide, cresylic acid, and xanthate used, or low-grade concentrates will result.

Report No. 294

EXPERIMENTAL TESTS FOR THE RECOVERY OF GOLD FROM THE ORES OF THE GRACE MINE, HAWK JUNCTION, ONT.

A. K. Anderson

Shipment.—Two bags of ore, gross weight 200 pounds, were received at the Ore Testing Laboratories, August 13, 1928, from the Power and Mines Corporation, Ltd., Hawk Junction, Ont. The material consisted of a hard, white quartzite carrying free gold and small amounts of iron sulphides.

Sampling and Analysis.—Approximately 100 pounds of the ore were fed through jaw crushers and rolls and crushed to pass a 14-mesh screen. A representative head sample was secured by passing the crushed ore through a Jones riffle sampler. This sample was found to contain 0·46 ounce gold per ton, and 0·15 ounce silver per ton.

Purpose of Tests.—The object of the investigation was to determine what recovery of the gold can be obtained by amalgamation, the proper fineness of crushing to secure the best results, and the effect of concentrating the contained sulphides.

EXPERIMENTAL TESTS

All tests, unless otherwise specified, were conducted on 1,000-gramme lots of ore ground in porcelain mills containing iron balls. Amalgamation was accomplished by removing the balls after grinding, adding 100 grammes of mercury, and rotating the mill for one hour. The details of the tests follow:

Test No. 1

A lot of 1,000 grammes of ore, −14 mesh, was ground ½ hour in a pebble jar. The pebbles were removed and amalgamated with 100 grammes of mercury.

Results.—

—	Au, oz./ton	Ag, oz./ton
Heads	0·46	0·15
Amalgamation tailing	0·05	0·03
Recovery	89·1 per cent	80·0 per cent

Grinding.......................12·9 per cent + 150 mesh; 87·1 per cent − 150 mesh.

The test shows that most of the gold can be recovered by amalgamation.

Test No. 2

One thousand grammes of ore, −14 mesh, ground ¾ hour in pebble jar; pebbles were removed, and ore amalgamated with 100 grammes of mercury.

Results.—

Heads	0·46 Au, oz./ton
Amalgamation tailing	0·05 "
Recovery	8·91 per cent

Grinding.................5·3 per cent + 150 mesh; 94·7 per cent − 150 mesh.

This test shows that no increase in recovery is obtained by grinding finer than that shown in Test No. 1.

Test No. 3

One thousand grammes of ore, −14 mesh, ground 1 hour in pebble jar; pebbles were removed, and ore amalgamated with 100 grammes of mercury.

Results.—

Heads	0·46 Au, oz./ton
Amalgamation tailing	0·05 "
Recovery	89·1 per cent

Grinding.................1·2 per cent + 150 mesh; 98·8 per cent − 150 mesh.

This test bears out the fact shown in Test No. 2, that no benefit is derived by extremely fine grinding.

Test No. 4

In this test an endeavour was made to determine the recovery obtained by amalgamating a product closely approaching that produced by a stamp mill. The ore was crushed and screened to pass 35 mesh and amalgamated in a pebble jar.

Results.—

Heads	0·46	Au, oz./ton
Amalgamation tailing	0·11	"
Recovery	76·1	per cent

Screen Analysis of Tailing.—

Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
− 35+ 65	38·54	0·15	47·4
− 65+100	12·04	0·12	11·8
−100+150	11·52	0·12	11·3
−150+200	9·37	0·11	8·4
−200	28·53	0·09	21·1

It is to be noted that 47·4 per cent of the values remaining in the amalgamation tailing is contained in the +65-mesh material and 70·5 per cent of the values in the +150-mesh product. It appears from the previous tests that fine grinding is necessary to free the gold in order to recover it by amalgamation.

Test No. 5

This test is similar to Test No. 4, except that the ore was crushed through 65 mesh prior to amalgamation:

Results.—

Heads	0·46	Au, oz./ton
Amalgamation tailing	0·08	"
Recovery	82·6	per cent

Screen Analysis of Tailing.—

Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
− 65+100	20·40	0·08	19·9
−100+150	18·18	0·11	24·4
−150+200	14·48	0·09	15·9
−200	46·94	0·07	39·8

This test, compared with the preceding one, shows that finer grinding yields a higher recovery of the gold. The analysis of the tailing also shows that gold is not freed in sizes larger than 150 mesh.

Test No. 6

This test was performed with the object of determining what recovery could be obtained by passing the amalgamation tailing over a concentrating table. One thousand grammes of the ore were crushed to pass a 35-mesh screen and amalgamated in a pebble jar. After removal of the mercury the tailing was passed over a small Wilfley table:

Results.—

Product	Weight, per cent	Assay, oz./ton	Per cent of values
Heads	....	0·46	....
Amalgamation	....	....	80·4
Amalgamation tailing	100·0	0·09	....
Concentrate	1·8	1·66	5·9
Tailing	98·2	0·07	13·7

Screen Analysis.—

—	Weight, per cent	Assay, oz./ton	Per cent of values
Table tailing—			
− 35+ 65	20·4	0·13	30·1
− 65+100	15·8	0·10	17·9
−100+150	11·2	0·06	7·6
−150+200	9·9	0·05	5·6
−200	42·7	0·08	38·8

This test shows that an additional recovery of the gold amounting to 40 cents per ton can be secured by concentration following amalgamation. A pyrite concentrate is obtained which represents 1·8 per cent of the total weight of ore treated and which yields 1·66 ounces gold per ton. The screen analysis of the table tailing shows 48·0 per cent of the gold not recovered to be contained in the +100-mesh material which is 36·2 per cent of the total by weight. The higher assay of the −200-mesh product may be due to fine float gold passing over the table into the tailing.

Test No. 7

This test is similar to Test No. 6 except an attempt was made in grinding to reduce the percentage of the +100-mesh material, keeping the percentage of finer material approximately the same. One thousand grammes of ore, −14 mesh, were ground 20 minutes in a pebble jar with iron balls and amalgamated. The tailing, after removal of mercury, was passed over a Wilfley table:

Results.—

Product	Weight, per cent	Assay, oz./ton	Per cent of values
Heads	....	0·46	....
Amalgamation	....	....	80·4
Concentrate	1·47	2·44	7·5
Tailing	98·53	0·06	12·1

Screen Analysis.—

Product	Weight, per cent	Assay, oz./ton	Per cent of values
+100	19·5	....	....
+150	26·5	....	....
+200	17·4	....	....
−200	36·6	....	....

It will be noticed that in both tests the amount passing 150 mesh is approximately the same, but the material remaining on 100 mesh in Test No. 7 is much lower than that in Test No. 6, being 19·5 and 36·2 per cent respectively. In each case the recovery by amalgamation is the same. Test No. 7 shows a slightly lower final tailing with a table concentrate higher in gold, and, although of less bulk than that of Test No. 7, shows a higher recovery. It appears from these two tests that better results are to be expected from material crushed finer than 35 mesh, without excessive production of fines.

Test No. 8

In this test the effect of cyaniding the amalgamation tailing was studied. A lot of 750 grammes of ore, −14 mesh, was ground in a pebble jar in a 1:1 pulp with 0·15 per cent cyanide solution and 2·6 pounds lime per ton. After grinding mercury was added and the ore amalgamated. The amalgam was then removed and the tailing agitated for 24 hours with 0·15 per cent cyanide solution and an additional 2·6 pounds lime per ton.

Results.—

	Assay, oz./ton
Head	0·46
Tailing after cyaniding	0·03

Recovery, amalgamation + cyanidation, 93·5 per cent.

Screen Analysis.—

Mesh	Weight, per cent	Assay, oz./ton
+200	1·7	0·12
−200	98·3	0·03

Consumption.—KCN, 3·75 lb./ton; CaO, 4·9 lb./ton.

This shows that the ore is amenable to cyanidation, an additional recovery of 40 cents per ton being secured by cyanidation of the amalgamation tailing.

CONCLUSIONS

Test No.	Grinding +150	Per cent recovery by amalgamation
1	12·9	89·1
2	5·3	89·1
3	1·2	89·1
4	62·1	76·1
5	38·6	82·6

The above table shows that up to a certain point the gold recoverable by amalgamation is not entirely freed in the coarser meshes.

The fineness of crushing necessary appears to be at least −100 mesh, preferably −150 mesh as indicated in Tests Nos. 4, 5, and 6. In Test No. 6 where the ore was coarsely crushed, amalgamated, and concentrated, 48 per cent of the values remaining in the residues was found in the +100-mesh material.

This fine crushing cannot be obtained with high efficiency from stamp mills as −30 mesh is about as fine as can be profitably produced. Tests Nos. 4 and 6 show that from 76 to 80 per cent of the gold can be recovered by amalgamation at this mesh. An additional recovery of 5·9 per cent can be secured by concentration of the amalgamation tailing. This product, however, is in a form which requires further treatment to recover the contained gold. No benefit is to be expected by returning this product to the mill.

As indicated in Test No. 8 the ore yields to treatment by cyanide, showing a higher recovery than by amalgamation and concentration combined.

It appears that stamp mills for crushing will not yield the best results, as fine grinding is necessary. Crushing in ball mills to the required degree of fineness and passing the thick pulp over amalgamation plates would appear to be preferable. Blankets replacing the amalgamation plates may be found to be profitable. Cyanidation of the tailings from amalgamation will doubtless be found to yield the highest recovery.

Report No. 295

EXPERIMENTAL TESTS ON GOLD ORE FROM MALARTIC MINE, FOURNIÈRE TOWNSHIP, WESTERN QUEBEC

J. S. Godard

Shipment.—An ore shipment of 5,000 pounds was received April 30, 1928, from the Malartic Gold Mines, Ltd., Fournière township, Quebec.

Characteristics and Analysis of the Ore.—The shipment received was a fairly high-grade gold ore. Eighty-five per cent of the gold is freed on grinding to 85 per cent through 200 mesh, the remainder is associated with small quantities of the sulphides of iron, copper, zinc, and arsenic. Of the sulphides, a very fine-grained iron pyrite greatly predominates, the others are present only in very small amounts. The gangue is siliceous. The analysis of the head sample was as follows:

Gold	0·90 oz./ton	Zinc	0·03 per cent
Silver	0·56 "	Copper	0·07 "
Lead	Nil	Arsenic	0·02 "

Purpose of Tests.—The purpose of the experimental tests was to determine the best metallurgical treatment for the ore.

EXPERIMENTAL TESTS

CONCENTRATION

Tests were made on 1,000-gramme lots.

Results.—

Test No.	Product	Weight, per cent	Assay, oz./ton	Per cent of values
1	*Flotation and tabling the flotation tailing—*			
	Flotation concentrate	16·0	4·71	86·6
	Table concentrate	2·0	3·34	7·6
	Table middling	3·9	0·13	0·6
	Table tailing	50·6	0·067	3·9
	Table slime	27·5	0·04	1·3
	Table tailing +200	9·3	0·14	19·4
	−200	90·7	0·06	80·6
2	*Flotation—*			
	Concentrate	10·9	7·45	82·0
	Tailing	89·1	0·20	18·0
	Tailing +200	16·8	0·90	75·1
	−200	83·2	0·06	24·9
	Sample of the +200-mesh tailing was amalgamated and the tailing assayed—			
	Before amalgamation	....	0·90	....
	After amalgamation	....	0·04	....
	Amalgamated, 95·5 per cent.			
3	*Flotation—*			
	Concentrate	9·2	6·94	78·6
	Tailing	90·8	0·192	21·4
	Tailing +200	21·2	0·61	67·2
	−200	78·8	0·08	32·8
4	*Flotation—*			
	Concentrate	9·8	6·79	73·4
	Tailing	90·2	0·267	26·6
	Tailing +200	10·7	1·66	66·6
	−200	89·3	0·10	33·4

Reagents.—

Test No. 1.—Lime 2·0 lb./ton added to ball mill. Xanthate 0·2, pine oil 0·04 lb./ton added to cell. Froth dirty.

Test No. 2.—Soda ash 2·0, coal-tar creosote 0·08 lb./ton added to ball mill. Xanthate 0·1, $CuSO_4$ 0·4; xylidine 0·04, pine oil 0·08 lb./ton added to cells. Froth good.

Test No. 3.—Soda ash 1·0 lb./ton added to ball mill. Xanthate 0·2, $CuSO_4$ 1·0, pine oil 0·08 lb./ton added to cell. Froth good.

Test No. 4.—Soda ash 1·0 lb./ton added to ball mill. Xanthate 0·2; $CuSO_4$ 2·0, pine oil 0·08 lb./ton added to cell. Froth good.

Test No. 5

Flotation Test in Continuous Unit

Procedure.—The ore was dry crushed to −10 mesh and fed from a bin by means of a belt feeder into a small rod mill in closed circuit with an Akins classifier. The oversize from the classifier was returned to the rod mill and the overflow emptied into a small contact tank adjusted for a 10-minute contact. The discharge from the tank emptied into a 10-cell, mechanically agitated flotation machine. The rougher concentrate was cleaned in a 3-cell Denver machine. The middling from the cleaner cell was returned to the contact tank.

Results.—

Head, oz./ton	Classifier overflow, oz./ton	Concentrate, oz./ton	Tailing, oz./ton	Recovery	
				Head	Per cent based on classifier overflow
1·09	0·31	5·70	0·063	95·2	80·7

Sample of classifier overflow screened on 200 mesh:

Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
+200	24·4	0·19	14·9
−200	75·6	0·35	85·1

Sample of tailing screened on 200 mesh:

Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
+200	24·8	0·07	27·8
−200	75·2	0·06	72·2

Reagents.—

Soda ash 0·75 lb./ton added to ball mill. Xanthate 0·1 lb./ton, pine oil 0·08 lb./ton added to contact tank. $CuSO_4$ 0·4 lb./ton added to No. 1 mixing chamber on flotation machine.

Test No. 6

Procedure as in Test No. 5. Test divided into two parts, A and B.

Results.—

Head, oz./ton	Classifier overflow, oz./ton	Concentrate, oz./ton	Tailing, oz./ton	Recovery	
				Head	Per cent based on classifier overflow
0·90	0·33	10·06	0·056	94·4	83·5

Sample of classifier overflow screened on 200 mesh:

Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
+200	24·8	0·25	19·1
−200	75·2	0·35	80·9

Sample of tailings screened on 200 mesh:

Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
+200	31·0	0·07	38·6
−200	69·0	0·05	61·4

Reagents.—

Xanthate 0·18, pine oil 0·1 lb./ton added to contact tank. $CuSO_4$ 0·5 lb./ton added to mixing cell. Coal-tar creosote 0·04 lb./ton added to ball mill. Soda ash 0·75 lb./ton added to ball mill for first two hours then cut off. Conditions improved when no soda ash was added.

Test No. 7

Procedure as in Test No. 5. Test divided into two parts, A and B.

Part	Head, oz./ton	Classifier overflow, oz./ton	Concentrate, oz./ton	Tailing, oz./ton	Recovery	
					Head	Per cent based on classifier overflow
A	0·90	0·31	7·18	0·072	92·8	77·6
B	0·90	0·31	10·66	0·073	92·5	77·0

Sample of tailings screened on 200 mesh:

Part	Mesh	Assay, oz./ton	Per cent of values	Weight, per cent
A	+200 −200	0·08 0·07	20·3 79·7	18·2 81·8
B	+200 −200	0·08 0·07	32·0 68·0	29·2 70·8

Reagents.—

Part A—Soda ash 0·75 lb./ton added to rod mill. Xanthate 0·05 lb./ton added to contact tank. Cresylic acid 0·28 lb./ton added to mixing cell.

Part B—Xanthate 0·05 lb./ton added to contact tank. Cresylic acid 0·28 lb./ton added to mixing cell.

AMALGAMATION

Tests were made on 1,000-gramme samples.

Test No. 8

Results of Screen Test on Amalgamation Tailings.—

Product	Weight, per cent	Assay, oz./ton	Per cent of values
+100	5·6	0·12	5·4
+150	6·8	0·13	7·1
+200	28·8	0·13	30·3
−200	58·8	0·12	57·2
Average	....	0·125	

Test No. 9

Product	Weight, per cent	Assay, oz./ton	Per cent of values
+200	9·0	0·14	11·2
−200	91·0	0·11	88·8
Average	....	0·113	

Summary of Tests Nos. 8 and 9.—

Test	Head, oz./ton	Tailing, oz./ton	Per cent of gold amalgamated
8	0·90	0·124	86·2
9	0·90	0·113	87·5

AMALGAMATION AND FLOTATION OF AMALGAMATION TAILING

Test No. 10

Results of Flotation.—

Product	Weight, per cent	Assay, oz./ton	Per cent of values
Concentrate	4·6	2·11	61·4
Tailing	95·4	0·064	38·6
Tailing +150	8·1	0·11	13·9
+200	31·3	0·08	38·9
−200	60·6	0·05	47·2

Test No. 11

Product	Weight, per cent	Assay, oz./ton	Per cent of values
Concentrate	4·5	2·00	64·0
Tailing	95·5	0·053	36·0
Tailing +200	8·9	0·08	13·4
−200	91·1	0·05	86·6

Summary of Tests Nos. 10 and 11.—

Test No.	Head, oz./ton	Amalgamation tailing, oz./ton	Flotation tailing, oz./ton	Recoveries, per cent		
				Amalgamation	Flotation	Total
10	0·90	0·158	0·064	82·5	10·75	93·25
11	0·90	0·141	0·053	84·3	10·05	94·38

CYANIDATION

Tests were made on 750-gramme lots of ore ground in Abbé pebble mills in 0·05 per cent KCN and CaO 3 lb./ton. Length of grinding periods varied. Pulp density during grinding, 1:1; during agitation, 1: 2·5.

Results of Screen Tests on Cyanide Tailings.—

Test No.	Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
12	+150	2·0	3·69	27·6
	+200	16·6	0·48	29·8
	−200	81·4	0·14	42·6
	Average		0·268	
13	+200	1·9	4·45	46·3
	−200	9·1	0·10	50·7
	Average		0·183	
14	+200	1·0	8·87	42·8
	−200	99·0	0·12	57·2
	Average		0·208	

Summary of Tests Nos. 12, 13, and 14.—

Test	Heads, oz./ton	Tailing, oz./ton	Extraction, per cent	Reagent consumption	
				KCN, lb./ton	CaO, lb./ton
12	0·90	0·268	70·2	0·71	6·6
13	0·90	0·183	79·7	0·86	6·7
14	0·90	0·208	76·9	0·82	7·1

AMALGAMATION AND CYANIDATION OF AMALGAMATION TAILING

Tests were made on 1,000 grammes of ore. Grinding done in Abbé ball mills in 1:1 pulp. Amalgamation in 1:1 pulp. Cyanidation in 1:2·5 pulp. Cyanide strength, 0·05 per cent KCN.

Tests Nos. 15 and 16

Results of Screen Tests on Amalgamation Tailings.—

Test	Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
15	+200	17·1	0·14	18·1
	−200	82·9	0·13	81·9
16	+200	6·6	0·11	8·0
	−200	93·4	0·09	92·0

Results of Screen Tests on Cyanide Tailings.—

Test	Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
15	+150	4·1	0·06	6·1
	+200	29·6	0·04	28·9
	−200	66·3	0·04	65·0
16	+200	9·4	0·05	14·7
	−200	90·6	0·03	85·3

Summary: Tests Nos. 15 and 16.—

Test No.	Head, oz./ton	Tailings, oz./ton		Recoveries, per cent			Reagent consumption	
		Amalgamation	Cyanide	Amalgamation	Cyanide	Total	KCN, lb./ton	CaO, lb./ton
15	0·90	0·132	0·041	85·3	10·2	95·5	0·63	5·0
16	0·90	0·091	0·032	89·8	6·8	96·6	0·79	5·1

Test No. 17

Cyanide tests were made on 140 pounds of ore. The ore was dry crushed to −35 mesh, then fed continuously from a mechanical feeder at the rate of about 10 pounds per hour to a small rod mill in closed circuit with a drag classifier. The classifier overflow was pumped into a Pachuca tank where it was agitated for 48 hours. The cyanide tailing was filtered and washed in a plate and frame press, then repulped and refiltered.

Results.—

Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
+200	10·3	0·07	16·7
−200	89·7	0·04	83·3
Average		0·043	

Test No. 18

Procedure as in Test No. 17, with an adjustment made to the feeder to obtain a more even feed, and an improvement made in Pachuca tank to improve the agitation.

Results.—

Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
+200	6·3	0·02	6·3
−200	93·7	0·02	93·7
Average		0·02	

Summary of Tests Nos. 17 and 18.—

Test No.	Head, oz./ton	Tailing, oz./ton	Extraction, per cent	KCN, per cent		Time of agitation, hours
				Grinding	Agitation	
17	0·86	0·043	95·0	0·05	0·05	48
18	0·95	0·020	97·9	0·35	0·05	40

Test No. 19

Cyanidation test was made on a 500-gramme sample of flotation tailings from Test No. 6. Pulp density during agitation 1: 2·5. Cyanidation tailings screened on 200 mesh.

Results.—

Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
+200	28·6	0·02	44·5
−200	71·4	0·01	55·5
Average		0·013	

Test No. 20

Flotation tailings reground in Abbé pebble mill in 0·05 per cent KCN, 1:1 pulp before agitation. Pulp density during agitation 1: 2·5. Cyanide tailings screened on 200 mesh.

Results.—

Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
+200	7·5	0·03	20·0
−200	92·5	0·01	80·0
Average		0·012	

Summary of Tests Nos. 19 and 20.—

Test No.	Head, oz./ton	Tailing, oz./ton	Extraction, per cent	KCN, per cent		Reagent consumption		Time of agitation, hours
				Grinding	Agitation	KCN, lb./ton	CaO, lb./ton	
19	0·053	0·013	75·5		0·05	0·53	4·9	24
20	0·053	0·012	77·4	0·05	0·05	0·57	5·2	24

Test No. 21

A 140-pound sample of tailings from Test No. 6 was agitated in a Pachuca tank for 24 hours in 1: 3 pulp, 0·05 per cent KCN. Tailings were filtered, washed, and repulped. Cyanide tailings screened on 200 mesh.

Results.—

Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
+200	30·7	0·02	52·6
−200	64·3	0·01	47·4

Head, 0·053 oz./ton; tails 0·014 oz./ton; extraction, 73·6 per cent.

CONCLUSIONS

The tests indicate that the most satisfactory method of treating this ore is cyanidation. Good extractions were obtained with low cyanide consumption on ore ground to about 90 per cent −200 mesh.

The ore is amenable to concentration by flotation. As such a large percentage of the gold is freed on grinding it would seem advisable to amalgamate before flotation and to table the flotation tailing. The table concentrate would be too low grade for shipment and would have to be returned to the grinding circuit or reground in a separate unit, and refloated.

Report No. 296

CONCENTRATION OF A COPPER-MOLYBDENUM ORE FROM THE GOLCONDA MINE, KEREMEOS, BRITISH COLUMBIA

J. S. Godard

Shipment.—A shipment of 100 pounds of ore was received March 3, 1928, from the Golconda mine near Keremeos, B.C. D. McEachern was the shipper.

Characteristics and Analysis of the Ore.—The shipment consisted of a molybdenum-copper ore. The molybdenum is present as very fine-grained molybdenite and in some specimens occurs in patches free from chalcopyrite and iron pyrite, while in others the chalcopyrite and pyrite are scattered throughout the molybdenite forming a complex mixture of the three sulphides. Chalcopyrite is also found free from molybdenite. Considerable iron pyrite is present, and the gangue is siliceous. The analysis of the head sample was:

Molybdenite	2·84 per cent
Copper	4·66 "

Purpose of Tests.—The purpose of the tests was to determine whether a satisfactory separation of the molybdenite and chalcopyrite could be made, and the sulphides concentrated into marketable products by selective flotation.

EXPERIMENTAL TESTS

The following tables give the results of selective flotation tests on the ore, and the reagents used in the tests:

Test No.	Product	Weight, per cent	Analysis		Per cent of values	
			MoS_2 per cent	Cu per cent	MoS_2	Cu
1	Molybdenite concentrate	6·8	34·50	6·83	79·4	9·5
	Copper concentrate	21·6	1·67	18·00	12·2	79·9
	Tailing	71·6	0·35	0·72	8·4	10·6
	Tailing +100	8·9	0·48	1·13	12·2	14·0
	−100	91·1	0·34	0·68	87·8	86·0
2	Molybdenite concentrate	7·8	30·87	8·84	82·4	14·5
	Copper concentrate	16·4	1·52	23·44	8·5	82·1
	Tailing	75·8	0·35	0·21	9·1	3·4
	Tailing +150	4·3	0·32	0·21	4·0	4·3
	−150	95·7	0·35	0·21	96·0	95·7
3	Molybdenite concentrate	5·4	21·66	3·84	43·2	4·4
	Copper concentrate	19·2	2·74	23·00	19·4	93·4
	Tailing	75·4	1·34	0·14	37·4	2·2
4	Molybdenite concentrate	5·1	25·19	4·88	45·7	5·2
	Copper concentrate	18·9	2·21	23·28	14·9	92·1
	Tailing	76·0	1·46	0·17	39·4	2·7

Test No.	Product	Weight, per cent	Analysis		Per cent of values	
			MoS_2 per cent	Cu, per cent	MoS_2	Cu
5	Rougher concentrates from six lots combined and cleaned—					
	Molybdenite concentrate	4·6	46·94	3·42	77·4	3·3
	Molybdenite middlings	4·3	3·04	7·04	4·7	6·4
	Copper concentrate	13·2	0·54	29·75	2·5	83·4
	Copper middlings	6·4	1·16	1·82	2·6	2·5
	Tailing	71·5	0·50	0·29	12·8	4·4
	Tailing +150	9·5	0·60	0·37	11·4	12·2
	−150	90·5	0·49	0·28	88·6	87·8
6	Molybdenite concentrate	6·8	34·20	4·22	75·5	6·1
	Copper concentrate	18·7	1·01	22·36	6·1	89·3
	Tailing	74·5	0·76	0·29	18·4	4·6
7	Molybdenite concentrate	6·3	34·75	3·42	75·6	4·5
	Copper concentrate	20·0	1·29	21·24	8·9	89·3
	Tailing	73·7	0·61	0·40	15·5	6·2
8	Molybdenite concentrate	5·5	39·28	2·70	73·0	3·1
	Copper concentrate	20·5	1·44	21·46	10·0	92·2
	Tailing	74·0	0·68	0·30	17·0	4·7
9	Molybdenite concentrate	8·0	30·65	6·86	79·2	11·3
	Copper concentrate	17·7	1·20	23·26	6·9	84·3
	Tailing	74·3	0·58	0·29	13·9	4·4
10	Molybdenite concentrate	6·2	38·22	5·22	82·2	6·8
	Copper concentrate	18·7	1·10	22·68	7·1	88·7
	Tailing	75·1	0·41	0·29	10·7	4·5
11	Molybdenite concentrate	4·4	35·54	11·56	54·7	10·9
	Copper concentrate	22·5	1·68	17·36	13·2	83·9
	Copper middlings	6·7	5·46	1·30	12·8	1·9
	Tailing	66·4	0·83	0·23	19·3	3·3
12	Molybdenite concentrate	5·8	38·51	5·06	78·9	6·1
	Copper concentrate	20·0	1·39	21·70	9·8	90·2
	Tailing	74·2	0·43	0·24	11·3	3·7
13	Molybdenite concentrate	5·9	41·86	3·68	79·5	4·5
	Copper concentrate	19·5	1·63	22·12	10·2	90·2
	Tailing	74·6	0·43	0·34	10·3	5·3
14	Molybdenite concentrates from six lots combined and cleaned—					
	Molybdenite concentrate	4·1	49·9	3·26	71·9	2·9
	Molybdenite middlings	1·5	10·4	3·36	5·5	1·1
	Copper concentrate	17·7	1·6	24·14	10·0	92·8
	Tailing	76·7	0·47	0·19	12·6	3·2
	Tailing +200	16·4	0·80	0·13	28·2	11·3
	−200	83·6	0·40	0·20	71·8	88·7

Reagents.—

Test No. 1.—Soda ash 3·0, NaCN 0·6, kerosene 0·2, pine oil 0·08 lb./ton added to ball mill. Copper flotation $CuSO_4$ 0·6, xanthate 0·3, pine oil 0·04 lb./ton added to cell.

Test No. 2.—Soda ash 4·0, NaCN 0·6, kerosene 0·24, pine oil 0·08 lb./ton added to ball mill. Copper flotation: $CuSO_4$ 0·2, xanthate 0·3, xylidine 0·04 lb./ton added to cell.

Test No. 3.—Lime 4·0, NaCN 1·0, kerosene 0·16, pine oil 0·04 lb./ton added to ball mill. Copper flotation: $CuSO_4$ 0·3, xanthate 0·3, xylidine 0·04 lb./ton added to cell.

Test No. 4.—Lime 3·0, NaCN 1·0, kerosene 0·32, pine oil 0·04 lb./ton added to ball mill. Copper flotation: $CuSO_4$ 0·2, xanthate 0·3, xylidine 0·04 lb./ton added to cell.

Test No. 5.—Soda ash 3·0, NaCN 0·6, kerosene 0·28, pine oil 0·04 lb./ton added to ball mill. Copper flotation: $CuSO_4$ 0·4, xanthate 0·3, xylidine 0·04 lb./ton added to cell. Cleaning molybdenite concentrate: soda ash 1·0, NaCN 0·25 lb./ton added to cell. Cleaning copper concentrate: CaO 1·5 lb./ton added to cell.

Test No. 6.—Soda ash 1·0, NaCN 1·0, kerosene 0·28, pine oil 0·04 lb./ton added to ball mill. Copper flotation: $CuSO_4$ 0·5, xanthate 0·15, Aerofloat 0·08 lb./ton added to cell

Test No. 7.—Soda ash 2·0, NaCN 1·0, $ZnSO_4$ 2·0, kerosene 0·28, pine oil 0·04 lb./ton added to ball mill. Copper flotation: $CuSO_4$ 0·2, xanthate 0·15, Aerofloat 0·08 lb./ton added to cell.

Test No. 8.—Bleaching powder 2·0, NaCN 1·0, kerosene 0·28, pine oil 0·04 lb./ton added to ball mill. Copper flotation: $CuSO_4$ 0·2, xanthate 0·15, Aerofloat 0·04 lb./ton added to cell.

Test No. 9.—Bleaching powder 2·0, soda ash 3·0, kerosene 0·28, pine oil 0·04 lb./ton added to ball mill. Copper flotation as in Test No. 8.

Test No. 10.—Bleaching powder 2·0, sodium silicate 2·0, NaCN 1·0, kerosene 0·28, pine oil 0·04 lb./ton added to ball mill. Copper flotation as in Test No. 8.

Test No. 11.—Bleaching powder 2·0, H_2SO_4 1·0, kerosene 0·28, pine oil 0·04 lb./ton added to ball mill. Copper flotation: $CuSO_4$ 0·8, xanthate 0·15, Aerofloat 0·04 added to cell. Cleaning copper concentrate: Aerofloat 0·04 lb./ton.

Test No. 12.—NaCN 1·0, kerosene 0·32, pine oil 0·04 lb./ton added to ball mill. Copper flotation as in Test No. 8.

Test No. 13.—NaCN 2·0, kerosene 0·32, pine oil 0·04 lb./ton added to ball mill. Copper flotation as in Test No. 8.

Test No. 14.—NaCN 2·0, kerosene 0·32, pine oil 0·04 lb./ton added to ball mill. Copper flotation: CaO 2·0, $CuSO_4$ 0·5, xanthate 0·15, pine oil 0·04 lb./ton added to cell. Cleaning molybdenite concentrate: NaCN 0·25 lb./ton added to cell.

CONCLUSIONS

The results from 14 tests indicate that by selective flotation a recovery of 72 per cent of the molybdenite may be obtained in a concentrate containing 50 per cent molybdenite and 3 per cent copper, and that 93 per cent of the copper may be recovered in a concentrate containing 25 per cent copper. Slightly better results may be expected in large-scale operations.

Report No. 297

CONCENTRATION OF A COMPLEX COPPER-ZINC ORE FROM THE SHERRITT-GORDON MINE, COLD LAKE, MANITOBA

C. S. Parsons, A. K. Anderson, and J. S. Godard

Shipments.—Two shipments were received, designated in this report as No. 4 and No. 5. Lot No. 4, received in March 1928, consisted of 115 bags of ore weighing about 3 tons. Lot No. 5, received in April, consisted of a carload of ore weighing about 20 tons. The shipments were made by the Sherritt-Gordon Mines, Ltd. of Toronto and were from the Sherritt-Gordon mine near Cold Lake, northern Manitoba.

Characteristics and Analysis of the Ore.—The shipments consisted of a complex copper-zinc-iron sulphide ore containing low values in gold and silver. The analyses of the two shipments were as follows:

—	Lot No. 4	Lot No. 5
Copper	1·24 per cent	3·08 per cent
Zinc	8·94 "	6·49 "
Gold	0·02 oz./ton	0·05 oz./ton
Silver	0·26 "	0·90 "
Iron	....	17·92 per cent

Purpose of Experimental Work.—These shipments were made for the purpose of having concentration tests made on the ore to determine what economic results could be expected from concentrating the ore. A high-grade copper concentrate was desired with a minimum of zinc in it, and a high-grade zinc concentrate with a minimum of copper and iron. Information as to the control of the conditions which would produce these products was also desired. Special tests were requested to determine whether the pyrrhotite carried an appreciable amount of gold.

EXPERIMENTAL TESTS

The work on Lot No. 4 was divided into three sections. The first section consists of selective flotation tests made in a small continuous flotation unit having a capacity of 100 pounds of ore per hour. The second section consists of bulk flotation tests made in the same unit, and the third section, of special work done to isolate the pyrrhotite for the determination of its gold content.

The work on Lot No. 5 was also divided into three sections. The first section consists of bulk flotation tests made on a one-ton sample taken at random from the shipment and run in a small flotation unit, and the second section of selective tests made in this unit. The third section consists of large-scale tonnage tests made on the large flotation unit treating the ore at the rate of 1,000 pounds an hour.

LOT NO. 4

Section 1: Selective Flotation

The ore was crushed to 10 mesh and fed to an 18- by 24-inch rod mill in closed circuit with a 12-inch Akins classifier. The overflow of the classifier passed to a mechanically agitated conditioning tank giving from 20 to 30 minutes contact to the pulp and reagents before entering an 8-cell standard Minerals Separation type of flotation cells for the flotation of the copper. The first two cells were used as mixers, the other six as flotation cells. A rougher concentrate was made which was cleaned in a 3-cell, Denver Sub-A machine, the cleaner tailing being returned to the head of the rougher. The rougher tailing passed to the zinc flotation circuit. The feed to the zinc circuit was conditioned for 20 to 30 minutes in a conditioning tank, the overflow from which passed to a 6-cell, Denver Sub-A machine where a rougher zinc concentrate was made which was cleaned in a small Callow cell. The cleaner tailing was returned to the head of the rougher cell. Results:

Test No. 1.—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate	5·25	20·46	7·57	0·10	3·82	86·99	4·47	90+	66·66
Zinc concentrate	15·07	0·46	52·93	0·01	0·29	5·69	89·26	....	13·33
Copper tailing	....	0·19	8·54	trace	0·10	....	....	....	....
Zinc tailing	79·68	0·12	0·71	trace	0·08	7·32	6·27	....	....

Test No. 2.—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
1st sample (9.30 a.m.—2.00 p.m.)									
Copper concentrate	5·86	18·49	5·97	0·10	3·30	87·10	3·91	81·08	65·51
Zinc concentrate...	11·96	0·42	49·78	trace	0·11	4·03	66·55	13·55	3·45
Zinc tailing........	82·18	0·13	3·21	trace	0·12	8·87	29·54	5·37	31·04
2nd sample (2.00—4.30 p.m.)									
Copper concentrate	5·33	20·93	6·27	0·10	3·63	90·24	3·69	71·43	65·52
Zinc concentrate...	14·41	0·36	51·41	trace	0·17	4·07	82·89	14·28	6·89
Zinc tailing........	80·26	0·09	1·49	trace	0·10	5·69	13·42	14·29	27·59

Reagents (in lb./ton).—

Time	Rod mill, Na_2CO_3	Conditioner, NaCN	Zn cells, $CuSO_4$	Rod mill, thiocarbanilide	Cu cell, cresylic acid	Rod mill, $ZnSO_4$	Conditioner, NaCN	Classifier overflow, density
Test No. 1								
9.30.......	....	....	....	0·14	0·14	1	....	....
10.00.......	2·3	0·51	....	....	0·70	1	0·32	1 : 2·8
10.30.......	....	0·58	....	0·14	....	....	....	....
10.40.......	....	0·25	0·80	....	....	1	0·32	....
10.50.......	....	....	....	0·14	....	1	0·32	....
11.00.......	2·3	0·25	0·80	....	0·56	1	0·32	1 : 2·4
11.30.......	2·3	....	....	0·14	0·56	....	....	1 : 2·5
12.00.......	2·3	0·24	0·80	....	0·56	1	0·32	1 : 2·2
12.30.......	....	0·29	0·80	0·14	0·56	1	0·32	1 : 2·5
1.15.......	2·3	0·29	0·80	0·14	0·56	1	0·32	1 : 4·2
1.45.......	2·3	0·43	0·80	0·14	0·56	1	0·32	1 : 3·7
2.00.......	2·3	0·43	0·80	0·14	0·56	1	0·32	1 : 3·3
2.30.......	2·3	0·43	0·80	0·14	0·56	1	0·31	1 : 4·8
2.40.......	....	0·29	0·80	0·14	....	....	....	1 : 3·1
3.00.......	2·5	0·29	0·80	....	0·56	1	0·31	1 : 3·0
3.30.......	2·5	0·43	0·80	0·14	0·56	1	0·81	1 : 2·8
4.00.......	2·5	0·43	0·80	0·14	0·56	1	0·30	1 : 2·7
4.30.......	2·5	0·43	0·80	0·14	0·56	1	0·30	1 : 2·7
Test No. 2								
9.45.......	2·37	0·44	1·18	0·07	0·56	1	0·24	1 : 5·2
10.00.......	2·37	0·44	1·18	0·07	0·56	1	0·24	....
10.30.......	2·37	0·45	1·18	0·07	0·56	1	0·24	....
11.30.......	2·37	0·45	1·18	0·07	0·56	1	0·24	1 : 2·5
12.00.......	2·37	0·45	1·18	0·07	0·56	1	0·24	1 : 2·5
12.30.......	2·37	0·45	1·18	0·07	0·56	1	0·20	1 : 2·7
1.00.......	2·37	0·45	1·18	0·07	0·56	1	0·30	1 : 3·3
1.30.......	2·37	0·45	1·18	0·07	0·56	1	0·30	1 : 3·3
2.00.......	2·18	0·40	1·09	0·07	0·56	1	0·31	1 : 3·3
2.30.......	2·32	0·43	1·16	0·07	0·56	1	0·33	1 : 3·5
3.00.......	2·32	0·43	1·16	0·07	0·56	1	0·33	1 : 3·1
3.30.......	2·32	0·43	1·16	0·07	0·56	1	0·33	1 : 3·3

Section 2: Bulk Flotation

The crushing circuit was the same as that used in the selective tests. The standard Minerals Separation cells were used to make the bulk float. The pulp was conditioned in the No. 1 conditioner for 5 minutes. The rougher concentrate was cleaned as in the selective tests. Results:

Test No. 3.—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Concentrate No. 1	19·15	6·06	42·84	0·04	1·36	92·06	91·72	50·00	81·25
Concentrate No. 2	....	1·78	40·49	0·02	0·44	....	....	....	....
Tailing	80·85	0·12	0·91	trace	0·07	7·94	8·28	50·00	18·75
Head	....	1·24	8·94	0·02	0·26	....	....	....	....

Reagents (in lb. /ton.)—

Time	To rod mill			To head of cells			Feed, lb./hr.	Classifier overflow, density
	CaO	NaCN	Thiocarbanilide	$CuSO_4$	Cresylic acid	Xanthate		
9.30	5	0·44	0·07	1·18	0·56	....	101·5	1 : 4·0
10.00	5	0·44	0·07	1·18	0·56			
10.30	5	0·44	0·07	1·18	0·56	0·04	....	1 : 3·0
11.00	5	0·44	0·07	1·18	0·56	0·04	....	1 : 2·8
11.30	5	0·44	0·07	1·18	0·56	0·04	....	1 : 3·0
12.00	5	0·32	0·07	1·21	0·56	0·04	....	1 : 2·7
12.30	5	0·32	0·07	1·18	0·56	0·04	....	1 : 3·5
1.00	5	0·32	0·07	1·18	0·56	0·04	....	1 : 3·3
1.30	5	0·44	0·07	1·18	0·56	0·04	....	1 : 3·5
2.00	5	0·32	0·07	1·18	0·56	0·04	....	1 : 3·7
2.30	5	0·30	0·07	1·18	0·56	0·04	108	1 : 4·0
3.00	5	0·30	0·07	1·18	0·56	0·04	....	1 : 6·1
3.30	5	0·30	Ball mill plugged.					
4.00	5	0·30	0·07	1·11	0·56	0·04	....	1 : 4·8

Section 3: Isolation of Pyrrhotite

The tailings from the selective flotation tests were tabled. The table concentrate containing the sulphides was passed through a laboratory magnetic separator, which is designed to thoroughly wash out any entangled non-magnetic particles. The results are as follows:

No. 1, pyrrhotite product (magnetic)—	
Iron	57·87 per cent
Sulphur	36·47 "
Gold	0·004 oz./ton
No. 2, pyrite product (non-magnetic)—	
Iron	20·52 per cent
Arsenic	trace
Copper	0·22 per cent
Gold	0·01 oz./ton

This definitely indicates that the gold is not associated with the pyrrhotite. In this connexion we find a little free gold present in the ore, probably due to oxidation of the ore taken from near the surface.

Conclusions and Remarks on Lot No. 4.—The head analysis was: copper, 1·24 per cent; zinc, 8·94 per cent; gold, 0·02 ounce per ton; silver, 0·26 ounce per ton.

Selective Tests: There was no particular difficulty in making the separation outside of the control of the operation of so small a unit. The tests show that exceptionally sharp separations can be expected if this sample is representative of the type of ore to be mined.

Bulk Test: This test was run for the information of the Base Metal Extraction Co. As can be seen from the results a very excellent recovery and grade of concentrate was made. No difficulty of importance was encountered.

LOT NO. 5

Section 1: Bulk Flotation

The flow-sheet was the same as that used on Lot No. 4, Section 2.

