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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)

1923

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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, 1923

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm

Chief of Division

During the year 1923, the laboratories of the Division were devoted entirely to experimental and research work in connexion with the treatment of Canadian ores and metallurgical products. The results of the various investigations are given in the reports by the officials of the Division directly engaged on the work. A brief review of the investigations is given below.

Report No. 185

THE CONCENTRATION OF THE ORE FROM THE ROSSLAND VELVET MINES, LTD., ROSSLAND, B.C., BY R. K. CARNOCHAN

The experimental work showed that the ore could be concentrated with good recoveries of the gold and copper values by table concentration and flotation, or by flotation alone. Tabling and flotation gave the higher recoveries; straight flotation gave the higher grade product.

Report No. 186

THE CONCENTRATION OF THE ORE FROM THE COLUMBIA-KOOTENAY MINE, ROSSLAND, B.C., BY R. K. CARNOCHAN

The shipment submitted was very low grade, badly weathered and oxidized. The experimental tests, however, showed that the copper and gold values could be concentrated by table concentration and flotation, or by straight flotation. On higher grade ore, results similar to those obtained on other Rossland ores are possible.

Report No. 187

SELECTIVE FLOTATION TESTS ON SULLIVAN MINE ORE, KIMBERLEY, B.C., BY C. S. PARSONS

The experimental work on Sullivan ore was done in order to obtain a comparison of results using various flotation reagents. Tests were made using reagents other than those used at the Trail plant and using their reagents, for a comparison of results. From the results it was found that Z cake and Fumol were very satisfactory for the flotation of the lead, and light hardwood creosote and No. 34 gravity fuel oil gave good results for the flotation of the zinc. Better results were obtained with these reagents than by using the reagents in practice at Trail.

Report No. 188

FLOTATION TESTS ON COPPER MOUNTAIN ORE, COPPER MOUNTAIN, B.C., BY R. K. CARNOCHAN

The test work was done to determine the flotation reagents suitable for use in the concentration of this ore by flotation. The copper minerals present being chalcopyrite and bornite, permits the production of a high-grade concentrate. The recovery of the copper values varied with the degree of fineness to which the ore was ground. Of the flotation reagents used, TT mixture gave the higher grade concentrates, showing a selective action for the copper minerals.

Report No. 190

CONCENTRATION OF A COPPER GOLD ORE FROM SURF INLET, B.C., BY R. K. CARNOCHAN

The test work on the Belmont Surf Inlet ore was done in order to determine the results obtained by different methods of treatment, and to study the effects of different oil mixtures used in the flotation of the ore. Table concentration and flotation gave much better results than flotation alone. High recoveries of the gold, silver, and copper values were made in a high-grade concentrate. Of the flotation reagents used, TT mixture had a better selective action for the copper minerals, giving a higher grade concentrate than the other flotation reagents.

Report Nos. 191-192

GOLD ORE FROM CRANBERRY HEAD, YARMOUTH CO., N.S., BY C. S. PARSONS

Experimental test and research work was done on two shipments of ore, to obtain the necessary data for determining the best metallurgical treatment. It was found that amalgamation was essential in the treatment of this class of ore; that the amalgamation tailing could be cyanided without difficulty, with the recovery of practically all the remaining gold values, or that the amalgamation tailing could be concentrated on tables with the production of a concentrate, marketable for its gold and arsenic content, or the gold recovered by pan amalgamation and the residue marketed for its arsenic and gold content. The reports describe in detail the various methods that could be applied, and give the results obtained from each method. The investigator has described very fully the different methods that could be applied to Nova Scotia gold ores.

Report No. 193

AMALGAMATION AND CYANIDE TESTS ON THE ORE FROM THE OPHIR LODE MINE, B.C., BY R. K. CARNOCHAN

Experimental tests were made to determine if amalgamation at 80 mesh was preferable to amalgamation at 40 mesh, and if it would be necessary to cyanide after amalgamation. It was found that amalgamation at 40 mesh was as efficient as amalgamation at a finer mesh and that cyanidation was necessary after amalgamation to obtain a high recovery of the gold values. The results show high recoveries by amalgamation and cyanidation.

Report No. 194

THE CONCENTRATION OF THE COMPLEX SULPHIDE ORE OF THE EUSTIS MINE, QUEBEC, BY C. S. PARSONS

The experimental work was done on a carload shipment, and consisted of making semi-commerical runs to determine the correct conditions for the selective flotation of the chalcopyrite from the pyrite, in this heavy sulphide ore. The results showed that a copper concentrate containing better than 20 per cent copper could be made with a recovery of over 90 per cent of the copper values. These results were obtained by accurate control of the proper conditions in the ball mill, classifier and flotation circuits, of the grinding, pulp density, reagents, etc.

Report No. 195

EXPERIMENTAL TESTS ON GOLD ORE FROM THE KIRKLAND LAKE GOLD MINING CO., KIRKLAND LAKE, ONT., BY R. K. CARNOCHAN

Tests were made on three samples, one of high-grade ore containing molybdenite and tellurides, one of very low-grade ore, and one of mill tailings. The experimental work on the high-grade sample was conducted to determine whether the presence of molybdenite and tellurides had any effect on the recovery of the gold values, and what recoveries could be obtained. Considering the grade of the ore, good recoveries were made by cyanidation. The molybdenite does not seem to have any deleterious effect on the recovery of the gold by the cyanide process. The presence of tellurides requires that the ore be ground extremely fine for cyanidation. Cyanidation gave good results on the low-grade ore, and also on the tailing sample. The high tailings seem to be due to the coarseness of the grinding, as shown in the screen tests.

Report No. 196

EXPERIMENTAL TESTS ON GOLD ORE FROM THE WRIGHT HARGREAVES MINES, LTD., KIRKLAND LAKE, ONT., BY R. K. CARNOCHAN

Tests were made on a high-grade sample, containing molybdenite, tellurides and free gold, to determine whether the presence of molybdenite and tellurides had any effect on the recovery of the gold values. Considering the grade of the sample, good recoveries were made by cyanidation. The molybdenite does not seem to have any deleterious effect on the recovery of the values. The presence of tellurides requires that the ore be ground extremely fine for good extraction of the values by cyanidation.

Included in the report is a microscopic examination of selected specimens of the ore by E. A. Thompson, showing the presence of the various minerals.

Report No. 197

GOLD ORE FROM DASSERAT ROUYN GOLDFIELDS, LTD., ROUYN TOWNSHIP, TÉMISCAMINGUE DIST., QUE., BY R. K. CARNOCHAN

Experimental tests were made to determine whether the ore was amenable to treatment with high recoveries of the precious metal values. The tests show the ore as submitted to be very amenable to treatment by the cyanide process. A very high extraction of the values was obtained.

Report No. 198

CONCENTRATION TESTS ON GRAPHITE FROM THE CANADIAN GRAPHITE CORP., GUENETTE, QUE., BY C. S. PARSONS AND R. K. CARNOCHAN

Tests were made to check up the results being obtained in the company's concentrator and to determine the percentages of recoverable graphite in as coarse a form as possible, with high recoveries of the graphite. The tests show that there should be no difficulty in recovering a good percentage of the graphite in coarse flake; that good recoveries can be made; that high-grade flake can be produced by flotation and subsequent refining of the flotation concentrate. The point to which concentration should be carried before refining operations begin, is discussed in the report.

Report No. 199

SELECTIVE FLOTATION OF THE COPPER-NICKEL ORE OF SHEBANDOWAN LAKE DISTRICT, ONTARIO, BY C. S. PARSONS

Experimental tests were conducted to determine whether the ore could be concentrated and a high-grade concentrate obtained carrying the copper-nickel, cobalt, and precious metal values, with good recoveries of these values. The tests show that the ore can be concentrated by selective flotation, with the production of a concentrate assaying 15 per cent in copper, nickel, and cobalt, with a recovery of 90 per cent of these values. Further work will be conducted to determine whether the precious metal values are reporting in the flotation concentrate.

Report No. 200

THE CONCENTRATION OF MOLYBDENITE ORE FROM THE PROPERTIES OF THE MOLYBDENITE REDUCTION CO., NEAR AMOS, QUE., BY C. S. PARSONS

The object in view in conducting experimental tests was to determine whether the ore was amenable to treatment by the flotation of the molybdenite from the gangue minerals, with the production of a high-grade marketable product with a high recovery of the molybdenite values. The results showed the ore to contain an average of 2 per cent molybdenite from which a high-grade concentrate, averaging 90·8 per cent molybdenite was obtained with a recovery of 95 per cent of the molybdenite values in the ore. The tailings contained 0·10 per cent molybdenite. No difficulty was experienced in obtaining the above results which were from tonnage check tests.

Report No. 201

THE CONCENTRATION AND TREATMENT OF ARSENICAL GOLD ORE FROM BATHURST, N.B., BY J. S. GODARD

Gravity concentration tests were carried out by means of jigs and tables, and by flotation, to determine whether a high-grade arsenical concentrate could be produced from the ore. Flotation gave the better results, producing a concentrate containing 30 per cent arsenic with a recovery of 90 per cent of the arsenic values. The ore contains small amounts of gold and silver values, about $2.50 per ton. The arsenopyrite concentrates contain $3 to $3.50 per ton in gold and silver. Leaching tests made on the concentrates showed that the cost of leaching and consumption of cyanide was too high to make the extraction of precious metal values by this method feasible.

Report No. 202

THE TAILING DUMPS OF THE NAUGHTON GOLD MINE, NAUGHTON, ONT., BY C. S. PARSONS

The report contains the estimated reclaimable tonnage in the dumps, the average assay of arsenic and gold content, and the results of simple concentration tests. The assays and results of the concentration tests indicate that it will not pay, at present, to reclaim and concentrate the dumps, especially the lower dump, as the grade and amount of concentrate obtainable would be too low to stand the cost of concentration, treatment charges, and transportation to the smelter.

Report No. 203

CONCENTRATION OF THE ZINC-IRON MIDDLING FROM THE DUMP AT NOTRE-DAME DES ANGES, QUE., BY C. S. PARSONS

The purpose of the experimental work was to determine whether the zinc, lead, gold and silver values could be concentrated by the elimination of the iron sulphides and any gangue minerals. Magnetic separation and flotation tests were made. By magnetic separation after giving the material a magnetic roast, a high recovery of the zinc was obtained. The zinc concentrate contains all the siliceous material and this prevents a high-grade zinc product from being made. Table concentration would improve the grade of the zinc product. The flotation tests indicate that a 40 per cent zinc product can be obtained by selective flotation of the zinc from the iron sulphides, but attention is drawn to the difficulty in obtaining and maintaining the correct flotation conditions in the treatment of a limited tonnage of partly oxidized dump material.

Section III

The investigations carried on by the staff of the chemical laboratories of the Division, were as follows:—

THE VALUE OF GIVING THE SILVER RESIDUES FROM THE DOMINION REDUCTION COMPANY, COBALT, ONT., A WEAK SULPHURIC ACID WASH PREVIOUS TO CYANIDING, BY B. P. COYNE.

It was found that a 1 per cent sulphuric acid wash was as effective as a stronger acid wash for removing soluble salts; that it was easier to wash out the soluble salts when a weak solution was used, and that this treatment reduces the cyanide consumption.

DETERMINATION OF SOLUBLE CYANICIDES IN CRANBERRY HEAD GOLD ORE, AND THE EFFECT OF CERTAIN REAGENTS ON CYANIDE CONSUMPTION, BY B. P. COYNE.

It was found that the ore does not contain any cyanicides soluble under working conditions, and that the use of Na_2O_2 reduced the cyanide consumption.

DETERMINATION OF SOLUBLE SALTS IN EUSTIS COPPER ORE, AND THE EFFECT OF ADDING LIME FOR SELECTIVE FLOTATION OF THE CHALCOPYRITE, BY B. P. COYNE.

It was found that the oxidized ore contained a much greater amount of soluble salts than the unoxidized ore, and that the addition of lime prevented the iron from going into solution.

THE PRECIOUS METAL VALUES IN PRODUCTS FROM SMELTING TESTS ON COPPER-NICKEL ORE OF SHEBANDOWAN LAKE, ONT., BY H. C. MABEE.

The matte produced was low grade, representing 37·5 per cent of the weight of the ore used. Analysis of the matte showed it to contain gold 0·03 oz., platinum 0·085 oz., palladium 0·131 oz., and that almost all the total precious metal values in the ore were contained in the matte.

THE ELIMINATION, BY SELECTIVE FLOTATION, OF BARREN PYRRHOTITE FROM THE NICKELIFEROUS PYRRHOTITE ORES OF THE SUDBURY DISTRICT, BY H. C. MABEE.

A study was made of the flotation products to determine the amounts of barren pyrrhotite that were eliminated in the tailings by selective flotation of the ores. This was found to be 30 to 50 per cent of the pyrrhotite in the ores. This elimination of barren pyrrhotite, together with the gangue minerals, is significant, and means a saving in several directions if the ores were concentrated before smelting.

THE HYDROMETALLURGICAL TREATMENT OF PYRRHOTITE AND PYRITE ORES WITH THE PRODUCTION OF ELECTROLYTIC IRON, AND THE RECOVERY OF SULPHUR AND OTHER METALS AS BY-PRODUCTS, BY R. J. TRAILL.

The results of the test work carried on during the year are given in the report. The test work was carried to a point where better facilities for carrying on the investigation were required. A new laboratory was built and is being equipped, so that a complete cycle of operations can be maintained. Although the equipment is on a small scale and only small scale tests can be conducted, it was deemed advisable to work out the problems on a small laboratory scale before embarking on any extensive large scale work. The process has possibilities of being commercially applicable to Canadian pyrrhotite and pyrite ores.

Section IV

As a large part of the experimental work done in the laboratories is on the flotation of ores, a summary of the work on selective flotation as applied to Canadian ores was compiled by C. S. Parsons. This report is inserted as Section IV.

Section V

A considerable amount of experimental work has been done in the laboratories on the beneficiation of Canadian iron ores. A summary of the more important tests was compiled by W. B. Timm. It is inserted under Section V.

Section VI

During the last two years, experimental tests were made on the selective flotation of the lower grade nickeliferous pyrrhotite ores of Ontario. The ores were from the Sudbury nickel area and the Shebandowan Lake district of Ontario. The results of the various tests have been collected and compiled in one report. It is inserted as Section VI.

Section VII

During the year a number of ore concentration plants in Canada were visited by officials of the Division, and a study made of the processes being applied to the concentration of Canadian ores. A description of some of the important ones is included in this section.

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

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

Report No.	Ore or product	Source of shipment	Shipper and address	Weight pounds
185	Gold-copper..	Rossland, B.C..........	Rossland Velvet Mines, Ltd., Rossland, B.C................................	160
186	Gold-copper..	Rossland, B.C..........	Columbia-Kootenay Mines, Ltd., Rossland, B.C..........................	60
187	Lead-zinc.....	Kimberley, B.C........	Cons. Mining and Smelting Co. of Canada, Ltd., Kimberley, B.C...........	100
188	Copper.......	Copper Mountain, B.C..	Canada Copper Corporation, Copper Mountain, B.C......................	100
190	Gold-copper..	Surf Inlet, B.C..........	Belmont Surf Inlet Mines, Ltd., Surf Inlet, B.C...........................	150
191	Gold.........	Cranberry head, Yarmouth co., N.S.......	S. F. Johnson, Boston, Mass., U.S.A....	280
192	Gold.........	Cranberry head, Yarmouth co., N.S.......	S. F. Johnson, Boston, Mass., U.S.A....	380
193	Gold.........	Ophir Lode mine, Lardeau, B.C............	H. B. Morrison, Nelson, B.C...........	20
194	Copper.......	Eustis, Que............	Eustis Mining Co., Eustis, Que..........	40,000
195	Gold.........	Kirkland Lake, Ont.....	Kirkland Lake Gold Mining Co., Kirkland Lake, Ont.......................	200
196	Gold.........	Kirkland Lake, Ont.....	Wright Hargreaves Mines, Ltd., Kirkland Lake, Ont.......................	150
197	Gold.........	Rouyn township, Témiscamingue dist., Que.	Dasserat-Rouyn Gold Mines, Ltd., Ottawa, Ont.........................	950
198	Graphite.....	Guenette, Que..........	Canadian Graphite Corporation, Guenette, Que............................	300
199	Copper-nickel	Shebandowan lake, Ont.	Jamieson and Peacock, Duluth, Minn., U.S.A..............................	400
200	Molybdenite .	La Corne and Malartic tps., Abitibi dist., Que.	Molybdenite Reduction Co., Montreal, Que..................................	20,186
201	Arsenic-gold..	Bathurst, N.B..........	L. D. Densmore, Bathurst, N.B........	200
202	Arsenic-gold..	Naughton, Ont.........	Mines Branch, Department of Mines, Ottawa, Ont.........................	200
203	Zinc..........	Notre-Dame des Anges, Que..................	British Metals Corporation, Montreal, Que..................................	110

II

REPORTS ON INVESTIGATIONS CONDUCTED

Report No. 185

THE CONCENTRATION OF THE COPPER-GOLD VALUES IN THE ORE FROM THE ROSSLAND VELVET MINES, ROSSLAND, B.C.

R. K. Carnochan

Shipment.—A shipment of ore, 157½ pounds net, was received at the Ore Dressing and Metallurgical Laboratories, October 12, 1922, from the Rossland Velvet Mines, Ltd., Rossland, B.C.

Purpose of experimental tests.—The ore contains values in gold, silver, and copper, and tests were desired to determine a suitable method of concentrating these values.

Analysis of shipment.—The shipment was divided into three lots. Samples of these lots were taken and found to contain:—

	Gold oz. per ton	Silver oz. per ton	Copper per cent
Milling ore	0·26	0·32	2·58
No. 1 dump ore	0·18	0·26	1·01
Oxidized dump ore	0·26	0·34	1·12

Experimental Tests

Test No. 1—No. 1 dump ore; tabling and flotation.—A lot of 2000 grammes of No. 1 dump ore −20 mesh, was screened on 80 mesh. This gave:—

−20+80	1,310 grammes
−80	690 "

The −20+80 was run over a small Wilfley table making a concentrate, middling, and tailing. The −80 and the table middling were ground for 15 minutes in a small ball mill with 25 drops of water-gas tar and then floated in a small Ruth machine with 2 drops of P.T.T. No. 350.

Test No. 2—Milling ore; tabling and flotation.—A lot of 1417 grammes of milling ore, −20 mesh, was screened on 40 and 80 mesh. This gave:—

−20+40	640 grammes
−40+80	286 "
−80	491 "

The three sizes were tabled separately making in each case a concentrate and a tailing. The −20+40 table tailing was ground for 5 minutes in a small ball mill with 25 drops of water-gas tar, the −40+80 tailing was then added and the ball mill run for 5 minutes more, the −80 tailing was now added and the charge ground again for 5 minutes. The ball mill charge was then floated in a small Ruth machine with 5 drops P.T.T. No. 350.

Test No. 3 on oxidized dump ore; tabling and flotation.—A lot of 1655 grammes of oxidized dump ore −40 mesh was screened on 80 mesh. This gave:—

−40+80	690 grammes
−80	965 "

Each of the above sizes was tabled, making a concentrate and a tailing. The −40+80 table tailing was ground for 5 minutes in a small ball mill with 25 drops of water-gas tar. The −80 tailing was then added to the mill and the whole charge ground for 5 minutes more. The ball mill charge was then floated in a small Ruth machine with 10 drops of P.T.T. No. 350.

Test No. 4 on No. 1 dump ore; tabling and flotation.—This test is similar to Test No. 2.

−20 +40	692 grammes
−40 +80	379 "
−80	557 "

The flotation concentrate was re-run to clean it up. This gives from flotation a concentrate, a middling, and a tailing.

Test No. 5 on milling ore; flotation.—The ore, −20 mesh, was ground for 30 minutes with 25 drops of water-gas tar in a small ball mill and then floated in a small Ruth machine with 3 drops P.T.T. No. 350. The concentrate was re-run to clean it up. This gives 3 products from flotation.

Test No. 6 on No. 1 dump ore; flotation.—Similar to test No. 5.

Test No. 7 on oxidized dump ore; flotation.—Similar to test No. 5.

Test No. 8 on oxidized dump ore; flotation.—The ore, −20 mesh, was ground for 30 minutes in a small ball mill with 0·6 c.c. heavy hardwood creosote oil (F.P.L. 26) and 0·4 c.c. of a mixture of 40 per cent coal tar and 60 per cent coal tar creosote. The ball mill charge was then floated in a small Ruth machine, the concentrates being re-run to clean them.

Test No. 9 on oxidized dump ore; flotation.—The ore, −20 mesh, was ground for 30 minutes in a small ball mill with 3 grammes soda ash and then floated in a small Ruth machine with 10 drops of TT (Callow alphabetical reagent). The concentrate was re-run to clean it.