Test No. 4.—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Sample No. 1									
Concentrate No. 1.	13·45	13·78	27·95	0·09	2·69	92·50	73·29	52·63	69·23
Concentrate No. 2.		2·68	45·68	0·03	0·70				
Tailing...........	86·55	0·17	1·58	0·01	0·19	7·50	26·71	47·37	30·77
Sample No. 2									
Concentrate No. 1.	16·67	11·10	28·05	0·08	2·24	92·50	82·83	43·33	71·15
Concentrate No. 2.		1·92	38·25	0·03	0·63				
Tailing...........	83·33	0·18	1·17	0·02	0·18	7·50	17·17	56·67	28·85

Reagents (in lb./ton).—

Time	Rod mill			Head of cells			Feed, lb./hr.	Classifier overflow density
	CaO	NaCN	Thio.	$CuSO_4$	Cresylic	Xanthate		
Sample No. 1								
10.15.......	5·0	0·28	0·07	1·17	0·56	0·03	114	
10.30.......	5·0	0·28	....	1·17	0·56	0·03	....	1 : 4·2
11.00.......	5·0	0·28	0·07	1·17	0·56	0·03	....	1 : 2·5
11.30.......	5·0	0·28	0·07	1·17	0·56	0·03	....	1 : 2·5
12.00.......	....	0·28	0·07	1·17	0·56	0·03	....	1 : 2·8
12.30.......	5·0	0·28	....	1·17	0·56	0·03	....	1 : 2·1
1.00.......	....	0·28	0·07	1·17	0·56	0·03	....	....
1.30.......	....	0·28	....	1·17	0·56	0·03	....	....
Sample No. 2								
2.00.......	5·0	0·28	0·07	1·17	0·84	0·04	....	1 : 3·0
2.30.......	....	0·28	....	1·17	0·84	0·04	....	1 : 2·0
3.00.......	5·0	0·34	0·07	1·22	0·84	0·04	98	1 : 3·3
3.30.......	5·0	0·34	....	1·22	0·84	0·04	....	1 : 4·0
4.00.......	5·0	0·34	0·07	1·22	0·84	0·04	....	1 : 3·1

Section 2: Selective Flotation

The flow-sheet was the same as that used for Lot No. 4, Section 1.

Results:—

Head: Cu, 2·00 per cent; Zn, 5·51 per cent; Au, 0·04 oz./ton; Ag, 0·48 oz./ton.

Test No.	Product	Weight, per cent	Assay				Per cent of values			
			Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
5	Copper concentrate......	10·83	17·06	4·64	0·20	3·44	92·50	9·07	66·67	75·51
	Zinc concentrate.........	10·67	0·70	43·55	0·02	0·36	3·50	84·39	6·67	8·16
	Copper tailing...........	....	0·14	4·64	0·01	0·12	......	.....	.....	......
	Zinc tailing..............	78·50	0·10	0·46	trace	0·10	4·00	6·54	26·66	16·33
6	*Sample No. 1*									
	Copper concentrate......	8·46	20·04	3·52	0·21	3·63	85·00	5·44	54·05	70·45
	Zinc concentrate.........	18·83	1·19	24·28	0·015	0·37	11·00	82·94	27·00	15·90
	Copper tailing...........	....	0·34	5·40	0·01	0·15	......	.....	.....	......
	Zinc tailing..............	72·71	0·11	0·88	0·01	0·09	4·00	11·62	18·95	13·65
	Sample No. 2									
	Copper concentrate......	10·44	17·75	3·62	0·16	3·50	92·50	6·90	64·00	80·00
	Zinc concentrate.........	13·44	0·63	36·31	0·01	0·27	4·00	88·57	4·00	8·89
	Copper tailing...........	....	0·16	5·76	0·01	0·15	3·50	4·53	32·00	11·11
	Zinc tailing..............	76·12	0·08	0·33	0·01	0·07	......	.....	.....	......

Reagents (in lb./ton).—

Time	Rod mill				Condition tanks		Head of cells			Feed lb./hr.	Classifier overflow density
	Na$_2$–CO$_3$	Thio.	Na-CN	Zn-SO$_4$	No. 1 Na-CN	No. 2 Na-CN	Cu, cresylic acid	Zn, CuSO$_4$	Zn, xanthate		
Test No. 5											
9·20.....	4·44	0·14	......	0·92	0·30	......	0·42	....	....	108	....
10.00......	4·72	0·14	......	0·92	0·29	......	0·42	....	....	......	....
10.30......	4·72	0·14	......	0·92	0·30	......	0·70	....	....	......	....
11.00......	4·72	0·14	......	0·92	0·33	......	0·28	....	....	......	1 : 7·3
11.15......	4·72	0·14	......	0·92	......	0·22	0·56	....	....	......	1 : 3·3
11.30......	4·85	0·142	......	0·95	0·40	0·22	0·56	....	....	105	1 : 3·7
12.00......	4·85	0·142	......	0·95	0·31	0·22	0·56	1·19	....	......	1 : 4·0
12.30......	4·85	0·142	......	0·95	0·31	0·22	0·56	1·19	0·04	......	1 : 2·0
1.00......	4·85	0·142	......	0·95	0·31	0·22	0·56	1·19	0·04	......	1 : 2·2
1.30......	4·71	0·142	......	0·95	0·30	0·22	0·56	1·19	0·04	......	1 : 3·6
2.00......	4·85	0·147	......	0·90	0·30	0·23	0·56	1·26	0·04	102	1 : 3·6
2.30......	4·85	0·147	......	0·90	0·32	0·23	0·42	1·26	0·04	......	1 : 3·3
3.00......	4·85	0·147	.....	0·90	0·29	0·23	0·56	1·20	0·04	......	1 : 2·8
Test No. 6											
9.30......	4·6	0·14	0·27	0·95	......	0·28	0·56	1·28	0·075	......	1 : 1·8
10.00......	4·6	0·14	0·23	0·95	......	0·28	0·56	1·14	0·075	......	1 : 3·5
10.30......	4·8	0·14	0·31	0·95	......	0·28	0·56	1·14	0·075	105	1 : 2·2
11.00......	4·6	0·14	0·27	0·95	......	0·28	0·56	1·14	0·075	......	1 : 2·2
11.30......	4·7	0·14	0·31	0·95	......	0·28	0·56	1·14	0·075	......	1 : 2·8
12.00......	4·7	0·14	0·31	0·95	......	0·28	0·56	1·06	0·075	......	1 : 2·3
12.30......	4·7	0·14	0·31	off	......	0·28	0·56	1·06	0·05	......	1 : 3·0
1.00......	4·7	0·14	0·31	......	......	0·28	0·56	1·06	0·05	......	*1 : 3·0
2.00......	4·7	0·14	0·23	......	......	0·28	0·56	1·06	0·05	......	1 : 3·0
2.30......	4·3	0·14	0·33	......	......	0·28	0·56	1·37	0·05	......	1 : 3·5
3.00......	4·3	0·14	0·38	......	......	0·28	0·56	1·37	0·05	......	1 : 3·5
3.30......	4·6	0·14	0·43	......	......	0·28	0·56	1·37	0·05	......	1 : 3·0

*Sample No. 2.

Section 3: Large-scale Tests

The ore was crushed to ½-inch and fed to a 6-foot by 3-foot Marcy rod mill in closed circuit with a 14-inch by 12-foot Dorr classifier. The feed rate was from 800 to 1,000 pounds of ore an hour. The overflow of the classifier passed directly to an 8-cell, Greenawalt flotation machine; finished copper concentrate was taken from the first two cells, the remaining cell concentrates being returned to the head of the machine. The concentrate was not cleaned. The tailing from this operation was then elevated by an air lift to a 4-cell, Callow flat-bottom type unit consisting of two rougher cells in series and two cleaner cells in series. The tailing from both cleaner cells was returned to the head of the machine. Results of Tests Nos. 7, 8, 9, 10, and 11 follow:—

Test No.	Product	Weight, per cent	Assay				Per cent of values			
			Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
7	Head	....	3·08	7·11	0·06	0·95	......	......	......	......
	Sample No. 1									
	Copper concentrate	16·62	18·07	6·71	0·16	4·64	97·40	15·75	66·66	83·69
	Copper tailing	....	0·08	6·68	0·01	0·30	......	......	......	......
	Zinc concentrate	10·54	0·26	44·79	0·01	0·30	0·97	66·38	2·22	3·26
	Zinc tailing	72·84	0·06	1·75	0·02	0·16	1·63	17·87	31·11	13·05
	Sample No. 2									
	Copper concentrate	13·27	22·06	5·01	0·12	3·24	95·13	9·28	61·54	72·88
	Zinc concentrate	9·96	0·74	52·40	0·02	0·27	2·27	73·42	7·69	5·08
	Copper tailing	....	0·18	7·51	0·02	0·12	......	......	......	......
	Zinc tailing	76·77	0·11	1·60	0·01	0·17	2·60	17·30	30·77	22·04
8	Head	....	3·08	7·11	0·06	0·95	......	......	......	......
	Sample No. 1 (10 a.m.-2.50 p.m.)									
	Copper concentrate	10·57	26·41	3·81	0·27	5·05	90·88	5·63	56·60	81·54
	Copper tailing	....	0·36	7·26	0·01	0·13	......	......	......	......
	Zinc concentrate	13·12	1·54	45·00	0·02	0·45	6·51	83·10	5·66	9·23
	Zinc tailing	76·31	0·10	1·05	0·02	0·19	2·61	11·27	37·74	9·23
	Sample No. 2									
	Copper concentrate	13·28	22·04	3·91	0·20	4·52	95·13	7·31	63·83	77·92
	Copper tailing	....	0·17	7·31	0·02	0·18	......	......	......	......
	Zinc concentrate	13·45	0·64	46·29	0·02	0·38	2·92	87·62	6·38	6·49
	Zinc tailing	73·29	0·09	0·50	0·02	0·16	1·95	5·07	29·79	15·59
9	Head	....	3·10	5·91	0·045	0·81	......	......	......	......
	Sample No. 1 (10.30 a.m.-12.10 p.m.)									
	Copper concentrate	17·03	17·18	4·40	0·20	3·80	94·52	12·69	71·43	81·25
	Copper tailing	....	0·20	7·11	0·02	0·23	......	......	......	......
	Zinc concentrate	10·58	0·83	44·89	0·02	0·39	2·90	80·37	4·76	5·00
	Zinc tailing	72·39	0·12	0·57	0·02	0·16	2·58	6·94	23·81	13·75
	Sample No. 2									
	Copper concentrate	14·51	19·81	3·00	0·15	4·01	92·58	7·29	64·52	72·50
	Copper tailing	....	0·36	6·96	0·03	0·33	......	......	......	......
	Zinc concentrate	11·83	1·04	43·48	0·01	0·45	4·19	87·12	3·23	6·25
	Zinc tailing	73·66	0·14	0·45	0·02	0·23	3·23	5·59	32·25	21·25

Test No.	Product	Weight, per cent	Assay				Per cent of values			
			Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
10	Head..............	....	3·10	5·91	0·045	0·81	......	......	......	......
	Sample No. 1 (10.10 a.m.-4.00 p.m.)									
	Copper concentrate..	15·48	18·25	3·20	0·16	3·94	91·29	8·29	62·50	76·25
	Copper tailing.......	....	0·31	5·81	0·015	0·25	......	......	......	......
	Zinc concentrate.....	11·72	1·47	43·08	0·01	0·48	5·48	85·45	2·50	7·50
	Zinc tailing..........	72·80	0·14	0·50	0·02	0·18	3·23	6·26	35·00	16·25
	Sample No. 2									
	Copper concentrate..	11·78	22·97	3·71	0·20	5·00	87·42	7·43	68·96	79·73
	Copper tailing.......	....	0·45	6·71	0·01	0·39	......	......	......	......
	Zinc concentrate.....	16·61	1·55	27·35	0·01	0·25	8·39	77·03	6·90	5·40
	Zinc tailing..........	71·61	0·17	1·30	0·01	0·15	4·19	15·54	24·14	14·87
11	Head..............	....	3·10	5·91	0·045	0·81	......	......	......	......
	Sample No. 1 (10.35 a.m.-12.00)									
	Copper concentrate..	14·92	19·07	4·41	0·22	4·12	91·61	11·17	78·57	80·26
	Copper tailing.......	....	0·23	6·71	0·02	0·17	......	......	......	......
	Zinc concentrate.....	10·67	1·48	35·97	0·02	0·46	5·16	64·97	4·76	6·58
	Zinc tailing..........	74·41	0·13	1·90	0·01	0·14	3·23	23·86	16·67	13·16
	Sample No. 2 (12.00-1.55 p.m.)									
	Copper concentrate..	15·57	19·07	5·21	0·18	3·96	95·81	13·71	75·68	82·67
	Copper tailing.......	....	0·15	6·66	0·02	0·16	......	......	......	......
	Zinc concentrate.....	10·76	0·52	35·76	0·02	0·32	1·93	65·14	5·41	4·00
	Zinc tailing..........	73·67	0·10	1·70	0·01	0·14	2·26	21·15	18·91	13·33
	Sample No. 3									
	Copper concentrate..	14·85	19·86	5·11	0·24	4·30	95·16	12·86	81·82	81·01
	Copper tailing.......	....	0·17	6·66	0·01	0·15	......	......	......	......
	Zinc concentrate.....	10·57	0·52	39·57	0·015	0·34	1·61	70·73	2·27	5·06
	Zinc tailing..........	74·58	0·13	1·30	0·01	0·15	3·23	16·41	15·91	13·93

Reagents (in lb./ton).—

Test No.	Time	Rod mill				Head of cells						Air lift		Feed, lb./hour	Per cent solids		pH
		Na_2CO_3	NaCN	$ZnSO_4$	Thiocarbanilide	Copper		Zinc				Xanthate	NaCN		Rod mill discharge	Classifier overflow	
						Cresylic	Aerofloat	Cresylic	$CuSO_4$	TT	Xanthate						
7...........	9.30	2·7	0·27	0·92	0·09	0·13	....	....	....	....	....	....	0·36	3,100	....	....	
	10.00	2·7	0·27	0·92	0·09	0·13	....	....	0·46	....	....	....	0·36	776	34	....	
	10.30	2·7	0·27	0·92	0·09	0·10	....	....	1·13	....	....	0·06	0·36	1,060	43	31	
	11.00	2·7	0·27	0·92	0·09	0·10	....	....	1·13	....	....	0·06	0·36	880	40	33	
	11.30	2·7	0·27	0·92	0·09	0·11	....	....	1·13	....	....	0·06	0·36	870	49	37	
	12.00	2·7	0·27	0·92	0·09	0·08	....	0·13	1·08	....	....	0·19	0·36	870	52	....	7·2
	12.30	3·0	0·27	0·92	0·09	0·06	....	0·27	1·16	....	....	0·19	0·18	870	....	27	
	1.00	3·0	0·27	0·92	0·09	0·06	....	0·27	1·16	....	....	0·19	0·18	870	56	23	7·6
	1.30	3·0	0·33	0·92	0·09	0·08	....	0·31	1·16	....	....	0·21	0·09	870	55	14	
	2.00	3·0	0·33	0·92	0·09	0·08	....	0·31	1·16	....	....	0·21	0·09	870	51	29	
	2.30	3·0	0·33	0·92	0·09	0·11	....	0·31	1·16	....	....	0·21	0·09	870	46	24	8·6
	3.00	3·0	0·33	0·92	0·09	0·11	....	0·31	1·16	....	....	0·21	0·09	870	58	28	8·6
	3.30	3·0	0·33	0·92	0·09	0·11	....	0·31	1·16	....	....	0·21	0·09	870	50	29	
	4.00	3·0	0·33	0·92	0·09	0·11	....	0·31	1·16	....	....	0·21	0·09	870	55	28	
8...........	9.00	2·3	0·26	1·20	0·079	0·15	....	0·05	0·92	....	....	0·18	0·07	....	....	....	
	9.15	2·8	0·30	1·20	0·079	0·15	....	0·05	0·92	....	....	0·18	0·07	1,020	....	....	
	9.30	2·8	0·30	1·20	0·079	0·15	....	0·05	0·92	....	....	0·18	0·07	1,020	48	43	
	10.00	3·07	0·32	1·20	0·079	0·15	....	0·05	0·92	....	....	0·18	0·07	....	51	18	
	10.30	3·07	0·32	1·20	0·079	0·15	....	off	0·92	0·15	....	0·18	0·07	....	53	28	
	11.00	3·07	0·32	1·20	0·079	0·15	....	....	0·92	0·15	....	0·18	0·07	....	53	33	
	11.30	3·07	0·32	1·20	0·079	0·15	....	....	0·92	0·15	....	0·18	0·07	....	54	29	
	11.45	3·07	0·32	1·20	off	0·15	....	....	0·92	0·05	....	0·18	0·14	....	....	....	
	12.00	3·07	0·32	1·20	....	0·20	....	....	0·97	0·05	....	0·18	0·14	....	55	26	
	12.30	3·07	0·32	1·20	....	0·20	....	....	0·97	0·05	....	0·18	0·14	....	55	29	
	1.00	3·07	0·32	1·20	....	0·20	....	....	1·00	0·05	....	0·18	0·14	....	57	31	
	1.30	3·07	0·36	1·20	0·065	0·20	....	....	1·00	0·05	....	0·18	0·14	....	58	23	
	2.00	3·07	0·36	1·20	0·065	0·20	....	....	1·00	0·05	....	0·18	0·14	....	54	24	
	2.15	3·07	0·26	1·20	0·065	0·20	....	....	1·00	0·10	....	0·20	0·14	....	....	....	
	2.30	3·07	0·26	1·20	0·09	0·20	....	....	1·00	0·10	....	0·20	0·14	....	59	24	
	3.00	3·07	0·26	1·20	0·09	0·20	....	....	1·00	0·10	....	0·20	0·14	....	59	23	
	3.30	3·07	0·26	1·20	0·09	0·20	....	....	1·00	0·10	....	0·20	0·14	....	59	22	
	4.00	3·07	0·26	1·20	0·09	0·20	....	....	1·00	0·10	....	0·20	0·14	....	59	21	
	4.30	3·07	0·26	1·20	0·09	0·20	....	....	1·00	0·10	....	0·20	0·14	....	....	....	
9..........	9.45	3·42	0·28	1·3	0·11	0·06	....	....	1·09	0·03	....	0·24	0·17	....	....	....	
	10.00	3·42	0·28	1·3	0·11	0·06	....	....	1·09	0·03	....	0·24	0·17	926	....	....	
	10.30	3·20	0·27	1·2	0·10	0·06	....	....	1·02	0·03	....	0·23	0·16	990	43	16	
	11.00	3·20	0·27	1·2	0·10	0·06	....	....	1·02	0·03	....	0·23	0·16	....	64	27	
	11.30	3·20	0·33	1·2	0·10	0·06	....	....	1·02	0·05	....	0·23	0·16	....	56	46	
	12.00	3·23	0·33	1·25	0·10	0·06	....	....	1·03	0·05	....	0·23	0·16	980	59	38	
	12.30	3·23	0·33	1·25	0·10	0·06	....	....	1·03	0·05	....	0·23	0·16	....	59	38	

	1.00	3·23	0·33	1·25	0·10	0·06	....	....	1·03	0·05	....	0·23	0·16	....	63	36
	1.30	3·23	0·33	1·25	0·10	0·06	....	....	1·03	0·02	....	0·23	0·16	....	64	43
	2.00	3·23	0·33	1·25	0·10	0·06	....	....	1·03	0·02	....	0·23	0·16	....	66	42
	2.30	3·23	0·33	1·25	0·10	0·06	....	....	1·03	0·02	....	0·23	0·16	....	65	41
	3.00	3·23	0·33	1·25	0·10	0·06	....	....	1·03	0·02	....	0·23	0·16	....	65	41
	3.15	2·8	0·23	1·23	0·09	0·05	....	....	0·90	0·02	....	0·21	0·14	1,125	....	....
	3.30	2·8	0·23	1·23	0·09	0·05	....	....	0·90	0·02	....	0·21	0·14	....	....	....
	4.00	2·8	0·23	1·23	0·09	0·05	....	....	0·90	0·02	....	0·21	0·14	....	67	40
10...........	9.15	3·0	0·31	1·16	....	....	0·06	....	1·06	0·02	....	0·27	0·15	....	....	....
	9.30	3·0	0·31	1·16	....	....	0·06	....	1·06	0·02	....	0·27	0·15	1,050	....	....
	10.00	3·0	0·31	1·16	....	....	0·06	....	1·06	0·02	....	0·27	0·21	....	47	24
	10.30	3·0	0·31	1·16	....	....	0·06	....	1·06	0·02	....	0·27	0·21	....	52	23
	11.00	3·0	0·31	1·16	....	....	0·06	....	1·06	0·02	....	0·27	0·21	....	46	36
	11.30	3·0	0·31	1·16	....	....	0·06	....	1·06	off	....	0·27	0·21	....	45	29
	12.00	3·0	0·31	1·16	....	....	0·06	....	1·06	....	....	0·27	0·21	....	47	28
	12.30	3·0	0·31	1·16	....	....	0·06	....	1·06	....	....	0·27	0·21	....	47	26
	1.00	3·0	0·31	1·16	....	....	0·06	....	1·06	....	....	0·27	0·21	....	47	26
	1.30	3·0	0·31	1·16	....	....	0·06	....	1·06	....	....	0·27	0·21	....	46	30
	2.00	3·0	0·35	1·16	....	....	0·06	....	1·06	....	....	0·11	0·25	....	45	29
	2.30	3·23	0·38	1·25	....	....	0·06	....	1·03	....	....	off	0·26	980	47	25
	3.00	3·23	0·34	1·25	....	....	0·06	....	1·03	Aero-float	....	....	off	....	50	29
	3.30	3·23	0·34	1·25	....	....	0·06	....	1·03	0·13	....	....	....	....	....	....
	4.00	5·12	0·34	1·25	....	....	0·06	....	1·03	off	....	....	....	....	52	29
	4.35	5·12	0·34	1·25	....	....	0·06	....	1·03	....	....	....	....	....	....	....
11...........	10·00	5·0	0·33	....	0·08	0·06	....	0·02	1·0	....	0·08	....	0·22	....	....	....
	10.15	5·0	0·33	....	0·08	0·06	....	0·02	1·0	....	0·08	....	0·22	....	....	....
	10.30	5·0	0·33	....	0·08	0·06	....	0·02	1·0	....	0·08	....	0·22	1,005	55	36
	11.00	5·0	0·33	....	0·08	0·06	....	0·02	1·0	....	0·08	....	0·22	....	50	41
	11.30	5·0	0·33	....	0·08	0·06	....	0·02	1·0	....	0·08	....	0·22	....	53	33
	12.00	5·0	0·33	....	0·08	0·06	....	0·02	1·0	....	0·08	....	0·22	....	50	33
	12.30	5·0	0·37	....	0·08	0·06	....	0·02	1·0	....	0·08	....	0·26	....	56	36
	1.00	5·0	0·37	....	0·08	0·06	....	0·02	1·0	....	0·08	....	0·26	....	56	31
	1.30	5·0	0·37	....	0·08	0·06	....	0·02	1·0	....	0·08	....	0·26	....	58	33
	2.30	5·0	0·37	....	0·08	0·06	....	0·02	1·0	....	0·08	....	0·26	....	58	33
	3.00	5·0	0·37	....	0·08	0·06	....	0·02	1·0	....	0·08	....	0·26	....	58	34
	3.20	5·0	0·37	....	0·08	0·06	....	0·02	1·0	....	0·08	....	0·26	....	58	34

Conclusions and Remarks, Lot No. 5.—The head sample for Tests Nos. 7 and 8 was: copper, 3·08 per cent; zinc, 7·11 per cent; gold, 0·05 ounce per ton; silver, 0·90 ounce per ton; iron, 17·92 per cent. For Tests Nos. 9, 10, and 11 the head sample was: copper, 3·10 per cent; zinc, 5·91 per cent; gold, 0·045 ounce per ton; silver, 0·81 ounce per ton. The reason for this variation was that the ore was crushed and sampled in two separate lots.

The small-scale tests in Sections 1 and 2 are not so good as those on Lot No. 4. This is chiefly due to manipulation of the small unit, although the poor grade of the concentrate may be partly due to over-oiling. The large-scale tests are exceptionally good and give a very excellent idea of what can be expected from the operation of a mill.

SUMMARY AND CONCLUSIONS

1. It will be possible to make a 20 per cent copper concentrate with over 90 per cent recovery.

2. A 45 to 50 per cent zinc concentrate with over 80 per cent recovery.

3. The gold recovery will vary with the grade of the copper concentrate and for a concentrate of 20 per cent copper will be over 60 per cent.

4. The silver recovery will also vary, but will be over 70 per cent.

5. The zinc in the copper concentrate can be kept between 3 and 4 per cent without much difficulty.

6. The use of zinc sulphate was found to lower the zinc in the copper concentrate but it also interfered with the subsequent flotation of the zinc, making it difficult to obtain a high-grade zinc concentrate and low tailing at the same time.

7. The alkalinity of the pulp during flotation was found to affect the results. A *p*H of about 8 to 9 was found to give the best results in these tests.

8. It was found necessary to reclean the zinc concentrate twice, that is to use two cleaners in series.

9. Although no conditioning tanks were used in the large-scale tests there was evidence to indicate that better and more uniform conditions would be obtained by adding the cyanide to a conditioning tank instead of to the rod mill, and conditioning for about 30 minutes.

10. Conditioning was found to be important between the copper and zinc separations. The well of the air lift gave a short period of contact for additional cyanide before the zinc was floated. This was found to be an important feature of the separation. If sufficient cyanide was added before the flotation of the copper to inhibit the iron during both the flotation of the copper and zinc, the copper was decidedly acted on by the cyanide, resulting in a poor recovery and high copper in the zinc concentrate. From experience the writers strongly recommend conditioning and the use of additional cyanide between the copper and zinc sections.

RECOMMENDATIONS

Although the results of our tests indicate that the ore can be readily concentrated, it is recommended that a pilot plant be built at the mine and operated for a sufficient time to obtain the necessary data on crushing, grinding, conditioning, uniformity of the milling action of the ore, and the seasonal effect, if any, of the water supply. After obtaining this data the mill can be designed and erected with the assurance that no extensive alterations will be found to be necessary after it is put into operation. It is believed that in the long run a pilot plant will pay for the cost of construction and operation many times over.

Report No. 298A

THE CONCENTRATION OF THE COPPER-ZINC ORE FROM THE AMULET MINE, ROUYN, QUEBEC

C. S. Parsons, A. K. Anderson, and J. S. Godard

Shipment.—A shipment of 50 bags of ore was received on March 5, 1928, from the Amulet Gold Mines, Limited, Rouyn, Que.

Characteristics and Analysis of the Ore.—The sample received consisted of a heavy sulphide ore containing pyrite and pyrrhotite, chalcopyrite and sphalerite; gold and silver values are also present. The analysis of the shipment was as follows:

Copper	5·30 per cent	Gold	0·04 oz./ton
Zinc	3·71 "	Silver	3·10 "

Purpose of Tests.—Selective flotation tests were requested on this sample with a view of producing marketable concentrates of both copper and zinc, and bulk flotation tests to produce a combined copper-zinc concentrate. The flotation tailings were also to be concentrated and analysed for tin.

EXPERIMENTAL TESTS

For the selective flotation tests the ore was crushed to 8 mesh and fed by a belt feeder to an 18- by 24-inch rod mill in closed circuit with a 12-inch Akins classifier. The overflow of the classifier passed to a mechanically agitated conditioning tank where a contact of 30 minutes was obtained before the pulp passed to the copper flotation. The copper was floated in an 8-cell, Minerals Separation type machine with two extra boxes for mixing ahead of the flotation; the cells are constructed of Monel metal.

The concentrates from this machine were cleaned in a 3-cell, Denver Sub-A machine, the tailings from which were returned to the head of the rougher. The tailings from the rougher went to a conditioning tank similar to the first conditioner where a 30-minute contact was given the pulp before it passed to the zinc flotation.

The zinc was floated in a 6-cell, Denver Sub-A machine, the rougher concentrate being cleaned in a 24-inch, Callow pneumatic cell; the tailing from the cleaner was returned to the head of the rougher cells.

The following tables give details and results of the experimental tests:—

Small-scale Tests

Test No.	Product	Weight, per cent	Analysis				Per cent of values			
			Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
2	Head	......	4·24	4·67	0·02	2·28	......	......	......	......
	Copper concentrate	18·56	20·70	5·28	......	......	95·28	22·12	......	......
	Zinc concentrate	5·85	1·27	29·38	......	......	1·73	63·43	......	......
	Zinc middling	3·70	0·85	8·53	......	......	0·74	7·22	......	......
	Tailing	71·89	0·13	0·58	......	......	2·23	7·23	......	......
3	Head	......	4·24	4·67	0·02	2·28	......	......	......	......
	Copper concentrate	13·47	20·34	4·67	......	......	91·04	19·33	......	......
	Zinc concentrate	5·05	2·67	48·77	......	......	3·14	55·28	......	......
	Zinc middling	2·71	2·11	7·67	......	......	1·45	4·71	......	......
	Tailing	73·77	0·25	1·25	......	......	4·36	20·67	......	......

Test No.	Product	Weight, per cent	Analysis				Per cent of values			
			Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
4	Head	......	4·24	4·67	0·02	2·28	......	......	......	......
	Concentrate	21·84	17·66	18·19	......	......	94·36	88·03	......	......
	Middling	6·49	1·80	3·86	......	......	2·94	5·54	......	......
	Tailing	71·67	0·16	0·41	......	......	2·70	6·43	......	......

2,000 grammes ore —14 mesh ground 35 minutes in ball mill, added copper sulphate 1·2 lb./ton

Lime	5·0 lb./ton	Xanthate	0·1 lb./ton
Cyanide	0·2 "	Cresylic acid	0·3 "
Thiocarbanilide	0·15 "		

NOTE.—Copper would not float so made bulk concentrate.

Test No.	Product	Weight, per cent	Analysis				Per cent of values			
			Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
5	Head	......	4·24	4·67	0·02	2·28	......	......	......	......
	Copper concentrate	20·30	19·36	6·00	......	......	88·71	26·01	......	......
	Zinc concentrate	5·95	0·68	51·90	......	......	9·03	64·82	......	......
	Zinc middling	1·63	0·59	4·06	......	......	0·23	1·49	......	......
	Tailing	72·12	0·12	0·51	......	......	2·03	7·68	......	......

2,000 grammes ore —14 mesh ground 30 minutes in ball mill.

			10 min. contact
Soda ash	5·0 lb./ton	Cyanide	0·1 lb./ton
Thiocarbanilide	0·2 "	Copper sulphate	0·1 "
Cyanide	0·25 "	Aerofloat	0·1 "
Cresylic acid	0·2 "	Cresylic acid	0·2 "

Test No.	Product	Weight, per cent	Analysis				Per cent of values			
			Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
6	Head	....	4·24	4·67	0·02	2·28	......	......	......	......
	Copper concentrate	15·47	23·28	3·76	....	....	89·77	13·15	......	......
	Zinc concentrate	6·35	3·25	49·07	....	....	5·24	70·52	......	......
	Zinc middling	3·59	2·77	11·18	....	....	0·25	9·07	......	......
	Tailing	74·59	0·19	0·43	....	....	4·74	7·26	......	......

2,000 grammes ore −14 mesh ground 30 minutes in ball mill.

	To cells		To ball mill
TA mixture	10 drops	Soda ash	5·0 lb./ton
Cresylic acid	5 „	Cyanide	0·25 “

Zinc: Conditioned 10 minutes, cyanide 0·1 lb./ton, then added

Copper sulphate	1·5 lb./ton
Aerofloat	0·1 “
Cresylic acid	3·0 drops

Large-scale Tests

Test No. 1.—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate	20·9	25·10	4·8	0·06	11·14	98·3	33·2	....	....
Zinc concentrate	4·0	0·46	42·21	0·02	1·64	0·3	56·3	....	....
Tailing	75·1	0·10	0·42	trace	0·03	1·4	10·5	....	....
Head	....	5·35	3·00	0·04	3·10	....	....	....	....

Screen Tests.—

Mesh	Copper concentrate	Zinc concentrate	Tailing
+48	....	....	0·7
+65	0·2	....	1·9
+100	1·7	0·6	9·1
+150	8·2	2·0	8·3
+200	11·5	5·6	10·9
−200	78·3	91·8	69·1

Feed rate, 100 lb./hour. Run, 9 a.m. to 4 p.m.

Reagents.—

Time	Na_2CO_3 lb./ton	$CuSO_4$ lb./ton	Thio-carbanilide, lb./ton	Aerofloat, lb./ton	NaCN, lb./ton	Classifier overflow density	NaCN to Zn	Copper	
								Cresylic acid, lb./ton	Pine oil, lb./ton
8.45	5·5	....	....	....	0·15	....	....	....	....
9.10	....	....	....	....	0·23	1 : 4·2	....	....	....
9.30	....	....	....	....	....	....	0·11	....	....
9.40	....	....	....	....	....	1 : 3·3	....	....	....
9.55	....	1·1	....	....	....	....	....	....	....
10.00	5·6	....	....	....	0·21	1 : 3·5	0·11	....	....
10.35	....	1·1	....	....	....	....	....	....	....
10.45	....	....	....	....	....	1 : 3·7	....	....	....
10.55	5·6	....	....	....	0·21	....	0·11	....	....
11.15	....	....	....	0·4	....	1 : 4·2	....	....	....
11.40	5·8	1·2	....	....	0·25	....	0·10	....	....
11.50	....	....	....	....	....	1 : 3·5	....	....	....
12.00	....	....	....	0·3	....	....	....	....	....
12.15	....	....	....	0·23	....	....	....	0·54	0·12
12.35	5·9	1·2	....	....	0·25	1 : 2·8	0·10	....	....
1.15	5·6	....	....	0·23	0·25	....	0·10	....	....
1.50	5·8	1·0	....	....	0·24	1 : 3·0	0·14	....	....
2.55	....	....	....	....	....	1 : 3·2	0·14	....	....
3.15	5·9	0·9	....	0·16	0·27	....	0·14	....	....
3.50	....	0·9	....	....	....	1 : 2·7	....	....	....

Test No. 2.—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate	23·54	22·02	4·42	0·08	11·16	96·28	28·03	69·23	86·23
Zinc concentrate...	6·30	1·40	20·32	0·02	2·14	1·49	34·50	3·84	4·26
Tailing	70·16	0·17	1·98	trace	0·42	2·23	37·47	26·92	9·51
Heads	....	5·36	3·71	0·04	2·98	....	....	....	....

Feed rate, 105 lb. /hour.

Reagents.—

Time	Na_2CO_3 lb./ton	Thio-carbanilide, lb./ton	Cresylic acid, lb./ton	Pine oil, lb./ton	NaCN, lb./ton	Zinc reagents, lb./ton				Classifier overflow density
						$CuSO_4$	Aerofloat	Pine oil	NaCN	
9.00	5·9	0·15	0·83	0·15	0·31	0·80	0·21	0·15	0·14	....
9.30	5·9	....	0·83	0·15	0·29	0·80	0·21	0·15	0·14	1 : 3·5
10.00	5·9	0·15	0·83	0·15	0·29	0·85	0·21	0·15	0·14	1 : 3·0
10.15	5·6	0·15	0·83	off	0·27	0·83	0·21	0·15	0·13	1 : 2·8
10.45	5·6	0·15	0·83	....	0·25	0·83	0·21	0·15	0·12	1 : 3·3
11.00	5·6	0·15	0·83	....	0·26	0·83	0·21	0·15	0·29	....
11.30	5·9	0·15	0·83	....	0·24	0·83	0·21	0·15	0·29	1 : 2·2
12.15	5·7	0·15	0·83	....	0·27	0·89	0·21	0·15	0·29	....
12.45	5·6	0·15	0·83	....	0·27	0·85	0·21	0·15	0·29	....
1.30	5·6	0·15	....	....	0·29	0·85	0·21	0·15	....	1 : 3·2
2.00	5·4	0·15	....	....	0·29	0·91	0·21	0·15	0·29	....

*p*H of classifier overflow 8·6

Test No. 3.—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Head........	....	5·36	3·71	0·04	2·98	....	....	....	....
Sample No. 1									
Copper concentrate	17·91	28·12	3·20	0·08	10·84	94·03	15·36	60·87	81·52
Zinc concentrate...	11·08	2·08	24·79	0·02	2·34	4·29	74·12	8·69	10·92
Tailing...........	71·01	0·13	0·55	trace	0·25	1·68	10·52	30·44	7·56
Sample No. 2									
Copper concentrate	18·05	26·40	3·25	0·06	11·83	88·97	15·90	50·00	75·26
Zinc concentrate...	9·24	5·33	20·82	0·03	4·03	9·16	51·75	15·00	13·07
Tailing...........	72·71	0·14	1·65	0·01	0·45	1·87	32·35	35·00	11·67

Reagents.—

Time	Copper				Zinc			Feed, lb./ton	Classifier overflow density
	Na_2CO_3 lb./ton	A.T. mix, lb./ton	Cresylic acid, lb./ton	NaCN, lb./ton	$CuSO_4$, lb./ton	Aerofloat, lb./ton	NaCN, lb./ton		
9.00........	5·6	0·11	0·88	0·42	....	....	....	....	1 : 3·5
9.30........	5·6	0·11	0·88	0·42	1·1	....	0·05	....	1 : 3·0
9.45........	....	0·11	0·88	0·47	....	0·22	0·05	....	1 : 3·0
10.00........	5·6	0·11	0·58	0·47	1·1	0·22	....	108	1 : 2·44
10.15........	....	0·11	off	....	....	0·33	0·05	....	....
10.20........	5·6	0·11	....	0·42	....	0·22	....	....	....
10.45........	5·6	0·11	0·15	0·42	1·1	0·11	0·05	....	1 : 2·7
11.00........	5·6	0·11	0·15	0·50	1·1	0·11	0·05	....	....
11.15........	....	....	....	....	....	off	....	....	1 : 2·84
11.45........	5·6	0·11	0·15	0·50	1·1	....	0·09	....	1 : 2·84
12.10........	....	....	....	....	0·7	....	....	93	....
12.25........	5·3	0·11	0·73	2·3	0·65	....	0·07	100	....
12.45........	5·3	....	0·29	0·43	....	....	0·09	....	....
1.45........	....	....	....	....	1·1	....	....	....	....
2.00........	....	0·22	incr.	....	....	....	....	....	....
2.30........	5·3	0·22	0·58	0·22	1·1	0·11	0·07	....	1 : 2·7
3.00........	5·3	0·22	1·08	0·22	1·1	off	....	108	1 : 2·4
3.45........	4·0	0·11	0·88	0·22	1·1	....	0·07	....	1 : 2·4
4.30........	4·0	off	0·44	0·22	1·1	....	0·07	....	1 : 2·4

Water to ball mill 1,600 c.c./min.

Test No. 4.—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Head........	....	5·36	3·71	0·04	2·98	....	....	....	....
Sample No. 1									
Copper concentrate	32·51	16·00	5·36	0·08	8·14	97·39	47·17	78·79	91·69
Zinc concentrate...	5·40	1·16	33·45	0·02	2·30	1·12	48·79	3·03	4·15
Tailing...........	61·99	0·13	0·25	trace	0·20	1·49	4·04	18·18	4·16
Sample No. 2									
Copper concentrate	29·71	17·76	5·10	0·06	8·54	98·50	40·81	66·67	93·38
Zinc concentrate...	5·15	0·62	39·07	0·015	1·29	0·56	54·32	....	2·20
Tailing...........	65·14	0·08	0·28	trace	0·19	0·94	4·87	33·33	4·42

Reagents.—

Time	Copper					Zinc				
	Na_2CO_3, lb./ton	NaCN, lb./ton	$CuSO_4$, lb./ton	Aero-float, lb./ton	Cresylic acid, lb./ton	NaCN, lb./ton	Aero-float, lb./ton	Classifier overflow density	Feed, lb./hr.	Cresylic, lb./ton
8.45	3·5	0·2	0·7	0·22	0·14	0·15	....	1 : 4·2	....	....
9.00	....	....	....	....	....	....	....	....	102	....
9.15	3·5	0·2	0·7	0·22	0·14	0·15	....	1 : 4·9	....	....
9.30	....	....	....	0·11	0·14	0·19	....	....	....	....
10.00	3·5	0·2	0·7	0·11	0·14	0·19	0·22	1 : 2·8	....	....
10.20	3·5	0·2	0·7	0·11	0·28	0·19	0·22	....	....	0·14
10.30	....	....	....	0·11	0·28	0·19	0·22	1 : 3·0	....	0·14
11.50	3·0	0·2	0·7	0·11	0·28	0·19	0·22	1 : 3·5	....	0·14
2.00	....	....	....	0·11	0·28	0·19	0·22	1 : 3·7	....	0·14
2.40	....	....	....	0·11	0·28	0·19	0·11	....	....	0·14
3.20	....	....	....	0·11	0·28	0·19	0·11	1 : 3·0	102	0·14

*p*H of classifier overflow	9·6
Tailing	8·4

Test No. 5.—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Head	....	5·36	3·71	0·04	2·98	....	....	....	....
Sample No. 1									
Copper concentrate	25·87	20·30	6·00	0·08	11·52	98·50	41·89	99·00	88·43
Zinc concentrate	4·66	0·46	36·78	0·02	1·90	0·37	46·22	....	2·67
Copper tailing	....	0·10	4·42	trace	0·60	....	....	....	....
Zinc tailing	69·47	0·09	0·64	trace	0·44	1·13	11·89	....	8·90
Sample No. 2									
Copper concentrate	20·91	25·16	3·76	0·08	12·08	98·13	21·29	99·00	87·54
Zinc concentrate	5·60	0·60	45·52	0·02	1·46	0·56	68·73	....	2·77
Copper tailing	....	0·16	3·71	trace	0·46	....	....	....	....
Zinc tailing	73·49	0·09	0·51	trace	0·38	1·31	9·98	....	9·69

Reagents.—

Time	Copper						Zinc			
	Na_2CO_3, lb./ton	NaCN, lb./ton	$ZnSO_4$, lb./ton	$CuSO_4$, lb./ton	Cresylic acid, lb./ton	Thiocarbanilide, lb./ton	Aero-float, lb./ton	NaCN, lb./ton	Feed, lb./hr.	Classifier overflow density
9.30	5·1	0·20	1·06	....	0·22	0·15	....	....	105¾	1 : 3·7
10.00	5·1	0·23	1·06	....	0·33	0·15	....	....	....	1 : 4·2
10.30	5·1	0·28	1·08	0·83	0·33	0·15	....	0·11	107·8	1 : 3·5
11.00	5·1	0·28	1·08	0·83	0·44	0·15	....	0·11	....	1 : 3·7
11.30	5·1	0·28	1·08	0·83	0·44	0·15	....	0·11	....	1 : 3·7
12.00	5·1	0·28	1·08	0·83	0·44	0·15	....	0·11	....	1 : 3·7
12.30	5·1	0·28	1·08	0·83	0·44	0·15	....	0·11	....	1 : 3·3
1.00	5·1	0·28	1·08	0·83	0·44	0·15	....	0·11	....	1 : 3·3
1.30	5·1	0·28	1·08	0·83	0·44	0·15	....	0·11	....	1 : 3·3
2.00	5·1	0·41	1·08	0·83	0·44	0·15	....	0·11	....	1 : 3·1
2.30	5·1	0·41	1·03	0·88	0·44	0·15	....	0·11	103½	1 : 2·8
3.00	5·1	0·41	1·03	0·88	0·44	0·15	....	0·11	....	1 : 3·0
3.30	5·1	0·41	1·03	0·88	0·44	0·15	0·22	0·11	....	1 : 3·1

Test No. 6. Bulk concentration.

Product	Assay				Recovery, per cent			
	Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper + zinc concentrate	21·58	15·06	0·06	8·71	96·30	90·57	99·00	....
Tailing	0·26	0·45	trace	0·78	....	....	....	....

Reagents.—

Time	CaO, lb./ton	NaCN, lb./ton	$CuSO_4$, lb./ton	Xanthate, lb./ton	Cresylic acid, lb./ton	Feed, lb./hr.	Classifier overflow density
9.30	5·0	0·40	0·85	....	....	....	....
10.00	5·0	0·40	0·85	0·13	0·28	96	....
10.30	5·0	0·40	0·85	0·13	0·28	....	1 : 3·1
11.00	5·0	0·40	0·85	0·13	0·28	103¼	1 : 3·5
11.30	5·0	0·40	0·85	0·17	0·28	....	1 : 3·1
12.00	5·0	0·40	0·85	0·17	0·28	....	....
12.30	5·0	0·40	0·85	0·17	0·28	....	1 : 2·7
1.00	5·0	0·40	0·85	0·17	0·28	....	1 : 2·5
1.30	5·0	0·40	0·85	0·17	0·28	....	1 : 2·5
2.00	5·0	0·40	0·85	0·17	0·28	....	1 : 3·0
2.30	5·0	0·40	0·85	0·21	0·28	100·9	1 : 3·0
3.00	5·0	0·40	0·85	0·21	0·28	....	1 : 3·0
3.30	5·0	0·40	0·85	0·21	0·28	....	1 : 3·1
4.00	5·0	0·40	0·85	0·21	0·28	....	1 : 2·8

Tabling Flotation Tailings for Tin

Eighteen bags (wet) were tabled on a Wilfley slime deck and the concentrates sampled and assayed for tin.

Results.—

Sample No. 1	0·04 per cent Sn.
Sample No. 2	0·06 "
Sample No. 3	0·05 "

CONCLUSIONS

The results of the selective and bulk flotation tests can on the whole be considered satisfactory. Special attention is drawn to selective Tests Nos. 1 and 5, and to the bulk Test No. 6.

The control of the reagents and of the period of conditioning the pulp were found to be the principal factors influencing the results. In the use of the cyanide reagent, which if added in excess has a decided deadening effect on the flotation of the copper, it was found that the minimum quantity of the reagent with a long period of conditioning gave the sharpest separations. Adding the cyanide to the conditioning tank instead of to the grinding mill gave more uniform results in flotation. It was also found advisable to condition the copper tailing pulp with an additional amount of cyanide before floating the zinc.

A 30-minute period of conditioning was the maximum time that could be given, but the indications were that even a longer period than this would improve the results.

The flotation tailing was tabled and the concentrate analysed for in; only traces were found.

Report No. 298B

FLOTATION OF COPPER-ZINC ORE FROM THE AMULET MINE, ROUYN DISTRICT, QUEBEC

C. S. Parsons, A. K. Anderson, and J. S. Godard

Shipment.—A shipment of one carload of ore, consisting of 20 tons in sacks, was received at the Ore Testing Laboratory, September 4, 1928. The shipment consisted of a special sample of copper-zinc ore from the Amulet Mines, Ltd., Rouyn, Quebec.

Characteristics of the Ore.—The ore is a complex copper-zinc sulphide requiring relatively fine grinding to free the sulphides. A small amount of lead, less than 1 per cent, is present, together with some silver and gold.

Arrangements for Tests.—A number of shipments of Amulet ore have been made from time to time to the Mines Branch for experimental work. Unfortunately the occurrence of the ore in the mine makes it extremely difficult to obtain a sample of the average grade of the ore the mine will produce under actual mining operations. As development work proceeded it was found that the samples previously submitted did not contain the correct ratio of copper to zinc. The management, under the direction of Alderson, McKay, and Armstrong, consulting engineers for the company, applied for permission to ship a new sample. This sample was taken under the direction of Prof. MacKay of Queen's University, who was also present during the running of the experimental tests.

Purpose of Tests.—The purpose of these special tests was to obtain data for the erection of a mill on the property and also to obtain accurate figures on the smelter returns which could be expected on the products from the mill, namely the copper and zinc concentrates.

Sampling and Analysis.—A number of bags of waste rock were included in the shipment in order to vary the grade of the ore if it was found that the sample did not truly represent the mill grade of ore. Separate samples were taken from day to day and represent only the feed and heads for each test.

EXPERIMENTAL TESTS

The experimental work consisted of large-scale tests, the ore being treated at the rate of approximately ½ ton per hour. The flow-sheet used was as follows:

The ore was crushed to ½ inch in a jaw crusher and rolls, fed to a Hardinge, conical mill, 4 feet 6 inches by 13 inches, containing a charge of 4,000 pounds of 2- to 1-inch balls, which was in closed circuit with a small Dorr classifier; classifier overflow was approximately 80 per cent −200 mesh. The copper was floated in an 8-cell, Greenawalt flotation machine of the mechanical agitation type, 6 cells being used to produce a rougher concentrate which was cleaned in the first two cells, producing a finished copper concentrate. The tailing was elevated by an air lift to a Callow flat-bottom unit consisting of two roughers in series and two cleaners in series. The zinc was floated in this unit, the rougher concentrates being cleaned twice.

Results of the Tests

For the purpose of figuring the returns on a bulk concentrate of the copper and zinc, the last test, No. 4, was run to produce this type of concentrate. The bulk concentrate was desired by the Base Metal Extraction Company who are contemplating the erection of a smelter plant in eastern Canada. The following tables give the results of the tests and the reagents used:—

Test No.	Sample No.	Product	Weight, per cent	Analysis				Per cent of values			
				Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
1	1a	Copper concentrate	8·93	23·16	6·53	0·04	13·40	91·59	6·78	40·0	75·74
		Copper tailing	......	0·16	7·57	trace	0·41	......	......	....	......
		Zinc concentrate	14·41	1·11	49·25	0·01	1·41	7·08	83·04	....	12·86
		Zinc tailing	76·66	0·04	1·14	trace	0·24	1·33	10·18	....	11·40
	1b	Copper concentrate	8·62	23·97	5·30	0·04	14·44	91·59	5·37	35·0	76·10
		Copper tailing	......	0·25	8·56	trace	0·52	......	......	....	......
		Zinc concentrate	14·67	1·11	49·28	0·01	1·41	7·08	84·47	....	12·65
		Zinc tailing	76·71	0·04	1·14	trace	0·36	1·33	10·16	....	11·25
	2	Copper concentrate	9·76	22·18	6·44	0·04	13·00	95·15	7·36	39·0	74·82
		Copper tailing	......	0·14	8·65	trace	0·46	......	......	....	......
		Zinc concentrate	9·78	0·55	50·60	0·01	1·05	2·20	57·83	....	8·14
		Zinc tailing	80·46	0·08	3·71	trace	0·36	2·65	34·81	....	17·04
	3	Copper concentrate	8·58	24·54	4·60	0·05	15·23	92·10	4·56	40·0	76·74
		Copper tailing	......	0·31	10·39	trace	0·74	......	......	....	......
		Zinc concentrate	11·77	1·01	50·30	"	1·29	5·26	69·16	....	3·61
		Zinc tailing	79·65	0·08	2·82	"	0·42	2·64	26·28	....	19·65
2	1	Copper concentrate	11·10	21·38	7·25	0·04	14·28	97·53	6·82	44·0	80·61
		Copper tailing	......	0·11	8·80	......	0·42	......	......	....	......
		Zinc concentrate	17·97	0·25	54·00	......	0·83	1·65	82·62	....	7·65
		Zinc tailing	70·93	0·03	1·75	......	0·33	0·82	10·56	....	11·74
	2	Copper concentrate	8·92	25·04	4·65	0·04	14·94	91·77	3·49	36·0	73·89
		Copper tailing	......	0·15	8·60	......	0·47	......	......	....	......
		Zinc concentrate	19·40	0·84	50·30	......	1·32	6·58	83·06	....	14·44
		Zinc tailing	71·68	0·06	2·20	......	0·30	1·65	13·45	....	11·67
3	1	Copper concentrate	8·36	24·31	6·85	0·04	15·40	94·86	6·10	33·0	61·4
		Copper tailing	......	0·12	9·00	......	......	......	......	....	......
		Zinc concentrate	16·30	0·38	50·60	......	......	2·80	88·23	....	......
		Zinc tailing	75·34	0·06	0·70	......	......	2·34	5·67	....	......
	2	Copper concentrate	8·50	25·55	5·95	0·04	16·28	92·34	5·46	34·0	65·7
		Copper tailing	......	0·16	11·20	......	......	......	......	....	......
		Zinc concentrate	15·48	0·56	51·0	......	......	3·83	84·38	....	......
		Zinc tailing	76·02	0·12	1·25	......	......	3·83	10·16	....	......
	Special	Copper concentrate	8·65	25·55	5·95	......	......	94·04	5·45	....	......
		Copper tailing	......	0·16	11·20	......	......	......	......	....	......
		Zinc concentrate	15·64	0·51	51·4	......	......	3·40	85·99	....	......
		Zinc tailing	75·71	0·08	1·05	......	......	2·56	8·56	....	......
		Screen test on classifier overflow—									
		+200	9·5	......	......	......	......	......	......	....	......
		−200	90·5	......	......	......	......	......	......	....	......
4	....	Concentrate	31·77	8·17	37·50	0·02	5·50	97·9	97·5	63·5	83·2
		Tailing	68·23	0·08	0·45	......	......	2·1	2·5	....	......
		Heads—									
		Test No. 1	....	2·27	8·56	0·01	1·70	......	......	....	......
		Test No. 2	....	2·43	11·75	0·01	1·95	......	......	....	......
		Test No. 3	....	2·35	9·35	0·01	2·10	......	......	....	......
		Test No. 4	....	2·65	12·20	0·01	2·10	......	......	....	......

REAGENTS USED

(Pounds per ton)

Test No.	Time	Na_2CO_3	NaCN	$ZnSO_4$	TT	Xanthate	Pine oil	$CuSO_4$	Xanthate	Pine oil	Feed, lb./hr.	Densities		Remarks
												Ball mill	Classifier overflow	
1	9.00	1·0	0·15	....	0·10	0·04	0·08	0·50	0·175	....	860	23·0	8·0	Started ball mill 8.17 a.m.
	9.30	1·5	0·16	....	0·10	0·04	0·11	0·34	0·175	....	....	27·0	12·0	
	10.00	1·5	0·19	....	0·10	0·04	0·07	0·36	0·175	....	......	39·0	15·0	Began $ZnSO_4$ to ball mill.
	10.30	1·6	0·19	0·40	0·07	0·027	0·06	0·36	0·175	....	....	44·0	26·0	pH 9·6.
	11.00	1·6	0·19	0·40	0·07	0·027	0·08	....	0·06	....	....	55·0	37·0	Xanthate to zinc cut down.
	11.30	1·6	0·25	1·63	0·07	0·027	0·04	0·36	0·06	....	....	49·0	37·0	11·55 a.m. end of No. 1a copper sampling.
	12.00	....	....	....	....	....	....	....	....	....	....	50·0	39·0	
	12.30	1·6	0·25	1·63	0·07	0·025	0·02	0·33	....	....	....	50·0	35·0	Pine oil and xanthate taken off copper.
	1.00	1·6	0·25	1·63	0·25	off	off	0·35	0·08	....	....	53·0	37·0	End of No. 1b copper and No. 1 zinc samples.
	1.30	1·6	0·25	1·63	0·13	....	Cresyl	0·35	0·14	....	....	....	....	Cresylic acid replacing pine oil.
	1.50	....	....	....	0·06	....	off	....	0·175	....	....	....	....	Began No. 2 samples.
	2.00	1·6	0·25	1·63	....	....	....	0·35	0·21	....	....	35·0	37·0	Xanthate increased.
	2.15	....	....	....	....	....	....	0·82	....	....	....	....	....	Grab sample copper concentrate, 5·84 per cent zinc.
	2.30	....	....	....	....	....	....	0·59	....	....	....	55·0	41·0	Reduced $CuSO_4$ —2·35, No. 3 sample.
	3.00	1·6	0·25	1·63	0·06	....	0·03	0·59	0·30	....	....	55·0	43·0	pH 9·6.
	3.50	1·6	0·25	1·63	0·06	....	0·03	0·73	0·18	....	....	....	....	Copper appearing in zinc cells.
	4.20	....	....	....	....	....	0·03	....	....	....	....	....	....	Closed down feed and sample No. 3.
	4.25	Total feed, 7,486 lb. Running time, 8 hours........												Ball mill shut down.
2	10.00	1·5	0·31	1·6	off	0·02	0·09	0·36	0·09	....	890	47·0	33·0	Started ball mill 9.30 a.m.
	10.30	1·5	0·33	1·6	....	0·02	0·09	0·40	0·09	....	....	44·0	26·0	Copper in zinc cells.
	11.00	1·5	0·25	1·6	0·08	0·03	0·09	0·40	0·09	....	....	45·0	30·0	Added TT
	11.30	1·5	0·25	1·6	0·08	0·03	0·12	0·40	0·09	....	....	50·0	33·0	Xanthate raised. No. 1 copper sample started at 11.40 a.m.
	12.00	1·5	0·25	1·6	0·08	0·04	0·12	0·59	0·17	0·05 Aero-float	....	52·0	38·0	
	12.30	1·5	0·25	1·6	0·08	0·04	0·12	0·45	0·17	0·07	....	57·0	47·0	
	1.00	1·6	0·25	1·6	0·08	0·04	0·12	0·45	....	....	....	57·0	47·0	Began No. 1 zinc sample.
	1.30	1·6	0·25	1·6	·0·08	0·03	0·12	0·45	0·17	0·07	....	60·0	35·0	Increased water in classifier at 1.20 p.m.
	1.45	1·6	0·28	1·48	....	....	0·12	0·45	0·17	0·07	....	....	....	
	1.50	1·6	0·28	1·6	....	0·03	0·12	0·54	0·20	0·10	885	59·0	35·0	
	2.00	1·6	0·28	....	0·08	0·03	0·12	0·54	0·20	0·10	....	....	....	
	2.20	....	....	1·7	....	....	....	....	0·23	....	....	....	....	
	2.30	1·6	0·28	1·7	0·06	0·015	0·09	0·54	0·23	0·10	....	59·0	34·0	
	3.00	1·6	0·28	1·7	0·06	0·015	0·09	....	....	0·10	....	60·0	34·0	
	3.10	....	....	....	....	....	....	0·59	0·26	....	....	....	....	Zinc reagents increased.
	3.15	....	....	....	....	....	....	....	....	....	....	....	....	Changed to No. 2 sample copper and zinc.
	3.30	1·6	0·28	1·7	0·06	0·015	0·09	0·59	0·26	0·10	....	59·0	35·0	
	4.15	....	....	....	....	....	....	....	....	....	....	....	....	Feed finished—last of No. 2 sample taken.
	4.25	Closing down												

REAGENTS USED

(Pounds per ton)

Test No.	Time	Na_2CO_3	NaCN	Thio-carbanilide	Cresylic acid	$CuSO_4$	Xanthate	Aerofloat	Pine oil	Feed, lb./hr.	Densities		Remarks
											Ball mill	Classifier overflow	
3	9.45	1·9	0·20	0·09	0·09	0·35	0·20	0·13	......	......	......	......	Began at 9.30 a.m.
	10.00	1·9	0·20	0·09	0·09	0·62	0·20	0·13	......	......	28	9	
	10.15	......	......	......	......	......	off	......	......	......	36	9	
	10.30	1·9	0·20	0·09	0·09	0·30	......	0·13	......	......	44	21	
	10.50	......	......	......	......	0·42	......	......	......	......	......	......	Began copper sample No. 1.
	11.00	1·9	0·30	......	0·09	0·46	0·12	0·13	......	......	52	38	
	11.25	......	......	......	......	0·46	......	......	......	......	......	......	Began zinc sample No. 1.
	11.30	1·88	0·30	......	0·09	0·46	0·12	0·13	0·05	......	51	42	
	11.50	......	0·30	......	0·09	0·46	0·13	0·13	......	......	54	30	
	12.00	......	......	......	......	......	0·17	......	0·12	933	53	28	Began copper and zinc sample No. 2.
	12.30	1·88	0·30	......	0·09	0·46	0·17	0·13	......	......	53	38	
	12.50	......	......	......	......	......	......	......	......	......	55	43	
	1.00	1·88	0·31	0·09	0·09	0·46	0·19	0·13	......	......	56	42	
	1.30	1·88	0·31	0·09	0·09	0·51	0·20	0·13	0·12	......	57	41	1.55 p.m. Began increasing $CuSO_4$ and xanthate.
	2.00	1·88	0·31	......	0·09	0·53	0·22	0·13	0·12	......	58	46	
	2.30	......	......	......	0·09	0·58	0·24	0·13	0·11	......	55	42	
	3.15	1·84	0·30	......	0·09	0·60	0·23	0·13	0·11	......	53	41	2.35 p.m. Began taking special sample of zinc concentrate and tailing.
	3.30	1·84	0·30	......	0·09	0·60	0·23	0·13	0·11	......	53	38	
	4.00	1·84	0·30	......	0·09	0·55	0·23	0·13	0·11	......	53	38	
	4.20	Feed finished, 4·25. Began closing down.											
4		CaO											
	9.30	7·5	0·20	......	......	1·1	0·20	......	0·12	900	......	......	9.20 a.m. Began ball mill.
	10.00	7·5	0·20	......	......	0·95	0·20	......	0·14	......	55	56	
	10.30	......	0·20	......	......	0·95	0·21	......	0·14	......	55	29	Began sampling.
	11.00	......	0·20	......	......	0·95	0·21	......	0·14	900	......	32	Greenawalt cell discharge, 19 per cent solids.
	11.30	......	0·20	......	......	0·93	0·21	......	0·14	......	55	44	
	12.00	......	0·20	......	......	0·93	0·21	......	0·14	......	56	41	
	12.30	7·5	0·20	......	......	0·91	0·19	......	0·14	......	56	41	
	1.00	......	0·20	......	......	0·91	0·21	......	0·14	......	56	43	
	1.30	......	0·20	......	......	0·91	0·21	......	0·14	......	58	40	
	2.00	......	0·20	......	......	0·91	0·21	......	0·14	......	60	40	
	2.30	......	0·20	......	......	0·91	0·21	......	0·15	......	60	43	
	3.00	......	0·20	......	......	0·91	0·21	......	0·15	......	61	45	
	3.30	......	0·26	......	......	0·91	0·21	......	0·15	......	57	45	
	4.00	......	0·25	......	......	0·91	0·21	......	0·15	......	55	41	
	4.15	......	......	......	......	......	......	......	......	911	57	42	Ceased sampling.
	4.20	Closing down. Feed, 6,357 lb. Running time, 7 hours.											

SUMMARY AND CONCLUSIONS

It seems hardly necessary to discuss in detail the results of these tests. The outstanding feature is the effect of the use of zinc sulphate. When this reagent is used in conjunction with cyanide for the purpose of lowering the zinc in the copper concentrate it was found that the depressive effect on the zinc was too severe and that in the subsequent flotation of the zinc a low tailing could not be obtained. It is possible that conditioning for a period of time between the copper and zinc separations might overcome this effect. Experience with other ores which have been run on a smaller unit of 100 pounds an hour capacity equipped with a conditioning tank between the copper and zinc cells, indicates that very much improved results can be expected by the use of a conditioning tank giving a contact of 30 minutes. The gold content of this sample was very low and according to the mine management, is not representative. The results on previous shipments indicate that if gold is in excess of $2.00 per ton that at least 60 per cent of the gold can be recovered in the copper concentrate.

RECOMMENDATIONS

For reagents to have on hand to start the mill up with, the following are recommended:—

Thiocarbanilide
Pine oil
Cresylic acid
Potassium xanthate
Orthotoluidine, for TT mixture
Soda ash
Cyanide
Copper sulphate
Zinc sulphate
Hydrated lime containing less than 1 per cent magnesium oxide.

For the 100-ton mill conditioning of the pulp for one-half to three-quarter hour before the flotation of the copper and one-half hour conditioning between the copper and the zinc separations are recommended.

The use of an additional cleaner on the copper concentrate (four cells) and the use of 2 four-cell cleaners on the zinc is desirable; the cleaners to be operated so that the concentrate from the preceding cleaner is recleaned in the second; the tailing from both to be returned to the head of the rougher cells.

Report No. 299

CONCENTRATION OF LEAD-ZINC ORE FROM LARCHWOOD, ONTARIO

A. K. Anderson and C. S. Parsons

Shipment.—Five bags of ore, weighing 365 pounds, were received January 5, 1928, from H. E. Harcourt of the Sudbury Concentrating and Mining Co., Ltd., through the Base Metals Extraction Company. The ore was from the Larchwood property.

Characteristics.—This material consisted of approximately 75 per cent of massive sulphides of iron and zinc with finely divided galena interspersed throughout. The latter mineral is apparently of a later origin than the former as it is segregated along joint-planes through the iron and zinc mineral. These planes also show indications of slight leaching and recrystallization of the gangue. Small amounts of graphite are present along slickensides which probably occur along the walls of the ore-body.

Purpose of Experimental Tests.—The primary object of this investigation was to determine whether it is possible to secure by flotation a recovery of the zinc and lead in a high-grade concentrate containing a low percentage of iron, such a product being suitable for treatment by the methods of the Base Metals Extraction Company. As no specified grade of concentrate was asked for, the investigation was conducted along the above lines.

Sampling and Analysis.—Approximately 25 pounds of the ore as received was crushed through a jaw crusher and rolls to −14 mesh, passed through a Jones riffle sampler and a representative head sample secured. This sample gave the following analysis:

Lead	4·17 per cent	Iron	29·89 per cent
Zinc	19·76 "	Gold	0·07 oz./ton
Copper	0·15 "	Silver	2·18 "

A second lot, weighing 50 pounds, sampled in a similar manner yielded the following analysis:

Lead	3·57 per cent	Iron	31·47 per cent
Zinc	17·82 "	Gold	0·06 oz./ton
Copper	0·29 "	Silver	1·92 "

Tests Nos. 1 and 2 were run on the first lot, and the remainder on the second lot.

EXPERIMENTAL TESTS

All tests unless otherwise noted were conducted on 1,000-gramme samples of the −14-mesh material ground in porcelain jars containing iron balls. All grinding was done in 1 : 1 pulp with the addition of reagents as indicated in the individual tests. Due to the finely divided condition of the galena it was found necessary to grind through 150 mesh. Flotation was carried out in a laboratory Ruth machine.

In Tests Nos. 1 to 4 the results of removing a graphitic concentrate were studied.

Test No. 9 was carried out to produce a preferential flotation of the lead and zinc.

The remaining tests were made with the object of obtaining one concentrate containing the value of the ore.

Test No. 1

This preliminary test was performed to note the behaviour of the ore under conditions calculated to yield results with normal material. Grinding was done in a circuit made alkaline with soda ash with the addition of sodium cyanide and zinc sulphate to depress the zinc and iron. The zinc was then reactivated by copper sulphate added to the flotation cell and a concentrate taken off by the addition of Aerofloat. After this concentrate was removed, potassium xanthate was added and a second concentrate which appeared to contain much zinc was secured. After removal of this concentrate a further addition of potassium xanthate was made and a third concentrate taken off.

Reagents.—

To ball mill—			To flotation machine—		
Soda ash	16·0	lb./ton	Copper sulphate	2·0	lb./ton
Cyanide	0·3	“	Aerofloat	0·12	“
Zinc sulphate	0·2	“	Potassium xanthate	0·6	“
			Pine oil	0·06	“

Results.—

Product	Weight, per cent	Analysis			Per cent of values		
		Pb, per cent	Zn, per cent	Fe, per cent	Pb	Zn	Fe
Head	....	4·17	19·76	29·89	....	....	....
1st concentrate	17·83	6·09	14·87	30·86	26·43	13·39	18·73
2nd concentrate	28·95	5·51	40·60	15·91	38·83	59·81	15·68
3rd concentrate	9·94	3·95	27·17	25·09	9·57	13·74	8·49
Tailing	43·28	2·39	5·93	38·76	25·17	13·06	57·10

The first concentrate carried considerable quantities of lead, zinc, and iron, showing that other reagents are necessary to eliminate them. The tailing also carried large amounts of both lead and zinc.

The second concentrate indicates that potassium xanthate is effective on the flotation of the zinc mineral with a tendency as shown in the third concentrate to lift the iron pyrite.

Test No. 2

As the graphite appears to rob the oil from the sulphides, an attempt was made in this test to remove this material. Kerosene and pine oil were added to produce a graphitic froth prior to the flotation of lead and zinc.

Reagents.—

To ball mill—			To flotation machine—		
Soda ash	20·0	lb./ton	Copper sulphate	2·0	lb./ton
Sodium cyanide	0·3	“	Potassium xanthate	0·3	“
Kerosene	0·1	“	Pine oil	0·06	“

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Pb, per cent	Zn, per cent	Cu, per cent	Fe, per cent	Pb	Zn	Cu	Fe
Head	....	4·17	19·76	0·15	29·89	....	....	....	....
1st concentrate	3·54	5·75	11·81	0·28	32·54	4·60	2·13	6·85	3·90
1st middling	3·88	5·58	15·25	0·24	32·03	5·15	3·01	6·16	4·21
2nd concentrate	15·97	7·32	50·24	0·25	8·72	27·86	40·83	27·40	4·21
2nd middling	7·74	5·18	14·34	0·16	32·95	9·53	5·65	8·22	8·64
Tailing	68·87	3·22	13·80	0·11	33·86	52·86	48·38	51·37	79·04

Under these conditions the removal of the graphite prior to floating the lead and zinc is of no benefit. The complete removal of graphite was not secured as it continued to float until the completion of the test. The tailing is quite high in all minerals.

Test No. 3

An attempt was made to secure a low-grade graphitic froth by decreasing the amount of kerosene used in Test No. 2, followed by preferential flotation of the lead and zinc.

Reagents.—

To ball mill—		To flotation machine—	
Soda ash............	16·0 lb./ton	Copper sulphate.....	2·0 lb./ton
Zinc sulphate.......	0·2 "	Potassium xanthate.	0·3 "
Cyanide............	0·3 "	Pine oil............	0·06 "
Kerosene...........	0·02 "		

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Pb, per cent	Zn, per cent	Cu, per cent	Fe, per cent	Pb	Zn	Cu	Fe
Graphitic concentrate............	21·80	5·28	12·67	0·35	33·32	30·51	15·49	29·80	23·25
Lead concentrate..	11·48	12·19	18·90	0·85	26·73	37·05	12·17	38·04	9·82
Zinc concentrate...	23·29	1·24	47·14	0·17	13·92	7·66	61·61	15·29	10·38
Tailing...........	43·43	2·15	4·40	0·10	40·67	24·78	10·73	16·87	56·55
Head.............	....	3·57	17·82	0·29	31·47	....	....	....	....

The graphitic concentrate in this test still carries high percentages of lead and zinc. The lead concentrate is high in zinc, while the tailing carries considerable lead and zinc.

Test No. 4

A further attempt was made to produce a low-grade graphite concentrate by using pine oil alone in a minute quantity, with no addition of kerosene. The lead and zinc were then recovered in one concentrate.

Reagents.—

To ball mill—		To flotation machine—	
Soda ash............	16·0 lb./ton	Copper sulphate.....	2·0 lb./ton
Cyanide............	0·3 "	Potassium xanthate.	0·3 "
Zinc sulphate.......	0·2 "	Pine oil............	0·06 "

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Pb, per cent	Zn, per cent	Cu, per cent	Fe, per cent	Pb	Zn	Cu	Fe
Head.............	....	3·57	17·82	0·29	31·47	....	....	....	....
Graphitic concentrate............	13·58	4·65	12·13	0·38	33·22	16·91	9·13	18·38	14·48
Lead and zinc concentrate.........	38·64	5·20	38·77	0·40	17·35	54·79	83·08	54·76	21·52
Tailing...........	47·78	2·21	2·98	0·16	41·73	28·30	7·79	26·86	64·00

It is apparent from the above tests that no benefit is derived from the removal of a graphitic concentrate as this carries high percentages of both lead and zinc. While the recovery of both metals is good the ratio of concentration, 1·9 : 1, is low.

Test No. 5

As considerable lead and zinc reported in the tailing from previous tests an attempt was made to recover this. A small amount of kerosene was added to oil the graphite, and TT, xanthate, and pine oil were used to float lead and zinc. The iron was depressed by sodium cyanide in a circuit made alkaline with soda ash.

Reagents.—

To ball mill—		To flotation machine—	
Soda ash	18·0 lb./ton	Copper sulphate	2·0 lb./ton
Cyanide	1·0 "	TT	0·12 "
Kerosene	0·02 "	Xanthate	0·3 "
		Pine oil	0·04 "

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Pb, per cent	Zn, per cent	Cu, per cent	Fe, per cent	Pb	Zn	Cu	Fe
Head	....	3·57	17·82	0·29	31·47	....	....	....	....
Concentrate	35·62	5·36	42·31	0·42	14·62	54·78	85·09	55·35	16·47
Middling	14·56	3·81	8·05	0·29	38·02	15·92	6·62	14·5	17·51
Tailing	49·82	2·25	2·95	0·16	41·90	29·30	8·29	29·15	66·02

No noticeable improvement over previous tests is observed in results secured in this one.

Test No. 6

In this test the effect of adding lime to the circuit prior to flotation was studied. The ore was ground with barely enough soda ash to neutralize the acidity, and after transferring to the flotation machine enough lime was added to make the pulp alkaline. A short period of contact was allowed before adding the other reagents.

Reagents.—

To ball mill—		To flotation machine—	
Soda ash	14·0 lb./ton	Lime	2·0 lb./ton
Cyanide	1·0 "	Copper sulphate	2·0 "
Kerosene	0·02 "	TT	0·40 "
		Xanthate	0·2 "
		Pine oil	0·06 "

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Pb, per cent	Zn, per cent	Cu, per cent	Fe, per cent	Pb	Zn	Cu	Fe
Head	....	3·57	17·82	0·29	31·47	....	....	....	....
Concentrate	54·83	4·84	29·64	0·34	23·37	74·85	91·18	77·18	43·39
Middling	16·44	2·13	4·51	0·13	41·85	9·85	4·17	8·71	20·51
Tailing	28·73	1·89	2·85	0·12	42·14	15·30	4·65	14·11	36·10

TT mixture added to the flotation cells yielded a black froth, low in zinc. To float this mineral it was found necessary to add potassium xanthate. The addition of lime to the circuit after grinding, followed by a short time contact has a beneficial result, yielding a good recovery of both lead and zinc. The tailing still carries appreciable amounts of both minerals.

Test No. 7

The effect of increased alkalinity obtained by a greater addition of lime was studied.

Reagents.—

To ball mill—		To flotation machine—	
Soda ash...........	12·0 lb./ton	Lime...............	4·0 lb./ton
Cyanide............	1·0 "	Copper sulphate.....	2·0 "
Kerosene...........	0·02 "	Xanthate...........	0·5 "
		Pine oil.............	0·1 "

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Pb, per cent	Zn, per cent	Cu, per cent	Fe, per cent	Pb	Zn	Cu	Fe
Head.............	....	3·57	17·82	0·29	31·47	....	....	....	....
Concentrate........	35·45	5·50	37·05	0·45	17·96	56·02	74·79	63·60	20·20
Middling..........	24·38	3·11	13·37	0·16	34·64	21·81	18·57	15·60	26·80
Tailing............	40·17	1·92	2·90	0·13	41·58	22·17	6·64	20·80	53·00

The deleterious effect of an excess of lime is clearly marked in this test. Although the grade of concentrate secured is higher than that of the preceding test, the recovery is lower due to a less bulky concentrate being recovered. The depressing action of lime on iron pyrite is clearly marked, more iron passing out in the tailing, thus raising the grade of the finished concentrate.

Test No. 8

This test was run to determine the effect of lime and an increased amount of cyanide added to the grinding circuit.

Reagents.—

To ball mill—		To flotation machine—	
Soda ash...........	12·0 lb./ton	Copper sulphate.....	2·0 lb./ton
Lime..............	2·0 "	Xanthate...........	0·5 "
Cyanide............	2·0 "	Pine oil.............	0·1 "

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Pb, per cent	Zn, per cent	Cu, per cent	Fe, per cent	Pb	Zn	Cu	Fe
Head.............	....	3·57	17·82	0·29	31·47	....	....	....	....
Concentrate........	17·46	4·92	42·54	0·35	14·95	23·87	44·95	22·76	8·04
Middling..........	19·46	4·00	18·04	0·35	31·00	21·62	21·23	25·37	18·72
Tailing............	63·08	3·11	8·86	0·22	37·30	54·51	33·82	51·87	73·24

A marked decrease in the bulk of concentrate produced together with a much lower recovery of the metals results from the use of lime and excess cyanide in the grinding circuit.

Test No. 8A

To check results secured in the foregoing test, much higher lime in the grinding circuit together with an increase in the cyanide was used. After grinding, the water was decanted and flotation conducted in a fresh water circuit with no further addition of alkaline reagents. Results were negative, no concentrate being obtained with flotation conditions the same as Test No. 8. All three of the minerals, lead, zinc, and iron were depressed to such an extent that they could not be reactivated and floated with xanthate and pine oil.

Test No. 9

The behaviour of the ore under conditions calculated to give a preferential flotation of the lead and zinc was studied. Grinding was carried on in a pulp made alkaline with soda ash. Cyanide was used to depress the iron and thiocarbanilide added as a promoter. Cresylic acid was added as a frother in the flotation of lead. Lime was added after removal of the lead and the depressed zinc reactivated by the addition of copper sulphate and floated by potassium xanthate and cresylic acid.

Results.—

Product	Weight, per cent	Assay		Per cent of values	
		Pb, per cent	Zn, per cent	Pb	Zn
Head	....	3·57	17·82	....	....
Lead concentrate	6·72	5·57	12·39	10·51	4·81
Zinc concentrate	16·66	2·48	54·06	11·61	52·01
Zinc middling	11·74	5·37	22·93	17·71	15·55
Tailing	64·88	3·30	7·37	60·17	27·63

Conditions under which this test was performed were apparently not equal to those of former tests.

Test No. 10

This test was run to determine what results could be obtained by removing soluble salts in the ore prior to flotation. The ore was ground in water, filtered, repulped in water, soda ash added to make alkaline, cyanide added and the pulp agitated for 15 minutes, followed by additions of copper sulphate, potassium xanthate, and pine oil.

Reagents.—

To flotation machine—

Soda ash	16·0 lb./ton	Cyanide	1·0 lb./ton
Xanthate	0·4 "	Pine oil	0·03 "

Results.—

Product	Weight, per cent	Assay			Per cent of values		
		Pb, per cent	Zn, per cent	Fe, per cent	Pb	Zn	Fe
Head	....	3·57	17·82	31·47	....	....	....
Concentrate	26·01	6·43	44·25	13·03	45·85	64·17	10·76
Middling	15·21	3·79	17·50	31·42	15·79	14·85	15·17
Tailing	58·78	2·38	6·40	39·70	38·36	20·98	74·07

These results show that a preliminary water wash is of no particular benefit to recovery or grade of concentrate secured.

Test No. 11

In this test, conducted along lines similar to the preceding one, the ore was given a quick grind in water, filtered, and returned to the grinding mill and ground to the required fineness in the presence of soda ash and cyanide. Flotation was conducted under conditions similar to those of Test No. 10.

Reagents.—

To ball mill—		To flotation machine—	
Soda ash	9·0 lb./ton	Copper sulphate	2·0 lb./ton
Cyanide	1·0 "	Xanthate	0·4 "
		Pine oil	0·03 "

Results obtained in this test point to the same conclusions as arrived at in Test No. 10, that no advantage is derived from a preliminary water wash.

Test No. 12

The effect of floating in solutions heated to 100°F. was studied in this test. Grinding was carried out in an alkaline circuit using cyanide to depress the zinc and iron and then reactivating the zinc by copper sulphate and floating with potassium xanthate and pine oil.

Reagents.—

To ball mill—		To flotation machine—	
Soda ash	18·0 lb./ton	Copper sulphate	2·0 lb./ton
Cyanide	1·0 "	Xanthate	0·4 "
		Pine oil	0·03 "

Results.—

Product	Weight, per cent	Assay		Per cent of values	
		Pb, per cent	Zn, per cent	Pb	Zn
Head	....	3·57	17·82	....	....
Concentrate	28·47	6·10	35·02	53·52	65·50
Middling	15·95	2·56	17·00	11·44	1·78
Tailing	55·58	2·25	8·96	35·04	32·72

The results of this test show that heating the solutions is of no material benefit.

Test No. 13

This test was run to determine whether or not the lead and zinc reporting in the tailing from preceding tests can be recovered by sulphidizing the minerals. The ore was ground with soda ash, cyanide, and a very small amount of sodium sulphide. This sulphidizing reagent was added a short time before completion of grinding. Copper sulphate, xanthate, and pine oil were added and a first concentrate removed. The tailing was then filtered and agitated in a 1 : 1 pulp with sodium sulphide, 3 pounds per ton, and a second concentrate recovered, using xanthate and pine oil.

Reagents.—

To ball mill—		To flotation machine—	
Soda ash	18·0 lb./ton	Copper sulphate	2·0 lb./ton
Sodium cyanide	0·5 "	Xanthate	0·7 "
Kerosene	0·02 "	Sodium sulphide	3·0 "
Sodium sulphide	0·1 "	Pine oil	0·06 "

Results.—

Product	Weight, per cent	Assay			Per cent of values		
		Pb, per cent	Zn, per cent	Fe, per cent	Pb	Zn	Fe
Head	....	3·57	17·82	31·47	....	....	....
1st concentrate	30·76	5·37	38·71	16·88	46·85	66·69	16·77
2nd concentrate	15·61	3·26	18·95	31·57	14·43	16·46	15·93
Middling from 2nd concentrate	15·35	3·01	8·19	38·71	13·10	7·04	18·83
Tailing	38·28	2·36	4·58	40·48	25·62	9·81	49·13

The addition of sodium sulphide to the tailing from the flotation operation has the effect of raising the iron in the concentrate. The tailing still contains the same percentage of lead and zinc as in former tests, showing that sulphidizing is of no benefit.

Test No. 14

In this test the ore was ground for the entire time with sodium sulphide in a pulp alkaline with soda ash. After grinding, the pulp was transferred to the flotation machine and agitated for a sufficient time to oxidize the sodium sulphide remaining in the solution. Cyanide was then added to depress zinc and iron, followed by an addition of copper sulphate to reactivate the zinc. A concentrate was then produced by xanthate and pine oil. As the quantity of concentrate produced seemed low, coal-tar creosote was added and a second concentrate taken.

Reagents.—

To ball mill—		To flotation machine—	
Soda ash	18·0 lb./ton	Sodium cyanide	1·0 lb./ton
Sodium sulphide	3·0 "	Copper sulphate	2·0 "
		Xanthate	0·5 "
		Pine oil	0·06 "
		Coal-tar creosote	0·1 "

Results.—

Product	Weight, per cent	Assay			Per cent of values		
		Pb, per cent	Zn, per cent	Fe, per cent	Pb	Zn	Fe
Head	....	3·57	17·82	31·47	....	....	....
1st concentrate	24·10	6·45	32·31	20·74	44·11	43·94	15·69
1st middling	3·13	3·16	11·58	35·95	2·81	2·06	3·53
2nd concentrate	4·34	3·26	39·82	17·61	4·00	9·76	2·36
2nd middling	18·27	2·93	23·57	29·02	15·19	24·25	16·68
Tailing	50·16	2·38	7·06	39·20	33·89	19·99	61·74

It is noticed from these results that the total recovery of lead in the combined concentrates and middlings is slightly higher than in the preceding test. Both tests (Nos. 13 and 14), however, show that no benefit is to be derived from the use of sodium sulphide to recover the minerals remaining in the tailing.

CONCLUSIONS

In all these tests, graphite is not entirely eliminated in the concentrate, some still remaining in the tailing. The presence of this material may account for the lead and zinc remaining in the tailing, or a certain amount of these sulphides lying along the joint-planes in the massive minerals may be sufficiently altered to render it difficult to float.

In Test No. 5 the percentage recovery of the metals in the combined concentrates is 71·7 per cent lead, 92·2 per cent zinc, and the ratio of concentration 1·9 : 1. In Test No. 6 the results are practically identical. In this test no graphitic concentrate was removed, the one concentrate secured containing 74·8 per cent of the lead and 91·2 per cent of the zinc with a ratio of concentration 1·9 : 1. This product assays 29·6 per cent zinc and 48 per cent lead. It is possible to obtain varying grades of concentrate, but as no analysis was specified most of the tests were run with the object of securing a high recovery.

It is interesting to note the conditions that produced the best results, as shown in Tests Nos. 5, 6, 7, and 8. The greater part of the acidity in the ore was neutralized with soda ash, followed by a short time contact with lime prior to flotation. Comparing results secured in Test No. 5, where no lime was added, with those of Test No. 6, it was found that lime increases the recovery of lead by 20 per cent and that of zinc by 6 per cent. The ratio of concentration, however, is lower with more iron reporting in the concentrate. The amount of lime added and also the time of contact has an important bearing on the results obtained as shown in Tests Nos. 7, 8, and 8A, an increase of this reagent over that used in Test No. 6 seriously affects the recovery.

As mentioned above, no specifications of grade of concentrate were furnished. The investigation shows that a fair recovery of the minerals is possible.

Report No. 300

EXPERIMENTAL TESTS ON GOLD ORE FROM THE SECOND RELIEF MINE, ERIE, B.C.

J. S. Godard

Shipment.—A shipment of 50 pounds of ore was received March 20, 1928, from the Second Relief mine, Erie, B.C.

Characteristics and Analysis of the Ore.—The shipment consisted of a gold ore carrying about an ounce of gold, per ton. Seventy-five per cent of the gold is freed on grinding to 90 per cent −200 mesh. The balance of the gold is associated with small amounts of the sulphides of iron, copper, and arsenic. The gangue is chiefly siliceous but some sericite is present. Analysis of head sample:

Gold..................	1·03 oz./ton	Lead..................	trace
Copper..................	0·05 per cent	Arsenic..................	"
Iron..................	7·89 "	Sulphur..................	8·05 per cent
		Insoluble..............	81·55 "

Purpose of Tests.—The purpose of the tests was to determine the association of the gold and a metallurgical treatment for the ore.

EXPERIMENTAL TESTS

Amalgamation Tests

Screen Analysis of Amalgamation Tailing.—

Test No.	Mesh	Weight, per cent	Assay, oz./ton	Per cent of values	Remarks
1............	+65	1·1	0·41	1·3	
	+100	10·5	0·53	16·3	
	+150	12·7	0·50	18·6	
	+200	11·7	0·44	15·1	
	−200	64·0	0·26	48·7	Per cent gold amalgamated....... 65·8
	Average..	....	0·34	....	
2............	+100	0·7	0·43	1·1	
	+150	5·1	0·51	10·6	
	+200	12·7	0·43	20·0	
	−200	81·5	0·23	68·3	Per cent gold amalgamated....... 72·6
	Average..	....	0·27	....	
3............	+200	7·4	0·44	13·6	
	−200	92·6	0·22	86·4	Per cent gold amalgamated....... 76·4
	Average..	....	0·24	....	

Concentration Tests

Flotation, Tabling the Flotation Tailing.—

Test No.	Product	Weight, per cent	Assay, oz./ton	Per cent of values
4	Flotation concentrate	20·3	3·78	82·5
	Table concentrate	2·2	2·14	5·1
	" middling	8·4	0·38	3·4
	" tailing	40·7	0·143	6·2
	" slime	28·4	0·09	2·8
	Tailing +200	19·0	0·16	21·0
	" −200	81·0	0·14	79·0
	" average	....	0·143	....
5	Flotation concentrate	9·7	7·03	74·8
	Table concentrate	6·8	1·54	11·5
	" middling	17·8	0·31	6·0
	" tailing	39·2	0·138	5·9
	" slime	26·5	0·06	1·8
	Tailing +100	3·9	0·21	6·0
	" +150	9·2	0·20	13·4
	" +200	16·8	0·16	19·5
	" −200	70·0	0·12	61·1
	" average	....	0·138	....
6	Flotation concentrate	13·6	5·54	83·3
	Table concentrate	6·2	1·14	7·8
	" middling	7·2	0·25	2·0
	" tailing	38·1	0·11	4·6
	" slime	34·9	0·06	2·3
	Tailing +200	8·1	0·13	9·4
	" −200	91·9	0·11	90·6
	" average	....	0·11	....