Test No. 10 on oxidized dump ore; flotation.—This test is the same as test No. 9, except that 3 grammes of lime were used in place of the soda ash and 10 drops of XY were used in place of the TT.

Test No. 11 on oxidized dump ore; flotation.—This test is the same as test No. 9, except that lime was used in place of soda ash, and thiofizzan in place of TT.

Test No. 12 on oxidized dump ore; flotation.—This test is the same as test No. 9, except that lime was used in place of soda ash.

Test No. 13 on oxidized dump ore; flotation.—This test is the same as test No. 9, except that XY was used in place of TT.

Test No. 14 on oxidized dump ore; flotation.—This test is the same as test No. 9, except that thiofizzan was used in place of TT.

Test No. 15 on milling ore; flotation.—This test is similar to test No. 9. Three grammes of lime were used in grinding the ore, and 5 drops of TT were used for flotation.

Test No. 16 on No. 1 dump ore; flotation.—This test is similar to test No. 9. Three grammes of lime were used in grinding the ore and 5 drops of TT used for flotation.

Results of Experimental Tests

Test No.	Product	Weight grms.	Analysis			Per cent of values		
			Au. oz.	Ag. oz.	Cu. per cent	Au.	Ag.	Cu.
1	Table concentrate	169	1·02	0·86	3·92	64·9	51·1	73·6
	Flotation concentrate	67	1·24	1·38	12·17			
	Table tailing	684	0·06	0·08	0·33			
	Flotation tailing	1,070	0·09	0·16	0·28			
	Loss	10	0·09	0·16	0·28			
2	Table concentrate +40	83	0·96	0·94	7·30	84·5	79·5	91·0
	“ “ +80	57	0·98	0·74	8·65			
	“ “ −80	72	0·84	0·88	10·12			
	Flotation concentrate	69	0·74	1·18	16·79			
	“ tailing	1,015	0·04	0·06	0·26			
	Loss	121	0·04	0·06	0·26			
3	Table concentrate +80	173	0·66	0·38	3·37	74·7	55·2	91·0
	“ “ −80	165	0·80	0·46	3·20			
	Flotation concentrate	65	0·76	1·14	8·15			
	“ tailing	1,070	0·08	0·14	0·13			
	Loss	182	0·08	0·14	0·13			
4	Table concentrate +40	127	0·68	0·48	2·87	78·8	68·9	88·1
	“ “ +80	62	1·08	0·70	3·67			
	“ “ −80	70	0·78	0·56	3·25			
	Flotation concentrate	62	0·73	0·95	7·62			
	“ middling	145	0·47	0·63	0·40			
	“ tailing	1,027	tr.	tr.	0·10			
	Loss	135	tr.	tr.	0·10			
5	Flotation concentrate	115	0·98	0·80	16·78	57·7	54·6	77·2
	“ middling	153	0·30	0·26	1·91	23·5	23·6	11·7
	“ tailing	734	0·05	0·05	0·38	18·8	21·8	11·1
6	Flotation concentrate	76	1·32	0·88	10·23	61·0	52·4	68·7
	“ middling	173	0·24	0·22	0·86	25·2	29·8	13·2
	“ tailing	758	0·03	0·03	0·27	13·8	17·8	18·1
7	Flotation concentrate	44	1·10	1·14	11·99	17·4	13·9	46·8
	“ middling	178	0·60	0·62	2·64	38·4	30·7	41·6
	“ tailing	768	0·16	0·26	0·17	44·2	55·4	11·6
8	Flotation concentrate	150	....	....	5·95	....	....	79·2
	“ middling	188	....	....	0·69	....	....	11·5
	“ tailing	655	....	....	0·16	....	....	9·3
9	Flotation concentrate	77	....	....	9·38	....	....	66·9
	“ middling	134	....	....	1·32	....	....	16·4
	“ tailing	781	....	....	0·23	....	....	16·7
10	Flotation concentrate	135	....	....	6·80	....	....	80·4
	“ middling	118	....	....	0·63	....	....	6·5
	“ tailing	744	....	....	0·20	....	....	13·1
11	Flotation concentrate	165	....	....	5·57	....	....	82·6
	“ middling	146·5	....	....	0·76	....	....	10·0
	“ tailing	695	....	....	0·12	....	....	7·4
12	Flotation concentrate	126	0·73	0·93	7·75	37·2	38·7	79·2
	“ middling	131	0·33	0·45	0·88	17·5	19·4	9·3
	“ tailing	746·5	0·15	0·17	0·19	45·3	41·9	11·5
13	Flotation concentrate	38	....	....	14·31	....	....	46·4
	“ middling	175·5	....	....	2·78	....	....	41·7
	“ tailing	775	....	....	0·18	....	....	11·9

Results of Experimental Tests—Concluded

Test No.	Product	Weight grms	Analysis			Per cent of values		
			Au. oz.	Ag. oz.	Cu. per cent	Au.	Ag.	Cu.
14	Flotation concentrate........	171·5	....	....	5·66	....	....	82·2
	" middling..........	118	....	....	1·05	....	....	10·5
	" tailing............	713	....	....	0·12	....	....	7·3
15	Flotation concentrate........	82·2	1·34	2·12	26·50	46·8	54·3	88·0
	" middling..........	77·1	0·63	0·69	2·23	20·7	16·6	6·9
	" tailing............	849·7	0·09	0·11	0·15	32·5	29·1	5·1
16	Flotation concentrate........	40·3	2·09	2·31	17·94	44·8	46·0	77·4
	" middling..........	78·8	0·75	0·71	1·43	31·5	27·6	12·1
	" tailing............	889·3	0·05	0·06	0·11	23·7	26·4	10·5

Screen Test of Flotation Tailings

Test No.	+35	+48	+65	+100	+150	−150	+200	−200
	Per cent	Per cent	Per cent	Per cent	Per cent	Per cent	Per cent	Per cent
2..........	0·4	2·4	15·8	27·0	9·7	44·7		
3..........	....	....	1·2	5·9	15·1	77·8		
4..........	....	....	10·0	25·6	13·6	50·8		
5..........	....	....	0·6	8·8	17·3	....	18·3	55·0
6..........	....	....	0·1	3·2	10·0	....	23·1	63·6
7..........	....	....	0·1	6·3	10·9	....	22·7	60·0

SUMMARY

Milling ore.—The results obtained on the milling ore are as follows:—

Test No.	Method used	Product	Analysis			Per cent of values		
			Au. oz.	Ag. oz.	Cu. per cent	Au.	Ag.	Cu.
2	Tabling and flotation...	Total conc....	0·88	0·94	10·63	84·5	79·5	91·0
5	Oil flotation...........	Concentrate..	0·98	0·80	16·78	57·7	54·6	77·2
		Middling.....	0·30	0·26	1·91	23·5	23·6	11·7
15	Flotation with alphabetical reagents.......	Concentrate..	1·34	2·12	26·50	46·8	54·3	88·0
		Middling.....	0·63	0·69	2·23	20·7	16·6	6·9

No. 1 Dump ore.—The results obtained on No. 1 dump ore are as follows:—

Test No.	Method used	Product	Analysis			Per cent of values		
			Au. oz.	Ag. oz.	Cu. per cent	Au.	Ag.	Cu.
4	Tabling and flotation...	Total conc....	0·79	0·63	4·02	78·8	68·9	88·1
		Middling.....	0·47	0·63	0·40	21·2	31·1	4·0
6	Oil flotation............	Concentrate..	1·32	0·88	10·23	61·0	52·4	68·7
		Middling.....	0·24	0·22	0·86	25·2	29·8	13·2
16	Flotation with alphabetical reagents.......	Concentrate..	2·09	2·31	17·94	44·8	46·0	77·4
		Middling.....	0·75	0·71	1·43	31·5	27·6	12·1

Oxidized dump ore.—The results obtained on oxidized dump ore are:—

Test No.	Method used	Product	Analysis			Per cent of values		
			Au. oz.	Ag. oz.	Cu. per cent	Au.	Ag.	Cu.
3	Tabling and flotation...	Total conc....	0·73	0·54	4·07	74·7	55·2	91·0
7	Flotation..............	Concentrate..	1·10	1·14	11·99	17·4	13·9	46·8
		Middling.....	0·60	0·62	2·64	38·4	30·7	41·6
8	Oil flotation............	Concentrate..	....	....	5·95	....	....	79·2
		Middling......	....	....	0·69	....	....	11·5
12	Flotation with alphabetical reagents.......	Concentrate..	0·73	0·93	7·75	37·2	38·7	79·2
		Middling.....	0·33	0·45	0·88	17·5	19·4	9·3

CONCLUSIONS

1. Tabling and flotation give higher recoveries and lower grade products than flotation alone.

2. Flotation with alphabetical reagents gives better products and a higher copper recovery than oil flotation, but the oil flotation gives better gold and silver recoveries.

3. Cyaniding at −200 mesh for 24 hours reduced the tailings from tests Nos. 2 and 3 to a trace in gold and silver.

4. The best method of treating this ore can only be decided by a careful consideration of freight rates and smelter charges.

Report No. 186

THE CONCENTRATION OF THE VALUES IN THE PYRRHOTITE FROM THE COLUMBIA-KOOTENAY MINE, ROSSLAND, B.C.

R. K. Carnochan

Shipment.—A shipment of 56 pounds of ore was received September 5, 1922, from the Columbia-Kootenay mine, Rossland, B.C.

Characteristics of shipment.—The chief mineral constituent was pyrrhotite, carrying small amounts of chalcopyrite, gold and silver. It was in a weathered and oxidized condition.

Purpose of experimental tests.—Tests were desired to determine if the copper and precious metal values could be concentrated to a marketable grade.

Analysis.—The ore was crushed to 4 mesh and a sample cut out. This sample gave upon analysis the following:—

Gold...........	0·06 oz. per ton	Copper..............	0·46 per cent
Silver............	0·08 "	Iron.................	36·35 "
Sulphur..........	20·46 per cent	Insoluble............	22·40 "

Experimental tests:

Tabling and flotation.—About 1,400 grammes of the ore, −20 mesh, was screened on 40 and 80 mesh. Each size was tabled separately on a small Wilfley table, making a concentrate and a tailing. The table tailings were mixed, ground in a ball mill, and then floated in a small Ruth machine. All products were dried, weighed, sampled and analysed. The table concentrates averaged 0·07 oz. gold per ton and 0·48 per cent copper. The flotation concentrate ran 0·12 oz. gold per ton and 2·65 per cent copper.

Flotation.—Nine flotation tests were made on the ore using different oils and reagents. In the best test, the concentrate ran 3·02 per cent copper and the copper recovery was 76·0 per cent. The next best test gave a concentrate of 1·80 per cent copper with a recovery of 84·5 per cent. The other tests gave very much poorer results.

Flotation and tabling.—Two tests were made to see if it would be possible to raise the grade of flotation concentrates by tabling. It was found that the table concentrate and tailing ran practically the same in copper.

CONCLUSIONS

1. The ore submitted contains very low values in copper, gold and silver, and even if it could be concentrated satisfactorily it could not be worked economically.
2. If ore of higher grade was found on the property it would likely be amenable to flotation, for with a higher head a higher concentrate and recovery would be obtained.
3. Tabling seems of little value in treating an ore of this kind, as due to the large sulphide content, 44 per cent of the heads go into the concentrate when tabled.
4. Cyaniding the tails from the tabling and flotation test at −200 mesh for 24 hours reduced them to a trace of gold and silver. With higher grade ore this method might be of use in treating flotation tailings, if they are too high in gold and silver to discard.

Report No. 187

FLOTATION TESTS ON SULLIVAN MINE ORE

C. S. Parsons

A shipment of 100 pounds of ore from the Sullivan mine, Kimberley, B.C., was received July 18, 1922, from the Consolidated Mining and Smelting Co. of Canada, Trail, B.C., for the purpose of comparing the results obtained from treating the ore with the reagents used in their practice, with those obtained from the use of other flotation reagents.

The accompanying table gives the results from a series of nine tests made by flotation. The first three of these tests were made on the ore as soon as it was received. No further work was done on the ore until December, when a test was run with the reagents now being used at the company's mill.

In the first three tests, a method of treatment was used that had previously been worked out for the separation of the lead and zinc in the North Star ore. Such excellent separation was obtained that it is doubtful whether it can be very much improved on by any method of straight flotation alone. No attempt was made to table the lead from the zinc concentrate, but no doubt this could be done and an increase in the recovery of the lead obtained. In running these tests it was necessary to make both a lead and zinc middling product, but in practice this would be returned continuously to the lead and zinc circuits, or treated separately for the recovery of the values.

Tests Nos. 1, 2 and 3.—The general procedure in making the tests was as follows:—

1,000 grammes of ore crushed dry to 30 mesh was placed in a small ball mill and in the case of the first test 0·1 gramme Z cake added together with 2½ grammes lime and ½ gramme of Selecto. The lime keeps down the iron and also tends to keep the zinc down, but the Selecto is added for that purpose. The charge was then ground to the required fineness and dumped into a Ruth flotation machine. A little Fumol was added to froth the cell and a lead concentrate was floated. The cell was then run out and the lead concentrate placed back in the cell and recleaned. A lead middling from this recleaning operation was obtained which was kept separate. The lead tailing, which in the meantime had settled, was decanted in order to get rid of the Selecto and lime.[1] Soda ash, about 5 pounds or less per ton and 1 pound per ton copper sulphate were added.[2] YZ mixture was then added and the dewatered pulp washed into the cell. The zinc was floated, a little Fumol being used to increase the froth. The zinc concentrate was recleaned, producing a zinc middling.

Considering the results obtained in the first three tests and from similar results obtained on the North Star ore, it appears that Z cake and Fumol are the best for floating the lead, and that light hardwood creosote oil from the Standard Chemical Co. and No. 34 gravity fuel oil from the Great North Western Oil Co., Cleveland, Ohio, are the most suitable oils for the zinc.

Test No. 6.—In this test the Selecto was not added, the object being to determine if a satisfactory separation of the lead and zinc could be obtained without it, and at the same time avoid the step of dewatering the lead tailing before attempting to float the zinc. The procedure was as follows:—

The ore was ground with 5 pounds per ton of lime and 1 pound per ton soda ash, and 0·1 gramme of Z cake. The cell was frothed to float the lead with Fumol No. 6. After eliminating the lead, the zinc was floated by adding 1 pound copper sulphate per ton, and K.K. oil and a little No. 5 pine oil to froth the cell. The results obtained were very poor. Referring to the table, it will be seen that 13·8 per cent of the zinc floated with the lead and that the lead concentrate was low grade. The zinc concentrate was very dirty and only a 10·4 per cent recovery was obtained. The lime present evidently prevented the zinc from floating.

Tests Nos. 4, 5, 7 and 8.—The reagents used at the company's plant were used. Mr. C. L. Dewar ran these tests and was unable to obtain satisfactory separation. Tests 4, 5 and 7 were run with water-gas tar from the Barrett Company. It was thought perhaps that the water-gas tar was the trouble, so a sample of the water-gas tar used at the company's mill was obtained and tried out in test No. 8. No improvement in the separation resulted.

Test No. 9.—This test was made to check the results obtained in tests Nos. 1, 2 and 3, and to see if the ore had altered in any way through oxidation since the first tests were made. When running tests on the North

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[1] Note particularly here that from experience on other ores, as well as this one, it has been found that the main bulk of the lime must be removed before floating the zinc, also that if sufficient lime is not added in the first place to insure the wetting effect on the iron remaining after decantation, the iron will tend to float with the zinc in the soda ash pulp.

[2] On the North Star ore equally good results were obtained without the addition of $CuSO_4$.

Star ore it was observed that if the ore, ground to 30 mesh, stood for a couple of weeks, the lead oxidized and a good recovery could not be obtained without the addition of sodium sulphide.

The shipment of Sullivan ore had all been crushed to one-fourth inch for sampling. Only sufficient ore for three or four flotation tests was crushed to 30 mesh.

In test No. 9 a sample was taken from the one-fourth inch material and crushed to 30 mesh for the test. One thousand grammes of the freshly crushed ore was ground in the ball mill with 0·2 gramme Z cake, 5 lb. lime per ton and 1 lb. Selecto per ton for 30 minutes, to pass 200 mesh. The cell was frothed with Fumol No. 6. The procedure was the same as that used in the first three tests.

When the flotation of the lead was attempted, it was found that only a small quantity of concentrate was obtained. It was evident that the lead had become oxidized, so a little sodium sulphide was added to the pulp. After a short period of mixing, the lead was floated without much difficulty, but considerable iron also came up. Sodium sulphide has this effect as a rule. It was also expected that more iron would tend to float with the zinc in the subsequent flotation for that mineral. This was found to be the case when the zinc was floated, resulting in the production of a low-grade concentrate.

The results from test No. 9 show that the lead mineral in the ore has been altered in some manner and is difficult to float. The results on tests 4 to 9, therefore, cannot be accepted as reliable. In order to do any further work, in comparing the reagents suggested by the Consolidated Mining and Smelting Co., and the ones that have been found satisfactory by the Mines Branch laboratories, it will be necessary to obtain a fresh supply of ore.

Sullivan Lead-Zinc Ore

Head sample: Lead..... 11·40 per cent
Zinc...... 11·40 "
Iron...... 32·25 "

Test No.	Product	Weight grms.	Weight per cent	Pb. per cent	Zn. per cent	Per cent weight by per cent assay		Recoveries		Remarks
						Pb.	Zn.	Pb.	Zn.	
1	Pb. conc... Pb. mids.. Zn. conc... Zn. mids.. Tailing....	138 48 234 24 563	13·7 4·8 23·2 2·4 55·9	52·60 21·90 8·30 9·80 1·67	6·1 10·0 37·4 11·1 1·7	706·9 105·2 192·6 23·5 93·4	83·6 48·0 867·9 26·7 95·0	63·0 9·4 17·1 2·1 8·4	7·5 4·3 77·3 2·4 8·5	1000 grms. —30-mesh ore ground for 30 mins. with 0·1 grm. Z cake, 2½ grm. lime and 1 lb/ton Selecto. Floated in Ruth with Fumol 3 drops (0·1 grm.) Tails dewatered and refloated with 5 lb/ton soda ash and 5 lb/ton copper sulphate, 4 drops YZ and 1 drop Fumol. Zn. conc. re-treated making Zn. mids. and Zn. conc. (Fumol 2 drops).
2	Pb. conc... Pb. mids.. Zn. conc... Zn. mids.. Tailing....	85 58 256 109 491	8·5 5·8 25·6 10·9 49·2	63·18 28·83 10·47 5·77 1·62	5·4 9·6 37·4 3·5 1·1	537·1 167·3 268·0 62·9 79·6	45·9 55·7 957·4 38·2 54·0	48·2 15·0 24·1 5·6 7·1	4·0 4·8 83·2 3·3 4·7	1000 grms. —30-mesh ore ground for 30 mins. with 6 lb/ton lime and 1 lb/ton Selecto. Lead floated with 10 drops light creosote (hardwood) No. 25 (No. 31). Tails dewatered and Zn. floated with 6 lb/ton soda ash, 1 lb/ton copper sulphate, using No. 34 gravity fuel oil with No. 25 for froth.