Summary.—

Test No.	Recoveries, per cent		
	Flotation concentrate	Table concentrate	Total
4	82·5	5·1	87·6
5	74·8	11·5	86·3
6	83·3	7·8	91·1

Amalgamation, Flotation, and Tabling.—

Test No.	Product	Weight, per cent	Assay, oz./ton	Per cent of values	Recoveries, per cent	
7	Flotation concentrate	14·1	1·09	52·5		
	Table concentrate	2·6	1·64	14·5		
	" middling	1·9	0·57	3·7	Amalgamation	70·7
	" tailing	46·3	0·14	22·1	Flotation	15·4
	" slime	35·1	0·06	7·2	Tabling	4·3
	Tailing +200	15·7	0·18	20·5	Total	90·4
	" −200	84·3	0·13	79·5		
	" average	....	0·14	....		

Cyanide Tests

Screen Test on Cyanide Tailings.—

Test No.	Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
8	+100 +150	2·1	4·00	19·1
	+200	8·4	0·73	13·9
	−200	89·5	0·33	67·0
	Average	....	0·44	....
9	+150	4·5	2·55	40·0
	−150	95·5	0·18	60·0
	Average	....	0·29	....

Summary.—

Test No.	Assay, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Head	Tailing		KCN	CaO
8	1·00	0·44	56·0	2·55	7·2
9	1·00	0·29	71·0	2·97	8·4

Dilution during agitation	1 : 2·5
KCN	0·05 per cent
Time of agitation	48 hours

Tests Nos. 10 and 11.—Amalgamation and cyanidation of tailing

Amalgamation Tailing Screened on 200 Mesh.—

Test No.	Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
10	+200	8·0	0·47	13·7
	−200	92·0	0·26	86·3
	Average	....	0·277	....
11	+200	4·7	0·40	8·7
	−200	95·3	0·21	91·3
	Average	....	0·219	....

Cyanide Tailing Screened.—

Test No.	Mesh	Weight, per cent	Assay, oz./ton	Per cent of values
10	+150	0·7	0·17	2·4
	−150	99·3	0·4	97·6
	Average	....	0·041	....
11	+200	10·3	0·10	27·0
	−200	89·7	0·03	73·0
	Average	....	0·037	....

Summary.—

Test No.	Assay, oz./ton			Reagents, lb./ton		Recoveries, per cent		
	Head	Amalgamation tailing	Cyanide tailing	KCN	CaO	Amalgamation	Cyanide	Total
10	1·00	0·277	0·041	2·2	7·8	72·3	23·6	95·9
11	1·00	0·219	0·037	2·6	7·4	78·1	17·2	95·3

Test No. 12.—In this test an attempt was made to separate the various sulphides and determine the association of the gold. Selective flotation, magnetic separation, and gravity concentration were the methods employed. In order that the gold freed on grinding would not interfere, the ore was first amalgamated under conditions that would remove about 75 per cent of the total gold. No further grinding was practised. No free gold was visible in any of the products when examined under a medium-powered microscope.

Results.—

Product	Cu, per cent	Fe, per cent	As, per cent	Au, oz./ton
Chalcopyrite (flotation)	6·30		0·02	3·64
Pyrite (flotation and magnetic separation)			0·01	1·24
Pyrrhotite (flotation and magnetic separation)		59·40	0·01	0·41
Middling from chalcopyrite flotation (high in pyrite)			0·01	2·29
Pyrrhotite from flotation (containing some pyrite)			0·01	0·67
Table concentrate from flotation tailing (high in pyrrhotite but containing some pyrite and a very small amount of arsenopyrite)			0·02	0·62

From the results of the above test, the gold after amalgamation appears to be associated with the following sulphides: chalcopyrite, pyrite, pyrrhotite. No information was obtained on the association of the gold with the arsenopyrite as the head sample contained but a trace of this mineral and no product assayed higher than 0·02 per cent arsenic and such a small quantity made it impossible to separate it from the other sulphides.

In the sample submitted the sulphides were present in the following order of amounts: pyrite closely followed by pyrrhotite, chalcopyrite, and arsenopyrite.

GENERAL CONCLUSIONS

Amalgamation.—The percentage of gold recovered by amalgamation increased on finer grinding until 76 per cent of the gold was amalgamated at 92 per cent −200 mesh.

Concentration.—A combination of flotation and tabling recovered from 86 to 91 per cent of the gold in the form of concentrate. The recoveries increased with finer grinding. Flotation alone was insufficient to obtain a suitable recovery, largely because of the oxidized condition of the ore, especially the pyrrhotite, which reported later in the table concentrate. No increase in recovery was made when amalgamation preceded concentration, the gold which is amalgamated otherwise reports in the concentrates. Whether amalgamation is practised previous to concentration will depend upon economic conditions.

Cyanidation.—Poor recoveries were obtained by cyanidation in Tests Nos. 8 and 9, due to the free gold which is not ground fine enough in these small batch tests. Where the free gold is coarse, batch tests are unreliable as to gold extractions but give a good indication of reagent consumption. To determine recoveries on such ores it has been found necessary to either amalgamate before cyanidation or to run continuous grinding tests in order to get the free gold in a finely divided state similar to that obtained in practical operations, when it is readily dissolved by the cyanide solution. There was not sufficient ore to run continuous tests, so that in Tests Nos. 10 and 11 the ore was amalgamated before cyanidation. The recoveries in these two tests were over 95 per cent of the gold values.

The cyanide consumption of about 2·5 pounds per ton of ore is fairly high, due to the presence of a little copper and the somewhat oxidized condition of the ore. An increase in the copper content would greatly increase the cyanide consumption, and might cause trouble in cyaniding the ore unless the copper was removed by flotation previous to cyanidation.

Report No. 301

EXPERIMENTAL TESTS ON A COPPER-GOLD ORE FROM OXFORD LAKE, NORTHERN MANITOBA

J. S. Godard

Shipments.—Two small samples of ore, weighing 9,170 grammes, were received from Ventures, Limited, 503 Excelsior Life Building, Toronto, on November 2, 1928. Although the samples were not marked, they were kept separately; the sample consisting of one piece was designated as No. 1, its weight being 3,160 grammes, and the other sample consisting of several pieces was designated as No. 2, its weight being 6,010 grammes.

Characteristics and Analysis.—The ore consisted of chalcopyrite, pyrite, some pyrrhotite, and a small amount of arsenopyrite in a quartz gangue. Appreciable amounts of gold and silver were associated with the sulphides. The analyses were:

	No. 1	No. 2
Gold	0·27 oz./ton	0·23 oz./ton
Silver	3·78 "	2·10 "
Copper	5·50 per cent	5·06 per cent
Iron	14·07 "	17·34 "
Arsenic	0·04 "	0·03 "
Antimony	Nil	Nil
Insoluble	62·06 "	53·83 "

Purpose of Tests.—The purpose of the experimental tests was to determine the percentage of the sulphides present and the association of the gold and silver.

EXPERIMENTAL TESTS

SAMPLE NO. 1

Test No. 1—Amalgamation, Flotation, and Tabling

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Fe, per cent	Au	Ag	Cu	Fe
Copper concentrate	23·9	0·42	14·72	22·65	33·25	77·1	92·0	96·9	54·5
Copper middling...	7·5	0·10	1·52	0·98	29·37	5·8	3·0	1·3	15·0
Iron concentrate...	9·7	0·07	1·06	0·77	37·23	5·2	2·7	1·0	24·8
Table concentrate..	1·1	0·09	0·39	....	....	0·8	0·1	....	....
Table tailing.......	29·7	0·03	0·12	0·01	0·51	6·8	0·9	0·1	1·0
Slimes............	28·1	0·02	0·17	0·14	2·44	4·3	1·3	0·7	4·7

Distribution of gold in per cent:—

Amalgamated..	51·8
Copper concentrate..	37·1
Copper middling..	2·8
Iron concentrate..	2·5
Table concentrate..	0·4
Table tailing..	3·3
Table slimes..	2·1

No silver was amalgamated.

SAMPLE NO. 2

Test No. 1—Amalgamation, Flotation, and Tabling

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Fe, per cent	Au	Ag	Cu	Fe
Copper concentrate	14·0	0·46	8·58	28·40	29·27	58·8	62·7	82·9	23·8
Copper middling..	9·1	0·18	4·14	6·83	21·62	14·9	19·7	13·0	11·4
Table concentrate..	17·4	0·08	0·88	0·68	39·78	12·6	8·0	2·5	40·3
Table tailing.......	30·5	0·03	0·21	0·04	5·08	8·4	3·3	0·2	9·0
Slimes............	29·0	0·02	0·42	0·23	9·18	5·3	6·3	1·4	15·5

Distribution of gold in per cent:—

Amalgamated..	52·1
Copper concentrate..	28·2
Copper middling..	7·2
Table concentrate..	6·0
Table tailing..	4·0
Slimes..	2·5

COMBINED SAMPLE

A combined sample was made by mixing 1,337 grammes of No. 1 sample and 4,120 grammes of No. 2 sample. The calculated assay of the combined sample was:—

Gold	0·24 oz./ton
Silver	2·51 “
Copper	5·17 per cent
Iron	16·54 “
Insoluble	55·85 “

Test No. 1—Selective Flotation of the Chalcopyrite and Iron Sulphides

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Fe, per cent	Au	Ag	Cu	Fe
Copper concentrate	19·3	0·56	4·48	22·90	24·17	43·2	32·7	87·5	28·1
Iron concentrate	30·5	0·40	5·47	2·05	38·35	48·8	63·1	12·4	70·6
Tailing	50·2	0·04	0·22	0·01	0·44	8·0	4·2	0·1	1·3

Test No. 2—Bulk Flotation of the Chalcopyrite and Iron Sulphides

Results.—

Product	Weight, per cent	Assay				Per cent of values			
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Fe, per cent	Au	Ag	Cu	Fe
Concentrate	46·4	0·46	5·22	10·88	34·07	88·8	97·0	99·7	96·8
Tailing	53·6	0·05	0·14	0·03	0·99	11·2	3·0	0·3	3·2

CONCLUSIONS

The percentage of sulphides is about 45 per cent of the total.

The gold and silver are mainly associated with the chalcopyrite, and to a lesser extent with the iron sulphides. Over 50 per cent of the gold is freed on grinding to about 85 per cent −200 mesh.

When selective flotation of the chalcopyrite is carried out without previously amalgamating, the iron concentrate is high in gold. When amalgamation precedes selective flotation this is not the case, which indicates that the gold is free but does not float with the copper.

The silver is not amenable to amalgamation.

Report No. 302

THE CONCENTRATION OF THE COPPER-ZINC-PYRITE ORE OF THE ALDERMAC MINE, ALDERMAC, QUEBEC

C. S. Parsons, A. K. Anderson, and J. S. Godard

Shipment.—Twenty-three bags of ore, gross weight 2,350 pounds, were received March 26, 1928, from Aldermac Mines, Ltd., Aldermac, Quebec.

Characteristics and Analysis of the Ore.—This material consisted of very heavy iron pyrites associated with small amounts of copper and zinc sulphides. The ore was crushed to pass an 8-mesh screen and a representative head sample cut out. This yielded the following analysis:

Copper	1·78	per cent
Zinc	1·71	"
Iron	39·88	"
Sulphur	38·85	"
Gold	0·025	oz./ton
Silver	0·80	"

Purpose of Tests.—The object of this investigation was to determine the adaptability of the ore to a selective flotation of the copper, zinc, and iron sulphides, producing a copper concentrate, a zinc concentrate, and an iron sulphide concentrate, carrying medium percentages of the other metals.

EXPERIMENTAL TESTS

The −8-mesh material was fed to a 12-inch by 24-inch rod mill in closed circuit with a 12-inch by 5-foot Akins classifier. Flotation was carried out in mechanically agitated flotation machines where copper, zinc, and iron concentrates were produced. The results of the tests follow:

Results.—

Test No.	Product	Weight, per cent	Assay						Per cent of values				
			Cu, per cent	Zn, per cent	Fe, per cent	S, per cent	Au, oz/ton	Ag, oz/ton	Cu	Zn	Fe	Au	Ag
1	Copper concentrate................	7·41	22·33	1·98	....	....	0·10	4·16	92·70	8·77	13·24	44·18	39·74
	Zinc "	5·06	0·57	19·48	....	....	0·02	0·70	1·68	57·31		5·88	5·13
	Iron "	61·89	0·25	0·86	44·94	48·77	0·01	0·47	4·49	29·24	69·73	....	....
	Zinc tailing......................	....	0·11	0·66	39·53	....	0·01	0·49	....	....	....	52·94	55·13
	Iron "	25·64	0·18	0·30	26·47	....	0·01	0·55	1·13	4·68	17·03	....	....
2	*Sample No. 1*												
	Copper concentrates..............	6·17	24·16	1·33	....	....	0·08	4·08	84·66	4·68	....	28·51	35·71
	Zinc "	4·71	1·00	13·36	....	....	0·02	0·62	2·84	36·84	12·62	0·71	4·28
	Iron "	53·24	0·23	1·28	45·14	49·12	0·01	0·39	7·39	42·69	60·25	....	....
	Zinc tailing......................	....	0·25	1·12	39·11	....	0·01	0·47	....	....	....	64·28	60·01
	Iron "	35·88	0·24	0·71	30·16	....	0·01	0·57	5·11	15·79	27·13	....	....
	Sample No. 2												
	Copper concentrate................	4·68	25·79	0·97	....	....	0·08	3·48	67·98	2·92	....	26·67	23·53
	Zinc "	7·47	3·07	18·05	....	....	0·03	0·94	12·92	78·95	....	13·33	10·29
	Zinc tailing......................	87·55	0·39	0·36	....	....	0·01	0·51	19·10	18·13	....	60·00	66·18

Sample No. 1 started 3.00 p.m.

Reagents (lb./ton) Test No. 1.—

Time	Na_2CO_3	NaCN	Cresylic acid	Pine oil	Thio-carbanilide	Zinc cells			Pyrite cells			Feed, lb./hr.	Classifier overflow density
						$CuSO_4$	Xanthate	Pine oil	$CuSO_4$	Xanthate	Na_2CO_3		
10.25	5·3	0·43	0·28	....	0·09	1·04	0·05	0·40	0·49	0·59	2·92	....	1 : 4·8
11.00	5·3	0·43	0·42	....	0·09	1·07	0·05	0·40	0·49	0·59	2·92	97·5	1 : 4·8
11.30	5·3	0·43	....	0·08	0·09	1·07	0·05	0·40	0·49	0·59	2·92	....	1 : 6·6
11.40	5·3	0·52	....	0·08	0·09	1·07	0·05	0·40	0·49	0·59	2·92	....	
12.00	5·3	·52	....	0·08	0·09	1·1	0·05	0·40	0·49	0·59	2·92	....	1 : 3·0
12.30	5·3	0·52	....	0·08	0·09	1·1	0·05	0·40	0·49	0·59	2·92	....	1 : 3·1
1.00	5·3	0·52	....	0·08	0·09	1·1	0·05	0·40	0·49	0·59	2·92	....	1 : 7·3
1.30	5·3	1·04	....	0·08	0·09	1·1	0·05	0·40	0·49	0·59	2·92	....	1 : 6·1
2.00	4·2	0·81	....	0·08	0·07	0·86	0·05	0·40	0·38	0·47	2·26	126	1 : 4·2
2.30	1·7	0·81	....	0·08	0·07	0·86	0·05	0·40	0·38	0·47	2·26	....	1 : 3·7
												....	
3.00	2·1	1·00	....	0·08	0·09	1·08	0·05	0·40	0·38	0·59	2·26	100	1 : 5·6

Reagents (lb./ton) Test No. 2.—

Time	Ball mill				Copper cells		Zinc cells			Pyrite cells			Feed, lb./hr.	Classifier overflow density
	Na_2CO_3	NaCN	Thio-carbanilide	$ZnSO_4$	Pine oil	TT	$CuSO_4$	Xanthate	Pine oil	$CuSO_4$	Xanthate	Na_2CO_3		
9.30	1·87	0·89	0·076	0·87	0·07	....	0·94	0·05	0·21	0·42	0·50	2·47	....	....
10.00	1·87	0·89	0·076	0·87	0·07	....	0·94	0·05	0·21	0·42	0·50	2·47	115	1 : 8·0
10·30	1·87	0·89	0·076	0·87	0·07	....	0·94	0·05	0·21	0·42	0·50.	2·47	....	1 : 5·2
10.40	2·87	0·89	0·076	0·87	0·07	....	0·94	0·05	0·21	0·42	0·50	2·47	....	....
11.00	2·87	0·89	0·076	0·87	0·07 off	....	0·94	0·05	0·21	0·42	0·50	2·47	....	....
2.00	2·87	0·44	0·076	0·87	....	0·11	0·94	0·05	0·21	0·42	0·50	2·47	115	....
2.30	2·87	0·44	0·076	0·87	....	0·11	0·94	0·05	0·21	0·42	0·50	2·47	....	1 : 7·3
3.00	2·87	0·22	off	0·87	....	0·11	0·94	0·05	0·21	0·42	0·50	2·47	....	1 : 3·5
3.30	2·87	0·22	....	0·87	....	0·11	0·94	0·05	0·21	0·42	0·50	2·47	....	1 : 4·8
4.00	2·87	0·10	....	0·87	....	off	0·94	0·05	0·21	0·42	0·50	2·47	....	1 : 1·6
4.30	2·87	0·10	....	0·87	....	lime	0·94	0·05	0·21	0·42	0·50	2·47	....	....
4.55	2·87	off	....	....	....	5·0	0·94	0·05	0·21	0·42	0·50	2·47	....	....

Sample No. 2 started 4.30 p.m.

CONCLUSIONS

These tests show that satisfactory results can be secured on the copper content of the ore. Due to what is suspected to be oxidation of the heavy sulphides during grinding, results secured on the zinc and iron are not satisfactory.

It will be necessary to do considerably more test work on this ore to determine the correct milling conditions to overcome difficulties encountered.

Report No. 303

CONCENTRATION OF THE SILVER-LEAD-ZINC-BARITE ORE OF THE HOMESTAKE MINE, ADAMS LAKE, NEAR KAMLOOPS, B.C.

J. S. Godard

Shipment.—A shipment of 150 pounds of ore, from the Homestake mine, Adams Lake, B.C., was received November 16, 1928, submitted by Wentworth F. Woods, Kamloops, B.C.

Characteristics of the Ore.—The ore is a complex mixture of galena, tetrahedrite, zinc blende, and pyrite, in a gangue composed of barite and silicates with the barite predominating. It contains high silver values and a small amount of gold apparently associated with the galena and tetrahedrite.

Analyses of the Ore.—Analyses of the ore showed it to contain:

Gold	0·09 oz./ton	Copper	1·18 per cent
Silver	112·81 "	Zinc	14·78 "
Lead	6·23 per cent	Barium sulphate	34·00 "

PURPOSE OF EXPERIMENTAL TESTS

The purpose of the experimental tests was to determine by means of selective flotation the degrees of separation that could be obtained between the galena and tetrahedrite, and the zinc blende, and the barite, and the silica, also the recoveries that could be obtained in the lead, zinc, and barite concentrates.

Results of Experimental Tests

No.	Product	Weight, per cent	Assay						Per cent of values					
			Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	BaSO₄, per cent	Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	BaSO₄, per cent
1	Lead concentrate........	17·7	0·30	543·10	6·10	25·19	24·25	9·25	62·3	85·3	89·2	73·8	29·3	4·8
	Zinc “	28·1	0·10	54·62	0·41	5·58	36·50	12·50	33·0	13·7	9·5	21·9	70·0	10·2
	Barite “	34·3	trace	1·62	0·02	0·02	0·18	81·64	....	0·5	0·6	0·1	0·4	81·2
	Tailing................	19·9	0·02	2·82	0·04	0·05	0·20	6·65	4·7	0·5	0·7	0·2	0·3	3·8
2	Lead concentrate.......	17·6	0·35	596·40	6·25	31·00	16·70	10·00	74·6	92·7	93·2	93·7	20·0	5·2
	Zinc “	32·3	0·06	22·96	0·21	1·10	36·19	15·32	23·5	6·5	5·8	6·1	79·3	14·6
	Barite “	34·0	trace	1·44	0·02	0·02	0·18	77·04	....	0·4	0·6	0·1	0·4	77·1
	Tailing................	16·1	0·01	2·56	0·03	0·05	0·26	6·51	1·9	0·4	0·4	0·1	0·3	3·1
3	Lead concentrate.......	18·6	....	....	5·15	32·87	19·3	....	....	....	....	....	....	....
	Zinc “	....	....	....	0·20	0·92	42·5	....	....	....	....	....	....	....
	Barite “	....	....	....	....	....	....	86·08	....	....	....	....	....	....
	Tailing................	....	....	....	....	....	....	....	....	....	....	....	....	....
4	Lead concentrate.......	20·7	....	....	4·91	30·86	21·1	....	....	....	....	....	....	....
	Zinc “	20·7	....	....	0·12	0·25	49·92	....	....	....	....	....	....	....
	Barite “	33·5	....	....	....	....	....	96·52	....	....	....	....	....	....
	Barite middling.........	4·4	....	....	....	....	....	....	....	....	....	....	....	....
	Tailing................	20·7	....	....	....	....	....	....	....	....	....	....	....	....
5	Lead concentrate.......	16·2	....	....	3·03	34·46	17·20	....	....	....	....	....	....	....
	Zinc “	23·3	....	....	0·54	2·39	50·12	....	....	....	....	....	....	....
	Barite “	20·7	....	....	....	....	....	96·52	....	....	....	....	....	....
	Barite middling.........	4·8	....	....	....	....	....	....	....	....	....	....	....	....
	Tailing................	35·0	....	....	....	....	....	....	....	....	....	....	....	....
6	Lead concentrate.......	12·8	0·50	763·20	7·49	40·12	15·80	2·74	66·4	88·0	92·1	80·3	13·6	1·0
	Lead middling..........	4·8	0·30	160·10	0·70	15·42	23·97	18·00	14·9	6·9	3·2	11·6	7·8	2·6
	Zinc concentrate........	20·2	0·03	20·08	0·21	2·20	57·20	3·18	6·3	3·7	4·1	7·3	77·7	1·9
	Zinc middling...........	3·7	0·04	11·64	0·01	0·12	1·66	34·25	1·6	0·4	0·1	0·1	0·4	3·8
	Barite concentrate......	36·2	trace	1·48	0·01	0·05	0·08	76·80	....	0·5	0·3	0·3	0·2	82·5
	Barite middling.........	8·4	0·01	2·50	0·01	0·10	0·13	22·30	0·8	0·2	0·1	0·1	0·1	5·6
	Tailing................	13·9	0·07	2·22	0·01	0·12	0·21	6·33	10·0	0·3	0·1	0·3	0·2	2·6

7	Lead concentrate.......	9·2	0·45	945·90	9·92	46·91	13·05	1·46	41·5	78·9	88·3	68·2	8·1	0·4
	Lead middling..........	5·7	0·30	232·90	1·77	19·27	20·44	32·19	17·1	12·0	9·8	17·4	7·8	5·4
	Zinc concentrate........	21·0	0·12	41·60	0·02	4·03	54·50	2·25	25·3	7·9	0·4	13·4	77·0	1·4
	Zinc middling..........	4·8	0·21	14·74	0·12	0·67	14·35	27·92	10·2	0·6	0·5	0·5	4·6	3·9
	Barite concentrate......	28·7	0·01	0·94	0·015	0·025	0·75	85·28	2·9	0·2	0·4	0·1	1·5	71·8
	Barite middling.........	10·2	0·01	1·08	0·01	0·025	0·86	35·44	1·0	0·1	0·1	0·1	0·6	10·6
	Tailing.................	20·4	0·01	1·78	0·025	0·10	0·34	10·93	2·0	0·3	0·5	0·3	0·4	6·5
8	Lead concentrate.......	9·9	0·50	882·20	8·04	38·45	14·40	2·44	56·3	79·0	79·9	60·7	9·7	0·7
	Lead middling..........	5·4	0·30	215·80	1·86	17·68	19·55	27·13	18·4	10·6	10·0	15·2	7·2	4·8
	Zinc concentrate	21·4	0·07	47·24	0·36	6·77	53·77	2·22	17·1	9·2	7·7	23·1	78·2	1·4
	Zinc middling...........	7·9	0·05	8·47	0·05	0·55	7·70	34·76	4·5	0·6	0·4	0·7	4·1	8·0
	Barite concentrate......	23·7	trace	0·32	0·01	0·025	0·05	96·14	....	0·1	0·2	0·1	0·1	66·8
	Barite middling.........	5·0	0·01	1·57	0·0295	0·025	0·44	43·68	0·6	0·1	1·5	....	0·1	6·4
	Tailing.................	26·7	0·01	1·75	0·01	0·05	0·36	15·89	3·1	0·4	0·3	0·2	0·6	12·4

Reagents Used

Test No.	—	Reagent	Lb./ton	Remark
1	Lead	Soda ash	3·0	Added to ball mill.
		Thiocarbanilide	0·06	" "
		Sodium cyanide	0·30	" "
		Pine oil No. 5	0·04	" cell.
	Zinc	Soda ash	2·0	Added to cell.
		Copper sulphate	1·5	"
		P. T. & T. Co. No. 1580	0·05	"
		Potassium xanthate	0·50	"
		Pine oil No. 5	0·04	"
	Barite	Water glass	1·1	Added to cell.
		Oleic acid	1·5	"
2	Lead	Soda ash	3·0	Added to ball mill.
		Thiocarbanilide	0·08	" "
		Sodium cyanide	0·30	" "
		Zinc sulphate	1·0	" "
		Cresylic acid	0·07	Added to cell.
	Zinc	Copper sulphate	1·5	Added to cell.
		Potassium xanthate	0·50	"
		Pine oil No. 5	0·10	"
		P. T. & T. Co. No. 1580	0·05	"
	Barite	Water glass	1·1	Added to cell.
		Oleic acid	1·1	"
		Pine oil No. 5	0·07	"
3	Lead	Soda ash	2·0	Added to ball mill.
		Thiocarbanilide	0·08	" "
		Sodium cyanide	0·40	" "
		Zinc sulphate	1·4	" "
		Cresylic acid	0·07	Added to cell.
	Zinc	Sulphuric acid	2·9	Added to cell.
		P. T. & T. Co. No. 1580	0·05	"
		Potassium xanthate	0·50	"
		Pine oil No. 5	0·07	"
	Barite	Water glass	1·1	Added to cell.
		Oleic acid	1·1	"
		Pine oil	0·10	"
4	Lead	Soda ash	3·5	Added to ball mill.
		Thiocarbanilide	0·04	" "
		Sodium cyanide	0·35	" "
		Zinc sulphate	1·2	" "
		Amyl xanthate	0·035	Added to cell.
		Cresylic acid	0·07	"
	Zinc	Sulphuric acid	2·4	Added to cell.
		Xylidine	0·05	"
		Copper sulphate	0·35	"
		Potassium xanthate	0·35	"
		Pine oil No. 5	0·07	"
	Barite	Water glass	0·45	Added to cell.
		Oleic acid	1·1	"
		Pine oil No. 5	0·04	"

Reagents Used—Continued

Test No.	—	Reagent	Lb./ton	Remark
5	Lead.....	Soda ash........................	4·0	Added to ball mill.
		Thiocarbanilide.................	0·06	" "
		Sodium cyanide................	0·50	" "
		Zinc sulphate..................	1·0	" "
		Cresylic acid..................	0·07	Added to cell.
	Zinc......	Soda ash........................	2·0	Added to cell.
		Copper sulphate................	2·0	"
		Potassium xanthate............	0·50	"
		Barrett's No. 634..............	0·25	"
		Pine oil No. 5..................	0·07	"
	Barite....	Water glass.....................	8·0	Added to cell.
		Oleic acid.......................	4·6	"
6	Lead.....	Soda ash........................	3·7	Added to ball mill.
		Thiocarbanilide.................	0·07	" "
		Sodium cyanide................	0·32	" "
		Zinc sulphate..................	0·90	" "
		Cresylic acid..................	0·04	Added to cell.
		Sodium cyanide................	0·025	Added to cell to clean lead rougher concentrate.
	Zinc......	Copper sulphate................	0·10	Added to cell.
		Barrett's No. 634..............	0·08	"
		Potassium xanthate............	0·36	"
		Pine oil........................	0·04	"
	Barite....	Water glass.....................	1·06	Added to cell.
		Oleic acid.......................	1·7	"
		Pine oil No. 5..................	0·02	"
		Water glass.....................	0·6	Added to cell to clean barite concentrate.
7	Lead.....	Soda ash........................	3·5	Added to ball mill.
		Thiocarbanilide.................	0·06	" "
		Sodium cyanide................	0·34	" "
		Zinc sulphate..................	1·10	" "
		Cresylic acid..................	0·07	Added to cell.
		Sodium cyanide................	0·05	Added to cell to clean lead rougher concentrate.
	Zinc......	Copper sulphate................	0·70	Added to cell.
		Potassium xanthate............	0·40	"
		P. T. & T. Co. No. 1580........	0·07	"
		Pine oil No. 5..................	0·02	"
		Sodium sulphide...............	1·0	"
	Barite....	Water glass.....................	1·25	Added to cell.
		Oleic acid.......................	1·12	"
		Water glass.....................	1·0	Added to cell to clean barite rougher concentrate.
8	Lead.....	Soda ash........................	2·0	Added to ball mill.
		Thiocarbanilide.................	0·06	" "
		Sodium cyanide................	0·34	" "
		Zinc sulphate..................	1·1	" "
		Cresylic acid..................	0·07	Added to cell.
		Sodium cyanide................	0·50	Added to cell to clean lead rougher concentrate.

Reagents Used—Concluded

Test No.	—	Reagent	Lb./ton	Remark
8	Zinc......	Sulphuric acid..................	2·4	Added to cell.
		Potassium xanthate.............	0·40	"
		Pine oil No. 5..................	0·10	"
	Barite....	Water glass.....................	1·3	Added to cell.
		Oleic acid......................	1·2	"
		Water glass.....................	0·5	Added to cell.
		Oleic acid......................	0·08	Added to cell to clean barite rougher concentrate.

CONCLUSIONS

In the separation and concentration of this ore two difficulties arise, namely, obtaining a clear cut separation of the galena and tetrahedrite from the zinc blende, and obtaining a barite concentrate sufficiently high in grade to market.

In the former it might prove more economical to allow a larger amount of zinc blende to appear in the galena-tetrahedrite concentrate than to obtain a sharper separation at the expense of a loss in lead and copper, and the dropping of silver and gold in the zinc concentrate.

In the latter a better grade of barite concentrate was obtained when the zinc blende was floated in an acid pulp and well roughed. The undesirable change from an alkali to acid pulp may possibly be avoided by the increase in the amount of water glass used.

From the tests on this type and grade of ore the following results may be expected in practical operations:—

1. A galena-tetrahedrite concentrate having assay values and recoveries as follows:

	Assay value	Recovery
Lead,	43 per cent	90 per cent
Copper,	9 "	94 "
Zinc,	14–15 "	
Silver,	900 oz./ton	94 "
Gold,	0·5 "	75 "

2. A zinc concentrate assaying: zinc 52 to 55 per cent, silver 20 ounces per ton, with a zinc recovery of 75 to 80 per cent.

3. A barite concentrate assaying: barite 94 per cent, with a recovery of 70 per cent of the barite in the ore.

These results will only be obtained in a properly designed and constructed mill under the control of a capable metallurgist.

Report No. 304

THE RECOVERY OF GOLD FROM THE ARSENICAL ORE OF THE CAMERON ISLAND MINE, SHOAL LAKE, LAKE OF THE WOODS, ONTARIO.

A. K. Anderson

Shipment.—A shipment of 125 pounds of ore was received at the laboratories on October 31, 1927, consigned by J. G. Cross, Port Arthur, Ont.

Characteristics of the Ore.—The ore consisted of a siliceous gangue containing arsenopyrite and iron pyrite. No free gold was visible.

Purpose of Experimental Tests.—The shipment was made to determine the most economical method to recover the gold and possibly the arsenic.

Sampling and Analysis.—The lot was crushed to pass 10 mesh, passed several times through a Jones riffle sampler between stage grinding, and a representative head sample secured. This gave the following analysis:

Gold	1·68 oz./ton	Lead	nil
Arsenic	5·86 per cent	Bismuth	nil
Nickel	0·06 "	Tellurium	nil
Cobalt	0·03 "	Platinoids	nil

EXPERIMENTAL TESTS

The test work performed included amalgamation and cyanidation of the raw and roasted ore. The effect of adding lime to the charge before and after roasting to convert the arsenic to calcium arsenate was investigated. As this compound is insoluble, non-fouling of cyanide solutions used for the extraction of gold should result. The ore also was given preliminary treatment with acid and alkaline solutions to note what results would be obtained by their use.

AMALGAMATION

Test No. 1

One thousand grammes of ore −10 mesh were ground in water to approximately 80 per cent −200 mesh and amalgamated with mercury.

Head assay, gold	1·68 oz./ton
Tailing assay, gold	1·18 "
Recovery	41·7 per cent

Amalgamation of the raw ore is not successful.

Test No. 2

One thousand grammes of ore −10 mesh were dead-roasted, ground to approximately 80 per cent −200 mesh and amalgamated with mercury.

Head assay, gold	1·68 oz./ton
Tailing assay, gold	1·28 "
Recovery	23·8 per cent

Amalgamation of the roasted ore is not so successful as treating the raw ore by the same process.

CYANIDATION

Test No. 3

Ore dry-ground to pass 200 mesh was agitated for 24 hours, 1 : 3 dilution with sodium cyanide solution, equivalent in strength to 3·0 pounds potassium cyanide per ton. Lime was added at the rate of 5·3 pounds per ton ore.

Head assay, gold	1·68	oz./ton
Tailing assay, gold	1·23	"
Recovery	26·8	per cent

The ore does not yield to cyanidation under the above conditions.

Test No. 4

The ore was ground to approximately 80 per cent −200 mesh in a 3·0 pound KCN per ton solution, 1 : 1 dilution. After grinding, the pulp was diluted to 1 : 3 and brought up in strength to 3·0 pound KCN per ton. Agitation was continued for 72 hours.

Head assay, gold	1·68	oz./ton
Tailing assay, gold	1·08	"
Recovery	35·7	per cent

Test No. 5

In this test, the effect of adding a large amount of lime while grinding in cyanide solution was studied. Lime at the rate of 16 pounds per ton of ore was added while grinding in 1 : 1 dilution with a 3·0 pound KCN solution. After grinding to approximately 80 per cent −200 mesh, the pulp was diluted to 1 : 3 with fresh cyanide solution and agitation continued for 72 hours.

Head assay, gold	1·68	oz./ton
Tailing assay, gold	1·10	"
Recovery	34·5	per cent

Cyanide consumption—2·24 lb. KCN per ton ore.

This test shows that no advantage is derived from the use of large amounts of lime. Fouling of solutions is more marked.

ROASTING

Tests Nos. 6 to 18

The ore was roasted until fumes of arsenic trioxide and sulphur dioxide were given off; 16·9 per cent of volatile matter passed off.

The roasted material was then ground in a 1 : 1 dilution with 3·0 pounds cyanide solution until 80 per cent passed 200 mesh, lime being added at the rate of 5·3 pounds per ton of ore. After grinding, the pulp was diluted to 1 : 3 with additional cyanide solution and a further addition of 5 pounds lime per ton made. After 43 hours' agitation, the pulp was filtered, washed, and amalgamated.

Cyanide heads, gold	2·02	oz./ton
Cyanide tailing, gold	0·52	"
Amalgamation tailing, gold	0·25	"

Or calculated to original unroasted ore:—

Cyanide tailing	0·43	oz./ton
Amalgamation tailing	0·20	"
Recovery by cyanidation	74·4	per cent
Recovery by amalgamation	13·7	"
Total recovery	88·1	"

This test, while showing a higher recovery than any of the preceding tests, still leaves a final residue of $5.00 per ton.

In this series of tests, the effect of lime added to the charge before and after roasting at different temperatures and for varying lengths of time was studied. After roasting, the ore was moistened with water and allowed to stand for three days before cyaniding. The mixture was then ground to pass approximately 80 per cent −200 mesh. Cyanide solution 3·0 pounds KCN per ton was used with a dilution of 1 : 3. In the last three tests, Nos. 16, 17, and 18, grinding was done in water, after which the pulp was filtered before applying cyanide solution.

Test No.	Size of material	Time of roast	Temp., °F.	Lime before roast, lb./ton	Lime after roast, lb./ton	Lime during cyaniding lb./ton	Reagent consumpt'n lb./ton KCN	Reagent consumpt'n lb./ton CaO	Tailing, Au, oz./ton	Extraction, per cent	Remarks
7	−14 mesh	30 min.	820	20	....	....	6·8	20·0	0·58	65·5	Ore ignited: white fume given off.
8	"	"	980	40	....	....	7·3	38·5	0·52	69·1	Ore ignited: white fume given off.
9	"	"	780	None	....	6·0	7·8	6·0	0·38	77·4	Ore ignited and burned with blue flame.
10	"	"	900	60	....	....	6·7	59·8	0·70	58·3	Ore ignited: white fume given off.
11	"	"	900	120	....	....	7·9	119·9	0·73	56·5	Ore ignited, small amount of fume given off.
12	"	"	580	None	....	12·0	8·9	11·9	1·10	34·5	Ore did not ignite, no fume given off.
13	"	60 min.	900	....	20	....	3·9	20·0	0·96	42·8	White fume coming off at end of roast.
14	"	"	900	....	40	....	3·6	35·7	0·88	47·6	White fume coming off at end of roast.
15	"	"	900	....	60	....	2·7	54·4	1·12	33·3	White fume coming off at end of roast.
16	"	"	900	20	....	8·0	3·3	26·8	0·54	67·7	Ground in water and filtered before cyaniding.
17	"	"	900	40	....	8·0	3·6	46·8	0·77	54·1	Ground in water and filtered before cyaniding.
18	"	"	900	60	....	8·0	3·6	66·0	·97	42·3	Ground in water and filtered before cyaniding.

Test No. 19

To determine if the gold is locked up in the gangue when ground to pass 100 mesh, a portion was treated with hot nitro-hydrochloric acid. After the sulphides were decomposed, the residue was filtered, washed, and assayed. This gave a value of 0·04 ounce gold remaining undissolved. It appears, therefore, that the gold is mostly associated with the sulphides.

Treating the ore with hot caustic soda solutions and hydrochloric acid does not decompose the sulphides. The residue contains the same amount of arsenic as the untreated material.

A further shipment of ore was requested which to date has not been received. No further work, therefore, has been done.

From the above tests it appears that the ore is extremely refractory. Possibly the gold is in solid solution with the arsenopyrite, and on roasting is coated with an impervious compound, rendering it immune to solution by cyanide.

The percentage recovery varies inversely with the arsenic remaining in the ore after roasting. This is shown in the following table.

Test No.	Gold, oz./ton	Arsenic, per cent	Test No.	Gold, oz./ton	Arsenic, per cent
7	0·58	0·51	13	0·96	3·94
8	0·52	0·70	14	0·88	2·64
9	0·38	0·30	15	1·12	4·82
10	0·70	1·48	16	0·54	1·25
11	0·73	2·00	17	0·77	1·25
12	1·10	5·88	18	0·97	3·46

Conditions under which roasting removes as much arsenic as possible and leaves the gold in a form readily soluble in cyanide solutions, apparently are yet to be established. Addition of lime to the ore before roasting was thought to have this effect, but, though apparently being of some benefit, was not found to solve the difficulty.

The problem is one on which a large amount of work will have to be done before this ore can successfully be treated for the recovery of the gold contained in it.

Report No. 305

THE CONCENTRATION OF THE COPPER SULPHIDE ORES OF THE HOLLINGER-KAMISKOTIA PROPERTY

C. S. Parsons and A. K. Anderson

Shipments.—Two bags of ore, gross weight 200 pounds, were received June 7, 1928, at the Ore Dressing Laboratories. This shipment was taken at the mine by Prof. G. J. MacKay of Queen's University, and forwarded to Ottawa for testing purposes.

Characteristics of the Ore.—The material consisted of a hard, dense iron pyrite carrying finely disseminated copper sulphides. A large portion of this latter mineral is so finely divided that a microscope is necessary to detect its presence.

Purpose of Tests.—The object of the investigation was to determine whether the copper in the ore can be concentrated by flotation and what recovery may be expected.

Sampling and Analysis.—The entire shipment was crushed to 14 mesh, passed through a Jones riffle sampler and a representative head sample secured. On analysis, this showed the following:

Copper	3·44 per cent
Zinc	0·25 "
Gold	trace
Silver	0·20 oz./ton
Nickel	nil

EXPERIMENTAL TESTS

The physical examination and chemical analysis of the ore indicated the problem to be one of single-stage flotation of the copper sulphides, so as to separate this mineral from the associated iron and zinc sulphides. It was evident that due to the finely divided condition of the chalcopyrite fine grinding would be necessary.

All tests were conducted on representative 2,000-gramme samples of the ore ground in a Denver ball mill with an equal weight of water. The flotation was done in a Denver Sub-A type, laboratory machine.

A number of tests were carried out, the results of which are not included in this report as they closely parallel those shown in the following. Grinding in all tests was such that 80 per cent of the material passed through 200 mesh. The detailed tests follow:—

Test No. 4

Two thousand grammes of ore, −14 mesh, ground 20 minutes in ball mill.

Reagents.—

To ball mill—	To cells—
Lime........5·0 lb./ton	Xanthate..... 0·20 lb./ton Pine oil...... to froth

Product	Weight, per cent	Copper, per cent	Per cent of values
Concentrate	16·89	16·98	83·12
Tailing	83·10	0·70	16·87

This test shows considerable copper remaining in the flotation tailing. The concentrate was not recleaned.

Test No. 5

Two thousand grammes of ore, −14 mesh, ground 20 minutes in ball mill.

Reagents.—

To ball mill—		To cells—	
Lime.....	2·0 lb./ton	Xanthate......	0·2 lb./ton.
Cyanide..	0·1 "	Pine oil........	to froth

Product	Weight, per cent	Copper, per cent	Per cent of values
Concentrate	18·59	16·48	84·32
Middling	11·74	2·24	7·23
Tailing	69·67	0·44	8·45

This test shows that by decreasing the amount of lime added, and using sodium cyanide the recovery of the copper is increased.

Test No. 7

Two thousand grammes of ore, −14 mesh, ground 20 minutes in ball mill.

Reagents.—

To ball mill—		To cells—	
Lime.....	2·5 lb./ton	Xanthate.......	0·2 lb./ton
		TT mixture.....	0·1 "
		Pine oil.........	to froth

Product	Weight, per cent	Copper, per cent	Per cent of values
Concentrate	14·11	18·84	78·15
Middling	7·89	3·66	8·50
Tailing	78·00	0·58	13·34

This test shows that an increase in the amount of lime used in the absence of cyanide lowers the recovery. A further increase in the lime, as shown in Test No. 4, has a markedly detrimental effect. Other tests were run at a later date on ore more finely ground than in those reported. In none of these could the results secured in Tests Nos. 4, 5, and 7, be duplicated owing to oxidation of the sulphides having commenced.

SUMMARY AND CONCLUSIONS

An examination of these three typical tests shows that the amount of lime and cyanide used must be closely controlled to secure the maximum recovery. In Test No. 5 where 2·0 pounds of lime and 0·10 pound cyanide per ton are used the recovery in a concentrate, 16·5 per cent copper, was 84·3 per cent, with 7·23 per cent of the total copper in a middling product. This probably would require separate treatment. A further reduction in the amount of cyanide used would doubtless yield better results.

Due to oxidation of the ore setting in between the series of tests, the results of tests run on very finely ground material yielded indifferent results. However, it is reasonable to assume that a −200-mesh feed would result in a higher recovery than that secured in the above tests.

No work was done on the middling product. This could be returned to the head of the flotation cells, or reground and floated in a separate circuit. The better method would have to be determined by further work.

From the tests carried out it would appear that better than an 85 per cent recovery can be obtained in a 20 per cent copper concentrate. It is believed that it would be safe to figure on this grade and recovery.

Report No. 306

BENEFICIATION AND LOW-TEMPERATURE REDUCTION OF HIGH SULPHUR IRON ORE FROM BRISTOL MINES, PONTIAC COUNTY, QUEBEC

R. J. Traill, W. R. McClelland, and J. D. Johnston

The ore was a shipment of 65 tons from Bristol Mines, Pontiac county, Quebec, submitted by J. M. Forbes, engineer for Blackburn Bros., Ottawa. The character of the ore is a magnetite with considerable hematite present, while distributed throughout and interbanded with the magnetite are found hornblende, micaceous and chloritic materials. Iron pyrite is disseminated in small patches throughout the ore.

Purpose of Tests.—The tests here reported are essentially preliminary to further work, and constitute only a preliminary survey on the behaviour of the ore to present-day methods of treatment.

The purpose of the test work is to determine whether the sulphur can be removed to a sufficiently low limit to produce a marketable iron ore, suitable for making iron directly by the blast furnace process, or for making sponge iron as a step in the ultimate production of iron or steel.

Procedures Employed.—The methods of treatment carried out in this preliminary study consisted of first, oxidizing roasting in a multiple-hearth, laboratory type Herreshoff furnace, and second, oxidizing roasting in a rotary kiln furnace, each to be followed by a reduction treatment to produce sponge iron, and third, sintering in pans to produce a sinter suited to blast furnace treatment.

EXPERIMENTAL WORK

Five tons of the ore were crushed in a jaw crusher and rolls to pass a $\frac{1}{8}$- by $\frac{7}{16}$-inch punched screen and a sample cut by an automatic sampler and further reduced for analysis. Analysis showed the following approximate composition:—

	Per cent
Iron (total)	53·35
Sulphur (total)	2·95
Phosphorus	Trace
Silica	8·08
Lime	5·10
Magnesia	0·32
Sulphur trioxide	0·22

The above constituted the head sample for the following tests.

Roasting in a Laboratory Type Herreshoff Furnace

Several short period runs were first made to determine the minimum temperature at which best results could be obtained, and when this was found a 48-hour continuous run was made. The short tests seemed to indicate that the best results were obtained with heat applied on hearths 4 and 5, and temperatures running 400° to 450°C. on hearth 2, 680° to 720°C. on hearth 4, and 850° to 900°C. on hearth 5. With temperatures lower than 550°C. on hearth 4, and 800°C. on hearth 5 the total sulphur in the roast was over one per cent, and similar results were obtained when the heat on hearth 2 was increased. In the table following the results of the 48-hour run are shown. Heat was applied on hearths 4 and 5 during the first 24 hours' running, and during the remaining 24 hours heat was applied on hearths 4, 5, and 6 (discharging hearth). The discharge was collected every 3 hours, riffle sampled and analysed for sulphur with an occasional sample analysed for sulphur trioxide. Analysis showed the remaining sulphur to be present in the oxidized state and a water-soluble test showed that it was present almost wholly as calcium sulphate. The total feed to the furnace amounted to 5,023 pounds with a discharge during operation of 4,260 pounds, 606 pounds remaining in the furnace.

The elapsed time between the feed and first discharge was about 5 hours.

Herreshoff Furnace Run

Average temperature			Total sulphur, per cent	Sulphur as SO_3, per cent
Hearth 2	Hearth 4	Hearth 6		
° C.	° C.	° C.		
—	690	880	0·90	—
445	720	855	0·78	0·72
475	710	885	0·85	0·77
465	720	905	0·87	—
490	720	890	0·84	0·80
485	700	885	0·80	—
460	715	885	0·75	0·64
455	680	870	0·85	—
625	720	880	0·96	0·94
585	760	880	1·11	—
610	750	765	1·15	1·15
630	800	870	1·10	—
610	700	870	0·91	—
600	730	880	1·01	0·92
620	750	860	0·99	—

A classification of roasted ore by sulphur content shows:

22 per cent under 0·80 per cent
33 per cent from 0·80 to 0·90 "
45 per cent from 0·90 to 1·15 "

Rotary Kiln Roasting Test

This furnace is described in Report No. 307, "Metallization of Iron Content in Ilmenite Ore", and the only alterations made for this test were in the discharge pipe, which was increased to 3 inches diameter, and the elimination of the combustion block. Temperatures within the kiln were read with a pyrometer and probably showed the maximum temperature at a point about 3 feet from the end of the kiln.

With the 3-inch discharge opening a bed of ore, 3 to 4 inches in depth, was maintained in the kiln.

The discharge was removed each hour; three such discharges were mixed, riffle sampled and analysed for sulphur, with sulphur trioxide determinations on occasional samples.

The ore charged was crushed to pass a $\frac{1}{8}$- by $\frac{7}{16}$-inch punched screen, and charged to the furnace from a hopper feed. No coal mixture was used, heat being applied by an oil burner. The temperature range was roughly 900° to 1,000°C. in the hottest part of the kiln and 250° to 300°C. at the small or feed end of the kiln.

No nodulization or agglomeration of fines was noticed in the roasted ore. The results obtained were slightly higher than those obtained on the Herreshoff furnace; the sulphur was present as calcium sulphate. The total feed amounted to 4,488 pounds, and the discharge during operation to 3,867 pounds. The elapsed time between the feed and first discharge was about one and one-quarter hours.

Rotary Kiln Run

Temperature ° C.	Total sulphur, per cent	Sulphur as SO_3, per cent
900	1·05	1·07
950	1·17	0·97
950	1·13	—
1000	0·83	—
960	0·80	0·78
940	0·79	—
970	0·81	0·81
950	0·96	—
900	1·06	—
920	0·80	0·82
930	0·88	—
1020	0·99	0·96
1025	1·07	—
1025	1·38	—
1010	1·16	—

A classification of roasted ore by sulphur content shows:—

36 per cent under 0·90 per cent
64 per cent from 0·90 to 1·38 "

Sponge Iron Tests

Preliminary to a furnace test, a crucible test and a muffle test were made.

The mix used consisted of roasted ore of approximately 0·8 per cent sulphur content, Alberta semi-bituminous coal and gas coke in the proportions of 4 : 2 : 1.

A graphite crucible was charged with 27 pounds of the mix and placed in a gas-fired, brass-melting, pot furnace. Five hours heating were required to bring the contents of the crucible up to 1,065°C. and this temperature was maintained for one and one-half hours. The crucible was left in the furnace overnight to cool. The product weighed 17 pounds and showed on magnetic separation 60 per cent magnetic concentrate, 6 per cent middling, and 31 per cent tailing. Analysis of the magnetic concentrate showed 79·4 per cent metallic iron and 83·66 per cent total iron, giving a metallization figure of 94·9 per cent. The sulphur content of the magnetic product was 0·60 per cent.

In the muffle test, the same mix was used and 29 pounds charged to a vitreosil muffle, placed in a muffle furnace of the wire resistance type. Seven hours were required to bring the contents up to 900°C. and this temperature was raised to a maximum of 915°C., a higher temperature being unavailable on account of the low line voltage prevailing during the test. This maximum temperature was maintained for one and one-half hours. The product after cooling weighed 20 pounds and gave 57 per cent magnetic concentrate, 5 per cent middling, and 35 per cent tailing.

The magnetic concentrate showed on analysis 69·08 per cent metallic iron and 78·56 total iron with 0·70 per cent sulphur and 0·03 per cent sulphur trioxide (SO_3).

A test for soluble sulphide such as calcium sulphide gave a negative result.

The metallization obtained was only 88 per cent, and was probably due to the low temperature used.

Rotary Furnace Sponge Iron Test

A description of the furnace and the procedure is given in Report No. 307, "Metallization of Iron Content in Ilmenite Ore", and the same applies to the following test with the exception that the discharge opening was reduced to 1¼ inches in diameter. The charge to the furnace consisted of roasted ore from the Herreshoff and rotary furnaces and averaged 0·80 per cent sulphur. The charge was made up of 10 parts ore, 7 parts Alberta coal, and 3 parts coke. The coal and coke were crushed to pass a ¼-inch screen.

Seven hours were required to bring the furnace and charge up to the minimum reducing temperature, namely 960°C.

The initial charge to the furnace was about 600 pounds. The discharge was withdrawn each hour for 4 hours, when a shutdown was necessary because of more pressing test work requiring the labour employed.

The average temperature for the 4-hour run was 1,020°C.

The total discharge amounted to 270 pounds during the 4 hours, and after magnetic separation averaged 38 per cent magnetic concentrate, 32·7 per cent middling, and 29 per cent tailing.

Analyses of the magnetic products gave the following results:—

Metallic iron, per cent	Total iron, per cent	Sulphur, per cent
61·28	—	—
73·71	—	—
77·02	80·29	0·66
77·29	80·46	0·67

The latter two show approximately 96 per cent of the iron as metallic iron.

The middling products ran somewhat high in metallic iron, namely 38 to 55 per cent, and further magnetic separation gave a product averaging 55 per cent metallic iron.

Of the total furnace discharge product, 73 per cent of the contained iron was metallized, a result which would probably be greatly improved upon on a longer run.

Sintering Tests

These tests were carried out in down-draught sintering pans of the double-pan laboratory type, the size of the pan grates being 18 inches by 12 inches.

The ore used was run-of-mine, containing 2·95 per cent sulphur, crushed to −35 mesh, and mixed with varying quantities of coal and water. The coal used was Welsh screenings. Each charge was distributed evenly over the grate through a quarter-inch screen, and given a slight ignition dressing of Welsh coal fines.

The charge was ignited by a gas burner moved over the surface of the charge; and air suction by fan to the extent of 2½ inches of water, was drawn through the grates and charge. When the sintering action appeared completed the charges were dumped into bins and the sinter and fines separated by means of an 8-inch screen. The following table shows the results of six tests employing various mixtures, the charge shown being the charge to each pan. The letters R and L designate the right and left pans, tests being run conjointly. Analyses of sinters are shown, the fines being discarded.

Sinter Tests

Mix	Ore, pounds	Coal, pounds	Water, pounds	Thickness of bed, inches	Time, minutes	Weight of sinter		Weight of fines		S, per cent		SO_3, per cent		Fe, per cent	
						R	L	R	L	R	L	R	L	R	L
1..............	25	2½	1¼	2	20	19	14	3½	8	0·79	0·83	0·10	0·12	56·17	58·50
2..............	25	1½	2½	2	20	16½	12½	6½	10	0·15	0·25	0·09	0·18	59·07	53·28
3.	50	2½	2½	3¾	20	24	27	21	19	0·38	0·50	0·16	0·18	56·97	57·75
4..............	50	3¾	3¾	4	30	27	28	17½	18	0·48	0·30	0·20	0·14	58·18	60·65
5..............	37½	1⅞	2⅛	2¾	20	19½	22½	15½	13½	0·32	0·47	0·25	0·32	58·97	54·96
6..............	25	2½	2½	2½	20	18	17	17	6	0·44	0·49	0·34	0·23	....	....

Nature of Sinters

(1) Hard sinter, much fused in small rounded balls, brownish, very porous.
(2) Fairly hard sinter, less fused but more massive and blacker than (1), porous.
(3) Softer sinter, some fusion, brownish, more porous than (2).
(4) Soft, friable sinter, fused considerably in centre, brownish, very porous.
(5) More friable than (4), centre massive and hard, remainder porous.
(6) Fairly hard sinter, highly fused in some portions, quite porous.

REMARKS

Sinters (1) to (5) were quite soft along the sides, while sinter (6) was even throughout except where fusion occurred. A charcoal ignition dressing was used in the latter case, which probably gave a more instant ignition.

The results obtained on each mix vary so markedly that it is difficult to make any definite comparison as to which mix gives the best results particularly as to desulphurization. Mixes (2) and (6), it will be noted, were identical and were, with the exception of the ignition dressing, carried out under similar conditions, yet the results vary quite markedly. Possibly the explanation lies in the fact that uniform ignition was seldom obtained due, perhaps, to the type of burner used for this purpose.

It is evident from these tests, however, that the sulphur can be reduced to below one half per cent by sintering, and that a fairly satisfactory sinter for blast furnace use can be made.

It is intended to continue this test work at the first favourable opportunity.

Report No. 307

METALLIZATION OF THE IRON CONTENT IN ILMENITE BY MEANS OF ROTARY KILN TYPE OF FURNACE

R. J. Traill, W. R. McClelland, and J. D. Johnston

In the report of Investigations of the Division of Ore Dressing and Metallurgy, 1925, a method was proposed for the treatment of ilmenite ore, whereby the iron would be recovered as electrolytic iron and the titanium content as high-grade, titanium dioxide concentrate. The tests, although carried out on a small-laboratory scale basis, proved quite encouraging and warranted further investigation on a semi-commercial or pilot-plant scale.

In the report referred to above, a brief summary of the complete process is given together with observations on operating conditions and results obtained. Subsequent to completion of the above test work, a kiln-type of furnace of semi-commercial proportions was installed as part of the equipment of the laboratories, for the purpose of making investigations in sponge iron production from Canadian iron ores. This kiln is of the two-diameter type used and recommended in the experimental work of the United States Bureau of Mines on sponge iron production, and as reported in U.S. Dept. of Commerce, Bureau of Mines Bulletin No. 270, 1927. The kiln installed in the Mines Branch laboratory consisted of a steel shell lined with firebrick, having the following dimensions: length of large section, 5 feet 6 inches, inside diameter, 2 feet 6 inches; one section tapering from diameter of 2 feet 6 inches to 1 foot 3 inches; the kiln rotates on two steel tires. The large end of the furnace was closed with a reinforced cement block lining and had a 7-inch circular opening in the centre to permit of direct heating by means of an Anthony Nebulyte oil burner, type L-1½. A circular combustion block, 13 inches in diameter, 9 inches long, and having a tapered combustion chamber opening, from 3 inches to 7 inches diameter, was used. The block was suspended on rails which allowed its easy removal when temperature readings were to be taken. With the block riding close to the furnace entrance of excess air was fairly well prevented.

A discharge pipe was inserted in the end wall of the furnace which fed into a hopper and thence to a bin, care being taken to prevent undue oxidation. The feed arrangement consisted of a rectangular hopper with a moving belt serving as the bottom which conveyed the mix through an adjustable opening at end of the hopper into a feed hopper to which was attached a 2½-inch pipe, leading about 8 inches inside the small end of the kiln. This end of the kiln opened into a dust chamber built of brick and having a baffle plate, the products of combustion thence passing on to the stack.

At the special request of Mr. R. H. Monk of Montreal, who represented a newly organized company interested in this process and holding ilmenite deposits in the province of Quebec, the following tests were run to demonstrate the suitability and the efficiency of such a kiln for this work. The magnetic product obtained was shipped to the plant of abovementioned company for experimental work on the producing of a titanium pigment product.

EXPERIMENTAL TESTS

The ore used in the tests was obtained from the deposit at Ivry, Terrebonne county, Quebec, and consisted of a hard, medium- to coarse-grained ilmenite with some showings of hematite and having the following analysis:

Total iron (Fe)	39·00	per cent
Titanium dioxide (TiO_2)	31·75	"
Alumina (Al_2O_3)	7·70	"
Lime (CaO)	0·45	"
Magnesia (MgO)	2·63	"
Silica (SiO_2)	7·07	"
Sulphur (S)	0·73	"

Weight of a cubic foot of −8-mesh ore was from 185 to 192 pounds. The coal used was obtained from a supply of Alberta sub-bituminous on hand at the Fuel Testing Laboratory, and had the following proximate analysis:

Moisture	9·00	per cent
Ash	9·30	"
Volatile matter	34·10	"
Fixed carbon	47·60	"
Sulphur	0·45	"
Gross calorific value	11,100	B.T.U.

The coke used was a gas coke purchased locally and of the following approximate analysis (air-dried basis):

Moisture	0·8	per cent
Ash	13·1	"
Volatile matter	1·9	"
Fixed carbon	84·2	"
Sulphur	1·1	"
Gross calorific value	12,000	B.T.U.

The fuel oil used was of ordinary market grade purchased locally.

The ore was crushed to pass an 8-mesh screen, and the coal and coke to pass 4 mesh. In the initial tests the mix consisted of ore and coal only, but as the metallization of the product was low and as it was thought that this was, perhaps, due to the low fixed carbon content of the mix, it was decided to replace some of the coal with coke. The size of the discharge opening also occasioned some trouble, in that too large an opening resulted in the mix passing through the furnace at too great a rate, with a resulting low metallization. Various diameters were tried between 1½-inch and 1-inch and finally 1⅛-inch discharge opening was adopted as a safe maximum. The feed mix consisted of ore and reducing material in the ratio of 5 to 4 and was mixed in batches by hand labour in the following proportions:

600	pounds	ore
320	"	coal
160	"	coke

The operating procedure was briefly as follows:—

The discharge opening was cemented over, the furnace temperature raised to around 1,000°C. and 300 pounds of mix charged into the wide diameter end through the opening in end wall of furnace, and a further 200 pounds was fed in through the hopper. This charge was sufficient to give a good bed of material in the furnace, which condition is very necessary to the successful operation of the furnace. When the temperature in the furnace had reached 990°C. this temperature was maintained for one to two hours before the discharge pipe was opened. About 20 minutes previous to opening the discharge, the feed was started and while the furnace was discharging the feed was so regulated that a bed of sufficient depth was maintained in the furnace to keep the top level with the burner opening in the end of the furnace. This proved an important factor in the operation of the furnace as when the depth of charge lessened, the rate of discharge and the percentage of metallization fell off quite noticeably. Temperature also proved to be an important factor and from the tests made it was found that a range of from 990° to 1,040°C. gave best results. Below 990°C. reduction falls off and above 1,040°C. the effect tends to produce sintering and sticking of fines to the lining of the furnace.

Several preliminary runs, averaging about 12 hours' operation, were made before sufficient data relative to the mechanical and operating conditions were obtained to warrant continuous operation. Two tests of the continuous type were run, the operating time being 100 and 104 hours respectively.

The continuous tests were run on the three-shift basis with one technical member of the staff in charge assisted by two mill men. Each shift was responsible for mixing up the charge and for magnetic separation of product and sampling in addition to the regular furnace attention.

The discharge from the furnace was collected in covered bins and was removed every two hours, weighed and allowed to cool several hours before magnetic separation. Magnetic separation was made on a Gröndal magnetic belt cobber at a belt travel of 300 feet per minute and with 22 to 24 amperes on the drum and 8 amperes on the rectifier magnets. This machine gave three products, namely a magnetic concentrate, a slightly magnetic middling, and a non-magnetic tailing. The analyses of these products are tabulated later. In plant operation the middling product would probably be returned with initial feed, but in the present tests it was not returned. The magnetic product from the cobber was riffled in a Jones riffle sampler and the resulting sample ground in a Braun pulverizer and analysed for the metallic iron. The method of analysis used for the metallic iron determination was the well known copper sulphate method.

Continuous Test No. 1

Following the general procedure outlined above, a continuous test run was made lasting 100 hours. Initial charge consisted of the regular mix as mentioned above, the temperature being held for one hour after rising to the desired reducing temperature, namely 990°C., before the discharge was opened. The reduction temperature within the furnace was kept as closely as possible within 990° and 1,016°C. during the whole run. The following table shows the daily record of feed, discharge, concentration, per cent metallic iron, and the oil consumption:

Hours	Feed, pounds	Feed per hour, pounds	Discharge, pounds	Discharge per hour, pounds	Magnetic, pounds	Middling, pounds	Non-magnetic, pounds	Metallic iron, per cent.		Oil consumed, gallons
								Low	High	
24........	2,550	106·2	1,776	74	972	276	534·5	30·54	41·46	45
24........	2,400	100	1,597	66·5	924·5	128	541	39·06	43·12	40
24........	2,550	106·2	1,626	67·7	785	279·5	537	39·26	44·78	35
24........	2,550	106·2	1,691·5	70·5	1,006·5	85·5	589·5	41·66	44·16	35
4........	250	....	287·5	71·8	178	13	96	43·32	43·43	
100........	10,300	103	6,978	69·78	3,866	782	2,298	....	....	155

During the first fourteen hours' operation some difficulty was experienced with the burner and with maintaining the proper depth of charge in the furnace due to feeder trouble with consequent irregular low metallization. During the rest of the run conditions were normal and the results were quite satisfactory, the test yielding 2,945 pounds of satisfactory product, showing a metallization of 90 to 95 per cent.

Time did not permit determining total iron and titanium dioxide on all samples, but several samples taken at random showed the following analysis:—

Metallic iron, per cent	Total iron, per cent	TiO_2, per cent	Metallization, per cent
41·61	45·28	39·49	92·1
42·67	45·86	39·78	93·0
42·88	45·96	39·39	93·4
43·03	44·98	39·98	95·8
43·35	46·15	40·07	93·9

The discharge from the furnace has a weight per cubic foot of 77 to 78 pounds, and the magnetic concentrate product a weight per cubic foot of 163 to 167 pounds.

Continuous Test No. 2

Preliminary to starting this test the furnace was emptied of practically all of the preceding charge. The initial charge to the large end of the furnace was made up from the discharge remaining in the furnace from the previous test, namely, about 280 pounds plus 50 pounds of coal, and the initial charge to the hopper feed consisted of some 140 pounds low metallic magnetics plus 90 pounds coal and 45 pounds coke. When the temperature in the furnace had attained 980°C. this temperature was maintained for two hours before the discharge pipe was opened. Conditions from beginning to end of the test were quite normal and no serious trouble was experienced, except in the discharge which blocked occasionally due to lumps in the furnace. The approximate average temperature throughout the run was 1000° to 1038°C. and the run lasted 104 hours. High metallization was obtained from start to finish as shown in the following table of the daily record.

Hours	Feed, pounds	Feed per hour, pounds	Discharge, pounds	Discharge per hour, pounds	Magnetic, pounds	Middling, pounds	Non-magnetic, pounds	Metallic iron, per cent		Oil consumed, gallons
								Low	High	
24	2,550	106·3	1,699	70·8	1,047·5	140	519	41·37	44·79	50
24	2,400	100	1,481	61·7	880·5	110	489	42·50	44·79	40
24	2,650	110·4	1,719·5	71·6	1,071	88·5	560	41·25	43·95	50
24	2,550	93·6	1,556	64·8	960·5	60·5	535	41·46	43·02	40
8	350	....	549	68·6	332	34	84	40·94	42·09	20
104	10,200	98·1	7,004·5	67·35	4,291·5	433	2,187	....	....	200

The total quantity of ore charged to the furnace was 5,660 pounds, of coal 3,015 pounds, and of coke 1,520 pounds. Of the furnace discharge it will be noted that 61·3 per cent is magnetic, 6·2 per cent middling, and 31·5 per cent tailing product. The middling product showed metallic iron content as follows:

First 24 hours	28·55 to 34·58	per cent metallic Fe
Second 24 "	29·90 " 33·75	" "
Third 24 "	26·25 " 27·18	" "
Fourth 24 "	19·37 " 26·25	" "
Final 8 "	24·79	" "

The total middling product would average about 28 per cent metallic iron and it is possible that by fine grinding and magnetic separation on a different type magnetic separator that a satisfactory recovery could be made of highly metallized concentrate. It might, however, be more economical and more convenient to pass this product with a coal mix back to the feed end of the furnace and treat with regular feed.

An examination of the tailing product which averages about 34 per cent ash showed the following results:

Mesh	Weight, per cent	Ash, per cent	TiO_2, per cent	Fe, per cent	Ash analysis	
					TiO_2, per cent	Fe, per cent
+14	52·54	26·40	1·51	3·59	5·71	10·11
−14+28	19·94	35·52	1·79	3·09	5·03	8·69
−28+48	14·34	48·80	3·26	4·85	6·68	9·93
−48+100	8·70	44·96	3·35	5·02	7·45	11·18
−100	4·46	47·70	3·39	5·82	7·11	12·20
Flue dust	....	70·35	17·40	22·86	24·73	32·48

The flue dust collected from the dust chamber amounted to about 3 per cent of the total feed charge.

Whether this tailing product can be advantageously utilized again has not been definitely determined. An attempt was made to use it mixed with low-grade magnetic concentrates and middlings in the proportion of 100 parts magnetic, 60 tailing, and 20 of coke, without the addition of any fresh coal, and run through the furnace in the usual manner. The result was not by any means encouraging, as none of the magnetically separated discharge product analysed above 39·4 per cent metallic iron, and, in comparison with the material used in the feed, appeared to be changed but little. There is, however, a possibility of the tailing product being utilized after screening or other method of separation, in place of coke, to a limited extent, but time has not permitted securing any data along this line and the suggestion is therefore just a matter of personal opinion.

Referring back to the discharge system and the possible effect of oxidation of metallic iron occurring in the falling of the discharge product through air into the discharge bin, a sample of magnetic concentrate was examined as follows: a 2,000-gramme sample was screened on an 8-mesh screen, the products weighed and analysed for metallic iron and the −8 mesh again screened through 48 and 65 mesh and the products analysed. A magnetic concentrate just below the desired grade in metallic iron content was used and the following table shows the results:

Mesh	Weight, per cent	Metallic iron, per cent
Head	....	38·74
+8	20	41·10
−8	80	38·14
+48	93·2	38·96
−48+65	3·7	26·72
−65	3·1	23·33

The result here obtained would indicate that the finer particles are the more readily oxidized as would be expected.

CONCLUSIONS

The rotary kiln type of furnace seems to be quite satisfactory for this metallization process. An improvement in the discharge rate would tend toward better efficiency. This might be brought about in several ways: (1) by reducing the size of coal and coke to the same size as the ore; (2) by increasing somewhat the large end diameter with a relative increase in the discharge pipe diameter. Finer grinding of the ore is not to be recommended in face of the results obtained and shown above. A good depth of bed charge must be maintained throughout, and the temperature must be held at not less than 990°C. and not over 1,050°C.

Segregation occurs to some extent in the large diameter of the kiln, but this is not serious, its only effect apparently being in a varying discharge period weight. During operation it was observed that the coke had a tendency to ride on the surface of the charge. This it is believed is a highly satisfactory condition as it protects the ore bed from oxidation through air entering the furnace at the burner opening.

The degree of metallization obtained, namely 90 to 95 per cent, should be quite satisfactory for producing a low iron titanium dioxide concentrate. A metallized product analysing 43·35 per cent metallic iron equivalent to 93·9 per cent metallization gave on leaching with 10 per cent sulphuric acid, a concentrate containing—titanium dioxide 75·8 per cent, iron 6·27 per cent, and carbon 0·80 per cent.

The reducing agents used in these tests are perhaps more expensive than the process would economically warrant, but there is no reason to believe that less expensive fuels would be less satisfactory, provided that fuel of non-coking or very slightly coking type is used. It would be interesting to determine if anthracite fines could be utilized, without coke admixture, and if a low volatile matter coal could be used with equally successful results.

These tests were productive of nearly 4 tons of magnetic concentrate analysing over 40 per cent metallic iron.

III

REPORTS OF INVESTIGATIONS: NON-METALLICS LABORATORY

Report No. 308

EXAMINATION OF ASBESTOS-BEARING ROCK FROM SPROAT MOUNTAIN, NEAR ARROWHEAD, B.C.

R. K. Carnochan

Shipment.—Five bags of asbestos-bearing rock, shipping weight 300 pounds, were received November 24, 1927, from the Lardeau Mines Exploration, Ltd., 704 North West Building, Vancouver, B.C. The rock was from the Asbestos group of six claims on Sproat mountain, near Arrowhead, B.C.

Purpose of Tests.—Tests were desired to determine the quantity and quality of the fibre in the shipment, and its suitability for the manufacture of asbestos boards and shingles.

Characteristics of the Rock.—The shipment consisted of greenstone, serpentine, a little calcite and foliated talc, and a large amount of asbestos fibre. The fibre is of different kinds, mostly soft white fibre, some soft brown fibre, and some hard, honey-coloured semi-fibre. There are intermediates between these different fibres, but these three are the general ones. Some fibre sent with the shipment as special specimens was 6 inches long; the longest in the regular shipment was about 3 inches in length.

Experimental Tests.—A small piece of soft white fibre picked out of the shipment was heated to 1000°C. in an electric furnace. On cooling it was found to have lost 35·8 per cent of its weight, to have become pink in colour, very brittle, and easily broken.

The shipment was crushed to 1-inch size in a small jaw crusher and cut into quarters by passing through a Jones riffle sampler. One quarter, 75 pounds, was screened on 2 mesh and the fibre picked out of the +2 mesh. The +2-mesh rock was crushed to pass through the 2-mesh screen and mixed with the −2-mesh material from the first screening. The −2-mesh material was then screened on 4-mesh and the fibre removed from the +4-mesh by suction. The +4-mesh rock was crushed to pass through the 4-mesh screen and mixed with the −4-mesh from the second sizing. The process was repeated at 8- and 12-mesh. The −12-mesh material was screened on 35-mesh and the fibre removed from the two sizes, −12+35 and −35-mesh, by tabling wet on a small Wilfley table. The overflow from the box catching the fibre during the tabling of the two sizes was put into a settling-tank and this tank caught the very fine fibre along with some fine sand. Some sand was also present in the regular −35 table fibre. The following table is a summary of the test:

	lb.	oz.
+2 fibre	0	12
+4 "	1	10·5
+8 "	1	4·5
+12 "	1	2·75
+35 "	1	15
−35 " and some sand	7	8
Tank " "	5	8
	19	12·75

The feed to the test was 75 pounds—the fibre recovered, 19 pounds 12·75 ounces. This means that 26·4 per cent of the feed was recovered as fibre. The amount of fibre without sand recovered was 6 pounds 12·75 ounces, or 9·1 per cent of the feed. The fibre with sand, asbestic, recovered was 13 pounds or 17·3 per cent of the feed.

CONCLUSIONS

The longer fibres, +2- and +4-mesh, are woody in texture, lacking in strength, and could not be used for spinning, shingle, nor paper making. The only possible marketable products that could be produced from this class of fibre is cement stock, grading 0–0–5–11, and fine fibre containing a little sand, known as asbestic and used in the manufacture of finishing plaster. The asbestos-bearing rock could be crushed and the fibre removed to give these two products. Cement stock sells for $20 to $25 per ton, and asbestic for $8 per ton. The following table shows the amounts of these two products that could be recovered from a ton of rock, and the value at the above prices:

Product	Recovery		Value	
	Per cent	Lb./ton	Per ton	Total
Cement stock	9·1	182	$20·00	$1·82
Asbestic	17·3	346	8·00	1·38
Total fibre	26·4	528	....	3·20

Report No. 309

THE PREPARATION OF A FINE-GRAINED LIMESTONE FROM L'ETANG, NEW BRUNSWICK

R. K. Carnochan

Shipments.—Two shipments of fine-grained limestone were sent in by J. Sutton Clark, St. George, N.B., owner and operator of the Canadian Calcite Mines, L'Etang, N.B., from whence the shipments originated. The first shipment of 500 pounds was received at the laboratories, November 22, 1927, and a second shipment of 820 pounds was received March 26, 1928.

Characteristics of the Limestone.—The limestone in the large pieces as received is greyish white in colour with a very small amount of rust stains in the fractures. It is not crystalline, but fine-grained and massive. When ground fine in an agate mortar the product is pure white in colour.

Sampling and Analysis.—The first shipment, designated Lot No. 1, was crushed in a gyratory crusher to about 2 inches, then in a hammer mill to about 4 mesh. A head sample was cut out by a Jones riffle sampler, which on assay gave:

Insoluble	2·28 per cent

This insoluble was found on analysis to be divided as follows:

Silica	1·73 per cent
Iron oxide and alumina	0·27 "
Lime magnesia and alkali earths	0·28 "

Purpose of Experimental Tests.—The purpose of the tests was to determine to what fineness the limestone could be ground by both wet and dry methods, and the nature of the products.

EXPERIMENTAL TESTS

Dry Grinding.—A part of Lot No. 1, 160 pounds, was prepared by grinding in a cylindrical mill, 2 feet by 3 feet, lined with porcelain, and carrying 120 pounds of pebbles, the discharge of the mill going to a 30-inch Gayco air separator and the oversize being returned to the mill. The product from the air separator was 99·85 per cent through 325 mesh, and was pure white in colour.

Wet Grinding.—The remainder of Lot No. 1 and all of Lot No. 2 were prepared by grinding wet in the 2- by 3-foot cylindrical mill in circuit with a classifier, the oversize being returned to the mill. The classifier overflow was thickened and filtered, giving a product 99·95 per cent through 325 mesh. The product was slightly off colour due to the small amount of material used in the large-size equipment which resulted in the limestone picking up a small amount of rust. Larger lots of similar material have been ground, water-floated, and thickened by the same method without any discoloration. Under proper conditions the product would have been pure white. The wet-ground, water-floated product was more uniform than the dry-ground, air-floated product.

CONCLUSIONS

The limestone submitted is of a very good grade and adaptable to fine grinding for the production of commercial products. It can be ground either dry or wet—wet grinding would give a slightly finer and more uniform product. The colour of the ground product would be pure white using either dry or wet methods of grinding. A report on samples submitted shows that it makes an excellent product for the manufacture of putty. It should also make an excellent substitute for imported whiting for use in the manufacture of paper, rubber, and pigment products.

Report No. 310

THE GRINDING OF A SOAPSTONE FROM ROBERTSONVILLE, QUEBEC

R. K. Carnochan

Shipment.—Nineteen bags of soapstone, shipping weight 3,600 pounds, were received on October 19, 1927, from the Robertsonville Soapstone and Quarry Company, Leeds Station, Beauce county, Quebec.

Purpose of Experimental Tests.—Tests were desired to determine the most suitable method of grinding the soapstone to nearly all −325 mesh, so that it could be used in paints, paper, rubber, textiles, etc. It was also required to find a suitable method to prepare a product from the soapstone, which would be all −60 mesh and contain as little fines as possible. This product was to be used in coating roofing paper.

Characteristics of the Rock.—The shipment received consisted of greenish grey soapstone free from cracks and joints, but having a laminated structure. This soapstone is at present being sawn into blocks to use in lining stoves, and some sawn faces on the rock received was evidence that it was the discard from the sawing operations. When broken or crushed the soapstone forms a lot of fine flakes, making it difficult to separate in an air separator, as the air will lift anything of a fibrous or flaky nature more readily than rounded or angular grains, which results in some flaky oversize going into the fine product.

EXPERIMENTAL TESTS

Coarse Crushing.—A few specimens were picked out of the shipment and the balance was crushed to about 1-inch size by passing it through a 12- by 8-inch jaw crusher.

Grinding with Pebble Mill and Gayco Air Separator.—A quantity of the 1-inch material, 165 pounds, was crushed to about 4-mesh by means of a small hammer mill. The hammer mill product was then fed to a small pebble mill, 2 feet by 3 feet. The mill discharge went to a 30-inch Gayco air separator, the Gayco fines were finished product and the Gayco oversize was returned to the pebble mill. The mill was lined with porcelain and carried 120 pounds of pebbles. The feed to the mill was 30 pounds per hour including the oversize returning from the separator. The separator produced 5 pounds of fines per hour. These fines were 99·98 per cent −325 mesh and were white in colour, not pure white, but a white having a slight greyish cast.

Grinding with Raymond Pulverizer.—A quantity of the 1-inch material, 250 pounds in weight, was crushed in the hammer mill to about 4 mesh and shipped to Raymond Brothers in Chicago. It was ground in one of their No. 00 pulverizers and the ground product returned. This ground product is 99·97 per cent −325 mesh and is the same colour as the soapstone ground in the pebble mill and the fines separated in the Gayco separator, that is, white with a slight greyish cast.

Grinding with Hammer Mill and Screen.—Fifty pounds of the 1-inch material was crushed in a hammer mill in closed circuit with a 60-mesh hammer screen. A screen test on the 60-mesh gave—

+ 65 mesh	4·5 per cent
−65 + 100 "	19·9 "
−100 + 150 "	16·1 "
−150 + 200 "	13·8 "
− 200 "	45·7 "

A screen test on a sample of ground talc used for coating roofing paper gave—

+ 65 mesh	6·4 per cent
−65 + 100 "	25·3 "
−100 + 150 "	20·2 "
−150 + 200 "	13·2 "
− 200 "	34·9 "

The ground talc is pure white in colour and the 60-mesh soapstone is greenish white. However, for coating roofing paper it is not necessary to have a pure white product.

It will be noticed that the colour of the soapstone depends upon its fineness. When very coarse it is greenish grey, when about 60 mesh it is greenish white, and when very fine it is white with a greyish cast. This shows that the finer it is ground the lighter its colour.

CONCLUSION

1. To produce a product nearly all −325 mesh two methods would be perfectly suitable. One of these would be to use the machine recommended by Raymond Brothers, namely a special five-roller, high-side mill; the other would be to grind in a pebble mill and separate the fines out by means of a Gayco air separator. In both these systems the soapstone would be first crushed in a jaw crusher and hammer mill.

2. To produce a product all −60 mesh and not containing too much fines, the soapstone could be broken in a jaw crusher and then reduced to 60 mesh by means of a hammer mill in closed circuit with a 60-mesh vibrating screen.

3. If it was desired to make both −325-mesh and 60-mesh material at the same time, it could be done by crushing in a jaw crusher and hammer mill and screening the hammer mill product on a 60-mesh vibrating screen. The −60 would be finished product and the oversize of the screen could be ground by one of the recommended methods to −325 mesh.

Report No. 311

GRINDING OF CALCITE FROM THE FRONTENAC LEAD MINES, PERTH ROAD, ONTARIO

R. K. Carnochan

Shipment.—A shipment of 140 bags, approximately 5 tons, of calcite was received January 23, 1928, from the Industrial Processes Development, Limited, Kingston, Ont. This calcite was from the Frontenac lead mine, Perth Road, Ont.

Characteristics of the Calcite.—The shipment consisted of white calcite containing about 2·5 per cent lead as galena, a very small amount of zinc as sphalerite, some pyrite, about 5 per cent white quartz, and some discoloured quartz.

Purpose of Experimental Tests.—Tests were desired to determine the best method for the grinding of the calcite to produce a fine white product. Large samples of the finished fine ground calcite were desired to submit to prospective users to determine its suitability for their requirements.

EXPERIMENTAL TESTS

Concentration methods for the recovery of the galena in a high-grade lead concentrate were worked out on a previous carload shipment (*see* Report No. 254, "Investigations in Ore Dressing and Metallurgy 1926", p. 60). Following this method of procedure the lead, zinc, and iron contents could be practically eliminated leaving a calcite tailing containing about 5 per cent of clean quartz and a small amount of discoloured quartz. This method for the elimination of sulphide impurities in the calcite was not followed as it entailed considerable work. A simpler method of crushing to 12 mesh and sizing on 26 mesh and 60 mesh with the concentration of the impurities from the three sizes on tables was followed. The ore was crushed in a gyratory crusher to about 1½-inch size and further reduced in a disk crusher to ½-inch size and screened on 12 mesh. The +12 mesh was crushed in rolls and returned to the screens. In this manner the whole lot was reduced to −12 mesh, screened on 26 mesh, and the −26 mesh screened on 60 mesh. This gave:

Product	Weight, pounds	CaO, per cent	Insoluble, per cent	Pb, per cent
−12 + 26	5,314	50·55	3·77	2·54
−26 + 60	2,449	49·26	4·69	2·70
−60	2,047	48·21	6·01	2·36

The above sizes, after removal from each of a small sample, were tabled making a lead concentrate and a clean calcite tailing from each size. The concentrates were tabled again making a clean lead concentrate and a middling containing most of the zinc blende and iron pyrite, in each case. The results of the table concentration tests were as follows:

Size	Product	Weight, pounds	Analysis			
			Pb, per cent	Zn, per cent	CaO, per cent	Insoluble, per cent
−12+26	Concentrate	129·5	78·55			
	Middling	87·0	5·68			
	Tailing	....	0·05	0·31	52·63	3·83
	Feed	5,298·0	2·54			
−26+60	Concentrate	77·0	71·88			
	Middling	53·5	4·22			
	Tailing	....	0·05	0·13	52·20	4·85
	Feed	2,444·0	2·70			
−60	Concentrate	43·5	81·81			
	Middling	56·0	1·83			
	Tailing	....	0·41	0·28	50·68	6·73
	Feed	2,039·0	2·36			

The tailings thus produced were fairly clean white calcite containing a few dark spots. These spots were discoloured quartz and zinc blende mostly and a very little galena and pyrite.

Wet Grinding.—The table tailings were sampled wet and a large portion ground wet in a cylindrical mill, 4 feet by 8 feet, in closed circuit with a 7-foot, Dorr bowl classifier. This mill was lined with porcelain and carried 3,000 pounds of porcelain balls. The classifier overflow went to a Dorr thickener (12-foot diameter, 2-tray, 3-compartment). When the grinding was completed the ground calcite, which had been allowed to remain in the thickener, was dewatered in a Patterson filter press (36-plate, 28-inch diameter). The filter cake was dried on steam coils and broken in a small hammer mill. The products from this test were:

Fine-ground calcite	6,358 pounds
Cleanout of mill	29 "
Cleanout of classifier	1,352 "

The fine-ground calcite produced is 99·99 per cent −325 mesh and is almost pure white in colour, the slight discolorization being of a grey cast and due to the small amount of galena and zinc blende remaining in the calcite after tabling.

Dry Grinding.—A lot of 190 pounds of the table tailings was dry-ground in two charges in a pebble mill, 2 feet by 3 feet, containing 260 pounds of pebbles, the product being separated into fines and coarse by a 30-inch Gayco air separator. To make a complete separation of the fines, the oversize was re-run through the Gayco twice. The final oversize was ground in the pebble mill and fed to the separator, the oversize from this operation being re-run four times in order to remove all of the fines possible. All the fines from the separator were combined, and the products from the test were:

Fine-ground calcite	101·5 pounds
Gayco oversize	46·5 "
Gayco cleanup	42·0 "

The fine-ground calcite produced by this method is 99·91 per cent −325 mesh and is of the same colour as that produced by wet grinding.

Grinding in Raymond Mill and Classifying in Gayco Separator.—A lot of 25 pounds of the table tailings was ground dry in a Raymond No. 0000 pulverizer, which is a beater type of machine. The product from the pulverizer was separated into fines and oversize in a 30-inch Gayco air separator, the oversize being returned to the Raymond three times. This gave:

Fine-ground calcite	14·5 pounds
Gayco oversize	2·5 "
Cleanup of machines	8·0 "

The fine-ground calcite produced is 96·19 per cent −325 mesh and grey in colour, the discolorization being due to iron wearing off the beater blades of the Raymond mill.

CONCLUSIONS

The best method of grinding the calcite will depend upon the class of product required. If an extremely fine product is desired then the calcite will have to be ground wet along the lines described under wet grinding. If, however, a product, say 99·8 per cent −325 mesh would be suitable then the calcite can be ground dry in a pebble mill and separated with an air separator.

Report No. 312

THE PREPARATION OF OYSTER SHELLS FROM ORANGEDALE, CAPE BRETON

R. K. Carnochan

Shipment.—A barrel of oyster shells, shipping weight 115 pounds, was received at the laboratories in November, 1927, from Wm. Mombourquette, Orangedale, Cape Breton.