Sullivan Lead-Zinc Ore—*Continued*

Test No.	Product	Weight grms.	Weight per cent	Pb. per cent	Zn. per cent	Per cent weight by per cent assay		Recoveries		Remarks
						Pb.	Zn.	Pb.	Zn.	
3	Pb. conc...	122	12·1	52·38	4·9	638·7	59·8	55·1	5·2	1000 grms. —30-mesh ore ground in ball mill for 30 mins. with lime 5 lb/ton and Selecto 2 lb/ton. Run in Ruth using pine oil No. 5, coal tar creosote 0·03 grm. (Davidson Co.). Tailing dewatered and re-run for Zn. using No. 1 K.K. oil 6 drops. CuSO$_4$ 1 lb/ton, Na$_2$CO$_3$ 5 lb/ton.
	Pb. mids..	44	4·4	28·18	10·9	124·0	47·7	10·6	4·2	
	Zn. conc...	241	23·8	11·23	37·7	270·8	908·6	23·3	79·7	
	Zn. mids..	80	7·9	6·80	7·7	54·4	61·2	4·7	5·4	
	Tailing....	523	51·8	1·40	1·2	73·2	62·8	6·3	5·5	
4	Pb. conc...	105	10·5	53·12	11·5	557·0	120·7	50·2	10·4	1000 grms. —30-mesh ore ground for 30 mins. with soda ash 5 lb/ton, water-gas tar 0·3 lb/ton and coal tar creosote 0·3 lb/ton. Made lead conc. and lead middling CuSO$_4$ 1 lb/ton and water-gas tar 0·2 lb/ton for zinc.
	Pb. mids..	97	9·7	27·20	13·53	264·0	131·4	23·7	11·4	
	Zn. conc...	196	19·6	8·13	21·65	159·3	424·3	14·3	36·8	
	Zn. mids...	242	24·2	3·90	0·25	94·4	223·8	8·4	19·5	
	Tailing....	360	36·0	1·52	7·00	37·8	252·0	3·4	21·0	
5	Pb. conc...	177	17·7	40·45	7·87	716·0	139·3	62·2	11·8	1000 grms. —30-mesh ore ground 30 mins. 5 lb/ton soda ash and 3 lb/ton water-gas tar and 3 lb/ton coal tar creosote to float lead. Zn. floated with a little water-gas tar and a drop of No. 350. CuSO$_4$ 1 lb/ton was added. Lead tailing not dewatered before floating Zn.
	Pb. mids...	174	17·4	13·26	15·20	230·7	264·5	20·1	22·5	
	Zn. conc...	158	15·8	9·75	36·40	139·1	575·1	13·4	49·0	
	Zn. mids...	138	13·8	2·27	9·20	31·3	127·0	2·7	10·8	
	Tailing....	355	35·3	0·52	2·00	18·5	71·0	1·6	6·0	
6	Pb. conc...	147	14·6	44·28	11·20	64·5	16·35	58·8	13·8	1000 grms. —30-mesh ore ground 30 mins. with Z cake 0·1 grm., lime 5 lb/ton, soda ash 1 lb/ton. Floated Pb. by adding Fumol No. 6, 3 drops. Floated Zn. with CuSO$_4$ 1 lb/ton and K.K. oil 3 drops, pine oil No. 5 1 drop. Pb. tailing was not dewatered before floating the zinc.
	Pb. mids..	160	15·9	13·88	12·60	22·1	20·04	20·2	17·0	
	Zn. conc...	47	4·7	14·03	26·03	6·6	12·4	6·1	10·4	
	Zn. mids...	42	4·2	7·74	7·38	3·3	3·1	3·0	2·6	
	Tailing....	610	60·6	2·17	10·96	13·15	66·4	11·9	56·2	
7	Pb. conc...	186	18·7	38·8	6·50	72·56	11·77	......	......	1000 grms. —30-mesh ore ground 45 mins. with soda ash 6 lb/ton, coal tar creosote 1 lb/ton, Barrett water-gas tar 1 lb/ton for lead. Zinc, CuSO$_4$ 1·5 lb/ton, water-gas tar 6 drops. Pb. tails not dewatered before floating zinc.
	Pb. mids..	74	7·5	14·7	14·51	11·03	10·88	......	......	
	Zn. conc...	41	4·2	21·0	22·04	8·82	9·26	......	......	
	Zn. mids...	93	9·4	8·8	12·30	8·27	11·56	......	......	
	Tailing....	599	60·2	1·14	11·16	6·86	67·18	......	......	
8	Pb. conc...	190	19·0	36·43	11·12	69·22	21·12	......	......	1000 grms. —30-mesh ore ground for 30 mins. with soda ash 5 lb/ton. Consolidated water-gas tar 0·3 lb/ton. Coal tar creosote 0·3 lb/ton. To float zinc, CuSO$_4$ 1 lb/ton, water-gas tar 0·2 lb/ton. Not dewatered between.
	Pb. mids..	129	12·8	10·78	15·55	13·82	9·90	......	......	
	Zn. conc...	56	5·6	23·20	15·94	12·99	8·92	......	......	
	Zn. mids...	105	10·4	7·59	14·61	7·89	15·20	......	......	
	Tailing....	525	52·2	0·76	9·94	3·97	51·90	......	......	
9	Pb. conc...	120·5	....	43·0	7·36	51·80	8·85	48·0	7·3	1000 grms. —30-mesh ore ground 30 mins. in ball mill with ·2 gr. Z cake, 5 lb/ton lime and 2 lb/ton Selecto. Floated for lead in Ruth machine. Lead did not float well so Na$_2$S was added—came up but brought some iron with it. Dewatered lead tails and floated zinc adding 5 lb/ton soda ash, 1 lb/ton CuSO$_4$ using No. 34 gravity fuel oil.
	Pb. mids..	42·5	....	33·20	10·30	14·10	4·38	13·1	3·6	
	Zn. conc...	273·5	....	10·50	35·30	28·60	96·5	26·6	79·5	
	Zn. mids...	122	....	4·02	4·03	4·80	4·92	4·6	4·1	
	Tailing.....	449	....	1·96	1·48	8·3	6·65	7·7	5·5	

SCREEN TEST OF FLOTATION FEED

Heads −30 mesh ground 30 minutes in ball mill, washed on 200-mesh screen, oversize run on Ro-tap for 10 minutes:—

Mesh	Grammes.	Per cent weight	Per cent cum.
+100	1	0·1	0·1
+150	5	0·5	0·6
+200	59	5·9	6·5
−200	935	93·5	100·0

Report No. 188

FLOTATION TESTS ON COPPER MOUNTAIN ORE, COPPER MOUNTAIN, B.C.

R. K. Carnochan

Shipment.—A bag of ore, gross weight 89 pounds, was received July 18, 1922, at the Ore Dressing and Metallurgical Laboratories, from the Canada Copper Corporation, Allenby, B.C.

Purpose of experimental tests.—Test work was carried out on this ore to determine the results that would be obtained by different methods of treatment and to study the effect of different oil mixtures when used in the flotation of the ore.

Characteristics of the ore.—The ore is chalcopyrite and bornite disseminated through granite-diorite. There are also present small amounts of hematite, magnetite, and pyrite.

Sampling and analysis.—The lot of ore was crushed to −20 mesh and a sample for analysis was cut out and crushed to 100 mesh. This sample was found to contain:—

Gold	0·01 oz. per ton
Silver	0·26 "
Copper	2·15 per cent
Iron	3·45 "

EXPERIMENTAL TESTS

Test No. 3.—A lot of 1000 grammes of ore, −20 mesh, was ground with 3 grammes of lime in a small ball mill for 30 minutes. The ground ore was then floated in a small Ruth flotation machine with five drops TT mixture. The first concentrate was re-run to clean it.

Product	Weight grms.	Analysis			Percentage of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Concentrate	45·9	0·10	3·77	32·7	59·7	91·1	71·6
Middling	77·5	0·04	0·22	3·23	40·3	8·9	11·9
Tailing	885·0	tr.	tr.	0·39			16·5

A screen test on the flotation tailing gave:—

− 48+ 65 mesh	0·6 per cent
− 65+100 "	11·1 "
−100+150 "	11·9 "
−150+200 "	17·7 "
−200 "	58·7 "

Test No. 11.—A lot of 1000 grammes of ore, −20 mesh, was ground in a small ball mill for 30 minutes with 0·3 c.c. of heavy hardwood creosote and 0·2 c.c. of a mixture of 40 per cent coal tar and 60 per cent coal tar creosote. The ground ore was then floated in a small Ruth flotation machine, the first concentrate being re-run to clean it.

Product	Weight grms.	Analysis			Percentage of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Concentrate........................	57·6	0·15	3·27	23·56	52·8	84·9	72·1
Middling...........................	128·9	0·06	0·26	1·60	47·2	15·1	10·9
Tailing............................	819·4	tr.	tr.	0·39	....	....	17·0

Test No. 12.—A lot of 1000 grammes of ore was ground in a small ball mill for 30 minutes with 12 drops of water-gas tar. The ground ore was then floated with 3 drops of crude pine oil in a small Ruth machine, the first concentrate being re-run to clean it.

Product	Weight grms.	Analysis Cu. per cent	Percentage of values Cu.
Concentrate...	32·8	33·82	58·1
Middling..	83·5	4·55	19·9
Tailing...	883·2	0·475	22·0

Conclusions from tests Nos. 3, 11, and 12.—In all of these three tests the concentrates are very good, but the tailings are all too high. This results in a low recovery of the copper. The screen test on the flotation tailing from test No. 3 shows that only 58·7 per cent of the tailing passes through 200 mesh, and that there is 0·6 per cent of the tailing +65 mesh. As the same amount of grinding was given to the ore used in all three tests, the tailings from tests Nos. 11 and 12 will be just as coarse as that of test No. 3. It was decided to make tests in which the ore would be ground finer, to determine whether this would improve the tailings.

Test No. 20.—A lot of 1000 grammes of ore, −200 mesh, was ground in a small ball mill for 30 minutes with 3 grammes of lime. The ground charge was then floated in a small Ruth machine with 8 drops TT mixture. The first concentrate was re-run to clean it.

Product	Weight grms.	Analysis Cu. per cent	Percentage of values Cu.
Concentrate...	112·5	14·10	81·7
Middling..	241·4	0·67	8·4
Tailing...	643·9	0·30	9·9

Test No. 21.—A lot of 1000 grammes of ore, −200 mesh, was ground for 30 minutes in a small ball mill with 0·5 c·c. heavy hardwood creosote and 0·3 c.c. of a mixture of 40 per cent coal tar and 60 per cent coal tar creosote. The ground ore was floated in a small Ruth machine, the first concentrate being re-run to clean it.

Product	Weight grms.	Analysis Cu. per cent	Percentage of values Cu.
Concentrate	186	9·00	85·7
Middling	291·5	0·50	7·5
Tailing	527·5	0·25	6·8

Test No. 23.—A lot of 1000 grammes of ore, −200 mesh, was ground in a small ball mill for 30 minutes with 5 grammes of lime and 10 drops of Barretts No. 634. The ground charge was floated in a small Ruth machine with 3 drops aldol. The first concentrate was re-run to clean it.

Product	Weight grms.	Analysis Cu. per cent	Percentage of values Cu.
Concentrate	146·1	10·78	83·2
Middling	195·2	0·71	7·3
Tailing	663·0	0·27	9·5

Conclusions from tests Nos. 20, 21, and 23.—The recoveries in these three tests are better than in tests Nos. 3, 11, and 12, but the grade of concentrates are poorer. The tailings are lower, but not as low as one might expect from grinding all to pass 200 mesh. It was thought that dry grinding all to pass 200 mesh and then grinding wet in the small ball mill might not be as good as all wet grinding. To determine whether this was true, it was decided to run a test in which the ore would be ground as in tests Nos. 3, 11, and 12, to make a screen test on the tailing, and to run each size from the screen test for copper.

Test No. 24.—A lot of 1000 grammes of ore, −20 mesh, was ground in a small ball mill for 30 minutes, and then floated in a small Ruth flotation machine with 8 drops of TT mixture, the concentrate being re-run to clean it.

Product	Weight grms.	Analysis Cu. per cent	Percentage of values Cu.
Concentrate	61·9	25·50	79·4
Middling	125·4	1·52	9·6
Tailing	813·4	0·27	11·0

A screen test on the tailing and analysis for copper on the different sizes from the screen test gave:—

Mesh	Per cent weight	Per cent copper
− 48+ 65	0·8	0·85
− 65+100	10·7	0·62
−100+150	15·6	0·46
−150+200	15·0	0·34
−200	57·9	0·17

Conclusions from test No. 24.—This test seems to show that wet grinding to 200 mesh is better than dry grinding, as the −200 from the screen test runs 0·17 per cent copper, which is much lower than the tailings made in any of the tests, when the ore was all ground dry to −200 and then ground wet for 30 minutes in a ball mill. It was decided to make tests in which the ore would be ground wet, all to pass 200 mesh.

Test No. 25.—A lot of 1000 grammes of ore, −20 mesh, was ground wet for one hour in a small ball mill. The ground ore was screened wet on 200 mesh. The oversize was put back into the mill and ground for another hour. The ore from the mill was screened on 200 mesh, and the oversize was put back into the mill and ground for the third hour, at the end of which time it was all fine enough to pass the 200-mesh screen. The ground ore was partly dewatered by means of a suction filter and floated with 8 drops TT mixture in a small Ruth machine, the first concentrate being re-run to clean it.

Product	Weight grms.	Analysis Cu. per cent	Percentage of values Cu.
Concentrate	54·7	31·10	83·0
Middling	134·1	1·37	9·0
Tailing	818·2	0·20	8·0

Test No. 26.—A lot of 1000 grammes of ore, −20 mesh, was ground wet, all to pass 200 mesh in a manner similar to that described under test No. 25. The ground ore was partly dewatered by means of a suction filter, put in a small Ruth flotation machine, and 3 grammes of lime mixed in. The ore was then floated with 8 drops of TT mixture, the first concentrate being re-run to clean it.

Product	Weight grms.	Analysis per cent Cu.	Percentage of values Cu.
Concentrate	65·0	26·40	86·5
Middling	158·7	0·88	7·0
Tailing	756·0	0·17	6·5

Conclusions from tests Nos. 25 and 26.—Both these tests gave high-grade concentrates, low tailings, and good recoveries. If these tests are compared with tests Nos. 20, 21, and 23, it will be seen that tests Nos. 25 and 26 are much better than tests Nos. 20, 21, and 23. This shows that to prepare the ore for flotation wet grinding all to pass 200 mesh is much better than dry grinding to the same size.

GENERAL CONCLUSIONS

1. The ore can be successfully treated by flotation. About 90 per cent of the copper value of the ore could be recovered in a concentrate running about 26 per cent copper.

2. The ore would have to be ground wet to nearly all pass 200 mesh in order to secure a low tailing and a high recovery.

3. Any of the oil mixtures or reagents used in the tests given in this report would give good results on ore ground to nearly all pass 200 mesh, but of all the different oil mixtures or reagents used TT gave the best results.

4. The gold and silver values in the sample submitted are very low, and for this reason little work was done to determine how much of the gold and silver might be recovered by flotation. However, from the small amount of work done in this connexion, it would seem that the gold and silver are associated with the sulphide minerals, and when the ore is floated, the greater part of these values report with the copper minerals in the concentrate.

Report No. 190

CONCENTRATION OF A COPPER GOLD ORE FROM SURF INLET, B.C.

R. K. Carnochan

Shipment.—A sample of ore, gross weight 150 pounds, was received at the Ore Dressing and Metallurgical Laboratories on July 24, 1922, from the Belmont Surf Inlet Mines, Surf Inlet, B.C.

Purpose of experimental tests.—Test work was carried out on this ore to determine the results that would be obtained by different methods of treatment, and to study the effect of different oil mixtures when used in the flotation of the ore.

Characteristics of the ore.—The ore is white quartz with pyrite and chalcopyrite. No free gold or silver are visible to the eye.

Sampling and analysis.—The whole sample of ore received was crushed to one-fourth inch. A portion was cut from the main lot by means of a riffle sampler and ground to −20 mesh. A sample for analysis was cut from the −20-mesh material and ground to −100 mesh. This sample upon analysis showed the ore to contain:—

Gold	0·81 oz. per ton
Silver	0·23 "
Copper	0·425 per cent
Iron	5·74 "

EXPERIMENTAL TESTS

Flotation

Test No. 1.—1000 grammes of ore, −20 mesh, were ground in a small ball mill for half an hour with one pound of coal tar and coal tar creosote per ton. The ground ore was then run in a small Ruth flotation machine making a concentrate and a tailing. The concentrate was re-run in the same machine making a concentrate and a middling.

Product	Weight grms.	Analysis			Per cent of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Concentrate	56	6·20	1·48	4·53	59·1	49·6	58·9
Middling	61	2·46	0·94	1·62	25·6	34·3	23·0
Tailing	897	0·10	0·03	0·087	15·3	16·1	18·1

A screen test on the flotation tailings from test No. 1 gave the following:—

+ 65	0·1 per cent
− 65+100	1·0 “
−100+150	8·0 “
−150+200	26·7 “
−200	64·2 “

Test No. 2.—1000 grammes of ore, −20 mesh, were ground in a small ball mill for half an hour with 3 grammes of lime. The ground ore was then floated in a small Ruth machine with 5 drops of General Engineering Co.'s TT mixture. The concentrate produced was re-run to clean it.

Product	Weight grms.	Analysis			Per cent of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Concentrate	14·5	22·55	4·17	20·62	62·1	40·7	72·4
Middling	49·6	3·08	1·40	1·73	29·0	46·6	20·8
Tailing	944·2	0·05	0·02	0·03	8·9	12·7	6·8

A screen test on the flotation tailings from test No. 2 gave:—

− 48+ 65	0·5 per cent
− 65+100	4·2 “
−100+150	16·8 “
−150+200	20·3 “
−200	58·2 “

Summary of Flotation Tests

1. Flotation gives very good concentrates but poor recoveries.
2. The TT mixture gave better results than coal tar and coal tar creosote in the above tests.

Tabling and Flotation Tests

Test No. 3.—Procedure:—A portion of the ore, 1200 grammes, was crushed to pass a 40-mesh screen and screened on 80 mesh. This gave:—

−40+80	442 grammes
−80	758 “

Each of the sizes was tabled separately on a small Wilfley table making a concentrate and tailing. The two concentrates were combined. The two tailings were also combined and then ground for 20 minutes in a small ball mill with 3 grammes of lime. After grinding, the tailings were floated in a small Ruth machine with 5 drops of TT mixture. The first concentrate was re-run to clean it.

Product	Weight grms.	Analysis			Per cent of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Table concentrate	135·8	1·37	0·32	2·24	37·7	56·4	67·1
Flotation concentrate	3·6	57·23	5·12	21·06	41·7	23·9	16·8
" middling	31·6	2·56	0·48	2·00	16·4	19·7	13·9
" tailing	916·4	0·02	trace	0·01	3·7	....	2·0
Loss	112·6	0·02	trace	0·01	0·5	....	0·2

A screen test on the flotation tailing from test No. 3 gave the following:—

− 48+ 65	0·8 per cent
− 65+100	4·4 "
−100+150	12·0 "
−150+200	11·2 "
−200	71·6 "

Test No. 4.—Procedure:—A sample of 1200 grammes of ore was crushed to −40 mesh and screened on 80 mesh:—

−40+80	463 grammes
−80	737 "

Each of the sizes was tabled separately on a small Wilfley table, making a concentrate and a tailing. The two concentrates were combined. The two tailings were combined and ground for 20 minutes with 0·6 c.c. of heavy hardwood creosote and 0·4 c.c. of a mixture of 40 per cent coal tar and 60 per cent coal tar creosote. The ground tailing was then floated in a small Ruth machine. The first concentrate was re-run to clean it.

Product	Weight grms.	Analysis			Per cent of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Table concentrate	107·2	3·18	1·32	2·56	53·8	50·5	61·2
Flotation concentrate	69·6	4·00	1·60	2·18	44·0	39·8	33·9
" middling	129·3	0·11	0·21	0·07	2·2	9·7	2·0
" tailing	820·5	trace	trace	0·015	....	....	2·7
Loss	73·4	trace	trace	0·015	....	....	0·2

Test No. 5.—Procedure:—A sample of 1200 grammes of ore was ground to −40 mesh and screened on 80 mesh. This gave:—

−40+80	470 grammes
−80	730 "

Each of the above sizes was tabled on a small Wilfley table, making a concentrate and a tailing in each case. The two concentrates were combined. The two tailings were combined and ground in a small ball mill for 20 minutes with 12 drops of water-gas tar. The ground tailings were floated with 6 drops of crude pine oil in a small Ruth machine. The first concentrate was re-run to clean it.

Product	Weight grms.	Analysis			Per cent of values		
		Au. oz. per ton	Ag. oz. per ton	Cu. per cent	Au.	Ag.	Cu.
Table concentrate	115	2·76	1·52	2·48	52·6	65·0	63·6
Flotation concentrate	8·5	23·00	6·00	10·74	32·4	19·0	20·3
" middling	43·2	2·10	1·00	1·29	15·0	16·0	12·5
" tailing	972·2	trace	trace	0·015	....	....	3·4
Loss	61·1	trace	trace	0·015	....	....	0·2

SUMMARY OF TABLING AND FLOTATION

1. Tabling and flotation gave very good concentrates and very good recoveries. In test No. 4, which seems the best test made on the ore, the combined table and flotation concentrates assayed 3·56 oz. per ton gold, 1·43 oz. per ton silver, and 2·41 per cent copper. The recoveries in this test were: gold 97·8 per cent, silver 90·3 per cent and copper 95·1 per cent. These figures do not include any of the values in the middling product.

2. The various oils used in the above tests gave good results. TT mixture seems to have a better selective action for the copper, giving a higher grade concentrate. Heavy hardwood creosote, coal tar and coal tar creosote gave the highest recoveries, but with a much lower grade of concentrate.

SUMMARY OF EXPERIMENTAL TESTS

Tabling and flotation give much better results on the ore than flotation alone.

Conclusions.—The ore can be successfully treated by tabling and flotation. This method of treatment gives a very high recovery of the gold, silver and copper, and a high-grade concentrate. It is a difficult matter to determine from the small scale tests which reagents are the more economical to use for flotation. This point can only be definitely determined by plant operations over a period of time. It is also governed to a large extent by local and market conditions.

No tests were made, by flotation and table concentration, of the flotation tailing. This method should give equally as good results as tabling followed by flotation, and it would simplify plant operations to a considerable extent.