Purpose of Tests.—The shipment was made to determine a suitable method for the preparation of the shells to meet market requirements for poultry feed.

Experimental Tests.—A small lot of the shells was dried, dry weight 18 pounds, and crushed in a small Sturtevant hammer mill fitted with a ½-inch mesh grating. The hammer mill product was sized by screening into the following sizes:

Mesh	Quantity		
	Lb.	Oz.	Per cent
+3	0	9	3·6
−3+6	2	9	16·3
−6+10	3	1	19·4
−10+20	3	9	22·6
−20+35	3	1	19·5
−35+65	1	8	9·5
−65	1	7	9·1
Total	15	12	100·00

Two samples of crushed oyster shells were purchased from a local dealer in Ottawa and screen tests made on them, with the following results:

Crushed shell for hens

Mesh	Per cent
+3	17·8
−3+6	47·6
−6+10	23·7
−10	10·9

Crushed shell for chicks

Mesh	Per cent
+10	7·3
−10+20	90·4
−20+35	1·5
−35	0·8

The remainder of the shells, 77·75 pounds, which had become dry on standing in the open barrel, was crushed in the hammer mill at the rate of 500 pounds per hour. The hammer mill discharge was screened on 10 and 20 mesh on a vibrating type of screen which was operated over capacity so as to leave some fines in the oversize. The rate of screening on 10 mesh was 325 pounds of feed per square foot of screen, and on the 20 mesh, 140 pounds of feed per square foot of screen. The products from screening were:

Mesh	Pounds	Per cent
+10	32	42·1
−10+20	21	27·6
−20	23	30·3
Total	76	100·0

Screen tests were made on the +10 and the −10+20 sizes with the following results:

+10-mesh shell for hens

Mesh	Per cent
+3	8·6
−3+6	28·5
−6+10	38·6
−10	24·3

−10+20-mesh shell for chicks

Mesh	Per cent
+10	0·3
−10+20	63·2
−20+35	32·4
−35	4·1

SUMMARY OF TESTS

The two products obtained do not exactly correspond to the samples purchased in Ottawa as to screen size, but are close enough for the purpose for which they are to be used. To the eye they appear to be the same. The products are, however, much whiter and cleaner, which is more important from a sales standpoint. If it is necessary to obtain a closer approximation as to screen size this can readily be accomplished by varying the hammer mill grate and adjusting the rate of feed to the screens.

CONCLUSIONS

The most suitable method for preparing oyster shells for poultry feed would appear to be as follows:

Dry in a suitable dryer,
Crush the dried shells in a hammer mill having a ⅜-inch grate opening, and
Size on 10- and 20-mesh screens.

The +10-mesh size should be suitable for hens, and the −10+20-size for chicks. The −20-mesh material would have to be discarded, or some other use found for it.

Report No. 313

THE WASHING OF A CHINA CLAY FROM ST. RÉMI, QUEBEC

R. K. Carnochan

Shipments.—Two shipments of crude china clay from the property of the Canadian China Clay Company, St. Rémi, Quebec, were submitted by R. A. Bryce of Toronto. The first shipment weighed 20 tons, and was received October 14, 1926; the second weighed 10 tons, and was received November 24, 1927.

Purpose of Experimental Tests.—Test work was desired to determine the most suitable method of refining the china clay, and also if possible to recover the coarse quartz sand present in the crude clay, as a clean sand suitable for glass-making.

Characteristics of the China Clay.—The crude clay consists of pure white china clay intimately mixed with fragments of sharp quartz. These fragments vary in size from about 1 inch to very fine sand.

Sampling and Analysis.—A head sample was taken from Shipment No. 1, and a wet screen test was made on a part of this sample. The screen products were analysed for silica, alumina, and iron. The following are the results:

Size	Weight, per cent	SiO_2	Al_2O_3	Fe_2O_3
+ 6	1·0	98·90	0·68	0·14
− 6+ 8	2·9	99·30	0·57	0·05
− 8+ 10	5·6	99·46	0·46	0·14
− 10+ 14	5·7	98·60	0·47	0·11
− 14+ 20	6·8	98·76	0·38	0·14
− 20+ 28	4·9	99·02	0·45	0·11
− 28+ 35	5·4	99·12	0·55	0·05
− 35+ 48	3·9	98·82	0·81	0·11
− 48+ 65	2·8	96·18	2·59	0·57
− 65+100	1·5	92·90	5·05	0·75
−100+150	2·4	89·78	7·10	2·16
−150+200	1·0	83·46	11·98	2·88
−200+300	1·0	84·78	11·15	3·03
−300+325	0·1	83·90	11·50	1·93
−325	55·0	47·26	37·39	0·75

The sizes +48 mesh are clear blue-grey sand. The sizes from +65 to +325 are sand, but of darker colour. The −325 mesh is very pure china clay.

EXPERIMENTAL TESTS

Shipment No. 1

Seven large-scale washing tests were run on Shipment No. 1, besides some medium-scale tests, and a large number of small tests. None of these tests was satisfactory, but they served to bring out difficulties, which were gradually overcome.

These difficulties were as follows:—

1. The coloured matter in the water used, which was Ottawa River water from the city mains, was taken up by the clay and gave it a slightly yellow appearance. This was overcome by treating the water with aluminium sulphate and filtering. The treatment removed most of the colour from the water and when the treated water was used to wash the clay, the clay remained white. The use of sulphuric acid instead of the aluminium sulphate was tried but it was found that the clay on drying became grey in colour.

2. When pebbles were used in the mill to blunge the clay, the silica sand was ground too fine, some clay still adhered to the sand particles and some lumps of hard clay were left in the sand product. It was found that when the mill was operated without pebbles and the coarse sand built up in the mill, it gave a much better blunging action, washing the sand particles free of clay.

3. When filtered water was used in washing the clay, the aluminium sulphate used coagulated the clay, enlocking fine sand particles which were carried over in the overflow of the bowl classifier. The addition of tri-sodium phosphate was tried to disperse the clay particles and permit the fine sand particles to settle out. This improved the clay product from the bowl classifier overflow.

4. The bowl classifier overflow contained some fine sand. Sand traps were installed between the classifier overflow and the thickener. A considerable amount of fine sand settled out in these and improved the quality of the washed clay.

Shipment No. 2. Washing and Separation Tests

A large-scale test was made on Shipment No. 2, following the preliminary work on Shipment No. 1. Filtered water was used. No pebbles were used in the mills. Tri-sodium phosphate was added to the feed to deflocculate the clay. Sand traps were used on the overflow of the bowl classifier. The flow-sheet given below was followed.

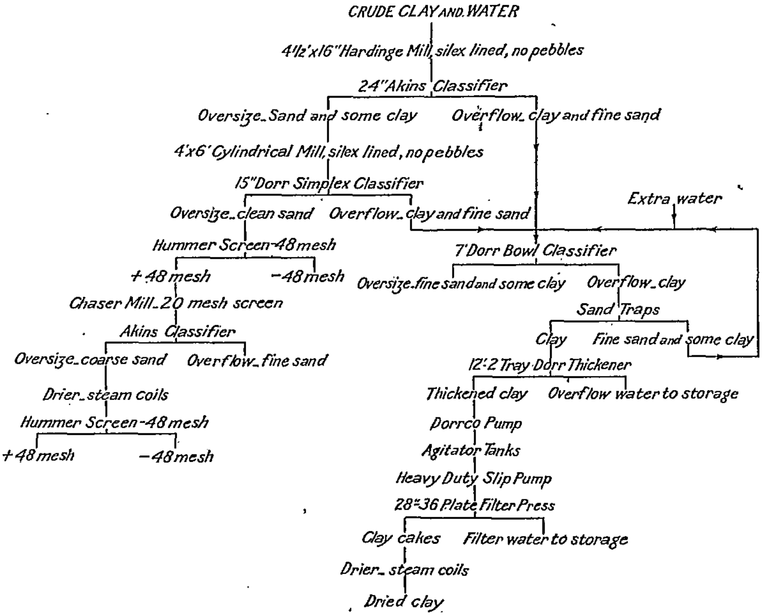

Figure 1. Flow-sheet, Shipment No. 2, washing and separation tests on china clay from St. Rémi, Quebec.

A sample, 18,178 pounds, of crude clay, having a moisture content of 11·26 per cent and therefore a dry weight of 16,131 pounds, was used for the test. The products obtained were as follows:

—	Pounds	Per cent feed
Washed clay	7,463	46·26
Coarse sand	7,534	46·71
Fine sand and clay	496	3·07
Loss	638	3·96
Total	16,131	100·00

The washed clay is pure white in colour and averages 99·95 per cent −325 mesh.

The coarse sand ranges in size from ½ inch to 150 mesh and contains very little clay.

The fine sand and clay is 99 per cent −100 mesh, and contains 21 per cent fine sand +325 mesh and 79 per cent −325 mesh, which is practically all clay.

The loss would most likely be china clay. Figuring it as such, the china clay content of Shipment No. 2 is approximately 52·64 per cent.

Treatment of Coarse Sand Product from Shipment No. 2

For the purpose of obtaining a sand product that could be used for the manufacture of clear glass, a test run was made on the coarse sand product, the oversize from the Simplex Dorr classifier. Preliminary tests on this product showed that the +48-mesh sand was low in iron and the −48-mesh sand high in iron. The coarse sand product was, therefore, screened wet on a Hummer screen fitted with a 48-mesh screen and the +48-mesh sand ground in a Chaser mill to 20 mesh and washed in an Akins classifier which washed out and overflowed the −100-mesh sand and impurities.

The results of this test were:

Product	Weight, pounds	Weight, per cent	Fe_2O_3, per cent
−48 from Hummer	816	6·11	1·43
Akins oversize	4,531	33·95	0·05
Akins overflow	887	6·65	0·58
Total	6,234	46·71	

The Akins classifier oversize analyses: Fe_2O_3, 0·05 per cent; TiO_2, 0·08 per cent; Al_2O_3, 0·11 per cent; SiO_2, 99·65 per cent.

Sand for use in making clear glass should not be over 0·06 per cent combined Fe_2O_3 and TiO_2. As the Akins classifier oversize contains 0·13 per cent Fe_2O_3 and TiO_2 it cannot be used for clear glass. A sample of the Akins classifier oversize was screened on a 48-mesh screen. The +48 from this screen test assayed less than 0·06 per cent Fe_2O_3 and TiO_2, and the −48 over 0·06 per cent.

The Akins classifier oversize, which had been dried, was run over a −48 mesh, Hummer vibrating screen, 5 feet by 3 feet, until the oversize of the screen assayed 0·02 per cent Fe_2O_3 and 0·04 per cent TiO_2. To secure this result the sand had to be put over the screen four times. The −48 mesh from the Hummer screen was not assayed for Fe_2O_3 nor TiO_2.

The results of this test were:—

Hummer screen products	Weight, pounds	Per cent of total weight	Fe_2O_3, per cent	TiO_2, per cent
+48 mesh	3,111	25·12	0·02	0·04
−48 mesh	1,093	8·83		
Total	4,204	33·95		

This shows that 25·12 per cent of the weight of the crude clay of Shipment No. 2 would be recovered as a sand assaying 0·02 per cent Fe_2O_3 and 0·04 per cent TiO_2, which would be low enough to allow the sand to be used to make clear glass.

To determine whether a suitable product for glass-making could be obtained by more careful screening of the sand before grinding in the chaser mill, a small quantity of sand was washed out of a head sample of Shipment No. 2. This sand was dried and a screen test made using 28-, 35-, and 48-mesh screens.

Assays of the products showed the +28 to be 0·10 per cent Fe_2O_3 and TiO_2; the +35, 0·13 per cent, the +48, 0·49 per cent, and the −48, 4·64 per cent. This shows that it is necessary to crush the sand in the chaser mill and screen on 48 mesh. An inspection of the sand from which the china clay was washed shows that it is very angular and many of the grains have little cavities in their sides. These cavities may contain some iron and titanium oxides. When the sand is crushed in the chaser mill the oxides are liberated and go into the fines when screened. This may explain why it is necessary to crush the coarse sand and screen it to reduce the iron and titanium content.

OPERATION NOTES ON CLASSIFICATION

In the test on Shipment No. 2, the fastest rate of feed to the bowl classifier that could be obtained, consistent with good work, was 100 pounds of crude clay in 20 minutes. At the time this rate was being maintained the moisture in the feed was 9·92 per cent, the dilution of the sand trap overflow was 58·8 to 1, and the overflowed clay was 99·77 per cent −325 mesh. The classifying rate of the bowl classifier figures out at 3 feet per hour. This is a very slow rate and is due to the aluminium sulphate used in the cleaning of the water. The effect of the sulphate in the water is to coagulate the clay, interlocking fine sand particles which are carried over in the bowl classifier overflow. To obtain fine clay the classifying rate has, therefore, to be greatly reduced. In order to correct this condition one pound of tri-sodium phosphate per 100 pounds of crude clay was added to the feed, but even this amount had only a slight effect.

Small tests made with the water of St. Rémi lake, which would be used to wash the clay in actual practice, show that the clay is also coagulated but not to the extent as when the treated water was used in the tests on Shipment No. 2. These small tests indicated that a classifying rate of 12 feet per hour could be obtained by using 16 pounds of tri-sodium phosphate per ton of crude clay if the water were not used over again. If the water is used over again, and it would be logical to do so, then the tri-sodium phosphate could be reduced to about one-tenth of a pound per ton of crude clay. The additional cost of this small amount of phosphate would be only 1 cent per ton of finished clay, figuring the phosphate at 5 cents per pound. A classifying rate of 12 feet per hour would enable the clay to be washed without too great an outlay for bowl classifiers.

The use of sand traps after the bowl classifier was found advantageous. In one test run partly without and partly with the traps it was found that the use of the traps reduced the sand content of the clay from 1·25 per cent to 0·75 per cent.

OPERATION NOTES ON THICKENING

A 2-tray, 3-compartment, 12-foot diameter Dorr thickener was used in all the large-scale tests, but was run to capacity only once, thickening 450 pounds of clay from a dilution of 60 : 1. When this capacity was obtained acid was used to coagulate the clay. Washing the clay without a coagulant or with the use of a dispersant such as tri-sodium phosphate, the capacity of the thickener would not be quite so great, but would be nearly so. The clay was thickened to a density of 3 : 1 and at certain times to 2 : 1 depending on how long the clay remained in the thickener. In 7 days the clay settles to 3 : 1 and in 13 days to 2 : 1.

OPERATION NOTES ON FILTERING

Most of the clay from the large-scale test was filtered on a 36-plate, 28-inch diameter, Patterson round-rail filter press. This press gave filter cake containing 35 per cent moisture.

Tests made on a laboratory, 3-foot diameter by 6-inch face, Oliver filter, showed that with a feed dilution of 3 : 1 and a speed of 23 r.p.m., cake containing 40 per cent moisture was produced at the rate of 40 pounds of dry clay per hour, or 8·5 pounds per square foot of canvas area.

OPERATION NOTES ON DRYING

All the clay produced from the tests was dried in a pan dryer or on steam coils. Both methods worked sufficiently well for testing purposes, but for regular commercial work a method involving less labour would be desirable. Samples of clay were submitted to firms making different types of dryers. From the reports received it would seem that a rotary indirect dryer would be the most suitable for the purpose.

CONCLUSIONS

1. The flow-sheet (Figure 2) is recommended for the treatment of the china clay to recover, besides the clay, a sand product suitable for glass-making.

2. The china clay can be separated from the sand in the crude clay and recovered as a pure white, fine product.

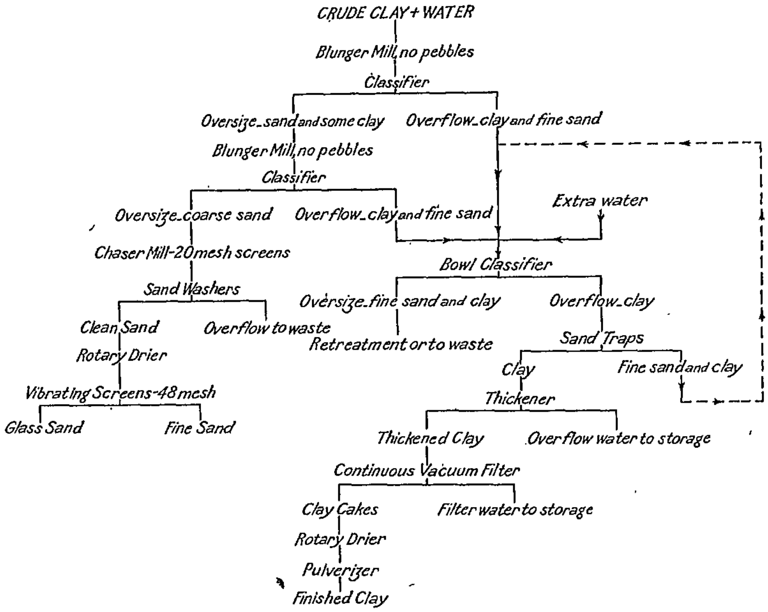

Figure 2. Flow-sheet recommended for treatment of crude china clay from St. Rémi, Quebec.

3. Shipment No. 1 produced 55·0 per cent and Shipment No. 2, 52·6 per cent china clay. As the weight of Shipment No. 1 was 20 tons and Shipment No. 2, 10 tons the average content of china clay in both shipments was 54·2 per cent.

4. The sand content of the crude clay can be separated from the china clay; and by further treatment a clean glass sand can be obtained, and a fine sand suitable for other purposes.

5. Shipment No. 1 contained 36·2 per cent sand +48 mesh and Shipment No. 2, 40·6 per cent. The average for both shipments is 37·7

per cent +48-mesh sand. In crushing and washing the sand of Shipment No. 2, 25·12 per cent of the crude clay was recovered as glass sand which would give an average of 23·3 per cent for the two shipments. The average amount of fine sand from the two shipments would be 22·5 per cent.

Report No. 314

THE CONCENTRATION OF A GRAPHITE ORE FROM BUCKINGHAM, QUEBEC

R. K. Carnochan and C. S. Parsons

Shipment.—A carload of graphite ore, net weight 37,115 pounds, was received May 31, 1927, from the Crucible Graphite Company, Buckingham, Quebec. The shipment was arranged for by H. S. Spence of the Mineral Resources Division.

Purpose of Experimental Tests.—The purpose of the concentration tests was primarily to produce a No. 1 high-grade flake graphite from the ore with which to determine its suitability for the manufacture of crucibles, and secondly to determine an economic method of concentration that will be least destructive of the flake and give the largest percentage of No. 1 flake, compatible with good recoveries. This report deals mainly with the concentration of the ore.

Arrangements for Experimental Work on Concentration.—Arrangements were made whereby H. S. Spence of the Mineral Resources Division would co-operate with R. K. Carnochan, in charge of the test work on non-metallics, and C. S. Parsons, in charge of ore dressing test work on metallic ores, who had previously done considerable work on graphite, including the operations of a graphite mill in the United States for the Joseph Dixon Graphite Company. H. C. Mabee, Chief Chemist of the Division, and B. P. Coyne were to conduct the chemical tests and analyses.

Characteristics of the Ore.—The graphite occurs in the ore in small flakes ranging in size from a tenth of an inch to very fine dust. The gangue minerals are pyrite, calcite, mica, pyroxene, and other silicates.

Sampling and Analysis.—The whole carload was crushed to about 1-inch size in a jaw crusher and a sample cut out by means of a Vezin sampler. This sample was run through the sampler and from the second sample a sample for assay was obtained which gave the following:

Carbon (C)	15·58	per cent
Silica (SiO_2)	23·78	"
Lime (CaO)	16·29	"
Magnesia (MgO)	1·20	"
Iron (Fe)	6·30	"
Sulphur (S)	1·70	"

A 1,000-gramme sample was obtained from the head sample crushed to 1-inch size for the purpose of examination of the flake as to its size and relation to the gangue constituents of the ore. This sample was treated with acids, including hydrofluoric, and the gangue minerals dissolved out as far as possible leaving the graphite flake in the size and condition as it occurred in the ore. A screen analysis of the flake obtained in this manner gave the following results:

Flake size	Weight, grammes	Per cent of total flake	Cumulative per cent	Carbon, per cent
− 8 + 10	1·3	0·8	0·8	89·60
− 10 + 14	4·8	2·9	3·7	94·36
− 14 + 20	17·3	10·6	14·3	98·04
− 20 + 28	23·4	14·4	28·7	97·68
− 28 + 35	30·1	18·5	47·2	89·04
− 35 + 48	25·4	15·6	62·8	95·88
− 48 + 65	19·3	11·8	74·6	91·68
− 65 +100	17·6	10·8	85·4	89·84
−100 +150	8·5	5·2	90·6	87·14
−150 +200	5·2	3·2	93·8	84·72
−200	10·1	6·2	....	85·92

This test shows the various sizes of flake present in the ore in its natural state, for instance, 74·6 per cent of the flake is retained on 65 mesh or is above 0·208 millimetre in size.

A microscopic examination of the various flake sizes shows the larger flake to be thick, irregular lumps, bent, warped, and pitted where gangue mineral has been dissolved out. The medium flake does not show this structure to the same extent. A greater proportion is flat firm flake the surfaces of which have a high lustre. In the finer sizes the flake is mostly flat, thin, and fragile.

EXPERIMENTAL TESTS

Test No. 1

Flotation

A lot of 5,200 pounds of ore, which was crushed to 1-inch size in the jaw crusher, was ground and the graphite concentrated following the flow-sheet given in Figure 3.

No reliable figures could be obtained from this test as too much of the feed went to fill up the circuit. It served, however, as a useful guide for the succeeding tests.

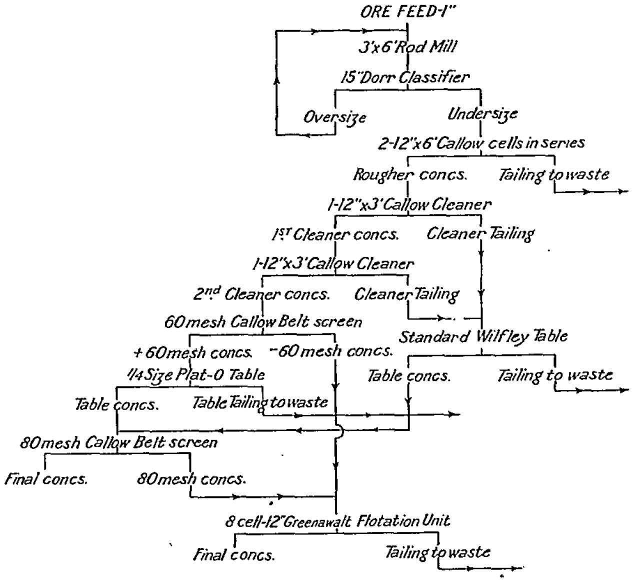

Figure 3. Flow-sheet, Test No. 1, flotation of Buckingham graphite ore.

Test No. 2

Flotation

A lot of 5,426 pounds of ore, which was crushed to 1-inch size in the jaw crusher, was ground and the graphite concentrated in a similar manner to Test No. 1, the only changes being that the first cleaner tailing was returned to the rougher cells; the second cleaner tailing was tabled on a Plat-O concentrator and the +60-mesh concentrates from the Callow belt screen was tabled on a Wilfley concentrator. The flow-sheet in Figure 4 was followed.

The results from this test were as follows:

Product	Weight, pounds	Carbon, per cent	Carbon content, pounds	Carbon, per cent of values
Ore	5,426·0	15·58	845·37	100·00
Flotation concentrates tabled and screened	444·5	87·05	386·95	45·77
Greenawalt flotation concentrates	735·5	61·09	449·30	53·15

This gives a total recovery in concentrates of 98·92 per cent. This recovery is high due to more ore being left in the mill circuit at the end of Test No. 1 than at the end of Test No. 2.

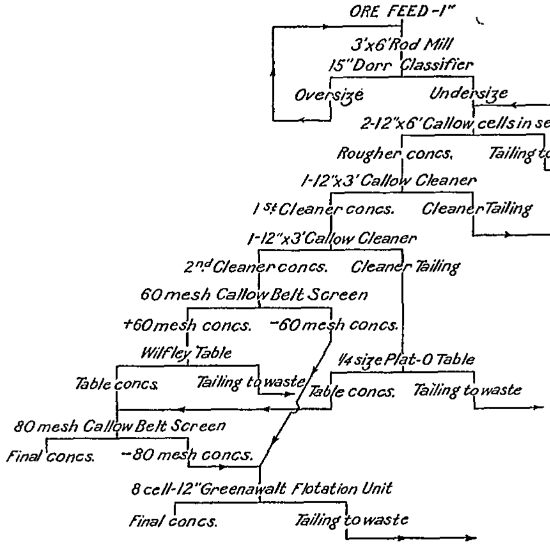

Figure 4. Flow-sheet, Test No. 2, flotation of Buckingham graphite ore.

A screen analysis of the two concentrates showed that 35·04 per cent of the total concentrates remained on a 65-mesh Tyler standard screen and that the +65-mesh concentrates analysed 88·77 per cent carbon.

A screen analysis of the Callow flotation tailing showed that considerable coarse flake was not being floated. The ore was not being ground fine enough for flotation of the very coarse flake. To recover this coarse flake, without unduly grinding it, some other means of concentration is necessary before flotation.

Test No. 3

Tabling and Flotation

This test was run for the purpose of determining whether the coarse flake could be recovered by concentration on tables before flotation. A lot of 11,241 pounds, which was crushed to one-inch size, was further reduced in rolls to pass a 10-mesh screen. It was then screened on 30 mesh. The −10+30-mesh size was tabled and the table tailing and −30-mesh size mixed and fed to the rod mill. The rod mill discharge was sized on a 26-mesh Hummer screen, the −26-mesh size going to flotation and the +26-mesh size to a table, the table tailing being returned to the mill for further grinding. The flow-sheet given in Figure 5 was followed.

The results from this test were as follows:—

—	Weight, pounds	Carbon, per cent	Carbon content, pounds	Carbon, per cent of values
Ore	11,241	15·88	1,784·60	100·00
+30 table concentrates	89	82·56	73·48	4·11
+26 table concentrates	250	91·72	229·31	12·85
Flotation Callow concentrates	477	88·36	421·48	23·62
Flotation Greenawalt concentrates	751	85·45	641·73	35·96
Total concentrates	1,567	87·17	1,366·00	76·54

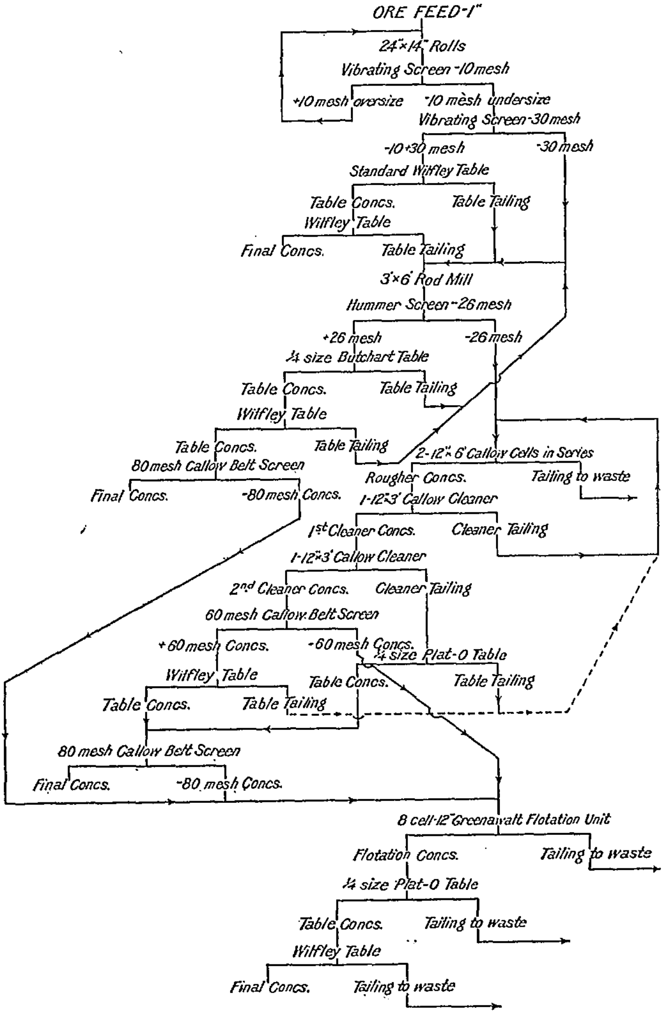

Figure 5. Flow-sheet, Test No. 3, tabling and flotation of Buckingham graphite ore.

Of the total concentrates obtained a screen analysis showed that 48·76 per cent is retained on a 65-mesh Tyler standard screen. The assay of the +65-mesh concentrates was 89·74 per cent carbon.

Conclusions from Results of Test No. 3.—The results given in the above table show very clearly that tabling out the coarse flake from the −10+30-mesh size produced by screening the ore after dry crushing in rolls to 10 mesh, is not an economic operation. Only 89 pounds of concentrates was obtained of a grade 82·56 per cent carbon representing 4·11 per cent of the carbon content of the total ore feed, or 12 per cent of the carbon content of the feed to the table which was 31·6 per cent of the total ore feed. A comparison of the results of the test with the preceding one proves conclusively that a system of closed circuiting the rod mill with a Hummer screen and tabling the oversize, returning the table tailing to the mill, will save a large amount of coarse flake from destruction.

The amount of flake obtained by tabling the Hummer screen oversize was 15·95 per cent of the total concentrates obtained. It analysed 91·72 per cent carbon, and 84·4 per cent was retained on a 65-mesh Tyler standard screen. Of the total concentrates obtained 48·76 per cent was retained on a 65-mesh Tyler standard screen as compared with 35·04 per cent in Test No. 2.

Test No. 4

Tabling and Flotation

From the results obtained in the preceding tests, Test No. 4 was run as being the most feasible and economic method of concentration to recover the coarse flake with least destruction and at the same time give high-grade products, good recoveries, and simplicity of operation. A lot of 12,802 pounds of the ore, one-inch size, was ground in the rod mill, which had in circuit with it a 26-mesh Hummer screen, and a table for concentrating the coarse graphite flake from the screen oversize and returning the table tailings to the mill. The −26-mesh material from the Hummer screen was floated. The flow-sheet given in Figure 6 was followed.

The results from this test were as follows:—

—	Weight, pounds	Carbon, per cent	Carbon content, pounds	Carbon, per cent of values
Ore	12,802	15·58	1,994·55	100·00
+26 table concentrates	416	87·04	362·07	18·15
Flotation Callow concentrates	657	84·77	556·91	27·92
Flotation Greenawalt concentrates	1,015	83·18	844·30	42·33
Total concentrates	2,088	84·45	1,763·28	88·40

Of the total concentrates obtained, a screen analysis showed that 48·94 per cent is retained on a 65-mesh Tyler standard screen and the assay of the +65-mesh flake was 87·04 per cent carbon.

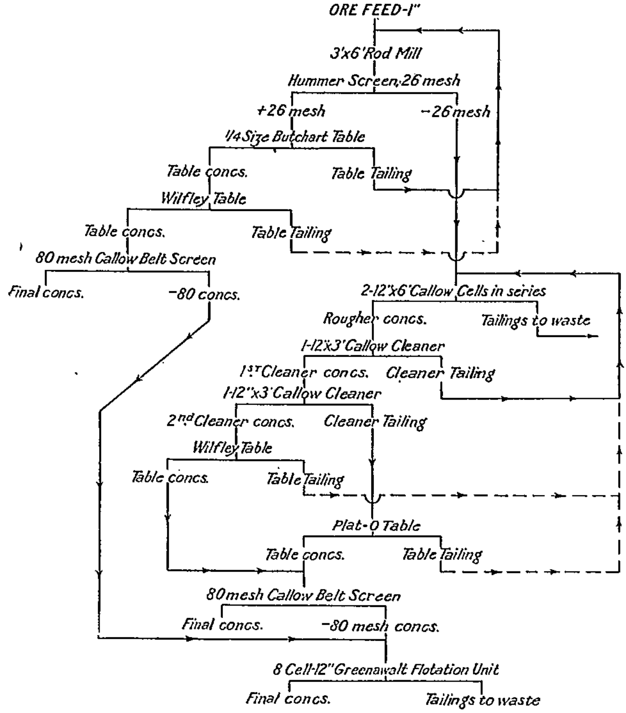

Figure 6. Flow-sheet, Test No. 4, tabling and flotation of Buckingham graphite ore.

Conclusions from Results of Test No. 4.—This test gave very good results. The concentrates are of good grade and contain a good proportion of coarse flake of No. 1 or crucible stock.

SUMMARY OF RESULTS AND CONCLUSIONS FROM EXPERIMENTAL TESTS

Test No.	Method	Recovery, per cent	Concentrates carbon, per cent	Concentrates +65 mesh, per cent	Concentrates +65 mesh, carbon, per cent
2	Flotation..........................	98·92	70·87	35·04	88·77
3	Tabling and flotation..............	76·54	87·17	48·76	89·74
4	" "	88·40	84·45	48·94	87·04

Tests Nos. 3 and 4 give the better results, in that almost 50 per cent of the flake recovered was retained on a 65-mesh Tyler standard screen. They show the least destruction of coarse flake. The flow-sheet followed in Test No. 4 was much simpler than in Test No. 3, and gave possibly slightly better results. These tests show that in order to recover the maximum amount of coarse flake, tabling operations are necessary to remove the coarse flake as soon as it is released on grinding and before it is fine enough for flotation.

The recovery figures given above are too high in the case of Test No. 2, and possibly too low for Tests Nos. 3 and 4. The high figure for Test No. 2 is due to graphite from Test No. 1 being left in the mill circuits. In Tests Nos. 3 and 4 tailing products containing considerable graphite were not returned as they would be in practice, hence the low recoveries.

Preparation of Crucible Flake

The total concentrate made from Tests Nos. 1, 2, 3, and 4 were mixed and screened on a Hummer screen fitted with a 60-mesh screen. The mesh opening of this screen corresponded with a 65-mesh Tyler standard. The following products were obtained:

—	+60-mesh product	−60-mesh product
	No. 1 or crucible flake	Fines
Weight	2,470·0 pounds	2,836·0 pounds
Per cent weight	46·55	53·45
Analysis	89·93 per cent C.	71·87 per cent C.

A screen analysis of the +60-mesh, or No. 1 crucible flake, gave the following:

Tyler standard screens	Weight, per cent	Assay, per cent carbon
+ 35 mesh	19·26	88·84
− 35 + 48 "	34·87	90·42
− 48 + 65 "	37·77	88·18
− 65 +100 "	8·00	88·13
−100 "	0·10	66·75

A screen analysis of the −60-mesh, or fine flake, gave the following:

Tyler standard screens	Weight, per cent	Assay, per cent carbon
+ 48 mesh	0·37	87·24
− 48 + 65 "	0·28	81·36
− 65 +100 "	23·38	80·29
−100 +150 "	18·29	71·80
−150 +200 "	18·41	70·21
−200 "	34·27	70·73

Report No. 315

THE CONCENTRATION OF A GRAPHITE ORE FROM GUENETTE, QUEBEC

R. K. Carnochan and C. S. Parsons

Shipment.—A carload of graphite ore, weight 28,000 pounds, was received August 4, 1927, from the Canadian Graphite Corporation, Guenette, Que. The shipment was arranged for by H. S. Spence of the Mineral Resources Division.

Purpose of Experimental Tests.—The purpose of the concentration tests was primarily to produce a No. 1 high-grade flake graphite from the ore with which to determine its suitability for the manufacture of crucibles, and secondly to determine an economic method of concentration that would be least destructive of the flake and give the largest percentage of No. 1 flake compatible with good recoveries. This report deals mainly with the concentration of the ore.

Arrangements for Experimental Work on Concentration.—Arrangements were made whereby H. S. Spence of the Mineral Resources Division would co-operate with R. K. Carnochan, in charge of the test work on non-metallics, and C. S. Parsons, in charge of the ore dressing test work on metallic ores, who had considerable experience both in the laboratory and in industrial plants on the concentration of graphite. H. C. Mabee and B. P. Coyne were to conduct the chemical tests and analyses.

Characteristics of the Ore.—The graphite occurs in the ore in small flakes ranging in size from a tenth of an inch to very fine dust. The flake is not so large as the Buckingham flake, nor is it so hard a flake. The gangue is siliceous.

Sampling and Analysis.—The whole carload was crushed to about 1-inch size in a jaw crusher and set on rolls, and a sample obtained by means of a Vezin sampler. The sample obtained was crushed to about ¼-inch size and sampled down in a Jones riffle sampler. By successive crushing and sampling, samples for analysis were obtained. The analysis showed the shipment to contain 16·34 per cent carbon.

EXPERIMENTAL TESTS

In previous work on a shipment of Buckingham graphite ore, several methods of concentration were tried out. The flow-sheet which gave the best results was followed on this ore. One test was made using a ball mill for grinding and the other using a rod mill with the same grinding charge, to determine which method of grinding was least destructive of the flake.

Test No. 1

Ball Mill Grinding

A lot of 16,972 pounds of ore, which was crushed to 1-inch size, was ground and the graphite concentrated, following the flow-sheet given in Figure 7.

Flotation Reagents Used per Ton of Ore.—

To ball mill	0·39 pound of pine oil;	0·35 pound of coal oil.
To Callow cells		0·02 " "
To Wilfley table		0·08 " "
To Greenawalt cells	0·11 pound of pine oil;	0·08 " "
Total	0·50 " "	0·53 " "

Feed Rate: 1,237 pounds of ore per hour.
Pulp Density: Ball mill discharge—46 per cent solids.

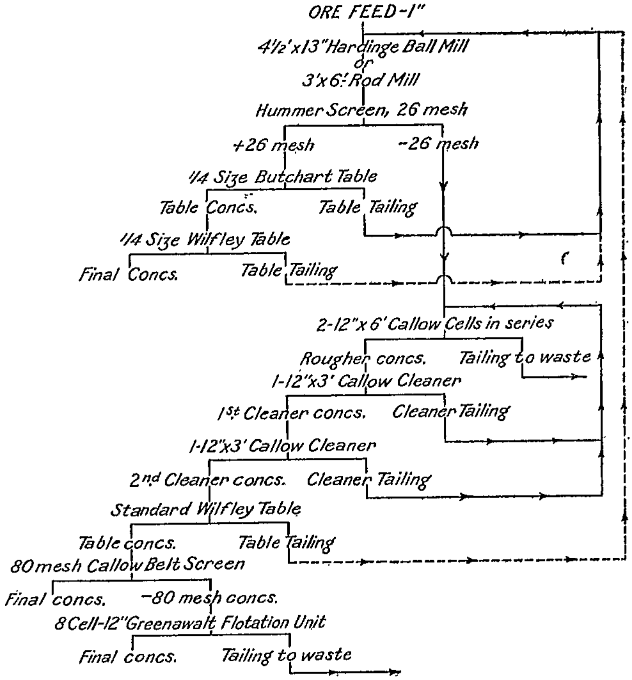

Figure 7. Flow-sheet, Tests Nos. 1 and 2, tabling and flotation of Guenette graphite ore.

Results from Test No. 1

Ore Feed.—

Weight	16,972·0 pounds
Assay	16·34 per cent C.
Content	2,773·22 pounds C.

Table Concentrates.—

Weight	320·0 pounds
Assay	54·47 per cent C.
Content	174·29 pounds C.

49·12 per cent of this table concentrate is retained on 65-mesh and assays 82·98 per cent C.

Results from Test No. 1—Concluded

Callow Flotation.—

Concentrate—

Weight	849·5 pounds
Assay	85·57 per cent C.
Content	726·89 pounds C.

61·20 per cent of this flotation concentrate is retained on 65-mesh and assays 88·04 per cent C.

Greenawalt Flotation.—

Concentrate—

Weight	1,244·0 pounds
Assay	84·08 per cent C.
Content	1,045·98 pounds C.

5·00 per cent of this flotation concentrate is retained on 65-mesh and assays 90·36 per cent C.

Total Concentrates.—

Weight	2,413·5 pounds
Assay	80·68 per cent C.
Content	1,947·16 pounds C.

30·63 per cent of this total concentrate is retained on 65-mesh and assays 87·16 per cent C.

Recovery of carbon values........................70·21 per cent.

This is the actual recovery made for the test but is not a true indication of the recovery which would be obtained in actual mill practice as it does not take into account the recovery that would be made from some fairly high-grade tailing products which would be returned to the circuit in milling practice.

The Callow flotation tailing assayed 3·18 per cent carbon. A screen test of this tailing showed 1·97 per cent retained on a 48-mesh screen, assaying 15·97 per cent carbon. This small amount of +48-mesh tailing showed that the large flake was floated. The screen test also showed that the grinding was 54·48 per cent through 200 mesh. The Greenawalt flotation tailing assayed 18·67 per cent carbon. The Wilfley table tailing assayed 10·35 per cent carbon. This tailing would be returned to the circuit in mill practice and would have raised the total recovery considerably and also given a greater amount of +65-mesh coarse flake.

Test No. 2

Rod Mill Grinding

A lot of 10,791 pounds of ore, which was crushed to 1-inch size, was ground and the graphite concentrated in like manner to Test No. 1, with the exception that a 3- by 6-foot Marcy rod mill was used for the grinding instead of a 4½-foot by 13-inch Hardinge ball mill. The rod charge was 2,500 pounds, the same as the ball charge in the Hardinge ball mill. The flow-sheet as given was followed.

Flotation Reagents Used per Ton of Ore.—

To rod mill	0·35 pound of pine oil;	0·34 pound of coal oil.
To Callow cells		0·03 " "
To Wilfley table		0·13 " "
To Greenawalt cells	0·11 pound of pine oil;	0·08 " "
Total	0·46 " "	0·58 " "

Feed Rate: 1,387 pounds of ore per hour.
Pulp Density: Rod mill discharge—47 per cent solids.

Results from Test No. 2

Ore Feed.—

Weight	10,791·0 pounds
Assay	16·34 per cent C.
Content	1,763·25 pounds C.

Table Concentrates.—

Weight	195·5 pounds
Assay	60·49 per cent C.
Content	118·26 pounds C.

58·66 per cent of this table concentrate is retained on 65-mesh and assays 83·62 per cent C.

Callow Flotation.—

Concentrate—

Weight	506·0 pounds
Assay	86·03 per cent C.
Content	435·29 pounds C.

69·85 per cent of this flotation concentrate is retained on 65-mesh and assays 91·31 per cent C.

Greenawalt Flotation.—

Concentrate—

Weight	791·0 pounds
Assay	87·56 per cent C.
Content	692·57 pounds C.

9·70 per cent of this flotation concentrate is retained on 65-mesh and assays 86·21 per cent C.

Total Concentrates.—

Weight	1,492·5 pounds
Assay	83·49 per cent C.
Content	1,246·12 pounds C.

36·51 of this total concentrate is retained on 65-mesh and assays 89·00 per cent C.

Recovery of carbon values........................70·67 per cent.

This is the actual recovery made for the test but is not a true indication of the recovery which would be obtained in actual mill practice. It does not take into account the recovery that would be made from the return to the circuit of the tailing products containing fairly high carbon content which would be the case in milling practice.

The Callow flotation tailing assayed 2·69 per cent carbon. A screen test of this tailing showed 1·54 per cent retained on a 48-mesh screen assaying 17·81 per cent carbon, which indicates that a very small amount of large flake did not float in the Callow cells. The screen test also showed that the grinding was 48·11 per cent through 200 mesh.

The Greenawalt flotation tailing assayed 44·15 per cent carbon. The Wilfley table tailing assayed 11·20 per cent carbon. In practice this tailing would be returned to the rod mill or preferably to a separate grinding and flotation circuit as it contains most of the sulphide and coarse rock material which might build up in the main circuit, making it more difficult to produce high-grade concentrate.

Comparison of Results—Tests Nos. 1 and 2

	Test No. 1, ball mill grinding	Test No. 2, rod mill grinding
	per cent	per cent
Recovery	70·21	70·67
Grade of concentrates	80·68	83·49
Concentrates, +65-mesh	30·63	36·51
Grade of +65-mesh flake	87·16	89·00

The above comparison of results shows that rod mill grinding is preferable to ball mill grinding for graphite ore, in that it is least destructive of the flake. In Test No. 2, 36·51 per cent of the graphite flake recovered, is in +65-mesh concentrates as compared with 30·63 per cent in Test No. 1. This point is further brought out by comparing the amount of carbon on the various screen meshes recovered from a ton of ore.

Cumulative Pounds of Carbon per Ton of Ore in Concentrates

Screen mesh	Test No. 1, ball mill grinding	Test No. 2, rod mill grinding
+28	6·90	1·55
+35	11·60	20·08
+48	36·74	43·99
+65	75·93	89·87
−65	153·53	141·09
Total	229·46	230·96

Retreatment of Table Concentrates from Test Nos. 1 and 2

The table concentrates +26-mesh from Test No. 1 assayed 54·47 per cent carbon and from Test No. 2, 60·49 per cent. To raise the grade of these concentrates they were combined and tabled. This operation gave the following concentrates:

Weight	220·0 pounds
Assay	91·06 per cent C.
Content	200·34 pounds C.

A screen test showed 94·30 per cent on 65 mesh, which assayed 92·54 per cent C.

The figures given under Tests Nos. 1 and 2 as to recovery, grade of concentrates, etc., would be changed by this operation. The final figures for Tests Nos. 1 and 2 combined would be:

	Per cent
Recovery	68·36
Grade of concentrates	85·89
Concentrates +65-mesh	33·78
Grade of +65-mesh flake	89·77

Recoveries.—The recoveries of about 70 per cent of the graphite content in the ore, made in these tests is low, as no account is taken of the graphite content in high tailing products which would be returned to the mill circuit in actual practice. Further, there is a loss in the mill circuit on the comparatively small amount of ore treated that would lower the recoveries. The greatest loss of coarse flake would be, however, in the table tailing which was not returned. It is doubtful if a very much higher recovery of +65-mesh flake could be expected than that indicated in these tests. The greatest loss would be in fine flake.

Preparation of Crucible Flake.—The total concentrates obtained were mixed and screened on a Hummer screen fitted with a 60-mesh screen. The mesh opening of this screen corresponded with a 65-mesh Tyler standard. The following products were obtained:

+60-mesh flake	1,449 pounds;	assay...... 89·72 per cent C.
−60-mesh flake	2,072 "	assay....... 80·05 "

A screen analysis of the +60-mesh, or No. 1 crucible flake, gave the following:

Tyler standard screen	Weight, per cent	Assay, per cent carbon
Mesh		
+ 28	1·55	89·90
− 28+ 35	10·40	92·65
− 35+ 48	31·15	89·68
− 48+ 65	42·46	89·05
− 65+100	14·11	89·75
−100+150	0·13	85·90
−150+200	0·10	73·90
−200	0·10	54·95

A screen analysis of the −60-mesh, or fine flake, gave the following:

Tyler standard screen	Weight, per cent	Assay, per cent carbon
Mesh		
+ 65	0·23	78·25
− 65+100	23·43	85·25
−100+150	24·93	85·65
−150+200	12·28	84·40
−200	39·13	74·75

Report No. 316

CONCENTRATION OF GARNET ORE FROM LABELLE, QUEBEC

R. K. Carnochan

Shipment.—A shipment of 10,363 pounds of garnet ore was received October 21, 1927, from the Labelle Nickel and Garnet Company, Ltd., Montreal. The ore was from the company's property in Joly township, Labelle county, Quebec.

Characteristics of the Ore.—The garnet crystals range in size from about three inches to very small grains and are of two different kinds: one, a dark red almost black garnet, glassy-like in appearance without fractures; and the other medium red showing fracturing. The gangue minerals are quartz, white feldspar, pyrrhotite, and a little mica and hornblende.

Purpose of Experimental Tests.—Test work was desired to determine a method for the concentration of the garnet into a high-grade product, the percentage of garnet that could be recovered from the rock and the quality of the garnet concentrates. It was also desired that clean garnet concentrates be prepared so that samples could be submitted to the garnet paper manufacturers.

Sampling and Analysis.—The shipment was crushed in a jaw crusher and rolls to about one-inch size. A sample cut of 1,070 pounds was obtained. This was further cut to 70 pounds from which a sample was obtained to determine the presence of base and precious metals which the ore was supposed to contain. This analysis gave:—

Copper	0·03 per cent
Nickel	trace
Gold	none
Silver	0·03 oz./ton

This shows that these metals are not present in commercial quantities and that the only economic value is in the garnet content.

EXPERIMENTAL TESTS

Test No. 1

The remainder of the 70-pound sample, which was crushed to pass 4 mesh, was sized on 8-, 14-, 28-, 48-, and 100-mesh screens. The −4+8 and −8+14 were jigged and the other sizes were tabled. As the jig and table concentrates contained a large amount of pyrrhotite they were run through an Ullrich magnetic separator using a weak magnetic field to lift the pyrrhotite and leave the garnet. The following table gives the weights of the products from this test:—

Size	Feed to table or jig		Table or jig concentrate		Table or jig tailing		Ullrich products			
							Pyrrhotite		Garnet	
	lb.	oz.	lb.	oz.	lb.	oz.	lb.	oz.	lb.	oz.
− 4 + 8	8	4¼	3	11	4	9¼	1	1	2	9
− 8 + 14	18	14	11	¼	7	13¾	2	13	8	1
− 14 + 28	17	3½	11	3¼	5	15¾	2	8	9	0
− 28 + 48	9	13¼	6	4¼	3	9	1	10	4	14
− 48 +100	5	4¼	2	13	2	7¼	−	8	2	2
−100	5	14½	2	0	3	14½	−	7	1	6
Totals	65	5¾	37	0¼	28	5½	8	15	28	0

The garnet concentrates equal 41·21 per cent of the ore treated. Each of the sizes contains a small amount of gangue mostly in the form of middling. The pyrrhotite concentrates equal 13·16 per cent of the ore treated and consist almost entirely of pyrrhotite.

Test No. 2

As there was a small amount of gangue in the concentrates of Test No. 1, it was decided to size closer in Test No. 2, to see if this would give better concentrates. A 69-pound portion was cut out of the 1,070-pound sample by means of a Jones riffle and crushed in a small breaker and rolls to all pass 4 mesh. The −4 mesh was sized on 6-, 8-, 10-, 14-, 20-, 28-, 35-, 48-, 65-, and 100-mesh screens. The sizes down to −14+20 were jigged and the remaining sizes were tabled. The jig and table concentrates were run over the Ullrich separator to remove pyrrhotite. The following table gives the weights of the products of this test:—

Size	Feed to table or jig		Table or jig concentrate		Table or jig tailing		Ullrich products			
							Pyrrhotite		Garnet	
	lb.	oz.	lb.	oz.	lb.	oz.	lb.	oz.	lb.	oz.
− 4 + 6	17	8	7	15	9	10	2	14	4	14
− 6 + 8	12	8	4	13	7	14	1	9	3	4
− 8 + 10	8	0	3	10	4	3	1	0	2	10
− 10 + 14	5	4	2	8	2	7	-	13	1	10
− 14 + 20	5	0	2	0	2	10	-	10	1	6
− 20 + 28	4	10	2	13	1	10	-	12	2	0
− 28 + 35	4	2	2	5	1	12	-	11	1	9
− 35 + 48	2	11	1	6	1	0	-	6	1	1½
− 48 + 65	1	14½	-	14	-	15	-	4½	-	9¾
− 65 +100	1	3½	-	10	-	9	-	2¾	-	6¾
−100	4	5	-	14	3	0	-	½	-	10¾
Total	67	2	29	12	35	10	9	2¾	20	1¾

The garnet concentrates equal 29·14 per cent of the feed. The two coarsest sizes, −4+6 and −6+8, contain a small amount of gangue and the finer sizes contain only a very little gangue. The pyrrhotite concentrates equal 13·29 per cent of the ore treated and consist almost entirely of iron except the two coarsest sizes, −4+6 and −6+8, which contain quite a lot of garnet. This partly explains why the recovery in this test is lower than that in Test No. 1.

Test No. 3

After examining the products of the two previous tests it seemed that the proper way to concentrate this ore would be to crush to all −8-mesh size as was done in Test No. 2, and then jig and table and run the concentrates over the magnetic separator twice, first to remove iron and secondly to lift the garnet and leave the gangue. However, as the trade likes the garnet concentrate as coarse as possible and the finer crushing and closer sizing would be more expensive, it was decided to run a large test crushing to 4 mesh and sizing on 8, 14, and 28 mesh so as to be sure that this crushing and sizing would not be suitable.

A part of the one-inch material, weight 4,471 pounds was obtained by taking every alternate bag. This portion was crushed in 24- by 14-inch rolls and screened on a Ferraris shaking screen fitted with an 8- and a 4-mesh screen. The screen oversize was returned to the rolls until it all passed through the screens. The −8-mesh material was fed to the screen fitted with a 28- and a 14-mesh screen. These operations gave four products, −4+8, −8+14, −14+28, and −28 mesh. The first two sizes were jigged and the last two were tabled. The jig and table concentrates were run over the magnetic separator twice, first to remove the iron, and the second time to lift the garnet and leave gangue. The following table gives the data obtained by this test:—

Size	Feed to jig or table, pounds	Table or jig concentrate, pounds	Ullrich products		
			Pyrrhotite, pounds	Garnet, pounds	Gangue, pounds
− 4 + 8	1,951	1,145	293¼	828½	16
− 8 +14	887	553	160½	383	5¼
−14 +28	603	418¾	82	318	16
−28	918	521	155	348	9¼
Totals	4,359	2,637¾	690¾	1,877½	46½

The garnet concentrate equals 41·99 per cent of the feed. The concentrates −4+8 contain a large amount of gangue in the form of middlings. The other sizes contain only a small amount of gangue. The iron concentrates equal 15·45 per cent of the feed. The size −4+8 contains quite an amount of garnet and gangue in the form of middlings. The other sizes are fairly good iron products. This test shows that the ore cannot be concentrated as coarse as 4 mesh and also that the sizing −4+8, −8+14, −14+28, and −28 is not close enough.

Test No. 4

This test was run using every device to obtain as clean garnet concentrates as possible. A lot of ore, 5,496 pounds, which had been crushed to about 1 inch by means of a 12- by 8-inch jaw crusher and 24- by 14-inch rolls, was crushed to all pass 8 mesh by means of an 18-inch Symons disk crusher, 3- by 4-foot Newaygo vibrating screen, and 12- by 12-inch rolls, the rolls being used to crush the oversize from the screen. The −8 mesh was sized on 10, 14, 20, 26, 35, 42, and 60 by means of a 3- by 5-foot, Hummer vibrating screen. The sizes −8+10 and −10+14 were jigged in a two, 12- by 10-inch compartment James jig. The other sizes were tabled on a large Wilfley table. The jig and table concentrates were run twice over the Ullrich separator, first with a light current to remove iron, and the second time with a strong current to lift the garnet and to leave the gangue. The following table gives the data obtained by this test:

Size	Feed to table or jig, pounds	Table or jig concentrate, pounds	Table or jig tailing, pounds	Ullrich products		
				Pyrrhotite, pounds	Garnet, pounds	Gangue, pounds
− 8 +10	1,738	1,034	701	200	807	18
−10 +14	607	405	201	100	278½	10½
−14 +20	898¾	595	301	141	427	27
−20 +26	562½	356	190	98	236	16½
−26 +35	388	232	150	61	156	9
−35 +42	345	195	134	50	134	9½
−42 +60	274½	138	107	35	95	9½
−60	570	160	344	31½	107	19
Total	5,383¾	3,115	2,128	716½	2,240½	119

The garnet concentrates equal 40·77 per cent of the ore treated and all sizes are very good. The iron concentrates equal 13·04 per cent of the feed and all the sizes are fairly pure pyrrhotite. The gangue products from the magnetic separator equal 5·04 per cent of the feed to the separator the second time when the garnet was lifted and the gangue left.

CONCLUSIONS

1. Test No. 4 shows that the ore can be successfully treated and very good concentrates equal to 40·77 per cent of the feed recovered.

2. Tests Nos. 3 and 4 when compared show that the ore must be crushed to 8 mesh and then sized closely before jigging and tabling.

3. A magnetic machine must be used to remove pyrrhotite from the garnet concentrate. This machine would not need to be of high intensity.

4. It would be very desirable but not absolutely necessary to use a magnetic machine of high intensity to lift the garnet and leave the gangue. The amount of gangue dropped from the garnet concentrates in this operation is 5·04 per cent of the product fed to the machine.

5. The following flow-sheet is recommended for the treatment of the ore:—

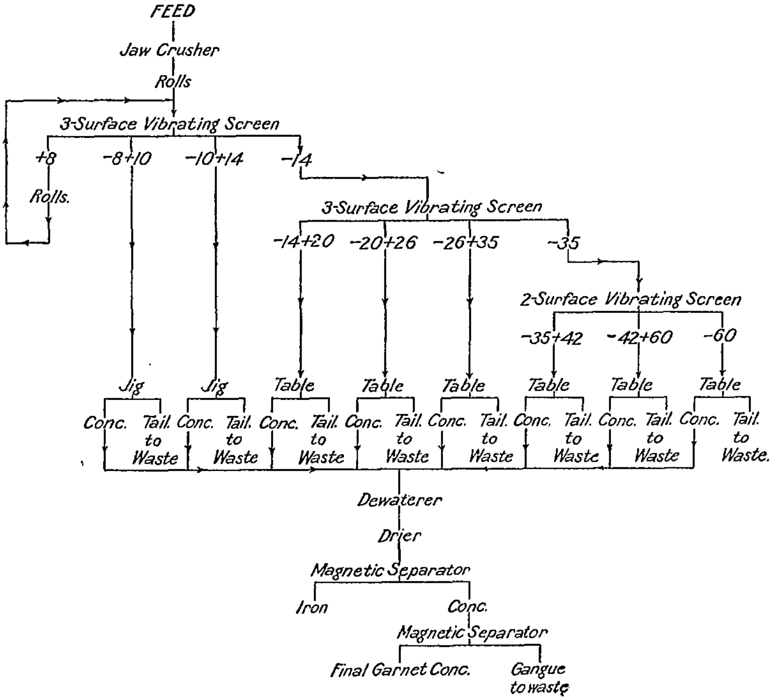

Figure 8. Flow-sheet, concentration of garnet ore from Labelle, Quebec.

THE TESTING OF GYPSUM AND GYPSUM PLASTERS

R. A. Rogers

Shipments.—During the latter part of the year 1927 and at various times during the year 1928, 61 samples of crude gypsum and 4 samples of calcined plaster were received. These samples were submitted by L. H. Cole, Mineral Resources Division, Mines Branch, and came from five provinces in Canada as follows:—

Nova Scotia	36 crude gypsum	
New Brunswick	12 " "	1 calcined gypsum
Ontario	4 " "	3 " "
Manitoba	2 " "	
British Columbia	7 " "	
	61 " "	4 " "

The average weight of the crude gypsum samples was 185 pounds, and of the gypsum plasters, 50 pounds.

Purpose of Tests.—The purpose of the tests was to find the suitability of the various gypsums for making plaster.

Methods of Testing

Washing and Drying.—As many of the samples were taken from surface deposits, some dirt and foreign material were included. The samples were, therefore, washed with water to remove these impurities. They were then dried at a low temperature.

Crushing and Grinding.—The gypsum as received was in lumps about 4 inches in diameter. It was first crushed to 1 inch in a No. 103 Austin gyratory crusher. This product was further reduced to ¼ inch in a No. 00 Sturtevant swing-sledge mill. The ¼-inch material was ground in a 24-inch Munson burr mill until 95 per cent passed through 100 mesh.

Sampling and Analysis.—The fine-ground gypsum was sampled with a Jones riffle. The samples were analysed by the Division of Chemistry. A complete table of the analyses will be found in the report "The Gypsum Industry of Canada" by L. H. Cole—Mines Branch, Rept. No. 714.

The average analysis of 61 samples was as follows:—

	Per cent
Insoluble	1·03
Ferric oxide (Fe_2O_3)	0·14
Alumina (Al_2O_3)	0·10
Lime (CaO)	32·67
Magnesia (MgO)	0·39
Sulphur trioxide (SO_3)	45·65
Water (H_2O)	19·01
Carbon dioxide (CO_2)	0·82
	99·81

Calcining

Kettle Employed.—A small-sized electrically heated kettle designed by L. H. Cole, Mining Engineer, Mineral Resources Division, and built in the machine shop of the Mines Branch, Booth Street, Ottawa, was employed to calcine the samples. It consists of a cylindrical chamber, 18 inches in diameter and 2 feet high with convex bottom. This chamber is made of $\frac{1}{4}$-inch boiler plate, with all joints electrically welded. This kettle is placed in a square heating box, which has four standard size oven elements bolted on the inside of each side and an 8-inch circular element on the inside of the bottom.

The heating box is surrounded by another square box of $\frac{3}{16}$-inch boiler plate which leaves 5-inch clearance on all sides and bottom between it and the heating chamber. This space is filled with diatomite for purposes of insulation. The three boxes are all the same height and are covered with a sheet of $\frac{1}{4}$-inch boiler plate having an opening the diameter of the inner kettle. A sheet of asbestos the same size is placed beneath this top plate. A sectional cover, lined on the underside with asbestos sheeting, covers the top of the calcining chamber when in operation. One section of this cover has a pipe leading from it to the outside of the building to allow the steam to escape.

Fifty pounds of ground gypsum was taken for each test. The kettle was filled when the temperature was about 150°F, and calcination carried to the first settle. The time required averaged about one hour and twenty minutes. The average temperature of the samples at the end of the calcination was 307°F.

The sample is agitated during calcination by a vertical shaft placed down the centre of the kettle and driven by worm gears from above, at 15 r.p.m. Two sets of paddles at right angles to each other are firmly bolted to this shaft.

The kettle is emptied through a 2-inch hole in the bottom, which is closed during calcination by a plug attached to a lever.

A more detailed description of this kettle and its operation is contained in the report previously referred to.

Testing of Calcined Plasters

Standard Methods.—The standard methods of testing gypsum and gypsum products used by the American Society for Testing Materials, Serial Designation C 26-27, were followed wherever practicable.

Water After Calcination.—After a sample was calcined, the remaining water was determined. The average percentage of water after calcination was 5·45.

Consistency, etc.—For each plaster, the following were determined: the testing consistency, the time of setting, the tensile strength, and the compressive strength.

A complete table of the results of the tests is contained in the report by L. H. Cole.

The average tensile strength of 63 samples was 390 pounds per square inch, and the average compressive strength, 2,021 pounds per square inch.

CONCLUSIONS

The results of these tests show that Canadian gypsums are suitable for the manufacture of all classes of structural materials having a gypsum base, and many of them are of a high degree of purity suitable for use in the manufacture of dental plaster, terra alba, pottery plaster, and the finest of finishing plasters.

It is probable that a greater range of plasters could be manufactured than are being prepared at present, by slight variations in the temperature of calcination, but this can only be determined by experiment. Our tests have shown in many cases that stronger plasters can be produced by higher calcination.

While many deposits are being worked at the present time in Canada, these tests show that there are many other deposits in which the material is equally as good. Provided transportation facilities are favourable, these can be operated commercially when market conditions warrant their exploitation.

IV

REPORT OF INVESTIGATIONS: HYDROMETALLURGICAL LABORATORY

R. J. Traill

The investigations carried out in the hydrometallurgical laboratory during the past year comprised: (1) the hydrometallurgical treatment of high-grade iron-copper concentrates, (2) preliminary test work on beneficiation and reduction of high-sulphur iron ore, and (3) metallization of the iron content in ilmenite by means of the rotary-kiln type of furnace.

For the test work on hydrometallurgical treatment of chalcopyrite concentrates, a concentrate obtained from ore of the Amulet mine, Rouyn, Quebec, was used.

The investigations made included, the reactivity of the ore concentrate to ferric chloride leaching, the separation and recovery of sulphur, the separation of copper, and the separation or elimination of lead from the leach liquor, the latter a necessary step in the preparation of a satisfactory electrolyte for producing high-grade iron.

The results on the whole, as may be observed from the report of tests, have proved highly satisfactory.

The elimination of lead from the ferrous liquor has, in the past, been a difficult problem, but the latest results are quite encouraging.

Further tests on electro-deposition are shown, particularly with regard to the effect of lead salts in the electrolyte on the deposited iron.

The laboratory during the last few years has been investigating the possibilities of a hydrometallurgical method of treating heavy sulphide ores of the pyrrhotite, pyrite, and chalcopyrite types, based on the extraction of the base metals by leaching with ferric chloride with the object of their ultimate recovery as metals, and the recovery of sulphur content as elemental sulphur.

Much data have been collected, many problems have been overcome, and encouraging results have been obtained.

The hydrometallurgical treatment of these ores, as described in previous reports of the laboratory, has many attractive features in its favour, judged from a laboratory scale basis, and sufficient data, it is thought, have now been obtained to warrant further investigation on a semi-commercial scale basis, to determine its adaptability as a continuous commercial process with economic possibilities. Such a scheme is quite outside the scope of this laboratory at present, and must be left to other interests.

In connexion with some proposed investigations into the beneficiation and reduction treatment of Canadian iron ores, experimental work of a preliminary nature was undertaken on a high-sulphur iron ore. The report shows the results of oxidizing roasting in the Herreshoff and rotary-kiln types of furnaces and by sintering in sinter pans. This work has not progressed to a sufficient degree to warrant any definite conclusions, and the results of the tests are merely set forth as a matter of record. The investigation will be continued.

Reports of 1925 and 1926 describe a process and show results of tests on a method for the treatment of ilmenite ore, whereby the iron and titanium contents can be separated and recovered as marketable products.

The present report describes tests on the metallizing aspect of the above process, a rotary kiln of semi-commercial proportions being utilized and operated in a manner comparable to commercial practice.

The results show the satisfactory application of the rotary kiln for this purpose, and the process should become a commercial possibility in the near future.

THE HYDROMETALLURGICAL TREATMENT OF HIGH-GRADE IRON-COPPER SULPHIDE CONCENTRATES

R. J. Traill, W. R. McClelland, and J. D. Johnston

The tests to be described were made on flotation concentrates obtained from tests on Amulet ore. The ore is a chalcopyrite, containing pyrite, pyrrhotite, and sphalerite with small quantities of galena and with gold and silver values. By selective flotation a high-grade copper concentrate of the following composition was obtained and used for the purpose of the tests here described:—

Iron	26·53 per cent
Copper	20·01 "
Zinc	9·13 "
Lead	0·67 "
Sulphur	30·04 "
Silver	11·22 oz./ton
Gold	0·80 "

Purpose of Tests.—The purpose of the tests was to determine which method, namely direct leaching, or roasting prior to leaching with ferric-ferrous chloride, would give the higher extractions of the contained metals.

Also, it was sought to determine how the gold and silver reacted, where they would be found, and if they were readily recoverable.

Further experiments on the problem of the lead elimination from the electrolyte liquor were carried out, and finally electro-deposition tests were made from electrolytes containing lead and electrolytes comparatively free from lead, to determine the effect of such impurity on iron deposition.

EXPERIMENTAL TESTS

Series F-1: Tests Nos. 1 and 2

In Table I the results by the method of roasting prior to leaching are shown. The procedure was briefly as follows. The concentrates were mixed with sulphur and roasted under non-oxidizing conditions for 3 hours at a temperature around 725°C. The roasted product was next leached with ferric-ferrous chloride liquor at 90° to 95°C. for 5½ hours and filtered. The leaching charge was about 15 per cent in excess of the theoretical requirement. The filtered leach liquor was treated with sponge iron to precipitate the copper. The copper precipitate was given an oxidizing roast, leached with sulphuric acid, and the residue assayed for gold and silver values. The leached material was dried, the free sulphur volatilized by roasting, and the roasted residue examined for gold, silver, copper, iron, zinc, and sulphur values.

Extractions of the heavy base metals were thus calculated on the difference between the metal contents of this final residue and the original charge.

With regard to the gold and silver values, the table shows that, in the respective tests, 85 and 90 per cent of the gold remains in the final residue with a little copper and zinc; a small percentage is obtained in the treatment of the copper precipitate, and 93·2 and 97·18 per cent of the gold are accounted for in the respective tests.

The silver is apparently quite soluble in the ferric-ferrous leaching liquor, the results showing 91·4 and 87·0 per cent as being present in copper residue with only 6·4 and 3·77 per cent retained in the residue and 98 and 90·7 per cent being accounted for in the respective tests. While the results do not check as well as might be desired, the general trend of the reactions is quite clearly shown in these tests.

TABLE I

Series F-1: Tests Nos. 1 and 2—Amulet Ore

Test No.	Charge to roast, grms.	Weight of roast, grms.	Loss in weight, per cent	Sulphur recovered, grms.	Charge to leach, grms.	Leaching liquor		Time, hours	Weight of leached residue, grms.	Per cent dissolved	Sponge iron added, grms.	Cement-copper residue, grms.	Residue after roast and H_2SO_4 leach	Charge to sulphur distillation roast, grms.	Weight roast residue, grms.	Loss in weight, per cent	Sulphur re-covered, grms	Analysis of roast		
						Fe''' grm. /l.	Fe'' grm. /l.											Fe, per cent	Cu, per cent	Zn, per cent
1	342 concentrate.................. 68 sulphur.	340	17	46	340	77	71	5½	187	45	140	169	84	187	116	38	60	34·73	2·93	3·67
2	342 concentrate.................. 68 sulphur.	331	19	48	331	77	71	5	159	52	130	147	59	159	76	52·2	90	30·52	3·35	4·96

Test No.	Extractions			Sulphur re-covered, grms.	Sulphur re-covery, per cent	Analysis ignited sulphur residue		Gold (in milligrams)							Silver (in milligrams)						
	Fe, per cent	Cu, per cent	Zn, per cent			Au	Ag	Head	Final residue	Per cent in residue	In cement-copper residue	Per cent	Gold accounted for	Per cent	Head	Final residue	Per cent	In cement-copper residue	Per cent	Silver accounted for	Per cent
1............	55·5	95·0	83·3	112	65·5	nil	trace	9·38	7·994	85·2	0·749	8·0	8·743	93·2	131·59	8·71	6·4	120·27	91·4	128·98	98·0
2............	74·4	96·2	90	138	80·9	nil	trace	9·38	8·476	90·3	0·640	6·88	9·116	97·18	131·59	6·78	3·77	113·6	87·0	119·38	90·7

Series F-3: Tests Nos. 2 and 3

In Table II results of tests by the method of direct leaching are shown. The same charge of ore concentrate and the same leaching liquor was used as in the above tests, the time of leaching was necessarily increased on account of the raw concentrates being less readily soluble than the roasted concentrate. The procedure was identical to that followed in the treatment of the roast product as described above. The extractions obtained in this test vary slightly from those obtained in Tests Nos. 1 and 2, inasmuch as the copper is slightly lower and the zinc slightly higher. This is also reflected in the final residue after sulphur distillation, the copper content in the residue being slightly higher and the zinc content lower in comparison with Tests Nos. 1 and 2.

Also it will be observed from Table II, that in the case of the precious metals, higher values are obtained in the residues and lower values in the copper precipitate, with 94 and 98 per cent of the gold and 99·5 and 98·2 per cent of the silver being accounted for.

A comparison of these two methods shows a slight advantage in favour of roasting prior to leaching in extractions obtained, but it is doubtful if the slight advantage gained in extraction would warrant the economic adoption of this method.

An important phase of the suggested process is the recovery of sulphur. In both methods the leached residue must be given a distilling roast to obtain the sulphur, and the roast product so obtained is of a similar composition in both cases. In the Series F-1 method, as shown in Table I, the whole ore charge is roasted and the leach residue is subsequently treated by distillation roast. This means practically two roasts, whereas in the Series F-3 method, only one roast is required, namely the distillation roast for recovery of sulphur. It would seem, therefore, that a more economical procedure would be to submit the ore to direct leaching, recover the sulphur in the leached residue by distillation roast and add the roast product so obtained to a subsequent ore charge for direct leaching. The distillation roast product is quite readily soluble in the ferric-ferrous leach liquor. To obtain maximum extraction of the desired metals this procedure of re-treating the residual product from sulphur distillation would be imperative in either case, and it is quite evident then that the direct leaching method would be the more economical in the treating of this ore concentrate.

Following this procedure, it is possible to obtain an extremely high extraction of the iron, copper, and other base metals, and the precious metal values are at the same time being concentrated in the leach residue to the maximum extent. The following tests illustrate quite definitely the results that might be obtained in carrying out this procedure.

TABLE II

Series F-3: Tests 2 and 3—Amulet Ore

Test	Charge raw concentrate, grms.	Leaching liquor		Time, hours	Weight leached residue, grms.	Per cent dissolved	Sponge iron added, grms.	Cement-copper residue, grms.	Residue after two roasts and two H_2SO_4 leaches, grms.	Charge to sulphur distillation roast, grms.	Weight of roast residue, grms.	Loss in weight, per cent	Sulphur recovered, grms.	Analysis of roast				
		Fe'' grm./l.	Fe''' grm./l.											Fe, per cent	Cu, per cent	Zn, per cent	Au, mgs.	Ag, mgs.
2...........	342	70·85	77·1	7⅚	163·0	52·3	125	127	46	163·0	78	52·1	73	30·74	9·98	3·03	8·658	11·258
3...........	342	70·85	77·1	7⅚	161·5	52·7	125	135	60	161·5	80	50·4	70	30·42	8·32	2·48	9·013	11·173

Test	Extractions			Sulphur recovery, per cent	Analysis ignited sulphur residue		Gold (in milligrams)							Silver (in milligrams)						
	Fe, per cent	Cu, per cent	Zn, per cent		Au	Ag	Head	Final residue	Per cent in residue	In cement-copper residue	Per cent	Gold accounted for	Per cent	Head	Final residue	Per cent in residue	In cement-copper residue	Per cent	Silver accounted for	Per cent
2...........	73·5	88·6	92·4	71	nil	trace	9·38	8·658	92·2	0·153	1·8	8·811	94·0	131·59	11·258	8·5	119·75	91·0	131·008	99·5
3...........	73·1	90·2	93·9	68·1	nil	0·015	9·38	9·013	96·0	0·18	2·0	9·193	98·0	131·59	11·173	8·5	118·0	89·6	129·188	98·2

Series F-3: Test 4

In this test 1,400 grammes of the ore concentrate was leached direct in four 4-litre leachings. The leach liquor was treated with sponge iron to remove copper and the copper-free liquor used for iron deposition tests. The results of these steps are shown in tabulations (a), (b), (c), in Table III. The residues from leaches A, B, C, D, were combined and comprised the first distillation roast; the sulphur recovered, analysis of roast and extraction of metals from ore charge being shown under (d); 5 grammes of the roast product being used for purpose of analysis. The roast product was again leached with excess ferric chloride, the purpose being to obtain a final residue of very low copper content. The leach liquor was treated with sponge iron to precipitate copper and the cement copper assayed for gold and silver values. Results are shown under secondary leach (e). This secondary leach residue was now roasted to recover free sulphur and the final residue analysed for base and precious metals. The extraction obtained in this secondary leach is shown together with analysis of final roast under (f).

A capitulation of the location of the two precious metals is shown in tabulation (g). The precious metal values shown under 1st and 2nd cement residues are the amounts remaining after the copper precipitates obtained from the leach liquors had been subjected to an oxidizing roast followed by sulphuric acid leaching and separation of the soluble copper. The results show 92·33 per cent of the gold and 92·8 per cent of the silver values accounted for in residues and readily recoverable by present-day standard methods.

The final capitulation at (h) shows the net result of this method of treatment with remarkable extractions of copper, iron, and zinc, and a high recovery of sulphur.

TABLE III

Series F-3: Test 4—Amulet Ore

(a) Primary Leach

Leach	Charge concentrate, grms.	Volume of leaching liquor, litres	Leached residue, grms.	Per cent leached out	Sponge iron added for cementation, grms.	Cement-copper residue, grms.
A	350	4	171	51·1	125	....
B	350	4	163	53·4	125	....
C	350	4	164	53·1	125	....
D	350	4	158	54·8	125	....
	1,400	16	656	53·1	500	515

(b) Analysis Cement-copper Residue—

Fe	37·08 per cent
Cu	47·92 "
Pb	nil
Gold	0·11 oz./ton—0·714 mg.
Silver	62·19 oz./ton—460·57 mg.

(c) Analysis Copper-free Liquor—

Fe''	172·75 grm./litre
Pb	0·61
Zn	7·26
*p*H	4·2

(d) First Distillation Roast

Charge, grms.	Weight of roast, grms.	Sulphur re-covered, grms.	Per cent sulphur re-covered	Analysis roast			Extraction		
				Fe, per cent	Cu, per cent	Zn, per cent	Fe, per cent	Cu, per cent	Zn, per cent
656..........	288	333	79	33·41	9·51	3·30	74·1	90·2	92·5

(e) Secondary Leach

Charge, grms.	Volume leaching liquor, litres	Analysis		Leached residue, grms.	Sponge iron added for cementation	Cement-copper residue, grms.	Gold in cement residue, mgs.	Silver in cement residue, mgs.
		Fe‴	Fe″					
283.........	4	90·57	39·26	157	135	79	2·17	36·46

(f) Second (Final) Distillation Roast

Charge, grms.	Weight of roast, grms.	Sulphur re-covered, grms.	Analysis final roast					Extraction		
			Fe, per cent	Cu, per cent	Zn, per cent	Gold, mgs.	Silver, mgs.	Fe, per cent	Cu, per cent	Zn, per cent
157........	88	55	11·10	1·06	0·64	32·61	2·87	89·8	96·6	94·1

(g) Precious Metals (milligrams)

—	Total in head	1st cement residue	Per cent	2nd cement residue	Per cent	Final residue	Per cent	Metal accounted for	Per cent
Gold..............	38·4	0·714	1·85	2·17	5·56	32·61	84·92	35·5	92·33
Silver.............	538·4	460·57	85·5	36·46	6·77	2·87	0·53	499·9	92·8

(h) Recapitulation

Initial charge raw concentrate....................................	1,400 grammes
Weight final residue....................................	88 “
Per cent sulphur and metals recovered..........................	93·7
Total sulphur recovered....................................	388 grammes
Per cent sulphur recovered....................................	92·2
Total extractions—	
Fe....................................	97·3 per cent
Cu....................................	99·6 “
Zn....................................	99·5 “
Gold accounted for....................................	92·33 “
Silver accounted for....................................	92·8 “

TABLE IV

Series F-3: Tests 6, 7, 8, and 9—Amulet Ore

(Volume leaching liquor per test, 55 litres.)

Test	Leaching				Copper cementation						Purification		Roasting				Results							
							Analysis, per cent										Analysis, per cent				Extractions, per cent			
	Time, hours	Charge raw concentrate	Weight leached residue	Per cent leached out	Sponge iron added	Weight cement residue	Fe	Cu	Copper oxide	Total Cu	CaS added, grms.	HCl acid added, c.c.	Charge	Weight of roast	Loss in weight, per cent	Weight of sulphur recovered	Fe	Cu	Zn	S	Fe	Cu	Zn	S
6.......	5	4,465	2,187	51·0	1,800	1,956	2·09	7·52	28·30	35·82	87	105	2,187	1,105	49·4	892	31·32	13·79	4·09	29·14	70·7	82·9	88·7	66·5
7.......	5½	4,465	2,306	48·3	1,700	1,636	......	8·12	30·94	39·06	65	115	2,306	1,277	44·6	829	31·72	15·79	5·26	29·10	65·7	77·4	83·5	61·8
8.......	5½	4,465	2,238	49·8	1,700	1,612	......	......	......	......	125	130	2,238	1,197	46·5	956	32·35	15·35	5·13	29·34	67·3	79·4	84·9	71·3
9*......	5½	3,579	1,740	51·3	1,300	1,174	......	4·10	36·72	40·82	110	80	1,740	802	53·9	858	10·99	1·76	2·03	7·66	......	......	......	......

* Test No. 9 comprised the total roasts of Tests Nos. 6, 7, and 8.

RECAPITULATION

Total raw concentrate charged	13,395 grms.
Weight final residue (roast)	802 "
Sulphur and metals removed, per cent	94
Sulphur removed by distillation	3,535 grms.
Recovery of sulphur, per cent	87·8
Total sponge iron charged	6,500 grms.
Total cement residue	6,378 "
Total CaS added	387 "
Total HCl acid added	430 c.c.
Total leaching liquor	220 litres

Total ultimate extractions

Fe	97·5 per cent
Cu	99·4 "
Zn	98·6 "
S	98·4 "

Analysis final liquor

Fe''	167·52 grm./litre
Pb	trace
Cu	nil
Zn	7·0–7·5
pH	4·2

Precious metals

Gold in total charges	367·2 mgs.
Gold in residue	328·82 "
	89·5 per cent

Gold with copper precipitate not determined.

Silver in total charges	5,152·2 mgs.
Silver in residue	39·03 "
Per cent	0·75

Silver with copper precipitate not determined.

Series F-3: Tests 6, 7, 8, 9

The following tests are really a duplicate of Test No. 4, using a much larger quantity of the ore charge. The main purpose was to prepare a liquor free from lead to provide electrolyte for iron deposition tests, and the only added step in the method of treatment was the addition of calcium sulphide and hydrochloric acid to the copper-free liquor for the precipitation of the lead. The extraction results are sufficiently interesting to be submitted and are, therefore, shown in Table IV.

Tests Nos. 6, 7, and 8 comprised direct leaching of ore concentrate and Test No. 9 was the combined leach residues of 6, 7, and 8 after the free sulphur was removed by a distillation roast.

The purification or lead precipitation and the electro-deposition will be dealt with later in this report.

In the recapitulation it will be observed that the final residue amounts to 6 per cent of the ore charge and that the extraction of the base metals is quite satisfactory.

With respect to the precious metal values, 89·5 per cent of the gold and 0·75 per cent of the silver were found in the final residue. Silver and gold values in the copper precipitate were not determined, but the results of preceding tests may be cited as a guide as to their disposition. The final liquor or electrolyte obtained, free from copper and showing a trace of lead, was used in iron deposition tests.

Conclusions

The above tests show the amenability of this ore to the ferric-ferrous chloride method of leaching for the extraction of the base metals, and the concentration of the precious metal values.

The gold is left in the leach residue in a convenient form for further treatment and recovery, while the silver is obtained in combination with the copper and its recovery should not present any serious difficulty. Copper is readily and completely separated from the leach liquor by precipitation on iron. Sponge iron was used in the above tests and produced a copper precipitate fairly high in iron. It would probably be more advantageous to use scrap electrolytic iron obtained from electrolysis of the ferrous liquor, as previous tests have proved that a high-grade copper precipitate can thus be obtained. (Compare copper recovery in Report of 1927.)

This proposed method of hydrometallurgical treatment for high-grade iron-copper ore or concentrates has now been tested out on three different types of ore—Eustis ore concentrate, essentially a pyrite chalcopyrite; Anyox ore concentrate, a pyrrhotite chalcopyrite; and Amulet ore concentrate, a pyrite pyrrhotite chalcopyrite—with equally satisfactory results.

The results obtained on Eustis and Anyox ores are described in the Report for 1927.

PURIFICATION OF ELECTROLYTE

ELIMINATION OF LEAD

The heavy sulphide ores of the type used in these investigations contain varying amounts of lead which is readily soluble in the ferric chloride leaching liquor. Its presence in the electrolyte has a decidedly adverse effect on the structure of the deposited iron. In the 1927 Report of Investigations an account is given of the elimination of lead and zinc from the iron electrolyte. This problem was not satisfactorily solved and tests on the elimination of lead were continued during the past year.

In previous experiments it has always been the practice to add the precipitating materials in slight excess of that required to react with the lead in the solution. It was decided to carry out tests using a much larger excess of the precipitating agents than had been used hitherto and to determine their effect on removing lead from the solutions. Three methods were tried out.

(1) The use of a metallic sulphide.
(2) The use of sponge iron.
(3) The use of calcium sulphide and hydrochloric acid.

In (1) the precipitation of the lead is based entirely on the replacement of one metallic sulphide by another, due to their respective solubilities. Freshly precipitated ferrous sulphide pulp was the metallic sulphide reagent used.

Theoretically, lead sulphide is more insoluble than iron (ferrous) sulphide in ferrous chloride solutions.

Sponge iron (2) is also a precipitant of lead from aqueous solutions due to the electro affinity of iron in relation to that of lead.

Both of these reactions are, however, complicated to some extent, by the high iron concentration of the solutions under investigation.

Under (3) the usual additions of calcium sulphide and hydrochloric acid were made, modified from previous practice by the elimination of agitation.

A number of tests were carried out on an electrolyte containing 0·8 gramme per litre of lead, using ferrous sulphide pulp. The pulp, suction dried, contained approximately 59 per cent FeS. Weighed quantities of the pulp were added to 500 c.c. of the electrolyte, agitated at room temperature for six or seven hours and allowed to settle overnight. The supernatent liquor was analysed for lead. One hundred, fifty, and twenty-five times excess of FeS, completely precipitated the lead. Ten and five times excess precipitated 61·2 per cent and 20 per cent respectively. Twenty-five times excess over theoretical represented 8·6 grammes of pulp per litre. The results show that a complete elimination of lead is effected by the addition to a ferrous chloride electrolyte of freshly precipitated iron sulphide in excess. The excess required, however, is so great that its economic application might be open to question.

The use of sponge iron as a precipitant is a well known practice in the hydrometallurgy of lead. The application of this method of lead precipitation to ferrous chloride leach liquors was attempted in several tests. Some interesting results were obtained which are briefly described in the following paragraphs. Co-precipitation of copper and lead in a single operation was found to be impossible. A possible explanation is that the iron particles become completely coated with copper, thus preventing any reaction between the soluble lead in the solution, and a clean iron surface. It was necessary, therefore, to carry out the precipitations in two stages, first the removal of the copper; followed by further additions of sponge iron for precipitation of the lead.

To 500 c.c. leach liquor containing approximately 17 grammes per litre of copper and 0·80 gramme per litre of lead, 25 grammes of sponge iron were added. This represented about 100 per cent excess iron over that required to precipitate all the copper present. The solution was agitated hot until the liquor was copper-free. To this copper-free liquor 12·5 grammes of sponge were added and agitation continued at room temperature. The sponge iron added in the second treatment represents an excess slightly over 90 times that theoretically required to react with the lead in the solution. One gramme of iron will precipitate 3·7 grammes of lead from solution and with the sponge analysing 79 per cent metallic iron, approximately 9·88 grammes of iron were used to precipitate the 0·4 gramme of lead. On analysis it was found that 82·5 per cent of the lead had been removed.

The iron-copper-lead residue from the above treatments was then added to fresh leach liquor and agitated hot. Analysis of the solution showed the copper completely precipitated but the lead content unaffected, despite the excess of iron present. A further excess of iron would be necessary to remove the lead.