Report No. 191

GOLD ORE FROM CRANBERRY HEAD, NOVA SCOTIA (SHIPMENT No. 1)

C. S. Parsons

A shipment of gold ore was received February, 1923, from Mr. S. F. Johnson, 570 Columbus ave., Boston. The sample consisted of 280 pounds of ore, and was taken from a property located at Cranberry head, Yarmouth county, Nova Scotia.

Character of the ore.—The ore carries free gold in a quartz gangue and is associated with a small amount of arsenical pyrites, galena, and chalcopyrite. The values in the ore are very spotty. Nuggets of free gold are plainly visible in the quartz, and their size ranges from a maximum of 0·0328 inch in diameter to 0·0029 inch, and smaller, the greater portion being about 0·0082 inch. The grains of gold are very dense and show no porosity, their surface being smooth and the edges rounded. When the gold is broken free from the quartz a thin film of gold seems to remain attached to the surface of the quartz. The sample contained no wall rock.

Object of experimental work.—Tests were performed on the sample in order to obtain data for determining the best metallurgical treatment of the ore. The sample received is supposed to be a representative sample of the ore deposit.

Ore Dressing and Metallurgical Tests

Sampling.—Great care had to be exercised when sampling the ore owing to the spotty distribution of the gold.

Head sample.—The assays of a number of carefully taken samples gave, gold 0·72 oz. per ton as the assay of this shipment.

Amalgamation and table tests.—These tests were run before the difficulty of sampling the ore was recognized. This accounts for the head samples of the ore used in the following tests containing less gold than the original sample from which they were cut.

Procedure of tests.—Four lots of ore were cut out from the shipment, and crushed respectively to 24, 30, 35, and 40 mesh. Each lot was amalgamated in a mercury-coated copper pan by panning. The tailing, from each after amalgamation, was tabled. These tests are numbered one to four, and are given in detail below.

Test No. 2

The ore was crushed through 30 mesh. A screen test of the amalgamation feed is given in table No. I*. The head sample assayed 0·57 oz. per ton. The recovery by amalgamation was 70 per cent. The amalgamation tailing was tabled, and the results are tabulated in table No. II below.

TABLE II

Product	Weight		Analysis		Per cent of gold values
	Grammes	Per cent	Au. oz.	As. per cent	
Concentrate	29	0·70	11·52	19·78	47·4 } 56·0
Middling	93	2·25	0·62	....	8·6 }
Tailing	3,870	93·00	0·08	....	44·0
Loss	168	4·05			
Total	4,160	100	0·18	....	100

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*See p. 36.

RECAPITULATION

Recovery of gold by amalgamation	70·0	per cent
Recovery of gold by tabling, in table concentrate	16·4	"
Recovery of gold by amalgamation and in table concentrate	86·7	"
Pounds table concentrate made from one ton of ore	14·0	pounds
Value of gold in one ton of table concentrate	$230·40	
Value of gold remaining in tailing, per ton	1·60	

Test No. 3

The ore was crushed through 35 mesh. A screen test of the feed is given in table No. I. Head sample assayed 0·75 oz. per ton. The recovery by amalgamation was 70·5 per cent. The tailing from the amalgamation was tabled. The results are given below in table No. III. A screen analysis of the table tailing was made and is given in table No. IV.

TABLE III

Product	Weight		Analysis		Per cent of gold values
	Grammes	Per cent	Au. oz.	As. per cent	
Concentrate	31	0·74	12·62	16·6	45·7 } 51·8
Middling	139	3·30	0·37	....	6·1
Tailing	3,764	90·40	0·11	....	48·2
Loss	233	5·60			
Total	4,167	100·0	0·22	....	100·0

RECAPITULATION

Recovery of gold by amalgamation	70·5	per cent
Recovery of gold in table concentrate	13·4	"
Recovery of gold by amalgamation and in table concentrate	83·9	"
Pounds table concentrate from one ton of ore	14·8	pounds
Value of gold in one ton table concentrate	$252·40	
Value of gold remaining in tailing, per ton	2·20	

TABLE IV

Mesh	Weight		Assay gold	Per cent content of gold	Per cent of gold total in table feed
	Grammes	Per cent			
−35+48	187	18·7	0·15	24·6	12·0
−48+65	224	22·4	0·12	23·5	11·6
−65+100	155	15·5	0·08	10·8	5·3
−100+150	79	7·9	0·07	4·8	2·3
−150+200	137	13·7	0·08	9·5	4·5
−200	218	21·8	0·14	26·7	13·2
Total	1,000	100·0	0·12	100·0	48·2

Screen test on table tailing, test No. 3.—This screen analysis shows that the gold in the coarse sizes was not freed by crushing. The +65-mesh sizes contain 48·1 per cent of the gold remaining in the tailing and the −200-mesh product 26·7 per cent. It is very evident that finer crushing is necessary to free the gold.

Test No. 4

The ore was crushed through 40 mesh. A screen test of the feed is given in table No. I. The head sample assayed 0·57 oz. per ton. The recovery by amalgamation was 65 per cent. The tailing from the amalgamation was tabled. The results are given below in table No. V.

TABLE V

Product	Weight		Analysis		Per cent of gold values
	Grammes	Per cent	Au. oz.	As. per cent	
Concentrate	26	0·6	14·26	20·85	45·4
Middling	66	1·6	0·64	....	5·2
Tailing	3,840	92·2	0·105	....	49·4
Loss	235	5·6			
Total	4,167	100·0	0·2	....	100·0

RECAPITULATION

Recovery of gold by amalgamation	65·0	per cent
Recovery of gold in table concentrate	15·7	"
Recovery of gold by amalgamation and in table concentrate	80·7	"
Pounds table concentrate made from one ton of ore	12·0	pounds
Value of gold in one ton of table concentrate	$285·20	
Value of gold remaining in tailing, per ton	2·20	

Test No. 9

A flotation test was made on the amalgamation tailing from test No. 4, which is −40-mesh material. The tailing without further grinding was mixed with the reagents and floated. The results are given in table No. VI. Head sample of flotation feed 0·20 oz. per ton.

TABLE VI

Product	Weight		Gold		Remarks
	Grammes	Per cent	Assay oz. per ton	Per cent of values	
Concentrate	73	7·3	1·66	59·2	Reagents used—coal tar and coal tar creosote, and pine oil. Amount 1 lb. per ton.
Tailing	929	92·7	0·09	40·8	
Total	1,002	100·0	0·205	100·0	

A screen test was made on the flotation tailing to determine the distribution of the gold remaining. The results are given below in table No. VII. The gold was found to be fairly evenly divided between all the sizes. It is evident that finer crushing is necessary to free the gold.

TABLE VII

Screen Analysis Flotation Tailing

Mesh	Weight		Gold oz. per ton	Per cent of total gold
	Grammes	Per cent		
− 40+ 65	96·7	14·1	0·13	22·5
− 65+100	171·2	25·1	0·07	21·4
−100+200	202·5	29·6	0·06	21·8
−200	213·0	31·2	0·09	34·3
Total	683·4	100·0	0·082	100·0

Test No. 12

This test was made to determine the maximum percentage of gold recoverable by amalgamation when the ore was crushed to pass a 35-mesh Tyler standard screen.

Procedure.—2000 grammes of ore was crushed through 35 mesh and amalgamated in a jar. The pulp dilution used was 1 part of ore to 1 part water. 20 grammes or 1·0 per cent mercury was added. A handful of pebbles was thrown into the jar in order to keep the pulp from sticking to the sides. The addition of the pebbles caused some of the ore to be ground finer. After agitation for 3 hours, the mercury was separated from the tailing by panning.

TABLE VIII

Screen Test of Sample Before Amalgamation

Mesh	Weight		Gold		Accumulative per cent Au.
	Grammes	Per cent	Oz. per ton	Per cent of total	
− 35+ 48	231	23·1	0·80	25·5	25·5
− 48+ 65	168	16·8	0·80	18·5	44·0
− 65+100	156	15·6	0·87	18·7	62·7
−100+150	94	9·4	0·97	12·6	75·3
−150+200	104	10·4	0·84	12·1	87·4
−200	247	24·7	0·37	12·6	100·0
Total	1,000	100·0	0·725	100·0	

TABLE IX

Screen Test on Tailing After Amalgamation

Mesh	Weight		Gold		Accumulative per cent Au.
	Grammes	Per cent	Oz. per ton	Per cent of total	
− 35+ 48	158·0	15·8	0·14	18·2	18·2
− 48+ 65	201·0	20·1	0·13	21·5	39·7
− 65+100	167·0	16·7	0·12	16·4	56·1
−100+150	92·5	9·2	0·11	8·4	64·5
−150+200	112·5	11·2	0·10	9·3	73·8
−200	265·5	26·5	0·12	26·2	100·0
Loss	3·5	0·3			
Total	1,000·0	100·0	0·12	100·0	

TABLE X
Gold Amalgamated

Mesh	Per cent of gold amalgamated from each size
− 35+ 48	88·0
− 48+ 65	80·5
− 65+100	85·2
−100+150	88·85
−150+200	87·1
−200	65·1
Total	83·2

Explanation of the above tables.—Table No. VIII is a screen analysis of the ore before amalgamation, and gives the distribution of the gold values in the different sizes. For example: the figures in the first column, headed mesh, means the material passing through 35 mesh and remaining on 48 mesh. Column 2, headed weight grammes, gives the quantity of, say, −35+48 material in the sample. Column 3, under weight, is that quantity expressed in per cent of the total weight of the sample. Column 4, under gold, is the assay of that portion of the sample, say, passing through 35 mesh and remaining on 48 mesh, and is given in ounces per ton. Column 5 is that quantity of gold expressed in per cent of the total gold contained in the sample. Column 6, called accumulative per cent, is obtained by adding the figures obtained in column five.

Table No. IX is similar to table No. VIII.

Table No. X gives the amount of gold amalgamated from each size.

Summary

Head sample assayed	0·73 oz. per ton
Amount of gold recovered by amalgamation	0·60 "
Amount of gold lost in tailing	0·13 "
Recovery by amalgamation	83·2 per cent

Discussion of above results.—Referring to and comparing the screen analyses on the amalgamation heads and tailing, it will be observed that the addition of the pebbles to jar produced a grinding action that reduced the quantity of material on the 48 and 65-mesh screen. This material caused a larger amount of gold to be amalgamated from these sizes than would otherwise have been the case. The figures for the quantity of gold which should actually amalgamate from the coarser sizes are therefore too high.

Test No. 13

AMALGAMATION AND CYANIDE TEST

This test was made with two objects in view. First, to determine the percentage of gold which would be recovered by a combined method of amalgamation and cyanidation from each respective size of a sample of the ore that had been sized on a series of screens given in table No. XI. Second, to obtain a comparison of the ratio of dissolution of the coarse gold in the 35, 48, and 65-mesh sizes with that of the fine gold on the 100, 150, and 200-mesh screens.

Procedure.—A 30-pound sample of ore was taken for the test and sized on the following screens, the weights being given in grammes (Table XI).

TABLE XI

Mesh	Weight		Assays		Per cent of total gold	Accumulative per cent total gold
	Grammes	Per cent	Gold oz. per ton	Arsenic per cent		
− 28+ 35	1,468	14·30	0·90	0·16	19·7	19·7
− 35+ 48	1,933	18·80	0·75	0·19	19·0	38·7
− 48+ 65	1,620	15·70	0·90	0·22	19·1	57·8
− 65+100	1,290	12·50	0·87	0·25	14·7	72·5
−100+150	806	7·80	0·87	0·16	9·2	81·7
−150+200	908	8·80	0·78	0·16	9·3	91·0
−200	2,277	22·10	0·30	0·22	9·0	100·0
Total	10,302	100·00	0·74	0·22	100·0	

Sampling.—It is extremely difficult to obtain correct samples of this ore, due to the spotty nature of the values, and to the coarse size of the gold particles.

Half of each of the above sizes down to the 100-mesh size had to be used as a sample to assay. Theoretically this amount is not sufficient, but in this case satisfactory checks were obtained.

Amalgamation tests.—The amount remaining after the samples were taken was used for the amalgamation tests. Each size was separately amalgamated in a revolving jar, a pulp of 1:1 being used, and with an amount of mercury equal in weight to 1 per cent of the ore in the charge. The contents of the jar were revolved for 2 hours. After amalgamation, the mercury was removed and dissolved in acid to recover the gold. The tailing was sampled and assayed for gold. The weight of gold obtained from the mercury plus the gold remaining in the tailing after amalgamation checked up very closely with assays obtained on the head sample of each of the sizes shown in table No. X. A recapitulation of these results is shown in table No. XII.

Discussion of results from amalgamation.—The tailings from the amalgamation of +200-mesh sizes were examined very carefully under the microscope. The gold remaining in the tailing was observed to be either attached to or enclosed in the sulphide and quartz particles. The amalgamation efficiency was practically 100 per cent on the free gold. In order to recover the remaining gold it would be necessary to crush each size finer. The curve shown in Fig. 1 shows very clearly that the recovery by amalgamation gradually increases as the sizes become smaller, until a maximum is reached at 150 and 200 mesh.

Referring to table No. XI, it will be observed that 57·8 per cent of the gold is in the sizes remaining on 65 mesh (that is coarser than 65 mesh) and that only 76·24 per cent of this gold is recoverable by amalgamation. In order to recover the remaining gold, this material would have to be re-crushed.

Cyanide tests.—Cyanide tests were made on the amalgamation tailing from each size. The object of these tests was to obtain a comparison of the rate of dissolution of the coarse and fine gold, and the maximum extraction of the gold in each size. A 0·20 per cent cyanide (KCN) solution was chosen for the test. The amount of lime used was equivalent to five pounds per ton of ore, and the ratio of solution was 3:1. These conditions were kept constant in all tests, the time of treatment being the only factor varied.

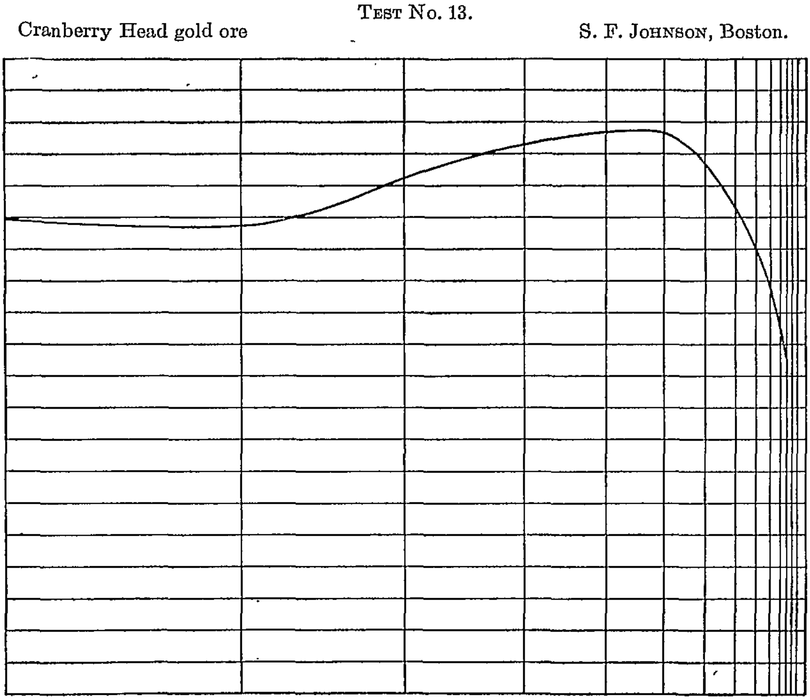

Test No. 13.
Cranberry Head gold ore
S. F. Johnson, Boston.

Fig. 1. Curve showing recovery by amalgamation on each size

The manner in which these tests were conducted was as follows: The amalgamation tailing from each screen product was divided equally into four lots. A lot from each size was given 1, 2, 4 and 5-days' treatment respectively. The third day period was skipped in every size, the reason being that it was desired to treat one sample for 5 days and there was only sufficient material on each size to make up four charges.

The results of the above tests are given in table No. XIV, and a series of curves from these figures plotted in Fig. 2. Table No. XIII gives the cyanide and alkali consumption for each test.

Discussion of cyanide results.—Referring to Fig. 2, it will be seen that in the −35+48-mesh material the rate of dissolution of the gold is rapid during the first 48 hours. At the end of the second day, the curve flattens out and very little increase in the extraction is obtained during the next 48 hours, and it is evident that this size material might be treated for an indefinite period without any appreciable increase in the extraction. It is also evident that it would hardly be economical to carry the treatment beyond 72 hours. Careful examination of the amalgamation tailing from

Cranberry Head gold ore.

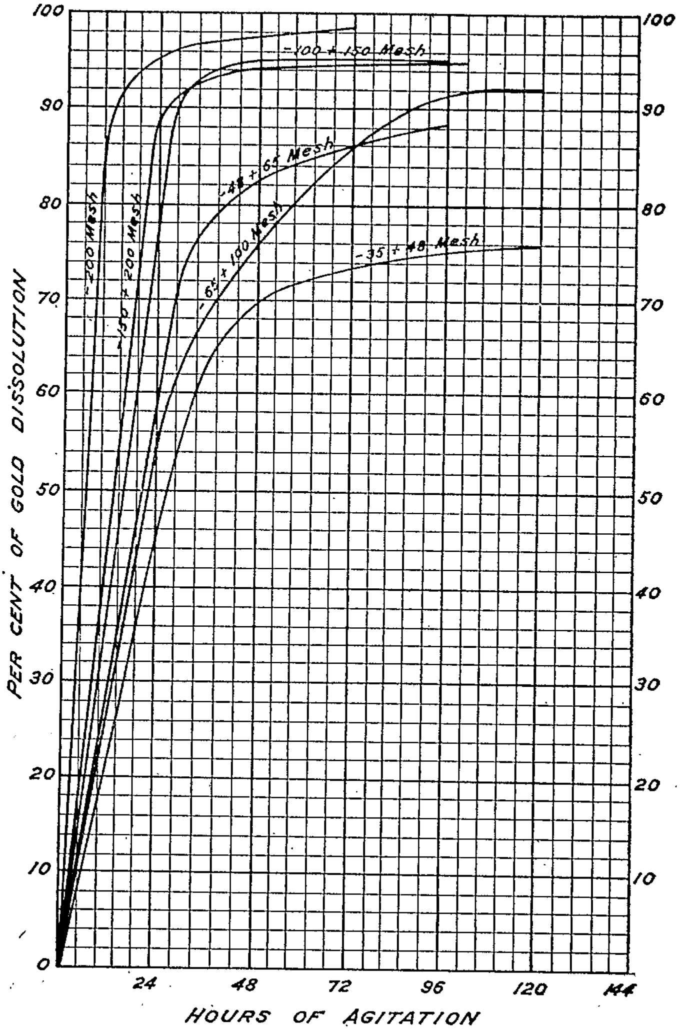


Fig. 2. Rate of dissolution of the gold in the different sized screen products obtained in equal intervals of time

this size, which constitutes the heads for the cyanide test, shows that no freed particles of gold were left unamalgamated in the tailing, but very thin coatings of gold could be seen on the surface of some of the quartz grains. The rapid extraction during the first 48 hours was, therefore, due to the dissolution of this gold. The flattening out of the curve in Fig. 2 is due to the inability of the cyanide to attack the gold still locked up in the quartz grains.

What has been said about the $-35+48$-mesh material applies to the $-48+65$ mesh, and to some extent to the $+100$-mesh material, but in both cases more gold is dissolved due to more surfaces being exposed by the reduction in size of the quartz grains.

In the $-100+150$ and $+200$-mesh sizes, all the gold freed from the quartz grains was amalgamated, the remaining gold left for recovery by cyaniding being either attached to the surface of the quartz grains or entirely embedded in them. The condition of the gold in -200-mesh product after amalgamation was not determined. The rate of dissolution of the gold was very rapid, and a maximum extraction was reached in the first 24 hours.

SUMMARY OF AMALGAMATION AND CYANIDE TREATMENT

1. By stamp milling the ore to 28 mesh and amalgamation on plates, a maximum extraction of 79·0 per cent of the gold is possible.
2. That in actual practice this result would be lower, probably about 70 per cent.
3. The fine gold does not amalgamate readily, but it is possible that the recovery might be increased by the addition of lime to the battery.
4. That there is no difficulty in obtaining a high extraction of the gold remaining in the amalgamation tailing by cyanidation.
5. That for maximum extraction by cyaniding, the amalgamation tailing should be re-crushed to pass 65 mesh.
6. That the rate of dissolution of the gold in the coarse sizes is so slow that it would not be practical to cyanide the gold by leaching methods.
7. That it is not necessary to continue cyanide treatment for a longer period than 48 hours.

Cyanide Test

Considerable detailed work was done on cyaniding this ore. A short synopsis of the findings is given under the following headings:—

Strength of solutions.—A number of tests were run using strengths of solutions varying from 0·05 to 0·25 per cent (KCN). The 0·05 per cent solution was found to give as high an extraction as the 0·25 per cent or stronger solution, in the same time of treatment. The consumption of cyanide was considerably lower with the 0·05 per cent solution. The consumption on the 0·05 per cent was approximately 0·6 pounds per ton, and with the 0·25 per cent solution 1·5 pounds per ton.

Alkalinity.—The alkalinity was found to have considerable influence on the cyanide consumption. If increased above a certain amount, more cyanide was consumed, and, if kept too low an increase in cyanide consumption was obtained. A protective alkalinity of about 0·02 per cent CaO should be maintained, and on the sample, 5 pounds of lime per ton of ore was sufficient to do this.

Time of treatment.—This depends entirely on the size to which the ore is crushed. The results from test No. 13 and the curves given in Fig. 2 show this very clearly.

Tests for Cyanide Consumption

Test No. 1.—To find if the ore contained water-soluble cyanicides, 1000 grammes ore was agitated for 24 hours with 600 c.c. water; 500 c.c. filtered off, 0·625 gramme KCN added, agitated 16 hours.

KCN found	0·124 per cent

No water soluble cyanicide present.

Test No. 2.—To find if CaO increases the solubility of cyanicides. Same procedure as No. 1 except that 1 gramme CaO (20 pounds per ton) was added.

KCN found	0·124 per cent

CaO does not increase the solubility of the cyanicides.

Test No. 3.—To find the cyanide consumption when no lime is added, 100 grammes ore agitated 24 hours with 600 c.c. cyanide solution (0·125 per cent KCN).

KCN found	0·096 per cent
KCN consumed	3·48 pounds per ton
Gold in tailing	0·01 ounce per ton

Test No. 4.—To find if NaOH increases the solubility of cyanicides. Same procedure as No. 1 except that half a gramme of NaOH was added.

KCN found	0·125 per cent

NaOH does not increase the solubility of the cyanicides.

Test No. 5.—To find the effect of Na_2O_2 on the cyanide consumption. Same procedure as No. 3, except that 0·15 gramme Na_2O_2 was added (3 pounds per ton).

KCN found	0·112 per cent
KCN consumed	1·56 pounds per ton
Au. in tailing	0·010 ounce per ton

Na_2O_2 decreases the cyanide consumption.

Cyanide Test "A"

These are preliminary tests. The ore used in the tests was the −40-mesh amalgamation tailing from test No. 4.

Weight of ore taken, 200 grammes. Solution 600 grammes. Ratio 1:3.

Cyanide added		Cyanide recovered		
	gramme KCN			gramme
Making up added	0·3	Titrated 100 c.c. at ·04 per cent KCN	=	0·04
Added 1st day	0·05	" 500 c.c. at ·046 "	=	0·23
Total	0·35	Total, 600 c.c.	=	0·27

Consumption 0·35—0·27=0·08 gramme, that is 0·8 pound per ton of ore.
Average strength of solution used, 0·05 per cent KCN.
Lime added at start 0·5 gramme=0·060 per cent CaO (lime 73 per cent CaO).
Lime in solution at end of 1st day=0·035 per cent CaO.
" " " 2nd day=0·030 per cent CaO.
The tailing from the test assayed 0·068 oz. per ton.
The extraction was 66 per cent.
Total time of agitation was 44 hours.

Cyanide Test "B"

The same material was used as in test "A".

Weight of ore taken, 200 grammes. Solution used 600 grammes, Ratio 1:3.

Cyanide added	gramme KCN	Cyanide recovered	gramme
Making up added	0·6	Titrated 100 c.c. at 0·038 per cent	= 0·088
Added 1st day	0·06	" 500 c.c. at 0·094 per cent	= 0·470
Total	0·66	Total, 600 c.c.	0·558

Consumption, 0·66—0·558=0·102, that is 1·02 pounds per ton of ore.
Average strength of solution, 1·0 per cent KCN.
Lime added at start 0·5 gramme, 73 per cent CaO=0·06 per cent CaO in solution.
Lime in solution the 1st day =0·046 per cent CaO.
" " 2nd day =0·043 " "
The tailing from the test assayed 0·07 oz. gold per ton.
The extraction was 65 per cent.
Time of agitation, 44 hours.

Cyanide Test "C"

Weight of ore used, 200 grammes. Solution, 600 grammes. Ratio, 1:3.

Strength of solution at start, 0·25 per cent KCN.
Cyanide consumption, 1·46 pounds per ton of ore.
Alkalinity maintained at end of test, 0·043 per cent CaO.
The tailing assayed 0·07 oz. gold per ton.
The extraction was 67 per cent.
Time of agitation, 44 hours.

A screen test was made on the cyanide tailing, with the following results:—

SCREEN TEST ON CYANIDE TAILING

Mesh	Weight		Gold			Per cent of total gold remaining in tailing
	Grammes	Per cent	Oz. per ton	Per cent	Accum. per cent	
− 35+ 48	8·1	4·15	0·072	4·5	4·5	1·58
− 48+ 65	36·7	18·80	0·095	27·2	31·7	8·90
− 65+100	40·4	20·70	0·101	31·7	63·4	10·45
−100+150	26·1	13·40	0·067	13·6	77·0	4·48
−150+200	25·7	13·20	0·047	9·4	86·4	3·10
−200	58·2	29·80	0·030	13·6	100·0	4·48
Total	195·2	100·0	0·066	100·0		

Test No. 14

Five samples of ore were cut out and crushed to pass 200 mesh. Head assay contained 0·72 oz. gold.

Test No. 1 was run with a 0·25 per cent KCN solution and with an alkalinity of 8 lbs. lime per ton.

Test No. 2 was run with a 0·10 per cent solution, and with an alkalinity of 8 lbs. lime per ton.

Test No. 3 was run with a 0·10 per cent KCN solution, and with an alkalinity of 5 lbs. lime per ton.

Test No. 4 was run with a 0·05 per cent KCN solution, and with an alkalinity of 8 lbs. lime per ton.

Test No. 5 was run with a 0·05 per cent KCN solution, and with an alkalinity of 5 lbs. lime per ton.

The results are given in table No. XV.

The tests were run for 24 hours each. The same extraction was obtained from each test, and amounted to 99·8 per cent of the gold.

It is evident that a 0·05 per cent KCN solution is as good as a 0·25 per cent KCN solution, and that if the alkalinity is increased above 5 pounds of lime per ton, the cyanide consumption is also increased.

TABLE I
Screen Tests

Test No. 1			*Test No. 2*		
Mesh	Weight grammes	Per cent	Mesh	Weight grammes	Per cent
+ 20	0·0	0·0	+ 35	1·35	1·4
− 20+ 28	0·2	0·2	− 35+ 48	19·00	19·0
− 28+ 35	24·5	24·0	− 48+ 65	18·20	18·2
− 35+ 48	22·3	21·8	− 65+100	15·60	15·6
− 48+ 65	13·7	13·4	−100+150	10·60	10·6
− 65+100	10·7	10·5	−150+200	11·30	11·3
−100+150	7·0	6·9	−200	23·8	23·8
−150+200	7·4	7·3			
−200	16·2	15·9			

Test No. 3			*Test No. 4*		
Mesh	Weight grammes	Per cent	Mesh	Weight grammes	Per cent
+ 35	0·15	....	+ 35	0·3	0·3
− 35+ 48	13·50	13·5	− 35+ 48	6·05	6·0
− 48+ 65	19·6	19·6	− 48+ 65	19·30	19·3
− 65+100	17·25	17·2	− 65+100	19·40	19·4
−100+150	11·80	11·8	−100+150	12·50	12·5
−150+200	12·00	12·0	−150+200	13·70	13·7
−200	26·05	26·0	−200	28·95	28·9

TABLE XII
Recapitulation of Amalgamation Results
Test No. 13.

Product	Heads				Amalgamation tailing		Recovery gold per cent
	Weight grms.	Weight per cent	Calcul. assay oz. per ton	Per cent total gold	Gold oz. per ton	Per cent loss, gold	
− 28+ 35...........	1,468	14·3	0·90	17·7	0·23	25·5	74·5
− 35+ 48...........	1,933	18·8	0·75	19·5	0·20	26·7	73·3
− 48+ 65...........	1,620	15·7	0·90	19·6	0·17	18·9	81·1
− 65+100...........	1,290	12·5	0·87	15·1	0·12	13·7	86·3
−100+150...........	806	7·8	0·87	9·4	0·10	11·5	88·5
−150+200...........	908	8·8	0·78	9·5	0·09	11·5	88·5
−200................	2,277	22·1	0·30	9·2	0·12	40·0	60·0
Total........	10,302	100·0	0·72	100·0	....	21·3	78·7

TABLE XIII

Table of KCN and CaO Consumption

0·25 per cent KCN solution
0·060 per cent CaO=5 lbs/ton lime

Test No. 13

Mesh	1st day 24 hours				2nd day 48 hours			4th day 96 hours			5th day 120 hours		
	KCN per cent	Lbs. per ton cons.	CaO per cent sol'n	Lbs. per ton CaO cons.	KCN per cent	Lbs. per ton cons.	CaO per cent sol'n	KCN per cent	Lbs. per ton cons.	CaO per cent sol'n	KCN per cent	Lbs. per ton cons.	CaO per ct. sol'n
- 35+ 48	0·224	1·6	0·034	1·56	0·228	1·32	0·042	0·222	1·68	0·035	0·226	1·44	0·037
- 48+ 65	0·238	0·72	0·030	1·8	0·244	0·36	0·041	0·240	0·6	0·038			
- 65+100	0·234	1·0	0·030	1·8	0·224	1·56	0·031	0·230	1·2	0·028	0·226	1·44	0·030
-100+150	0·226	1·4	0·030	1·8	0·226	1·44	0·029	0·228	1·32	0·027			
-150+200	0·226	1·4	0·024	2·2	0·23	....	0·030	0·232	1·08	0·026	0·220	1·80	0·029
-200	0·230	1·2	0·021	2·3	0·212	....	0·029	0·226	1·44	0·023	0·228	1·32	0·020

TABLE IV

Test No. 13

Mesh	Heads assay	1st day 24 hours		2nd day 48 hours		4th day 96 hours		5th day 120 hours	
		Tailing assay	Extractn. per cent	Tailing assay	Extractn. per cent	Tailing assay	Extractn. per cent	Tailing assay	Extractn. per cent
— 28+ 35......	0·23	This size was destroyed							
— 35+ 48......	0·20	0·11	45	0·06	70	0·05	75	0·048	76
— 48+ 65......	0·17	0·07	58	0·03	82·5	0·02	89		
— 65+100......	0·12	0·05	55	0·03	76	0·01	92	0·01	92
—100+150......	0·10	0·02	80	0·005	95	0·005	95		
—150+200......	0·09	0·01	89	0·005	94	....	95	0·005	
—200...........	0·12	0·005	95	0·005	95	....	95	0·005	95

TABLE XV

Test No. 14

Test No.	At start						At finish					Amt. consumed	
	Per cent KCN soln.	Grm. KCN	KCN lbs. per ton	Per cent CaO soln.	Grm. lime	CaO lbs. per ton	Per cent KCN soln.	Grm. KCN	KCN lbs. per ton	Per cent CaO soln.	CaO lbs. per ton	KCN lbs. per ton	CaO lbs. per ton
1	0·25	1·50	15	....	0·8	8	0·22	1·32	13·2	0·048	2·8	1·8	5
2	0·10	0·60	6	....	1·1	8	0·084	0·504	5·04	0·047	2·8	0·96	5
3	0·10	0·60	6	....	0·69	5	0·088	0·528	5·28	0·024	1·4	0·72	3·6
4	0·05	0·30	3	....	1·1	8	0·036	0·216	2·16	0·045	2·7	0·84	5·3
5	0·05	0·30	3	....	0·69	5	0·040	0·240	2·4	0·17	1·0	0·60	4

Report No. 192

GOLD ORE FROM CRANBERRY HEAD, NOVA SCOTIA (SHIPMENT No. 2)

C. S. Parsons

Shipment.—A second shipment of gold ore was received May 2, 1923, from Mr. S. F. Johnson, 570 Columbus ave., Boston, U.S.A. The shipment consisted of 380 pounds and was taken from a property located at Cranberry head, Yarmouth co., Nova Scotia. This second shipment was submitted at the request of the Division as it was thought that the first shipment was not a representative sample of the ore.

Purpose of experimental work.—The object in conducting test work was to determine a metallurgical treatment for the ore.

Characteristics of the ore.—The ore of the second shipment as compared with that of shipment No. 1, contained considerable slate from the walls of the vein. There was a larger percentage of arsenical pyrites and the gold was more finely disseminated. The slate contained some large crystals of arsenical pyrites.

Sampling and analysis.—Due to the spotty distribution of the gold in the ore great care was exercised in the sampling. The entire shipment was reduced to 10 mesh and cut in a riffle sampler to 93 pounds. This was further reduced to 20 mesh before cutting to 46·5 pounds. The samples for assaying were obtained from the 46·5 pounds by stage crushing and cutting until 500 grammes had been reduced to 150 mesh. The average assay of these samples gave 0·64 oz. gold per ton.

Experimental Tests.

Test No. 1A

The flow sheet as indicated by Fig. 3 was followed in this test. The ore was crushed to 30 mesh and amalgamated. The tailing from this first amalgamation was re-crushed to pass 100 mesh and again amalgamated. The tailing from the second amalgamation was passed over a Wilfley table and a concentrate containing arsenic and other sulphides was made and a final tailing was sent to waste.

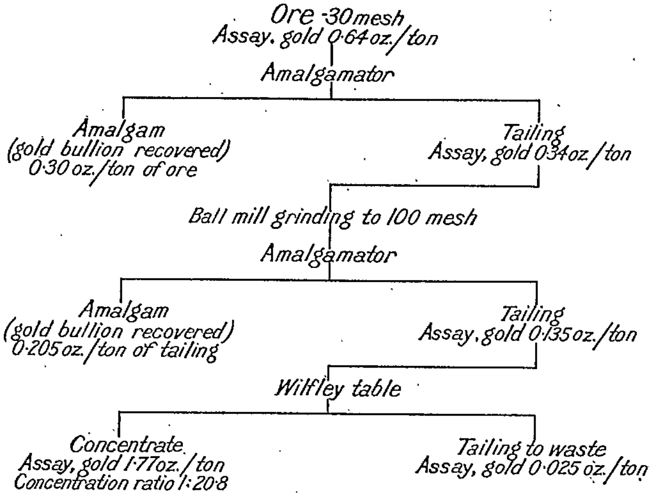


Fig. 3. Flow sheet used for test No. 1 A. Cranberry Head gold ore.

RECAPITULATION OF RESULTS

Assay of ore before treatment	0·64	oz. per ton
Tailing after first amalgamation	0·34	"
Tailing after second amalgamation	0·135	"
Tailing after tabling	0·025	"
Concentrate from table	1·77	"
Recovery of total gold by 1st amalgamation at 30 mesh	47·0	per cent
" " 2nd amalgamation at 100 mesh	32·0	"
" " table concentration	16·7	"
Recovery of total gold	95·7	"

The table concentrate represents 4·8 per cent by weight of the ore, that is, for every 100 tons of ore treated, 4·8 tons of concentrate will be obtained, assaying approximately 1·77 oz. gold per ton.

The above figures are a recapitulation of the results of the test which is given in detail below.

A 500-gramme sample was cut out, having first been crushed to pass 30 mesh, and a screen analysis made to determine the distribution of the gold values before amalgamation.

TABLE I
Screen Analysis Test No. 1A

Mesh	Weight		Gold	Per cent of total gold
	Grammes	Per cent	Assay oz. per ton	
− 30+100	838·2	55·9	0·665	59·9
−100+150	111·8	7·5	0·65	7·8
−150+200	159·0	10·6	0·96	16·4
−200	389·2	26·0	0·38	15·9

This material was amalgamated by placing in a jar with 1 per cent by weight of mercury and rotating for one hour. A pulp consistency of one part of water to one of ore was used. The mercury was separated by panning and retorted. The amount of gold amalgamated as above stated was found to be 47·0 per cent of the gold in the ore. The gold recovery was 0·30 oz. per ton. The tailing was assayed and found to contain 0·34 oz. per ton of gold. A screen analysis was made on this tailing to determine the distribution of the gold.

TABLE II
Screen Analysis of 1st Amalgamation Tailing

Mesh	Weight		Gold	Per cent of total gold
	Grammes	Per cent	Assay oz. per ton	
− 30+ 48	371·8	30·0	0·43	41·0
− 48+ 65	190·3	15·5	0·38	18·7
− 65+100	159·8	12·9	0·32	13·0
−100+150	94·5	7·5	0·225	5·4
−150+200	114·5	9·2	0·235	6·9
−200	308·5	24·9	0·19	15·0

TABLE III
Distribution of Gold Before and After 1st Amalgamation

Mesh	Before amalgamation	After amalgamation	Gold recovered from each	
	Oz. per ton	Oz. per ton	Oz. per ton	Per cent
− 30+100	0·665	0·398	0·267	40·1
−100+150	0·650	0·225	0·425	65·4
−150+200	0·960	0·235	0·725	75·5
−200	0·380	0·190	0·190	50·0

It is very evident from table No. III that the gold is not all freed in the coarser sizes, and that in order to obtain a high recovery by amalgamation it will be necessary to crush the ore to 100 mesh at least. The sample was crushed dry. Wet crushing, as will be shown later, gives much higher recoveries of gold.

Second amalgamation.—The second step in this test was to re-crush a portion of the first amalgamation tailing to 100 mesh and amalgamate. This lot was crushed dry in rolls and a disk crusher. The amalgamation was proceeded with as in the first part of the test. The tailing assayed 0·135 oz. of gold per ton. This is equivalent to a recovery of 60·3 per cent of the gold remaining in the first amalgamation tailing.

Tabling.—The second amalgamation tailing was concentrated on a small Wilfley table. A concentrate and tailing were made. The concentrate represented 4·8 per cent of the weight of the ore, and assayed 1·77 oz. per ton gold. The tailing was low in gold, assaying only 0·025 oz. per ton, equivalent to a value of 50 cents per ton. The table recovered 78 per cent of the gold remaining in the second amalgamation tailing. A total recovery of 95·7 per cent of the gold was made by this method of treatment.

Test No. 1B

Purpose of test.—The purpose of this test was to determine the relative merits of two methods of treatment, namely, double amalgamation followed by tabling of second tailing as in test No. 1A, and the method described below. The flow sheet as indicated by Fig. 4 was followed in this test. Half of the tailing from the first amalgamation test No. 1A was used in this test. This tailing was concentrated without further crushing,

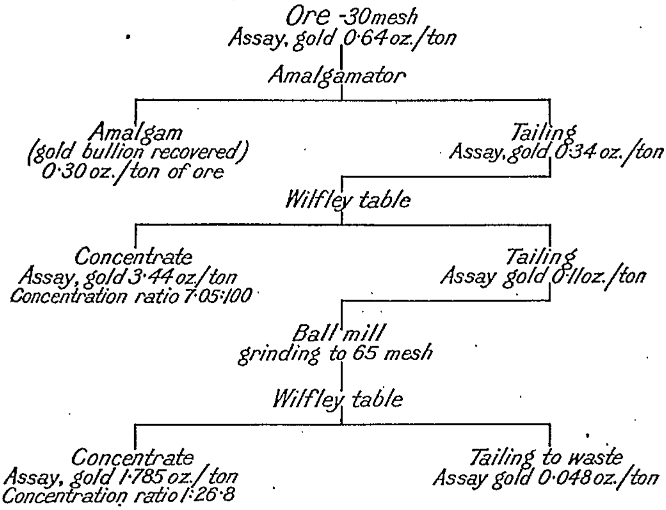


Fig. 4. Flow sheet used for test No. 1 B. Cranberry Head gold ore.

on a Wilfley table, and a concentrate and tailing produced. The tailing from this first tabling was crushed wet in a ball mill to pass 65 mesh, and again tabled. A recapitulation of the results follows:—

Assay of heads before treatment	0·64 oz. per ton
" tailing after first amalgamation	0·34 "
" concentrate from first tabling at 30 mesh	3·44 "
" tailing " " " 30 "	0·11 "
" concentrate " second " 65 "	1·785 "
" tailing " " " 65 "	0·048 "

The ratio of concentration was 7·05 : 100 in the first tailing.
" " 3·74 : 100 in the second tailing.

Total gold in heads recovered by amalgamation at 30 mesh	42·07 per cent
" " " in 1st table concentrate at 30 mesh	37·50 "
" " " 2nd " " 65 "	10·40 "
	94·90 "

Detailed description of test.—Part of the tailings from the first amalgamation at 30 mesh in test No. 1A was used in this test. This tailing which assayed 0·34 oz. per ton was concentrated on a Wilfley table, two products being made, a concentrate and tailing.