While the above method as outlined gives encouraging results from the standpoint of eliminating lead from leach liquors, its application on a large scale opens up many factors for consideration. The large quantities of sponge iron required together with the numerous settling-tanks and filters necessary for the solutions tend to raise serious difficulties in the application of such a method for electrolyte purification on an economic scale.

In the calcium sulphide hydrochloric acid method, the precipitating agent is most probably the hydrogen sulphide generated by the action of the acid on the alkali sulphide. To obtain a solution of the highest hydrogen sulphide saturation possible it would appear that the same could be obtained by reducing agitation to a minimum.

To test this out 4 litres of electrolyte analysing 173·75 grammes per litre ferrous iron and 0·458 gramme per litre of lead were treated in a pail with hydrochloric acid and calcium sulphide and occasionally stirred. An analysis of the electrolyte after standing overnight showed it to be free of lead.

This method was now applied to about 300 litres of this electrolyte in a large stoneware tank; 600 c.c. of hydrochloric acid made up to 1 : 1 with water were added and well mixed; 600 grammes calcium sulphide (commercial) were ground in water and added to the acidified liquor as a wet pulp and well stirred for a few minutes. After a period of two and a half hours the liquor was again stirred for a few seconds, and then allowed to settle. A sample, taken one hour after the final agitation, was analysed and gave negative results for lead.

The liquor was then run to the filter. Filtration was satisfactory.

The results were most encouraging and lead to the following assumptions.

In previous purifications, in which active agitation was carried out, a difficulty was always experienced in removing the last traces of lead from the solutions.

Concordant results were not always obtained. A possible explanation for these conditions now presents itself.

The reaction with H_2S and $PbCl_2$ is undoubtedly a time reaction, requiring in the presence of so strong a solution of ferrous chloride an excess of hydrogen sulphide. When the liquor is subjected to agitation a great deal of the hydrogen sulphide is driven off as it is formed and air is drawn into the solution. By allowing the solution to remain quiescent the hydrogen sulphide saturates the entire volume and reacts slowly with the lead, forming PbS, which is precipitated. Sufficient acid should be present to prevent the precipitation of iron.

A complete analysis of the purified electrolyte was as follows:—

Fe''	172·2 grm./litre
Pb	trace
Cu	nil
Zn	7·17
CaO	2·39
SO_3	3·29
Colloidal or sulphide S	0·04

The results of the analysis show that the iron losses are very low, little more than one gramme per litre. Any sulphur present in the colloidal state can be readily removed in the filtering operation by the use of filter-cel, or some similar filter aid.

Any hydrogen sulphide remaining in the solution after the reaction, can be easily removed by aeration.

This method offers several distinct advantages over the constant agitation method.

1. Large volumes of solution can be purified in one operation.
2. No agitation is required other than the dispersion of the reagents throughout the liquor.
3. Settling is effected during the period of purification, which fact simplifies filtration.
4. A satisfactory elimination of lead is attained.

It is quite possible that considerable economies can be effected in the consumption of calcium sulphide and hydrochloric acid.

In the test described above an excess of twelve times over theoretical was used. By further investigation the possibility of reducing this figure appreciably can undoubtedly be achieved.

PRE-LEACHING OF COPPER CONCENTRATE WITH SATURATED BRINE SOLUTION FOR ELIMINATION OF LEAD

During the past few years the leaching of lead ores with hot saturated brine solution has received considerable study. So encouraging have been these results that it has been commercially applied by at least two corporations.

In view of the success attained the question of its application to the leaching out of galena (lead sulphide) in the presence of other base metal sulphides presented itself.

A method of selective leaching was tried out and preliminary tests showed that solutions of saturated sodium chloride slightly acidified with hydrochloric acid have a strong solvent action on lead sulphide.

The ore under consideration was Amulet copper concentrate of the following analysis.

Fe	26·53 per cent
Cu	20·01 "
Zn	9·13 "
Pb	0·67 "
Ag	11·22 oz./ton
Au	0·80 "

A number of tests were carried out in which temperature, ratio of solution volume to weight of concentrate, per cent of volume of acid, and time were the primary factors.

The method adopted was to leach the concentrate in a 1·5-litre beaker using a motor-driven paddle for agitation with a saturated (cold) solution of brine acidified with HCl. After agitating hot or cold for a specified time, the leach was filtered. The residue on the filter was then washed with a saturated solution of brine equal in volume to that used in leaching. After the brine wash, the cake was washed free of sodium chloride or other salts with cold water. Analysis of the leach and washings for lead, iron, zinc, and copper were made and the extractions calculated.

The following is a summary of the tests carried out and the results obtained.

Test B-1

Charge—1,000 grms. concentrate
Brine leach—1,000 c.c. saturated NaCl+50 c.c. HCl (1·19 sp. gr.)
Brine wash—500 c.c. saturated NaCl
Time of agitation—5 hours
Temperature—50°–75°C.

—	Volume, c.c.	Fe, grms.	Pb, grms.	Zn, grms.	Cu, grms.	Weight of residue, grms.	Per cent leached out
Leach	920	16·84	4·629	7·452	nil	907	9·3
Brine wash	555	5·83	1·726	2·861	nil		
Water wash	590	2·18	0·095	0·186	nil		
Total metals	....	24·85	6·45	10·5	nil		
Extraction	....	9·3%	96·2%	11·5%			

There was no settling out of lead salts in the filtered brines. All the acid was used up.

Test B-2

Charge—1,000 grms. concentrate
Brine leach—500 c.c. saturated NaCl+25 c.c. HCl (1·19 sp. gr.)
Brine wash—500 c.c. saturated NaCl
Time of agitation—5 hours
Temperature—50°–75°C.

—	Volume, c.c.	Fe, grms.	Pb, grms.	Zn, grms.	Cu, grms.	Weight of residue, grms.	Per cent leached out
Leach	405	6·36	3·049	4·005	nil	925	7·5
Brine wash	690	5·04	2·704	3·193	nil		
Water wash	780	2·42	0·918	0·574	nil		
Total metals	....	13·82	6·671	7·77	nil		
Extraction, per cent	....	5·17	100	8·5			

The precipitation of lead salts in the leach and brine liquors presented a serious difficulty in this test. It was decided that a 2 : 1 ratio of concentrate to brine solution was altogether unsuitable and all further tests were carried out on 1 : 1 ratio.

The four tests following consisted of leaching 500 grammes of concentrate with 500 c.c. of saturated brine solution with or without additions of acid, and washing the residue with an equal volume of non-acidified saturated brine, followed by a water wash.

Test No.	Time	Temperature	HCl (1·19 sp.gr.) added	Extractions			Weight of filter cake, grm.	Weight of dry residue, grms.	Per cent moisture	Per cent leached out	Weight of salts in wash water	Remarks
				Fe, per cent	Pb, per cent	Zn, per cent						
			c.c.									*p*H leach
B-3	5	50–75	nil	2·4	84·0	8·5	615	471	23·4	5·8	43	6·0
B-5	5	60–80	5	3·9	93·7	8·1	615	469	23·7	6·2	....	....
B-6	5	60–80	10	4·9	94·3	9·1	604	467	22·6	6·6	....	....
B-4	5½	50–75	50	11·3	100	14·2	602	443	26·4	11·4	56	*

*Brine leach liquors contained H_2S.

The results of these tests show that a high acid brine has too great a solvent action on the iron and zinc content of the concentrate, while a non-acidified brine lowers the lead extraction considerably.

The results of three tests follow in which the agitation was carried out at room temperature. The 500 c.c. saturated brine solution contained 15 c.c. of 1·19 sp. gr. hydrochloric acid. The time factor was the variable.

Test	Time	Extractions				*p*H leach	Weight of dry residue, grms.	Per cent leached out	Weight of salts in wash water, grms.
		Fe, per cent	Pb, per cent	Zn, per cent	Cu, per cent				
B-10	2	3·6	91·3	7·9	1·52	3·0	465	7·0	48
B-11	4	4·3	100	8·1	2·07	3·2	463	7·4	55
B-8	5	3·78	93	8·52	1·51	3·0	464	7·2	49

It will be observed that the use of a hot solution is not essential for the extraction of the lead. The time of leaching does not appear to play an important part in the reaction. A feature to be noted here is the presence of copper in the leach liquors, for which no definite explanation can be offered. Previous tests showed no copper soluble, while these tests show an appreciable amount.

In order to effect a saving in the consumption of sodium chloride tests were run using the same solution over five times. After each leach the filtrate was gassed with H_2S to remove the bulk of the lead as PbS, filtered, aerated to remove H_2S and titrated against standard N/10 KOH to determine the acid regenerated. Prior to each leach the volume of the brine solution was made up to 500 c.c. with saturated brine and sufficient hydrochloric acid added to give an acidity of 15 c.c. HCl (1·19 sp. gr.) per 500 c.c. brine liquor. Five hundred grammes of fresh concentrate were used in each cycle and fresh saturated brine solution was used each time for washing the residue. In a subsequent series of tests, run as a check, the brine washes were also used over again, being subjected to the same treatment as the leaching liquor.

Very careful analyses of all solutions for lead were made at the end of each cycle and a percentage extraction calculated. The brine solutions only were analysed for iron and zinc and the total present after each cycle recorded. Each cycle was run for 5 hours and the temperature was between 60° and 80°C. The following table shows the results of this test.

Test B-7

Cycle	Total metals in liquors			Extra lead, per cent	pH leach	Leached residue	Per cent leached out	Weight of salts in wash water, grms.	Silver in H_2S precipitates, mgs.
	Fe	Zn	Pb						
1	10·11	7·047	3·129	93·4	4·6	462	7·6	53	1·14
2	14·763	10·406	3·093	92·3	3·4	468	6·4	46	13·04
3	16·451	13·18	3·004	89·6	4·0	465	7·0	50	0·50
4	20·252	14·911	3·341	99·7	3·2	467	6·6	48	5·58
5	22·80	16·61	3·073	91·7	3·0	465	7·0	50	50·70

In the above tests 2,500 grammes of concentrate were treated; 500 c.c. of saturated brine used in the first cycle required a total of 165 c.c. additional brine to maintain a constant volume for the subsequent cycles. A little over 12 per cent of the total acid required was regenerated in the H_2S treatment.

A trace of gold was recorded from the H_2S residue in Cycle No. 1. The remaining assays were negative.

Test B-9

This test was a duplicate of Test B-7 with several minor modifications. The brine wash was re-used after being gassed with H_2S, aerated and made up to 500 c.c. volume with fresh brine solution. Water added by slow drops from a dropping bottle overcame evaporation losses and maintained a constant volume during the run.

The temperature of each run was around 78°C. and the time of leaching 5 hours.

Cycle	Total metals in liquors				Extra lead, per cent	pH leach	Leached residue, grms.	Per cent leached out	Weight of salts in wash water, grms.	Silver in H_2S precipitates		
	Fe	Zn	Cu	Pb						Leach, mgs.	Wash, mgs.	Total, mgs.
1.......	7·67	4·402	....	3·357	100	3·8	458	8·4	54	31·70	8·9	40·60
2.......	11·958	7·984	1·173	3·041	90·8	3·0	468	8·4	54	34·58	7·8	42·38
3.......	17·428	10·98	1·147	3·483	100	3·0	459	8·2	49	38·44	7·84	46·28
4.......	20·676	12·309	0·649	3·148	95·5	....	465	7·0	49	22·66	27·16	49·82
5.......	24·496	14·225	0·969	2·924	87·2	3·8	459	8·2	....	....	....	....

No gold was found in the hydrogen sulphide precipitate. The results check fairly closely to those obtained in Test B-7.

Conclusion

The elimination of the lead from the ore concentrate by the method outlined above offers a possible solution to the purification problem. The extractions of lead are high, and the leaching agent, sodium chloride, is cheap.

By a re-use of the brine leach a saving is effected in salt consumption. This saving is partly offset by the use of hydrogen sulphide after each cycle.

The combined leach residues in which over 90 per cent of the lead had been extracted were mixed and analysed as follows:—

Fe	26·54 per cent
Cu	20·54 "
Zn	9·23 "
Pb	0·05 (approx.)

The tests that were carried out were on small volumes only, and at this stage it is not possible to state whether such a method would be economic if carried out on a larger scale.

A 55-litre ferric-ferrous chloride leach was made on the combined residues noted above. The object of this test was to determine whether the solubility of the copper concentrate would be affected in any way by the pre-leaching with sodium chloride. The results as given in Table V are highly satisfactory and show practically no difference in extraction from those tests in which the raw concentrate was leached directly.

TABLE V

Series F-4: Test No. 1—Amulet Ore

(55-litre leach of brine leach residues)

Leach.................Charge—4,000 grammes ($Fe=26·54\%$, $Cu=20·54\%$, $Zn=9·23\%$, $Pb=0·05\%$)
Leaching liquor—55 litres: Fe''' 65 g./l. Fe'' 77·3 g./l.
Time of leach—6 hours
Weight of leached residue—2,041 grammes
Per cent leached out—48·9

Cementation.......Sponge added—1,500 grammes
Weight of cement residue—1,641 grammes

Roasting..........Charge—2,041 grammes
Weight of roast—976 grammes
Per cent loss weight—52·1 grammes.
Sulphur recovered—981 grammes

Analysis of roast—

Fe............................	31·17	per cent
Cu............................	12·16	“
Zn............................	3·37	“
S............................	31·30	

Extractions—

Fe............................	71·3	per cent
Cu............................	85·5	“
Zn............................	91·0	“

ELECTRO-DEPOSITION OF IRON

The following paragraphs review a number of electro-deposition tests. The electrolyte used was in all cases obtained from the leaching of copper concentrates. The tests were carried out in two sizes of plate cells, small glass battery jars and a larger asbestos ebony cell. The details, observations, and results of the tests are set out in the following tables and paragraphs.

SMALL-SCALE DEPOSITION TESTS

A number of tests were carried out in small cells, primarily for the purpose of studying deposition from electrolyte containing lead derived from leaching Amulet copper concentrates.

The cells consisted of battery jars, 3¾ inches square by 8 inches high inside dimensions. Each jar was divided by an asbestos cloth diaphragm into two compartments. One graphite anode and one sheet steel cathode were used. The effective deposition area of each cathode was 0·158 square feet. The cells were partially immersed in an electrically heated copper water bath.

The results of these tests are summarized in Table VI. The electrolyte was a copper-free leach liquor analysing between 7 and 8 grammes per litre of zinc and under 1·0 gramme per litre of lead.

Tests 48P and 49P

These tests are duplicates. The deposits were rough and mossy, and on analysis show appreciable amounts of lead but little of zinc. Annealing of the deposits at 1000°C. for half an hour results in little change in the lead content.

Tests 50P and 51P

The electrolyte for these duplicate tests was largely the overflow catholyte from the two previous tests. The acidity was increased to determine the effect of this factor on deposition. There was an improvement in the nature of the deposits, the surfaces were fairly smooth with the exception of a few nodular growths. On analysis the lead content is found to be much lower than the two preceding tests. This improvement in the deposit is undoubtedly due to the lower lead in the electrolyte, whether the acidity is a contributing factor is difficult to determine.

Tests 52P and 53P

The electrolyte used in these tests was a fresh copper-free leach liquor analysing 0·61 gramme per litre of lead. The deposits were poor being covered with nodular, mossy growths.

Test 54P

The electrolyte was the same as the two previous tests with the exception that the acidity was increased. A very poor nodular-covered deposit was obtained, which on analysis gave over 2·0 per cent of lead. It is apparent that increased acidity has little or no effect on improving the deposit in the presence of over 0·5 gramme per litre of lead in the electrolyte.

A current efficiency of over 90 per cent was obtained in all the above tests. Little zinc is co-deposited with the iron due to the voltage employed being under the decomposition voltage of that metal. Lead in the electrolyte in amounts over 0·10 gramme per litre has a very decided influence on the nature of the deposits. The result is a rough, nodular, mossy surface. Small quantities of lead, under 0·05 gramme per litre, do not appear to exhibit this harmful influence as has been shown in the experiments on deposition reported in the Report of Investigations for 1927.

TABLE VI

Test No.	Analysis electrolyte, grm./litre.						Temperature in cell ° C	Voltage	Current density, amp./sq.ft.	Weight of deposit, grms.	Rate of flow of electrolyte, l./hr.	Nature of deposit	Analysis of deposit		Observations
	Fe''	Fe'''	Zn	Pb	*p*H	Time, hours							Pb, per cent	Zn, per cent	
48P	172·6	trace	7·34	0·4	3·6	7·5	74–91	3·25	53·8	61	0·486	Somewhat rough....	2·05	under 0·1	No chlorination. Very faint hydrolysis.
49P	172·6	trace	7·34	0·4	3·6	7·5	75–90	3·45	53·8	61	0·313	Smooth at top; bottom half rough.	2·12	under 0·1	ditto.
50P	180	....	7·89	0·18	2–2·5	7·5	73–88	3·4	53·8	66	0·320	Fair; nodular.......	0·31	0·065	No chlorination. Faint hydrolysis.
51P	180	....	7·89	0·18	2–2·5	7·5	81–89	3·4	53·8	54	....	Smooth; few nodules	0·45	0·06	ditto.
52P	172·8	....	7·26	0·61	3·5	5·5	87	3·15	53·8	52	0·387	Badly nodular......	0·98	0·09	No chlorination. Increased hydrolysis.
53P	172·8	....	7·26	0·61	3·5	6·5	88·8	3·56	53·8	62	0·450	Mossy; badly nodular.	1·51	0·07	ditto.
54P	172·8	....	7·26	0·61	2–2·5	5·5	86·5	3·33	53·8	52	0·548	Badly nodular; black.	2·21	0·03	ditto.

LARGE-SCALE DEPOSITION TESTS

The tests, the data of which are summarized in Table VII, were carried out in a cell of the following design. The cell consisted of an asbestos-ebony box, ½-inch wall thickness, 10½ inches by 9 inches and 17 inches deep. Three bakelite frames, 9⅝ inches by 1 inch and 17 inches deep, covered with asbestos cloth, constituted the anode compartments. The anodes were graphite sheets 7½ inches wide, ½-inch thick, 17½ inches deep. At first aluminium strips were used as anode holders. These were found to be quite unsatisfactory owing to the formation of aluminium compounds through corrosion between the graphite and the metal, thereby constituting a non-conducting medium, and causing irregularity in the current flow. Copper was substituted and current difficulties were overcome.

The cathodes were $\frac{1}{32}$-inch sheet steel plates. The effective deposition area of the two cathodes was 3·19 square feet, or approximately 30 square decimetres.

Two copper bus bars, one connected to the negative lead, the other to the positive, ran along opposite sides of the cell. The anode and cathode holders rested on their respective bus bars.

The electrolyte heated in a feed tank was fed in at the bottom of the cell, flowed across the face of the cathodes, and overflowed from near the top of the cell.

The anode overflow was syphoned through glass tubes from each anode compartment, screw cocks regulating the flow to any desired volume.

No initial treatment was given to the cathode surfaces, other than a superficial polishing with fine sandpaper to remove oxide or slight roughness.

Test 46P

The electrolyte was liquor from leaching of Anyox copper concentrates purified in the usual manner. *(See* Investigations in Ore Dressing and Metallurgy 1927.) Despite the irregular current due to the trouble caused by the aluminium anode holders noted above, an excellent deposit was obtained, which was stripped as sheets with comparative ease. There was no chlorination and little hydrolysis.

Test 47P

The electrolyte was the same as in the previous test. The rate of flow was reduced. The current was irregular. There was some evolution of chlorine. An excellent deposit of iron was obtained.

Test 56P

The electrolyte was composed of mixed liquors, the greater part of which was from Anyox concentrate leaches. The aluminium anode holders were replaced by copper and a more regular current flow was noted. An anode overflow of 3 litres per hour was maintained. A very good deposit of iron was obtained, very adherent, smooth, light grey in colour.

Test 57P

The electrolyte was purified liquor from leaches of Amulet copper concentrate. The electrolyte differed from previous solutions in carrying from 7 to 8 grammes per litre of zinc. The deposit was very good, light grey in colour, quite uniform, but very difficult to strip. Analyses gave only 0·029 per cent zinc.

Test 58P

The electrolyte was from liquor from leaches of Amulet copper concentrate, which had been subjected to brine leaching during which over 90 per cent of the lead was removed. The overflow from the cell passed through sponge iron in a regeneration tank from which it was pumped back into the feed circuit. Chlorination was excessive throughout the test. The deposit peeled badly early in the run. The plates were removed and this deposit stripped. The final deposit was very bright, badly pinholed and showed signs of cracking. Ferric iron was present in the inflow liquor, which probably would account for the peeling tendency.

TABLE VII

Test No.	Electrolyte	Analysis electrolyte, grm./lt.					Time, hours	Temperature in cell ° C	Voltage	Current density, amp./sq. ft.	Weight of deposit, grms.	Current efficiency, per cent	Rate of flow of electrolyte, litre/hr.	Nature of deposit	Analysis of deposit		Observations
		Fe″	Fe‴	Zn	Pb	*p*H									Pb, per cent	Zn, per cent	
46P.	Anyox copper concentrate.	147·7	trace	1·56	nil	3·9	6·75	80·2	2·84	31·09	670	94·4	12	Excellent...	0·046	0·01	No chlorination. Slight hydrolysis.
47P.	ditto	147·7	trace	1·3	nil	3·4	7	75·9	3·04	28·56	670	99·0	7·5	Excellent...	......	......	Some chlorination. No hydrolysis.
56P.	ditto	167·5	trace	......	nil	3·4	7	74·8	2·63	29·9	690	99·1	11	Very good; light grey	......	......	Chlorination ceased after 5th hour. Some hydrolysis.
57P.	Amulet copper concentrate.	167·5	trace	7–8	trace	3·4	7	77	2·84	29·3	682	99·9	11	Very good; smooth.	......	0·029	Some chlorination. No hydrolysis.
58P.	Amulet copper, brine leached concentrate.	144·8	trace	7–7·5	trace	3·6	6·5	67·6	2·86	31·0	665	99·2	....	Bright; pinholed.	......	......	Excessive chlorination.

REPORT ON THE TREATMENT OF MIXED (BULK) CONCENTRATES FROM BASE-METAL SULPHIDE ORE

W. E. Harris[1]

A considerable number of small laboratory tests and runs were made to clear up some of the difficulties met with in the large-scale tests described in the report of 1927. The electrolysis of mixed solutions and the treatment of the residues for the recovery of the significant metals were given special attention.

Leaches were made on roasted bulk concentrates obtained from Amulet ores; this concentrate assayed 32·92 per cent zinc, 6·3 per cent copper, enriched with a roasted copper precipitate containing 72 per cent copper which brought the copper content of the ore charge to approximately 12 per cent.

The leaching solution consisted of a 21 per cent sulphuric acid liquor; after leaching and precipitating the iron and silica the resulting solution contained 8·84 per cent zinc, and 3·09 per cent copper. This solution was acidified with sulphuric acid to 1 per cent, and electrolysed in a series of cells (cascade) using insoluble lead anodes. About 75 per cent of the copper was deposited out before any signs of deterioration or "burning" was seen in the deposit and the resulting outflowing solution ran 9 per cent zinc, 0·8 per cent copper, and 6·5 per cent acid.

The experiment was repeated with a synthetic solution made up from leach liquors enriched with copper sulphate, the head solution running 8·58 per cent zinc, 4·8 per cent copper, 1·0 per cent sulphuric acid.

This solution was electrolysed as before with lead anodes until signs of "burning" occurred, the resulting solution assayed, zinc 9·2 per cent, copper 0·82 per cent, acid 5·97 per cent, being equivalent to a deposition of 83 per cent of the copper present in the original solution. This copper assayed (electrolytic assay) 99·43 per cent. The current density of the top cells never exceeded 10 amperes per square foot, and of the lower cells 5 amperes per square foot.

In a large-scale plant where the effect of the solution flow (which would be much greater in proportion) would be felt, the copper content could doubtless be lowered to 0·5 per cent copper in solution.

The treatment of the residues followed the outlines of the German patents known as the Waelz process, which consists of mixing the leach (or other residues) with a reducing agent such as anthracite or coke dust and subjecting to great heat. This is done in a long, revolving furnace whereby the mixture is carried forward by each revolution of the furnace, and has but little depth.

At the Mines Branch Booth Street laboratory a small, revolving tubular furnace was used.

This furnace was 30 inches long, by about 12 inches in diameter, lined with firebrick so as to leave an inside diameter of about 6 inches. Owing to the small size of the furnace considerable difficulty was experienced in keeping the ore charge in the furnace, and in getting the

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required 1,100° to 1,200°C. as the air blast was so strong that it blew the charge back and much of the fines and the coal dust was blown into the flues. However, the greater part of this difficulty was eventually overcome.

The object of the experiments was to show that the zinc (and lead, if any) could be volatilized and removed as the zinc oxide (and lead oxide).

The residues treated came from Amulet ore and leaches and assayed 18·8 per cent zinc, 2·95 per cent copper, 0·14 ounce gold, and 4·24 ounces silver.

This was mixed with varying amounts of anthracite dust and in some cases powdered limestone. The resultant residues ran 0·55 per cent zinc, 3·78 per cent copper, 0·18 ounce gold, and 3·79 ounces silver; and the zinc oxide produced ran 56·45 per cent zinc with less than ½ per cent copper and equalled an elimination of 97 to 98 per cent of zinc from the residue.

A volatilization of some of the silver took place but as almost identical gold and silver contents were found in the flue dust, it was concluded that these were mechanical losses due to the heavy air blast before mentioned.

These losses would not occur in a commercial furnace, which is from 120 to 140 feet long and has only a natural draught. Again, the zone of maximum temperature in our experiment occurred approximately two-thirds the length of the furnace from the point of feeding and one-third from the source of heat—a gas blast. In a commercial furnace once the reaction starts, the zone of maximum temperature is only one-fifth of the distance from the source of feed and there is no blast so that the mechanical loss of fines is small; this can be collected in a special chamber and returned to the furnace.

However, the object of the experiments was to prove the volatilization of the zinc and the results show that this is possible.

The resultant residue containing the copper and the precious metals was a sinter of an open granular nature and of a size that should lend itself to smelting for the recovery of the copper as a matte. Some small-scale tests seemed to point to the possibility of recovering the copper direct as metal, but with the varying sulphur contents it is probable that a matte would have to be formed.

V

REPORT OF THE WORK AND INVESTIGATIONS OF THE CHEMICAL LABORATORIES OF THE DIVISION

H. C. Mabee, Chief Chemist

The activities of the chemical staff was, as in previous years, closely associated with the investigative work of the engineering staff on their ore treatment problems. Assistance was also extended the Base Metals Extraction Company in connexion with their experimental work on an electrolytic method for the treatment of base-metal sulphide ores and concentrates. Accommodation was provided for D. S. Halford, research fellow, maintained by the Cassel Cyanide Company of Canada, Ltd., in connexion with his research work on the use of cyanide as a reagent in the flotation of base-metal sulphide ores.

The chemical laboratory investigations were carried on under the supervision of H. C. Mabee, chief chemist of the Division: the analytical work connected with metalliferous ore investigations was performed by B. P. Coyne and H. L. Beer; the non-metalliferous, by R. A. Rogers, assisted by B. P. Coyne; and the furnace assay work by L. Lutes with the aid of laboratory assistants.

The routine analyses performed in connexion with the investigations in ore dressing and metallurgy exceeded that of any preceding year both in the number of samples treated and in the chemical determinations involved. A large number of samples of residues and the products resulting from experimental work on sulphide ores in the hydrometallurgical laboratory were also examined; in addition several samples were identified and valued.

A total of 1,736 reports of analyses were issued during the year, involving over 7,000 chemical determinations. Following is a list of ores and concentration products in connexion with which reports of analysis were issued:—

Calcite	24	Graphite	311
Calcium molybdate	8	Iron ore	5
China clay	55	Lead (galena)	7
Copper ore	32	Lead-silver-zinc	138
Copper-gold	87	Limestone	6
Copper-molybdenum	67	Molybdenite	13
Copper-silver	18	Silica sands (glass)	112
Copper-zinc-gold	491	Tin ore (cassiterite)	9
Garnet rock	21	Tungsten ore (scheelite)	4
Gold-arsenic	39	Type metal	2
Gold-silver	236	Identification and valuation	50
Gold-platinum	1		

THE MANUFACTURE AND USE OF CALCIUM MOLYBDATE

The use of molybdenum steel is now firmly established in several branches of engineering and as its application appears to be extending more widely, even to the replacing of mild steel, it is reasonable to assume that the increasing demand for molybdenum will be of great importance to Canada in the revival of the molybdenite industry.

At the end of 1924 the alloy steel manufacturers were using ferromolybdenum as a vehicle for the introduction of molybdenum into molybdenum steel. By the latter part of 1925 the entire steel industry had abandoned the ferro-alloy and in its place they were using molybdenum in the form of calcium molybdate. This material is a normal chemical compound easily and cheaply produced containing, theoretically, lime and molybdic oxide in the proportion of 28 per cent of the former, 72 per cent of the latter, and having a chemical formula of $CaMoO_4$.

The method commonly followed for the production of calcium molybdate, and the one adopted for its commercial manufacture, consists in roasting the sulphide molybdenite to molybdenum oxide and then calcining with a calcium-bearing compound such as calcium carbonate. A second method consists in mixing thoroughly the roasted oxide with powdered slaked lime, sufficient water being added to make a thick paste and to permit of the chemical formation of the molybdate compound; the mixture is then dried and pulverized.

The material used in these experiments was a concentrate containing 91 per cent molybdenite, the remainder consisting of small quantities of iron sulphide, silica, and other gangue material. Four or five pounds of this material were subjected to an oxidizing roast at a temperature of about 600°C. until no further odour of sulphur could be detected. After cooling a definite quantity—100 grammes in each test—was weighed and thoroughly mixed with sufficient pure calcite ($CaCO_3$) to yield the definite quantity of calcium oxide required to form $CaO\text{-}MoO_3$. This mixture was then calcined at a temperature of about 600°C. for 30 to 45 minutes, at the end of which time practically no free carbonate could be detected. Several tests were conducted on different lots under practically the same conditions with little variation in the results.

Following are the analyses of two composite samples made from a series of eight different tests:—

Sample No. 1

	Per cent
Metallic molybdenum (Mo)—46·93 per cent equivalent to molybdenum trioxide (MoO_3)	70·40
Lime (CaO)	22·93
Iron oxide (Fe_2O_3)	2·64
Silica (SiO_2)	3·20
Sulphur (S)	0·28
Alumina (Al_2O_3)	trace
Combined water (H_2O) and carbon dioxide (CO_2)	0·31
Total	99·76

Sample No. 2

	Per cent
Metallic molybdenum (Mo)—45·90 per cent equivalent to molybdenum trioxide (MoO_3)	68·85
Lime (CaO)	23·09
Iron oxide (Fe_2O_3)	2·57
Silica (SiO_2)	4·85
Sulphur (S)	0·20
Alumina (Al_2O_3)	trace
Combined water (H_2O) and carbon dioxide (CO_2)	0·33
Total	99·89

Another series of experiments was carried out with the object of determining what results could be obtained in applying the wet process referred to above. In each case 100 grammes of roasted molybdic oxide were used. This was mixed and ground thoroughly with powdered slaked lime in the proper proportions, water was then added to a thick paste and again ground. The paste was then allowed to stand for half an hour, dried, and pulverized.

An average sample of the products from this series of experiments gave the following analysis:—

	Per cent
Metallic molybdenum (Mo)—47·85 per cent equivalent to molybdenum trioxide (MoO_3)	71·79
Lime (CaO)	21·95
Iron oxide (Fe_2O_3)	2·47
Silica (SiO_2)	2·38
Sulphur (S)	0·12
Loss on ignition—H_2O+CO_2, etc.	1·12
Total	99·83

These results indicate that very little difference exists between this material and that obtained by the furnace method.

CONCLUSIONS

The formation or production of calcium molybdate presents no serious difficulties; it is more convenient as a furnace addition and at the same time cheaper in its production than a furnace alloy.

Owing to the comparative ease with which the lime base and molybdic acid are disassociated on being charged into the molten steel, the molybdic acid which immediately becomes reduced to the metal is readily available for absorption into a more uniform complex solution resulting in an improved quality of steel.

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NEW PYROMETALLURGICAL LABORATORY FOR TEST AND RESEARCH ON IRON AND STEEL

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In view of the growing importance of the iron and steel industry in Canada, the Department of Mines is providing laboratory facilities for test and research thereby extending to that industry the same degree of co-operation that has proved so advantageous to the non-ferrous mining industry. The Department has erected in Ottawa as an addition to its present Ore Testing Laboratories, a new pyrometallurgical laboratory, which is being equipped with laboratory-scale and semi-commercial roasting, calcining, sintering, metallizing, melting, heat-treating, and standard laboratory testing and metallographic equipment for conducting extensive test and research on ferrous and non-ferrous ores, metals and their alloys, especially iron and steel.

At the time the Mines Branch was organized into its various Divisions in 1907, the most pressing problem confronting the Division of Ore Dressing and Metallurgy was the beneficiation of Canadian iron ores and consequently the first laboratory to be built was equipped for beneficiation tests. Twenty-one ores from Nova Scotia, New Brunswick, Quebec, Ontario, and British Columbia were tested, the results showing that in many cases marketable products could be obtained, suitable for use in the iron blast furnaces. However, the work was not productive of establishing an iron ore industry as primary ores of foreign origin could be bought and laid down at Canadian furnaces cheaper than Canadian beneficiated ores.

During the war and post-war years, very little investigative work was done by the Mines Branch on iron ores or on any phase of ferrous metallurgy. This lack of attention may be attributed to two reasons: first, there appeared to be no hope for the immediate utilization of Canadian iron ores due to the ready availability of cheap foreign ores; and second, to the demands for ore treatment test and research by the rapidly growing non-ferrous industry, stimulated by the urgency for the production of war metals. The investigative work of the Division of Ore Dressing and Metallurgy was therefore directed to the treatment of non-ferrous ores. Nevertheless the laboratory equipment for the beneficiation of iron ores has been kept up to date by the purchase from time to time of new equipment as progress has been made in other countries. Developments have been closely watched for their application to Canadian ores.

No large deposits of primary iron ores available for use in blast furnaces are known in Canada, hence, she has to rely on foreign ores to supply her requirements. The two chief sources are the United States for Ontario, and Newfoundland for Nova Scotia furnaces. However, located within reasonable distances of her furnaces are large deposits of the following types, all of which require beneficiation or some special method of treatment for their utilization:—

1. Siderites or iron carbonates;
2. Low-grade, low-sulphur magnetites;
3. High-sulphur magnetites;
4. Magnetite-hematite mixtures;
5. Titaniferous magnetites.

It is felt that the time is approaching when it will be economically possible to utilize Canadian ores to supply Canadian furnace requirements and the iron and steel industry with products of Canadian origin. The annually increasing percentage of ores being beneficiated in other countries together with the developments in the technique of beneficiation processes and the vast amount of experimental work being done on direct reduction processes in the past few years have brought closer the time when Canadian ores can be utilized. It is believed that mixtures of Canadian beneficiated ores in the proper proportions will give grades of pig iron more adaptable to the production of diversified iron and steel products. It is, therefore, the purpose of the Division of Ore Dressing and Metallurgy to investigate the beneficiation of Canadian ores of the foregoing types and also their adaptability to direct reduction or sponge iron processes.

Direct reduction processes are under investigation in Norway, Sweden, Germany, Japan, and the United States, and large sums of money are being spent to put them on a commercial basis. In the European countries, where high-grade scrap is scarce, processes have been developed on a presumably economic basis and production plants have been erected. It is claimed that sponge iron is a superior product for the manufacture of high-grade irons and steels. The metallurgists of the Mines Branch have been and are watching closely the developments of these processes, in view of their application to Canadian ores and conditions. The plan of investigation as outlined is to determine, first the quality of the irons and steels produced from sponge iron made from Canadian ores in comparison with accepted grades made by present-day methods, and second, if superior products are obtained, which of the sponge iron processes would be more adaptable to Canadian ores and conditions. Direct reduction processes may provide a more suitable base for the manufacture of high-grade irons and steels and, while there is no immediate likelihood of such processes encroaching to any appreciable extent upon present blast furnace operations, it is possible that in time steel-making practice may be modified to permit of at least a partial substitution of sponge iron for pig iron in the manufacture of the more common grades of steel.

Canadian titaniferous magnetites are a source of iron upon which little investigative work has been done. These ores contain small amounts of vanadium, used extensively in the manufacture of alloy steels. The possibility of recovering the vanadium content lends interest to the problem of their commercial utilization, and will be investigated.

The making of alloy steels in Canada is in its infancy, but in other countries this branch of the steel industry has had a remarkable expansion in the last ten years. Each year sees the field for alloy steel widen and statistics show that progress in the alloy steel industry has been eighteen times as rapid as in the general steel industry during the past fifteen years. The development of compositions of alloy steels and the proper treatment of such steels to meet special requirements are constantly recurring problems. In these developments Canada cannot afford to lag behind, particularly in view of the fact that the alloy steel industry provides the most important market for our nickel and may provide important outlets for our copper, molybdenum, and other non-ferrous metals. We should be as well-informed as are the producers in other countries in the technique of making and treating alloy steels.

The electric melting furnace is particularly well adapted to the making of alloy steels, but since cheap electric power is not generally available the bulk of the production has been made by the open-hearth process. In spite of the lack of cheap power, the electric furnace has made and is making rapid strides, and certain grades of alloy steel are being produced in large electric melting furnaces at a cost that enables them to compete successfully with open-hearth steels on a price basis. It seems clear that with our cheap hydro-electric power we have one advantage that should offset to a considerable extent that of mass production held by the producers in other countries and it would appear logical that our development in the field of special steels should be with the electric furnace. With the facilities available on completion of the new laboratories, the Mines Branch can be of considerable service to the steel industry, in the working out of electric steel-making technique, in the development of new useful compositions, in the working out of methods of heat treatment, and in co-operating with steel producers in overcoming difficulties encountered in practical operations.

It is anticipated that the investigative work will follow three distinct but closely related lines:—

1. Broad problems connected with the beneficiation and utilization of Canadian iron ores, and the investigation of both new and established processes of iron and steel production and their applicability to Canadian ores and conditions.

2. The development of new steels and other alloys and the working out of the proper methods of melting, working, and treating such alloys for specific uses.

3. Co-operation with producers and consumers of iron and steel products in solving specific problems.

BUILDING AND EQUIPMENT

The new pyrometallurgical laboratory which has been erected is 105 feet long by 56 feet wide and will house the following furnace equipment:—

Melting Equipment

1. An Ajax-Northrup high frequency induction melting unit, consisting of the following main items:—

(*a*) Motor-generator set, induction type motor rated at 260 h.p., 3 phase, 60 cycle, 550 volt, 1,800 r.p.m. Generator rated at 150 kilowatts, output at 0·9 power factor; 900 volt, 960 cycle, single-phase. Complete with switchboard, condensers, etc.

(*b*) A 500-pound melting furnace (nose-tilt), complete with hoist for tilting.

(*c*) A 50-pound melting furnace, arranged to operate from the 150-kilowatt motor-generator set but not simultaneously with the 500-pound furnace.

2. A one-ton Heroult type, arc melting furnace complete with transformers and electrical equipment. Transformer equipment will consist of three single-phase, 250 K.V.A. units with special provisions to permit a selection of voltages over a wide range. This is to provide for the use of these transformers for additional electric furnaces.

3. A gas-fired crucible, brass melting furnace.

4. A small cupola furnace for cast iron.

The melting equipment will be served by a 3-ton overhead crane. Facilities for the pouring of ingots and for other purposes will be provided.

Roasting, Sintering, and Metallizing Furnaces

1. A 36-inch inside diameter, 6-hearth, mechanically rabbled, Herreshoff roasting furnace, with top drying hearth. This furnace has been in use for some years on non-ferrous ores.

2. A Dwight-Lloyd sintering unit, consisting of two pans. This unit has been in use for some years for making sintering tests on iron ore concentrates, flue dust, and non-ferrous ore concentrates.

3. A rotary-kiln type, oil-fired, metallizing furnace, 22 feet long, large diameter 42 inches for 6 feet at the discharge end, tapered section 4 feet long; small diameter 21 inches for 12 feet at the feed end. This furnace has been in use for metallizing the iron content in ilmenite ores.

4. A 60-kilowatt, rotating retort electric furnace, batch type, especially designed for making sponge iron by gaseous or solid fuel reduction of the ore. Capacity about 250 pounds of sponge iron.

Space is provided for supplementing the above equipment as required. Additional metallizing equipment will be installed to investigate the process or processes adaptable to Canadian iron ores and conditions.

Heat-treating Furnaces

A separate laboratory, 54 feet long by 14 feet wide, is provided for the heat-treating furnaces. It is located on a mezzanine floor at one end of the building, extending for its full width. The equipment will consist of:—

A box-type, electrically heated furnace, inside dimensions approximately 12 inches wide by 24 inches deep by 9 inches high.

A box-type, electrically heated furnace, inside dimensions approximately 20 inches wide by 36 inches deep by 18 inches high.

A pot-type, electric furnace, inside dimensions of pot 12 inches diameter by 18 inches deep.

An electrically heated tempering oven. Approximate dimensions 24 inches wide by 21 inches deep by 12 inches high.

An electrically heated oil tempering bath, approximate dimensions 30 inches long by 16 inches wide by 12 inches deep.

Quenching tanks and other miscellaneous equipment.

Additional furnaces and other equipment will be installed as required.

Pyrometer Equipment

A separate laboratory, 21 feet long by 14 feet wide, is provided for the pyrometer equipment. It is located above the room housing the motor generator set and other electrical equipment for the high frequency furnace, and is adjacent to the heat-treating laboratory. In addition to housing the temperature-recording controller for the heat-treating furnaces, it will also serve as a laboratory for the calibration and checking of pyrometer equipment. Each furnace is equipped with a Leeds and Northrup recording controller and there are provided for checking and calibrating purposes an electric furnace specially designed for checking thermocouples, a portable double-scale potentiometer indicator for rare and base metal thermocouples, and a triple range optical pyrometer.

Mechanical and Physical Testing Equipment

A laboratory, 21½ feet long by 18½ feet wide, is being provided for mechanical and physical testing equipment which will consist of the following:—

An Amsler tensile testing machine of 100 tons capacity, complete with arrangements for compression, transverse, and bending tests.

An Amsler single blow impact testing machine, capacity 240 ft.-lb., constructed for both Izod and Charpy tests.

An Olsen combination toughness and torsion testing machine.

An Olsen bend testing machine.

An Alpha Brinell hardness testing machine.

A Rockwell hardness tester—Model 3H.

A Shore scleroscope—Model D.

A Vickers hardness testing machine.

Other testing machines will be added as the occasion demands.

Metallographic Equipment

A laboratory, 21½ feet long by 18½ feet wide, is being provided for metallographic equipment which will consist of the following:—

A Zeiss, 8-inch by 10-inch, horizontal metallographic unit, complete with all necessary accessories.

A Bausch and Lomb binocular microscope.

A camera for macrophotography.

A Leeds and Northrup transformation point apparatus.

A Rockwell dilatometer, laboratory model.

Sample preparation machines, including cutting wheel, grinding wheels, and polishing machines, for which a separate laboratory is being provided.

A separate photographic dark room, completely equipped, is also being provided adjacent to the metallographic laboratory.

Chemical Laboratories

The chemical laboratories of the Division are fully equipped and staffed for the conducting of analyses and special chemical investigations in connexion with ores and metallurgical products.

Mechanical Shops

Complete facilities for the machining of test bars and other metallurgical samples are afforded by the mechanical division of the Mines Branch, which maintains machine shop, forging shop, and other services necessary for maintenance and development of the mechanical side of the Branch's activities.

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