TABLE IV

Product	Weight		Gold	Per cent of gold values
	Grammes	Per cent	Assay oz. per ton	
Concentrate	164·7	7·05	3·44	70·4
Tailing	2,171·0	92·95	0·11	29·6

A screen analysis of the first table tailing is given in table No. V. It is very evident from these figures that the gold is not freed from the coarser particles of gangue, and that crushing the whole to pass 100 mesh would probably do this sufficiently.

TABLE V

SCREEN ANALYSIS

Mesh	Weight		First table tailing		Per cent Au. recovered from each size	Per cent of total gold
	Grammes.	Per cent	Assay oz. per ton	Assay x weight per cent		
− 30+ 48	328·8	30·7	0·16	....	62·8	44·1
− 48+ 65	172·5	16·1	0·125	....	67·1	18·0
− 65+100	137·4	12·8	0·085	....	73·4	9·9
−100+150	86·7	8·1	0·054	....	76·0	3·6
−150+200	87·9	8·2	0·090	....	61·6	6·3
−200	257·0	24·0	0·085	....	55·2	18·1

The tailing from the primary table which contained 0·11 oz. gold was re-crushed in a ball mill to pass 65 mesh and again concentrated on a Wilfley table. The result of this operation is given in table No. VI:—

TABLE VI

Product	Weight		Gold	Per cent of gold values
	Grammes	Per cent	Assay oz. per ton	
Concentrate	41·2	3·7	1·785	58·9
Tailing	1,060·5	96·3	0·048	41·1

TABLE VII
SCREEN ANALYSIS SECOND TABLE TAILING

Mesh	Weight		Gold	Per cent of gold values
	Grammes	Per cent	Assay oz. per ton	
+200	374·0	47·7	0·03	29·2
−200	410·3	52·3	0·065	70·8

TABLE VIII
DISTRIBUTION OF GOLD BEFORE AND AFTER PRIMARY TABLING

Mesh	Gold, oz. per ton		Per cent of values recovered
	First amalgamation tailing	Primary table tailing	
− 30+ 48	0·43	0·160	62·8
− 48+ 65	0·38	0·125	67·1
− 65+100	0·32	0·085	73·4
−100+150	0·225	0·054	76·0
−150+200	0·235	0·090	61·7
−200	0·190	0·085	55·2

Discussion of results of tests Nos. 1A and 1B.—The recoveries were practically the same in both tests. Comparing the results of the two tests, it is evident that the treatment used in test No. 1A offers a number of advantages over the methods used in test No. 1B. By the double amalgamation, over 79 per cent of the gold was recovered as bullion, while in test No. 1B, only 47 per cent was obtained as bullion. The costs of operation of the two methods would be practically the same. The first cost of the machinery used in the flow sheet of test No. 1B would be greater than that used in test No. 1A on account of twice the number of tables being used. The chief advantage of the first flow sheet over the second is that 79 per cent against 47 per cent of the gold was obtained as bullion ready to market. In the second flow sheet 47·9 per cent of the gold is in a concentrate associated with refractory material, compared with only 16·7 per cent in the first flow sheet.

Pan amalgamation tests on table concentrates.—The concentrate obtained from the first tabling in test No. 1B was used for this test. The concentrate weighing 164·7 grammes was placed in a mortar with 10 per cent of its weight of mercury. Water was added to make a thick pasty pulp which would hold up the mercury and the mixture was ground for four hours. The pulp was gradually diluted, the grinding being continued for another hour until the concentrate had been all reduced to a slime. The slime was decanted and washed away and the mercury was collected and retorted to recover the gold. The following results were obtained:—

Amount of gold in concentrate before amalgamation	3·44	ozs. per ton
Amount of gold in concentrate after amalgamation	0·63	"
Amount of gold recovered as bullion	2·81	"

This means that 81·7 per cent of the gold in the concentrate was recovered as bullion ready for the market. The residue remaining contained 0·63 oz. gold per ton, valued at, roughly, $12.60, and 2·07 per cent arsenic. This concentrate could probably be sold to smelters in the United States owing to the favourable situation of the mine, which is understood to be on the sea coast.

AMALGAMATION TESTS

A series of amalgamation tests were run to determine the best size to crush the ore for the primary amalgamation. It had previously been observed that the recovery fell off if the ore was crushed finer than 35 mesh. The following tests were run:—

TABLE IX

Test No.	Size to which ore was crushed for amalgamation	Percentage of gold recovered by amalgamation as bullion
	mesh	per cent
3	−20	47·3
4	−28	48·9
5	−35	59·4
2	−50	47·5

The above table shows that the best size to crush to for the primary amalgamation is between 30 and 40 mesh.

LARGER SCALE TEST

A larger scale test was made on 10,545 grammes of ore, using flow sheet Figure 5. The ore was crushed wet in a ball mill to pass 40 mesh and then passed over amalgamation plates. The amalgamation tailing was then recrushed to 100 mesh and passed over a second series of amalgamation plates. The second amalgamation tailing was concentrated on a Wilfley table and a concentrate and a final tailing were made. Recapitulation of results:—

Assay of ore before treatment	0·64	oz. per ton
Assay of tailing after first amalgamation	0·18	"
Assay of tailing after second amalgamation	0·05	"
Assay of table tailing	0·035	"
Assay of table concentrate	1·33	"
Per cent of total gold recovered by amalgamation at 40 mesh	71·9	
Per cent of total gold recovered by amalgamation at 100 mesh	20·3	
Per cent of total gold recovered by amalgamation	92·2	
Per cent of total gold recovered by table concentration	2·6	
Per cent of total gold recovered by combined operations	94·8	

The table concentrate represented 6·34 per cent of the weight of the original ore. That is for every 100 tons of ore treated 6·34 tons of concentrate would be obtained assaying 1·33 ozs. per ton.

Corduroy blankets were used after the amalgamation plates in the second amalgamation at 100 mesh. The material remaining on the blankets was passed over the table with the second amalgamation tailing, hence the increase in amount of gold values shown in the table concentrate.

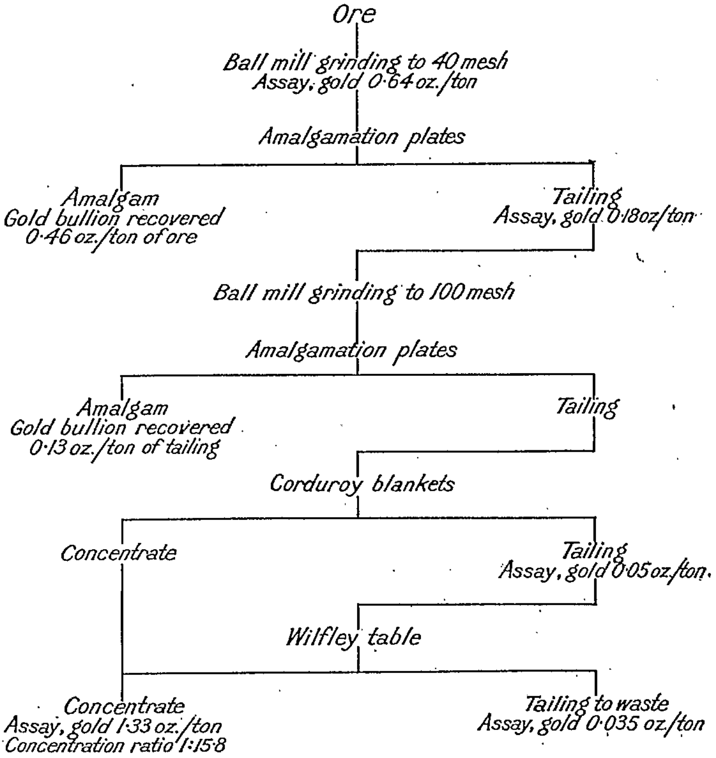


Fig. 5. Flow sheet used for larger scale tests, Cranberry Head gold ore.

Roasting and pan amalgamation of table concentrate.—An assay ton of the concentrate was roasted and the residue amalgamated by pan amalgamation, the idea being that the arsenic could be recovered in the form of a crude arsenical oxide by roasting, and the residue amalgamated for the recovery of the gold values. The results were as follows:—

Concentrate before roasting	1·33	oz. per ton
Concentrate after roasting	0·96	"
Loss in roasting	0·37	"
Loss in residue after amalgamation	0·23	"
Recovered by amalgamation	0·73	"
Recovered	55	per cent

This method of treating the table concentrate was not very successful.

CONCLUSIONS AND RECOMMENDATIONS

It is understood that the property from which these samples were taken has not been developed to any extent, and that no tonnage of ore has been blocked out. The samples received were taken from the surface, as a considerable portion of the rock showed surface weathering. Taking into consideration that the property is still in the development or prospect stage, the flow sheet as indicated by Figure 6 is recommended for the recovery of the gold.

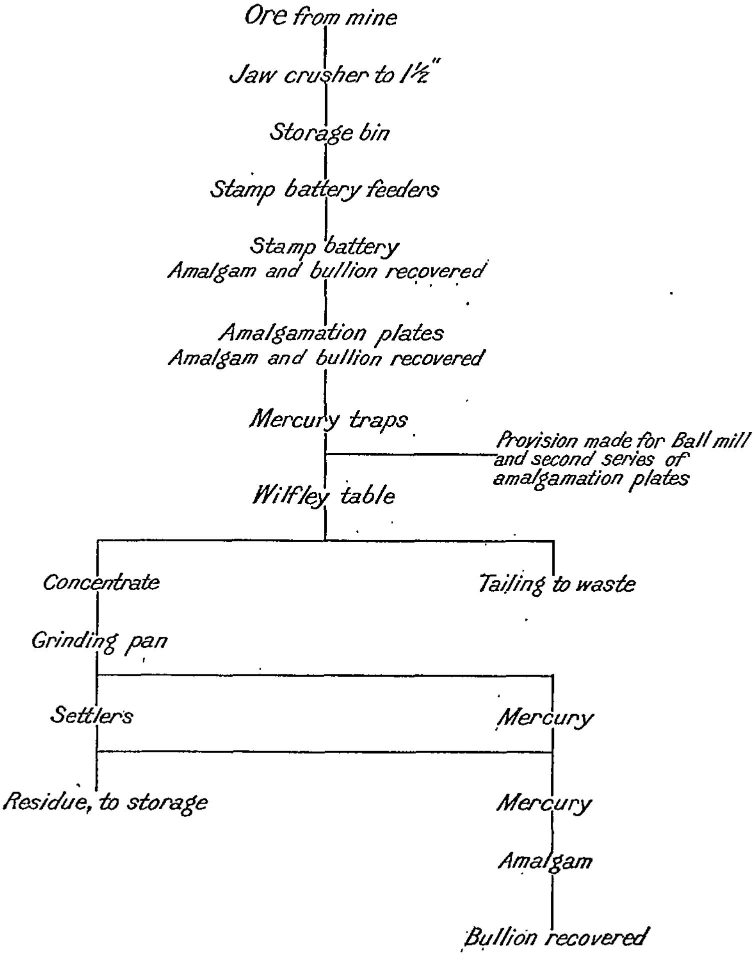


Fig. 6. Flow sheet recommended for treatment of Cranberry Head gold ore.

The flow sheet can be added to and improved later when sufficient ore has been developed in the mine to warrant a larger expenditure. It is advisable to install a 1000-pound ten-stamp battery of the California type with a wide and deep mortar for inside amalgamation. The mortar should be at least 18 inches wide at a 6-inch discharge. For the most satisfactory operation the battery should be fed by a feeder, of the Challenge type. A jaw crusher should be used to crush the ore to 1½ inches before it is fed to the battery. The plate area should be fairly large, at least 25 square feet per stamp, and the plates should be stepped. A good type of mercury trap sould be used at the end of the plates. The tailings from the stamps should be run over one Wilfley table for concentration. Provision should be made to treat the table concentrate, and for this purpose the use of a grinding pan with a capacity of 2 tons of concentrate per day, together with mercury settlers to recover the mercury, is recommended.

The results of the foregoing tests indicate that the table concentrate will contain 1·3 or more ounces of gold per ton, and that an 80 per cent recovery of this gold can be obtained by fine grinding and amalgamation in a pan.

When the mill is built, provision should be made for the installation of a ball mill for regrinding the tailing from the battery plates, and for further amalgamation on stationary plates, so that if later it is found advisable to regrind, this equipment can be installed with the minimum of expense.

Report No. 193

AMALGAMATION AND CYANIDE TESTS ON THE ORE FROM THE OPHIR LODE MINE, BRITISH COLUMBIA

R. K. Carnochan

A shipment of ore, gross weight 20 pounds, was received May 7, 1923, from Dr. H. B. Morrison, Nelson, B.C. This ore came from the Ophir Lode mine, situated on the divide between Gainer creek, a fork of the Lardeau river, and Marsh Adams creek, a fork of the Westfall river.

The ore contains values in gold and silver. The sample received was very much weathered and contained a considerable quantity of iron sulphide.

Tests were desired on the sample to determine if amalgamating at 80 mesh would be better than amalgamating at 40 mesh, and if it would be necessary to cyanide after amalgamating.

Head sample.—The ore was crushed to 40 mesh, some metallics being obtained. A sample for assay was cut out and ground to pass 80 mesh, more metallics being obtained. The −80 mesh material and the two lots of metallics were assayed with the following result:—

——	Au. oz. per ton	Ag. oz. per ton
+40 mesh metallics	0·38	0·06
+80 mesh metallics	0·80	0·08
−80 mesh	1·76	0·46
Heads	2·94	0·60

Amalgamation.—Amalgamation tests were made on the ore by mixing in a small pebble jar for three hours, about 1000 grammes of ore, 400 c.c. of water, and 100 grammes of mercury. After mixing, the mill charge was put through a hydraulic classifier to separate the mercury from the tailings. The tailing was then dried, weighed, and sampled for assay. Three tests were made on ore ground to −40 mesh, and three on ore ground to −80 mesh. The results were:—

——	Au. oz. per ton	Ag. oz. per ton
−40 amalgamation tailing	0·35	0·24
−40 amalgamation tailing	0·36	0·23
−40 amalgamation tailing	0·28	0·15
Average	0·33	0·21
−80 amalgamation tailing	0·38	0·20
−80 amalgamation tailing	0·28	0·16
−80 amalgamation tailing	0·33	0·21
Average	0·33	0·19

Amalgamation at 40 mesh gave in metallics and amalgam a gold recovery of 88·8 per cent and a silver recovery of 65·0 per cent. Amalgamation at 80 mesh gave in metallics and amalgam a gold recovery of 88·8 per cent and a silver recovery of 68·3 per cent.

Cyanidation.—Cyanidation tests were made on all amalgamation tailings. About 200 grammes of tailing was taken for each test, a 24 hour agitating period was used, and the tailings were ground to −100 and −200 mesh. The average results obtained are:—

——	Au. oz. per ton	Ag. oz. per ton
−40 amalgamation tailing cyanided at −100 mesh	0·01	0·03
−40 amalgamation tailing cyanided at −200 mesh	0·02	0·01
−80 amalgamation tailing cyanided at −100 mesh	0·00	0·05
−80 amalgamation tailing cyanided at −200 mesh	0·03	0·02

The above table shows that the lowest tailings in gold are apparently secured by cyaniding at −100 mesh. This is not the case, however, as cyaniding at −200 mesh will always give as good, and usually better, results than at −100. The tests show that cyaniding at either −100 or −200 will give a tailing running about 0·02 oz. gold and 0·03 oz. silver per ton. With this tailing, the recoveries made on the ore by amalgamating and cyaniding are, gold 99·3 per cent, silver 95·0 per cent.

The average cyanide consumption in the tests was 5·5 pounds per ton of amalgamation tailing treated, and the average lime consumption was 7·2 pounds per ton.

CONCLUSIONS

1. Amalgamation followed by cyanidation is a suitable method of treating the ore submitted, as the recoveries are very high, and the tailings very low.

2. Amalgamation at 40 mesh is preferable to amalgamation at 80 mesh, as the gold recovery at 40 mesh is the same as that at 80 mesh; the silver recovery is only slightly lower, and the mercury does not flower to nearly the same extent.

3. Cyanidation will be necessary after amalgamation, as the increase in recovery certainly warrants it, and the amalgamation tailings are too high to discard.

4. For cyaniding, the amalgamation tailings will need to be ground to 100 mesh. Finer grinding than 100 mesh is not necessary.

5. The consumption of cyanide is fairly high, probably due to the oxidized condition of the ore. It would be expected on fresh unoxidized ore that this consumption would be much less.

Report No. 194

THE CONCENTRATION OF THE COMPLEX SULPHIDE ORE OF THE EUSTIS MINE, EUSTIS, QUEBEC

C. S. Parsons

Shipment.—A carload shipment of about 20 tons of ore was received May 29, 1923, from the Eustis Mining Co., Ltd., Eustis, Que. The shipment was submitted as representative of the company's milling ore from the Eustis mine.

Purpose of experimental tests.—The object in conducting experimental test work was to determine a satisfactory and commercial method of separating the minerals, with the production of marketable copper concentrates with high recoveries of the values in the ore.

Previous experimental tests.—The company had, previous to making this shipment, carried on small laboratory tests for a year under the direction of Mr. J. H. Allen, of Knox and Allen, consulting metallurgists of New York. The results of these tests were at first erratic, but as the work progressed, more uniform and very encouraging results were obtained from the small scale laboratory tests.

Request for tonnage check tests.—In order to eliminate any possibility of failure, the company made application to the Department to have tonnage check tests conducted to verify the results of the small scale tests, before proceeding with the alterations to their concentrating plant.

Arrangements for tonnage check tests.—It was arranged that Mr. Allen, representing the company, should be present during the running of the tests and that the work should be carried on under the joint supervision of Mr. Allen and an official of the Department.

Previous work of a similar nature.—The production of a high-grade copper concentrate from complex sulphide ores has, in many cases, presented difficult problems, and it is only in the last few years that successful solutions have been obtained commercially. The Department has given considerable attention to these problems, and has obtained encouraging results on a number of Canadian ores by differential flotation. Experimental work on such ores as the Flinflon, Hidden Creek, and the Sudbury copper-nickel ores indicated that the chalcopyrite could be floated selectively from the iron sulphides by maintaining an alkaline pulp with lime.

Decision as to experimental work.—It was desired by the company that the work should include two particular lines of investigation—

First: The flotation of the chalcopyrite from the other minerals of the ore with the production of a marketable copper concentrate (at least 20 per cent copper) with high recoveries of the copper content, and,

Second: The flotation of the iron sulphide from the gangue with the production of a marketable concentrate of high sulphur content.

This report deals entirely with the differential flotation of the chalcopyrite from the iron sulphides and gangue minerals. The flotation of the pyrite will be taken up later under a separate investigation.

Characteristics of the ore.—The ore is a complex sulphide consisting of chalcopyrite associated with pyrite and small quantities of calcite, quartz, and other gangue minerals. The shipment received contained about 3 per cent copper. The chalcopyrite is finely disseminated throughout the pyrite, the sulphides constituting 80 per cent of the ore, the remaining 20 per cent being made up of gangue minerals. A careful examination of the ore indicated that it should be ground to pass 100 mesh to free the chalcopyrite.

Sampling and analysis.—Each run was sampled separately. Three tons were weighed out, crushed in a jaw crusher and rolls to one-quarter inch. A tenth part was cut out by a Vezin automatic sampler and further reduced for analysis. An approximate analysis of the shipment would be:—

Copper	2·9	per cent
Iron	40·0	"
Sulphur	45·0	"
Insoluble	7·5	"
Other gangue	4·6	"

EXPERIMENTAL TESTS

Small scale tests by the Department.—Before the large scale tests were run, a number of small scale tests were run in order to gain some idea of the characteristics displayed by the ore during flotation, such as the nature of the froth, concentrate, tailing, etc.

Summary of small scale tests.—A summary of the results of these tests is given in table No. I.

TABLE I

SMALL SCALE TESTS

Test No.	Product	Analysis per cent Cu.	Remarks
1	Concentrate Middling Tailing	25·2 6·3 0·59	1000 grammes ore −80 mesh, lime 10 lb/ton. Barretts No. 634, Pine oil GNS No. 5.
2	Concentrate Middling Tailing	29·11 4·78 0·855	1000 grammes ore −80 mesh, lime 10 lb/ton. Barretts No. 634, Pine oil GNS No. 5.
3	Concentrate	18·1	1000 grammes ore −80 mesh, lime 10 lb/ton. Barretts No. 634, Pine oil GNS No. 5.
4	Concentrate	29·69	1000 grammes ore −80 mesh, lime 10 lb/ton. Barretts No. 634, Pine oil GNS No. 5.

Conclusions from small scale tests.—These tests clearly indicated that a high-grade concentrate could be produced. A low tailing, however, was not obtained. The poor recovery was probably due to the high dilution of the pulp used in these tests, which was approximately 1: 4. When running the large scale tests, it was found that in order to obtain a low tailing, the pulp must be kept fairly dense, approximately 1:2·5 in the cells.

Large scale tests.—The same flow sheet was followed in all tests. The one-fourth-inch material was fed to a 4½-ft. Hardinge ball mill carrying a mixed ball charge of 4,000 pounds of 2-inch and 1½-inch balls. The mill was operated in closed circuit with a Dorr classifier, the overflow of the classifier going to Callow flat bottom cells. The rate of feed was maintained at about 1,000 pounds per hour. The results of only the first three tests are given in condensed form from the notes kept by Mr. Allen.

Test No. 1

Synopsis of run.—This test was started with a clean circuit, the ball mill being empty. Lime was added to the ball mill with the ore at the rate of 5 pounds per ton. When the ball mill started to discharge it was noticed that the pulp did not have the right appearance for flotation. The pulp was discharging fairly thick from the mill; more water was added, and the mill discharge reduced to 50 per cent solids. The appearance of the pulp improved slightly. By this time the pulp had commenced to enter the flotation cells, but conditions were not right as the froth was pasty and dark looking, and not lifting much mineral. The lime was gradually increased to 10 pounds per ton before correct conditions were obtained, the ball mill discharge being maintained at 50 per cent solids. The condition of the pulp discharging from the mill began to improve and a bright float of copper was obtained in the classifier. The character of the froth in the cells started to show a radical change, becoming brighter and lively with fast breaking small bubbles which surrounded larger slower breaking ones. Copper began to colour the froth heavily at the feed end of the cell, while the froth near the discharge end became white and watery. It was known from previous experience that the right conditions had been obtained.

After obtaining this correct condition, attention was paid to the oils. Barretts No. 634 oil was fed to the ball mill, and the General Engineering Company's YZ reagent was added to the classifier overflow to the cells.

Density samples were taken every 15 minutes on the ball mill discharge and the classifier overflow, and every 30 minutes on the cell discharge.

Samples for analysis.—No samples were taken until near the end of the run, as the chief object in this first test was control.

Analysis of samples.—A sample of the concentrate and tailing from the cells was taken and analysed:—

TABLE II

ANALYSIS OF SAMPLES—TEST NO. 1

Product	Cu. per cent	Fe. per cent	S. per cent	Insol per cent
Head sample	2·90	40·6	46·4	7·10
Concentrate	23·66			
Tailing	0·49			

Tests Nos. 2 and 3

Procedure.—The same procedure was followed as in the previous test.

Oils and reagents used.—The amount of lime was kept constant at 10 pounds per ton. The oils and soluble flotation reagents were varied. The best results were obtained with Barretts No. 634 and TT mixture of the General Engineering Co.

Samples.—Separate samples were taken during the use of each reagent, the samples being changed every time the reagents were changed. One continuous sample was, however, taken of the concentrate and tailing during the whole run. The analysis of these samples is given in table No. III.

TABLE III

ANALYSIS OF SAMPLES—TESTS NOS 2 AND 3

Product	Cu per cent	Remarks
Head sample, test No. 2	2·95	
Concentrate No. 1, " 2	24·42	
Concentrate No. 2, " 2	24·60	
Concentrate No. 3, " 2	19·58	
Tailing No. 2, " 2	0·55	
Final tailing, " 2	1·20	Taken during whole run.
Head sample, " 3	2·88	
Final concentrate, " 3	22·30	Taken during whole run.
Concentrate No. 1, " 3	21·16	
Tailing No. 3, " 3	0·20	
Tailing No. 4, " 3	0·45	
Final tailing, " 3	0·33	Taken during whole run.

TABLE IV

SCREEN TEST ON FINAL TAILING SAMPLE—TEST NO. 2

Size	Weight grammes	Per cent
− 65+100	21·1	4·2
−100+150	43·5	8·7
−150+200	194·5	38·8
−200	241·5	48·2

TABLE V

SCREEN TEST ON FINAL TAILING SAMPLE—TEST NO. 3

Size	Weight grammes	Per cent
− 65+100	5·2	1·0
−100+150	24·0	4·8
−150+200	102·8	20·6
−200	368·0	73·6

Test No. 4

Procedure.—Similar to the preceding tests.

Synopsis of run.—The test was started at 10.10 a.m. with a built-up circuit remaining from the previous run. The rate of feed was approximately 1,000 pounds per hour.

Oils and reagents used 10.10 a.m. to 2.00 p.m.:—

Barretts No. 634	54 drops per minute
General Eng. Co., YZ	15 “ “
No. 5 steam distilled pine oil	4 “ “
Lime	10 pounds per ton

The pine oil was varied slightly from time to time to regulate the quantity of froth in the cells.

Temperature of pulp ball mill discharge.—The pulp at 47 per cent solids was 90°F.

Sample No. 1, from 11.00 a.m. to 2.00 p.m.—Time of running on sample No. 1 was three hours.

Analysis:

Heads	3·00 per cent copper
Concentrate	24·70 “ “
Tailing	0·29 “ “

Pulp dilutions, 11.10 a.m. to 2.00 p.m.—The ball mill discharge varied from 47 per cent solids at the start to 55 per cent at 1.10 p.m., with an average of 29 per cent solids. The classifier overflow varied from 46 per cent solids at the start to 29 per cent at 1.45 p.m. The average range was 29 to 36 per cent, with an average density of 33 per cent solids.

The tailing from the cells ranged during the greater part of the time between 22 and 26 per cent solids with an average of about 24 per cent. The density samples on the ball mill and tailing from cells were taken every half hour, and the classifier overflow every fifteen minutes.

Oils and reagents used, from 2.00 p.m. to 4.45 p.m.:—

Barretts No. 634	54 drops per minute
General Eng. Co., TT	30 “ “
No. 5 pine oil	1 “ “
Lime	10 lbs. per ton

Sample No. 2, from 2.00 p.m. to 4.45 p.m.—The time of running on this sample was 2·75 hours.

Analysis:

Heads	3·00 per cent copper
Concentrate	22·77 “ “
Tailing	0·12 “ “

Pulp dilution from 2.00 p.m. to 4.45 p.m.—The ball mill discharge ranged between 50 to 55 per cent solids, with an average of 53 per cent. The classifier overflow varied between 29 and 31 per cent solids, with an average of 30 per cent. The tailing from the cells varied between 24 and 30 per cent, with an average of 27 per cent.

TABLE VI

SCREEN TEST ON FLOTATION TAILING—TEST NO. 4

Mesh	Weight grammes	Per cent
+ 65		
− 65+100	19·5	3·9
−100+150	31·7	6·34
−150+200	68·8	13·76
−200	380·0	76·00

Summary:—

Started test at	10.10 a.m.
Completed test at	4.30 p.m.
Running time	6 hrs. 20 mins.
Rate of feed	952 pounds per hour

Analysis of samples taken continuously from beginning to end of test:—

Heads	3·00 per cent copper
Concentrate	23·00 " "
Tailing	0·24 " "

Conclusions.—The density of the feed to the flotation cells was much higher and more constant in the last half of the test than during the first half, but owing to the change from YZ to TT reagents, it is impossible to determine definitely whether the lower tailing obtained during the latter part of the run was due to the more uniform and higher density of the pulp in the cells or to the change of reagents. The opinion of the writer is that the lower tailing was due to the more uniform and higher density of pulp. The reason for this is based on the observation that when the density of the pulp was the highest, the appearance of the cell indicated that the best separation was being obtained.

Test No. 5

Procedure.—Similar to previous test, with the exception that 1,500 pounds of balls were removed from the ball mill, leaving 2,500 pounds.

Synopsis of run.—The test was started at 10.15 a.m. with a built-up circuit remaining from previous run. The rate of feed was maintained at approximately 1,000 pounds per hour.

Oils and reagents used at start:—

Barretts No. 634	34 drops per minute
General Eng. Co., TT	30 " "
No. 5 pine oil	1 " "
Lime	10 lbs. per ton

Conditions of froth at start.—The froth was pasty and dead at the start, but gradually cleaned up as the alkaline circuit was built up.

Change of reagents.—The Barretts No. 634 was increased to 70 drops, and the TT mixture reduced to 18 drops. A change in the character of the froth was noted, indicating over oiling and the lifting of more iron sulphide. The Barretts oil was reduced to 54 drops per minute and the TT raised to 30 drops. At the time of starting No. 1 sample the TT was increased to 36 drops per minute.

Sample No. 1, from 11.40 a.m. to 12.30 p.m.—The time of running on this sample was 55 minutes.

Analysis:

Heads	2·98 per cent copper
Concentrate	19·10 " "
Tailing	0·22 " "

Pulp dilution between 11.40 a.m. and 12.30 p.m.—The ball mill discharge varied between the narrow limits of 53 to 54 per cent solids. The classifier overflow density ranged between the two limits of 27 and 34 per cent solids, with an average of 28 per cent. The tailing discharge from the cells averaged 28 per cent.

Change of samples at 12.30 p.m.—The cells were working better and the concentrate looked cleaner.

Oils and reagents, 12.30 p.m. to 2.30 p.m.—These remained as before,—

Barretts No. 634	54 drops per minute
General Eng. Co., TT	36 " "
Pine oil	1 " "
Lime	10 lbs. per ton

Sample No. 2, 12.30 p.m. to 2.30 p.m.—A total time of two hours.

Analysis:

Heads	2·98 per cent copper
Concentrate	19·50 " "
Tailing	0·14 " "

Pulp dilution, 12.30 p.m. to 2.30 p.m.—Ball mill discharge varied between 49 and 55 per cent solids, with an average of 51. The classifier overflow varied between 26 and 29 per cent solids, remaining constant the greater part of the time at 26 per cent solids. The tailing discharge from the cells averaged 28 per cent solids.

Change of operating conditions at 2.30 p.m.—The feed, instead of all passing through one rougher cell and being returned through the second rougher cell, was divided, half going to each cell. The tailing from each rougher cell was sampled separately.

Oils and reagents, 2.30 p.m. to 4.45 p.m.—The reagents and oils were not altered.

Sample No. 3, 2.30 p.m. to 4.45 p.m.—The tailing from the two rougher cells were marked A and B respectively:—

Analysis:

Heads	2·98 per cent copper
Concentrate	20·20 " "
Tailing A	0·15 " "
Tailing B	0·12 " "

Pulp dilution, 2.30 p.m. to 4.45 p.m.—The ball mill discharge varied between 51 and 58 per cent solids. The classifier overflow varied between 29 and 33 per cent solids. The greater part of the time the density remained constant at 29 per cent solids. The tailing from the cells averaged 28 per cent solids.

Summary:

Total ore fed	6,000 pounds
Total time of running	6·5 hours
Rate of feed	923 pounds per hour

Analysis of samples taken continuously from start to end of test:—

Heads	2·98 per cent copper
Concentrate	19·00 " "
Tailing	0·20 " "

Screen test on flotation tailing.—This screen test is given in table No. VII.

TABLE VII

SCREEN TEST FLOTATION TAILING—TEST NO. 5

Mesh	Weight grammes	Per cent
+ 65		
− 65+100	6·5	1·3
−100+150	25·0	5·0
−150+200	98·5	19·7
−200	370·0	74·0

It will be observed that the removal of 1,500 pounds of balls from the mill did not affect the crushing to any extent. The material −200-mesh was only 2 per cent less than in the previous test, while the amount remaining on 100 mesh was considerably less than when more balls were used in the mill. This was due in all probability to the more careful regulation of the classifier.

Conclusions.—The use of TT mixture in place of YZ mixture, which is difficult to obtain, gave satisfactory results.

FINAL SUMMARY AND CONCLUSIONS OF ALL TEST WORK

1. If the shipment submitted was representative of the ore to be mined, the results of the experimental work justify the erection of a mill, the flow sheet of which should conform to the procedure used in conducting the tonnage check tests.

2. The results of the experimental work show that a high-grade concentrate of at least 20 per cent copper, with a recovery of better than 90 per cent, can be expected.

3. That the ore should be crushed to pass 100 mesh, and so that 80 per cent will pass 200 mesh.

4. That an alkaline pulp, made so by lime, is essential to obtain selective flotation of the copper from the iron.

5. That when the mill is first started, Barretts No. 634 and TT mixture should be used until in perfect operation, then other and cheaper oils can be tried.

6. That the temperature of the pulp in the ball mill should be maintained as low as possible, not exceeding 90° F.

7. That a high pulp density should be maintained in the flotation cells. Samples taken from the rougher cell discharge should show a density of not less than 26 per cent solids.

8. That the pulp should be delivered direct to the flotation cells from the grinding circuit. No attempt should be made to thicken this feed by dewatering. The efficiency of the closed grinding circuit should, if necessary, be sacrificed to obtain this condition. The classifier overflow density should be maintained at 30 per cent solids.

9. That the lime should be added ahead of the ball mill, as long contact is necessary.

10. That one cleaning operation is all that is necessary.

11. That to obtain efficient operation, conditions must be kept uniform. This is very clearly indicated from the tests conducted. The pulp densities must be maintained constant. The amount of lime added should be controlled by chemical determinations of the alkalinity of the tailing water from the cells.

Report No. 195

EXPERIMENTAL TESTS ON GOLD ORE FROM KIRKLAND LAKE GOLD MINING CO., LTD., KIRKLAND LAKE, ONTARIO

R. K. Carnochan

Shipment.—Two samples of gold ore and one of mill tailings were received on December 22, 1922, at the Ore Dressing and Metallurgical Laboratories, from the Kirkland Lake Gold Mining Co., Ltd., Kirkland Lake, Ont. The samples as received were numbered 1, 2, and 3, and weighed 95, 60, and 32 pounds respectively.

Purpose of experimental tests.—In the milling of their ore, the company was not securing a very low tailing, and tests were desired on the samples to determine if anything could be done to improve the extraction.

Characteristics of the ore.—Sample No. 1 was high-grade ore containing molybdenite. The gangue of this ore was mostly red porphyry with some quartz. A small amount of sulphides was present as minute specks. The molybdenite occurs as a very thin coat in the fracture faces. No free gold was visible. Sample No. 2 was run-of-mine containing no visible molybdenite, but otherwise similar to sample No. 1. Sample No. 3 was mill tailing.

Sampling and analysis.—After selecting some specimens from lots Nos. 1 and 2, they were crushed to one-fourth inch. A small portion was cut out of each and crushed to 40 mesh. Samples for assay and analysis were cut out of the 40-mesh material and also out of sample No. 3. Assays and analyses were made on all three samples with the following results:—

	Sample No. 1	Sample No. 2	Sample No. 3
Gold	4·33 oz./ton.	0·12 oz./ton.	0·04 oz./ton.
Silver	0·55 "	0·04 "	0·05 "
Molybdenite	0·30 per cent	0·15 per cent	0·10 per cent
Iron	4·20 "		
Silica	53·60 "		
Sulphur	2·30 "		
Tellurium	0·10 "		
Alumina	17·15 "		
Lime and magnesia	6·00 "		
Carbon	Nil		
Loss on ignition	3·50 "		

Due perhaps to the presence of tellurium in the ore, difficulty was found in assaying both the samples received and the products from the test work. The figures given above for samples Nos. 1, 2, and 3 are quite reliable.

Tests on Sample No. 1

Cyaniding.—Samples of the ore were ground to −100, −150, and −200 mesh. Portions of each size were agitated for different lengths of time in a solution made up to titrate 0·25 per cent KCN and 0·10 per cent CaO by using sodium cyanide and soda ash. The following table shows the results of these tests. Unless otherwise stated, agitation was effected in a bottle closed by a rubber stopper.

Time of agitation hours	Tails −100		Tails −150		Tails −200		Remarks
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	
8	0·06	0·04	0·10	0·08	0·12	0·14	
8	0·19	0·08	0·13	0·09	0·12	0·09	
16	0·10	0·10	0·12	0·12	0·04	0·04	
16	0·20	0·12	0·16	0·12	0·12	0·11	
24	0·18	0·16	0·16	0·18	0·08	0·10	
24	0·16	0·08	0·13	0·06	0·12	0·17	
24	....	....	....	....	0·09	0·03	Agitation by revolving stirrer in a pail.
24	....	....	....	....	0·14	0·05	Agitation in open bottle revolving horizontally.
24	....	....	....	....	0·09	....	1 lb/ton sodium peroxide used.
24	....	....	....	....	0·09	0·14	3 per cent H_2SO_4 wash, then water wash before cyaniding.
48	....	....	....	....	tr.	....	Agitation by revolving stirrer in a pail.
48	....	....	....	....	0·07	....	

The above table shows:—

1. The difficulty of securing checks in assaying.

2. The ore can be readily cyanided to 0·20 oz. or less gold per ton. This means a recovery of 95·4 per cent of the gold. This is a good recovery on an ore running $89.50 in gold, but the tailing runs $4.13 and is too high to throw away.

3. Fine grinding, finer than −100 mesh, does not greatly lower the tailing.

4. Increasing the time of agitation beyond 8 hours does not lower the tailing to any great extent.

5. The same results were obtained in the test work when different methods of agitation were used.

6. Sodium peroxide does not improve the cyaniding of the ore.

7. A preliminary acid wash does not make the ore more amenable to cyanidation.

Flotation and cyanidation.—In view of the fact that cyanidation alone did not produce a tailing low enough to discard, it was decided to make some tests to determine if flotation used in conjunction with the cyanide process would lower the tailing to a suitable point. In making the tests a small laboratory Ruth flotation machine was used for the flotation work. In cyaniding, a solution titrating 0·25 per cent KCN and 0·10 per cent CaO was used. Agitation was effected in bottles closed by rubber stoppers.

—	Flot. concentrate		Tailing	
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton
Floating at −200 mesh with pine oil and coal oil and then cyaniding for 24 hours	65·93	5·67	0·05	
Floating at −200 mesh with TT mixture and then cyaniding for 24 hours	52·68	3·50	0·12	0·19
Floating with heavy hardwood creosote, coal tar and coal tar creosote at −200 mesh and then cyaniding for 24 hours	16·50	2·19	0·09	0·18
Cyaniding at −200 mesh for 24 hours, and then floating with heavy hardwood creosote, coal tar and coal tar creosote	0·33	0·47	0·16	0·15
Cyaniding at −200 mesh for 24 hours and then floating with TT mixture	tr.	0·17	tr.	0·31

The above table shows:—

1. The difficulty of securing a representative sample of the ore. Apparently the sample used in the last test shown in the table did not run as high as the average of sample No. 1, for as both the flotation concentrate and tailing run a trace in gold, cyaniding for 24 hours at −200 mesh must have reduced the ore to a trace in gold. This does not agree with the tests described under cyaniding, so it would seem that the sample taken for the test was not representative.

2. Flotation used with cyaniding improves the recovery somewhat, but does not give a tailing low enough to discard.

Cyaniding and Tabling.

—	Table concentrate		Tailing	
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton
Cyaniding at −200 mesh for 24 hours and then tabling the cyanide tailing	1·25	1·65	0·10	0·12

This test shows that tabling improves the extraction somewhat but does not give a low enough tailing to discard.

Conclusions, Sample No. 1

1. It is easy to make a high recovery of the gold in this sample by cyanidation.

2. When used with cyanidation, tabling and flotation both improve the extraction.

3. It has been found impossible to secure a very low tailing.

Tests on Sample No. 2

Cyaniding.—Cyaniding tests were made on sample No. 2. These tests were made in the same manner as the cyaniding tests on sample No. 1.

Time hours	Tails −100		Tails −200		Remarks
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	
24	0·05	0·15	tr.	tr.	
48	....	....	tr.	....	Agitation by revolving stirrer in pail.

The above table shows:—

1. The difficulty of securing a representative sample, the tails in the first tests being much too high.

2. Fine grinding and cyaniding for 24 hours gives a high recovery and a low tailing.

Flotation and tabling.—As sample No. 2 was low-grade ore, it was thought advisable to see what results could be obtained by floating and tabling, as this would cost less than cyaniding, and if it worked would give a larger margin of profit per ton.

—	Flotation concentrate		Table concentrate		Table tailing	
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton
Floating at −100 mesh with heavy hardwood creosote, coal tar and coal tar creosote and tabling flotation tailing....................	tr.	tr.	0·50	0·28	tr.	tr.
Floating at −65 mesh with heavy hardwood creosote, coal tar and coal tar creosote and tabling flotation tailing....................	tr.	tr.	0·70	0·43	tr.	tr.
Floating at −80 with Barretts No. 634 and sulphuric acid and tabling flotation tailing.	1·03	0·14	1·56	0·71	0·02	0·05
Floating at −80 with Barretts No. 634 and sulphuric acid and tabling flotation tailing.	1·10	0·33	0·60	0·34	tr.	tr.
Ore at −20 ground in ball mill to about −65 then floated with Barretts No. 634 and sulphuric acid and flotation tailing tabled..	0·48	0·69	0·28	0·29	tr.	tr.

The above table shows:—

1. Floating and tabling give very good results on sample No. 2.

2. Barretts No. 634 and sulphuric acid are suitable reagents to float with, much better than the creosote mixture used in the first two tests, as this mixture did not float any values.

3. It is a question which would be the better method to use on a low-grade ore like sample No. 2. Flotation and tabling would cost less than cyaniding, but there would result a concentrate which would have to be treated at the mill, as it is scarcely valuable enough to ship to a smelter.

Tests on the concentrate produced by floating and tabling show that it can be cyanided to 0·06 oz. gold per ton. This means that the recovery by floating and tabling, and cyaniding the concentrates would be just about the same as that obtained by straight cyaniding, so it would seem that as far as can be determined by the small scale tests conducted, floating and tabling followed by cyaniding the concentrates would be the better method to treat low-grade ore such as represented by sample No. 2.

Conclusions, Sample No. 2

1. Cyaniding gives a high recovery and a low tailing.
2. Floating and tabling, followed by cyaniding the concentrate gives a high recovery and a low tailing.

Tests on Sample No. 3

Sample No. 3 was taken from the cyanide tailings of the Kirkland Lake Gold Mining Company's mill. This sample was slightly damp when received, but it was not dried, as it has been found that even the slight heating necessary to dry a tailing will make it more amenable to cyanidation. A small portion of sample No. 3 was sized and each size assayed with the following results:—

Size	Per cent by weight	Au. oz. per ton	Ag. oz. per ton
+ 65	4·7	0·09	0·10
− 65+100	25·7	0·04	0·05
−100+150	19·8	0·02	trace
−150+200	22·4	0·03	0·04
−200	27·4	0·05	0·05

Cyaniding.—Cyaniding tests were made on sample No. 3. These tests were made in the same manner as the cyaniding tests on sample No. 1.

Time hours	Tails not ground		Tails −100		Tails −200		Remarks
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	
24	trace	trace	trace	trace	trace	trace	
48	....	....	....	....	trace	trace	Agitation by revolving stirrer in pail.

The above table shows:—

1. Sample No. 3 is readily cyanided to a trace in gold.
2. The sample did not need to be reground to secure a low tailing.

Conclusions, Sample No. 3

The Kirkland Lake Gold Mining Co. are having difficulty in securing a low tailing, and numerous tests on their mill tailing show that it is impossible to reduce it to a trace in gold by straight cyaniding. In consideration of the fact that the company has made a large number of tests on many samples, and that the Division has had only one sample of tailing to experiment with, their results should be regarded as more reliable and the conclusion drawn, that the sample received by the Division was not representative, or that the air drying it received made it amenable to cyanidation.

CONCLUSIONS ON SAMPLES SUBMITTED

It is very regrettable that none of the samples received corresponded to the ore now being milled by the Kirkland Lake Gold Mining Company. Sample No. 1 is much higher than their ordinary run-of-mine, and sample No. 2 much lower. However, from the tests made on the samples submitted, it would seem that similar ore carrying a moderate value in gold could be cyanided with good results.

The molybdenite in the ore seems to have no deleterious effect on the cyanide process.

Report No. 196

EXPERIMENTAL TESTS ON GOLD ORE FROM WRIGHT-HARGREAVES MINES, LTD., KIRKLAND LAKE, ONTARIO

R. K. Carnochan

Shipment.—A box of ore, gross weight 146 pounds, was received on February 1, 1923, at the Ore Dressing and Metallurgical Laboratories, from the Wright-Hargreaves Mines, Ltd., Kirkland Lake, Ont.

Purpose of experimental tests.—Tests were desired on the sample submitted, to determine what results the different methods of milling would give, and to see whether the molybdenite present in the ore interfered with the cyanide process.

Characteristics of the ore.—The ore consists of red porphyry and dark quartz intimately mixed, and carrying a small amount of sulphides, mostly chalcopyrite and molybdenite. The molybdenite occurs in a thin film on slickensided faces.

Sampling and analysis.—After the selection of suitable specimens the balance of the shipment was crushed to one-quarter inch. A portion was cut out by means of a riffle sampler from the $-\frac{1}{4}$-inch material and crushed to pass a 20-mesh screen. A sample was cut from the −20-mesh material and ground to −100 mesh for analysis and assay. This sample was found to contain on analysis:—

Gold	5·62 oz. per ton	Sulphur	1·16 per cent
Silver	0·54 "	Tellurium	0·04 "
Molybdenite	0·32 per cent	Alumina	7·49 "
Iron	2·86 "	Lime and magnesia	2·34 "
Silica	81·10 "	Carbon	trace
		Loss on ignition	1·80 per cent

Experimental Tests

Cyaniding.—A series of cyaniding tests were made on the ore ground to −100, −150, and −200 mesh. A solution made up to titrate 0·25 per cent KCN and 0·10 per cent CaO by using sodium cyanide and soda ash was used in all the tests. Different times of agitation were used. Unless otherwise stated, agitation was secured by the use of rubber stoppered bottles.

Time hours	−100 tailing		−150 tailing		−200 tailing		Remarks
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	
8	0·68	....	0·38	....	0·40	....	
16	0·41	0·20	0·34	0·19	0·33	....	
24	0·45	0·34	0·40	0·29	0·37	0·28	
32	0·45	0·21	0·39	0·17	0·33	0·13	
40	0·48	0·21	0·40	0·20	0·35	0·15	
48	0·41	0·18	0·33	0·16	0·29	0·12	
48	....	....	....	....	0·28	....	Agitation by means of an open bottle revolving horizontally.
48	....	....	....	....	0·41	....	1 lb. sodium peroxide per ton of ore used.
72	....	....	....	....	0·23	....	
96	....	....	....	....	0·21	....	

The table shows:—

1. The ore can be readily cyanided to 0·68 oz. gold per ton. This gives a recovery of 87·9 per cent.
2. Fine grinding increases the extraction.
3. Longer agitation increases the extraction.
4. The ore ground to −200 mesh and cyanided for 96 hours gives a tailing of 0·21 oz. gold per ton. This means a recovery of 96·3 per cent. This is a very fair recovery but the tailings are too high to be thrown away.
5. Agitation by means of the open bottle gives the same results as agitation by means of the stoppered bottle.
6. Sodium peroxide does not seem to improve the extraction.

Floating and cyaniding.—A number of tests were made using flotation as an aid to cyanidation. The flotation was done in a small Ruth laboratory flotation machine. The cyaniding procedure was similar to that used in the straight cyaniding tests.

Procedure	Concentrate		Tailing	
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton
Floating at −200 mesh with coal oil and pine oil and then cyaniding the flotation tailing for 48 hours................	20·50	2·22	tr.	tr.
Floating at −200 mesh with TT mixture and then cyaniding flotation tailing for 48 hours............................	83·68	3·91	0·16	0·12
Floating at −200 mesh with heavy hardwood creosote, coal tar and coal tar creosote, and then cyaniding flotation tailing for 48 hours....................................	63·72	4·68	0·18	0·08
Cyaniding at −200 mesh for 48 hours and then floating the cyanide tailing with heavy hardwood creosote, coal tar and coal tar creosote....................................	2·43	1·03	0·20	0·15
Cyaniding at −200 mesh for 48 hours and then floating the cyanide tailing with TT mixture........................	1·30	1·11	0·24	0·20

The above table shows:—

1. There is difficulty in either getting a representative sample of the ore, or in assaying, as it is hardly possible that the tailing in the first test would run only a trace in gold when two other tests using reagents usually found better for similar ores give tailings of 0·16 and 0·18 oz. gold per ton.

2. Floating either before or after cyanidation improves the extraction a little, but leaves a tailing still too high in gold.

Cyaniding and tabling.—A test was made to see what tabling would do as an aid to cyaniding.

Procedure	Concentrate		Tailing	
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton
Cyaniding at −200 mesh for 48 hours and then tabling cyanide tailing..	1·30	1·11	0·24	0·20

This test shows that tabling improves the extraction somewhat, but does not give a sufficiently low tailing.

SUMMARY OF EXPERIMENTAL TESTS

1. Cyaniding alone, on the ore submitted, gives a good recovery, but leaves a tailing too high to discard.

2. Flotation, used as an aid to cyaniding, improves the extraction and gives a lower tailing, but the tailing is still too high.

3. Tabling, used as an aid to cyaniding, improves the extraction and gives a lower tailing, but the tailing is still too high.

CONCLUSIONS

It is regrettable that the ore submitted did not assay about the same as the ore being milled by the Wright-Hargreaves Mines, Ltd. The sample tested ran 5·62 ozs. gold per ton. This is equivalent to $116.17 per ton in gold, and is much higher than the ore that is being milled at present.

On lower grade ore it would be easier to secure a lower tailing, and it is possible that on ore similar to that now being milled at the mine straight cyaniding would work very satisfactorily.

The molybdenite present in the ore seems to have no deleterious effect on the cyanide process.

MICROSCOPIC EXAMINATION OF SELECTED SPECIMENS OF WRIGHT-HARGREAVES ORE.

E. A. Thompson

Two specimens of Wright-Hargreaves ore, selected from a shipment of 100 pounds of high grade, one containing a considerable amount of amorphous molybdenite and graphite, the other practically free from these minerals, were found to be:—

Specimen No. 1.—Telluride ore, a brecciated red syenite porphyry, the fracture faces of which are generally slickensided. Native gold and gold telluride are visible, closely associated with small amounts of chalcopyrite, amorphous molybdenite and graphite. Pyrite is also present but seems to be more closely associated with the porphyry than the other metallics.

Specimen No. 2 —A brecciated syenite, the fissures of which are filled with carbonate and quartz stringers carrying native gold. Pyrite is quite abundant but does not occur with the fissure material, evidently being of an earlier generation.

Polished Specimens

Specimen No. 1.—The microscopic examination of polished samples shows the ore to have been much fractured. Native gold, gold telluride, small quantities of chalcopyrite, and amorphous molybdenite are found closely associated along the minute fissures. These minerals were deposited at the time of fracturing or at a later period. Invariably wherever telluride is found, native gold is intermixed. Pyrite is present, associated with the gangue, and does not as a rule occur with the other metallics in the fractures, so does not appear to be genetically related to these minerals. A few specks which could not be identified as galena are considered to be lead telluride, altaite.

Specimen No. 2.—Ore has been fractured, the fissures are filled with carbonate and quartz, with native gold associated. Pyrite is also present but is associated with the original rock and is not found along the fissures.

Thin Sections

Specimen No. 1.—Thin sections show the sample to be mostly alkali feldspar, quite weathered. Quartz occurs very sparingly. The rock has been much fractured or brecciated, causing minute fissures, which are invariably filled with amorphous molybdenite, with which native gold and gold telluride are intimately associated. Small quantities of chalcopyrite are also present. The iron pyrite is more closely connected with the feldspar and quartz and seems to have crystallized at the same time. Considerable sericite is associated with the feldspar generally along the contact of the weathered crystals. There is also some chlorite. Graphite is apparently intermixed with the molybdenite. The genetic relationship seems to be a feldspar quartz rock carrying pyrite and then a fracturing and deposition of native gold, telluride, chalcopyrite and molybdenite with a small amount of quartz.

Specimen No. 2.—This ore is mostly alkali feldspar, very much weathered, with considerable sericite, which is generally found around and along the edges of the feldspar crystals. A little chlorite is also present. Quartz of the first generation occurs in minor quantities. The pyrite is closely associated with the quartz and feldspar, and apparently crystallized at the same time as the quartz. The rock has been much fractured. Along the minute fractures or fissures, native gold has been deposited with stringers of carbonate, mostly calcite. In some cases native gold is deposited along the cleavage planes of the calcite. A small quantity of secondary quartz is also found in the fissures.

PLATE II.

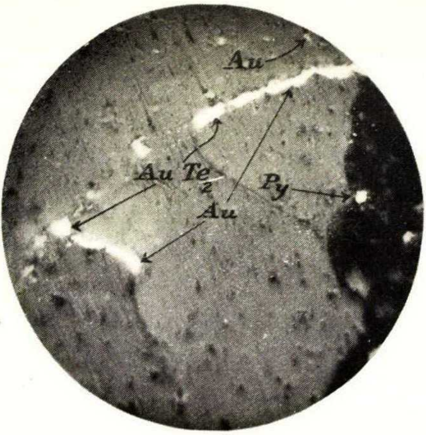


Fig. A.

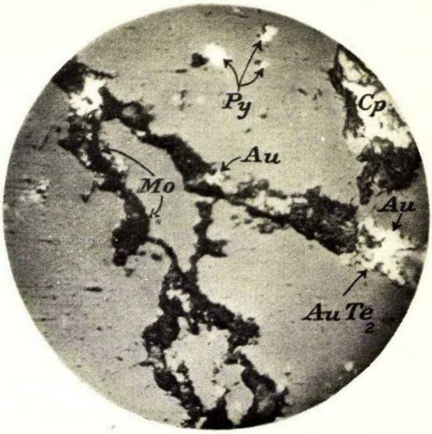


Fig. B.

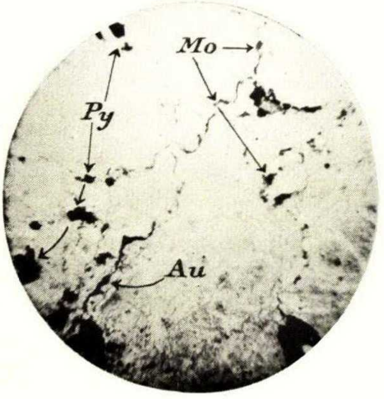


Fig. C.

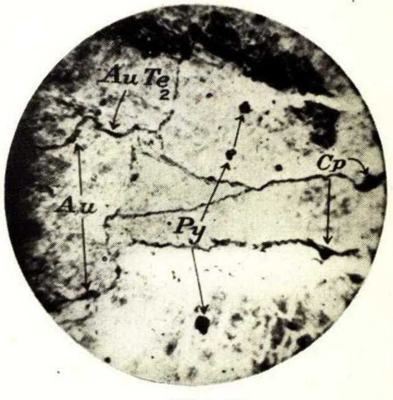


Fig. D.

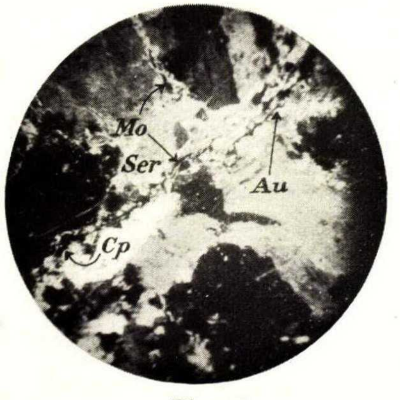


Fig. E.

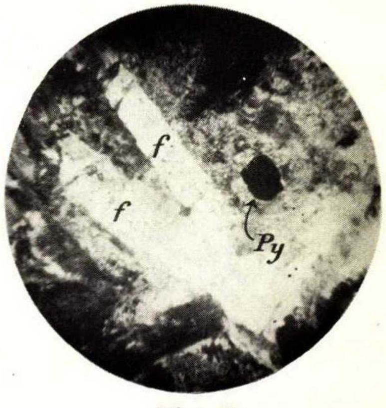


Fig. F.

PLATE III

Fig. A No. 2 ore, polished specimen, showing pyrite, Py, and native gold, Au, in a much fractured gangue, x 150.

Fig. B No. 2 ore, thin section, crossed nicols showing carbonate filling fissures, associated with native gold, Au, balance of section is weathered alkali, feldspar, sericite, a little chlorite and a few grains of pyrite, Py.

Fig. C No. 2 ore, thin section, ordinary light, showing pyrite, Py, in quartz, Q, and gold, Au, associated with calcite, Ca, balance of section is weathered feldspar and sericite.

Fig. D No. 2 ore, thin section, crossed nicols showing pyrite, Py, in quartz, Q, gold, Au, associated with calcite, Ca, balance of section is weathered alkali, feldspar and sericite.

PLATE III.

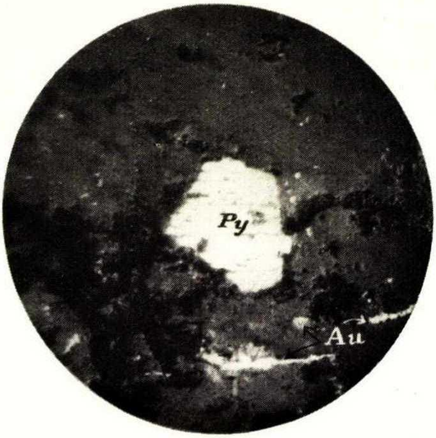


Fig. A.

Fig. B.

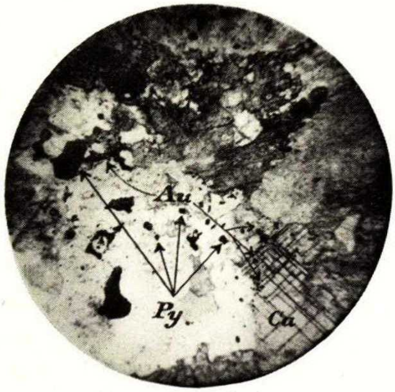


Fig. C.

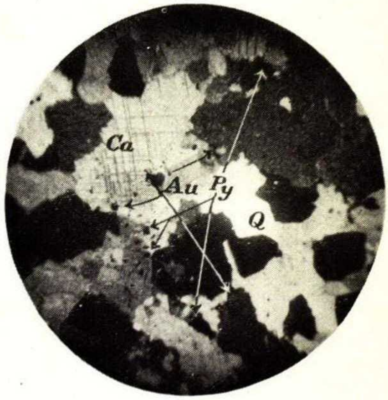


Fig. D.

Analysis of Specimen No. 1

	Per cent
Silica SiO_2	51·30
Alumina Al_2O_3	17·46
Ferric oxide Fe_2O_3	4·64
Lime CaO	1·30
Magnesia MgO	1·80
Soda Na_2O	0·14
Potash K_2O	8·18
Lead oxide PbO	0·14
Copper oxide CuO	1·33
Molybdenum trioxide MoO_3	1·96
Tellurium dioxide TeO_2	0·28=0·23 per cent Te
Gold Au	0·33=96·46 oz./ton
Carbon C	1·80
Sulphur trioxide SO_3	9·23
Water H_2O	0·50
	100·39

Metallics

	Per cent
Molybdenite MoS_2	2·18
Chalcopyrite $CuFeS_2$	3·09
Iron pyrite FeS_2	3·29
Altaite $PbTe$	0·23
Calaverite $AuTe_2$	0·23
Native gold Au	0·23
(Fe_2O_3)	0·81
Silver Ag	not determined

No carbon dioxide, antimony, mercury, bismuth, zinc or barium.

Analysis of Specimen No. 2

	Per cent
Silica SiO_2	55·00
Alumina Al_2O_3	19·89
Ferric oxide Fe_2O_3	5·11
Lime CaO	4·00
Magnesia MgO	1·80
Potash K_2O	7·23
Soda Na_2O	1·78
Sulphur trioxide SO_3	1·68
Carbon dioxide CO_2	1·80
Water H_2O	1·20
Carbon C	Nil
	99·49

Carbon dioxide calculated to calcite	4·09 per cent
Sulphur trioxide to iron pyrite	1·25 "
Calcite	4·09 "
Iron pyrite	1·25 "
Gold	79·02 oz. per ton
Silver, not determined.	

Traces of tellurium, copper, and manganese. No molybdenum or barium.

Report No. 197

GOLD ORE FROM THE PROPERTIES OF THE DASSERAT-ROUYN GOLDFIELDS, LIMITED, ROUYN TOWNSHIP, TEMISCAMINGUE DISTRICT, QUEBEC.

R. K. Carnochan

Shipment.—A shipment of 20 bags was received at the Ore Dressing and Metallurgical Laboratories, July 12, 1923, from the Dasserat-Rouyn Goldfields, Limited, 77 Trust Building, Ottawa. The shipment was from the company's claims, 3421a, b, and c, Rouyn township, Témiscamingue district, Quebec.

The shipment was in three lots, and each lot was assayed and tested separately, although the shippers claimed that lot No. 1, consisting of one bag lost in transit and afterwards located and re-shipped, was a part of the original shipment of 19 bags of ore, and that only two lots were submitted, one of ore, and the other consisting of one bag of wall rock.

The lots as received were as follows:—

Lot No. 1 (high grade), 1 bag	43·5 pounds net
Lot No. 2 (wall rock), 1 bag	54·5 "
Lot No. 3 (low grade), 18 bags	856·0 "

Characteristics of the ore samples.—A superficial examination showed the ore of lots 1 and 3 to possess very similar characteristics of mineralization and gangue constituents and it was quite impossible to distinguish which lot was the higher grade ore. The ores of both lots consisted of vein filling, quartz and a dark grey rock material with a considerable amount of iron pyrites. No copper pyrites or tellurides could be detected.

Purpose of experimental tests.—The object in view in submitting these samples to the testing laboratories was to determine if the ore was amenable to treatment with high recoveries of the precious metal values.

Sampling and analysis.—Each lot was crushed and sampled for assay. A considerable amount of flake gold was obtained from the sample of lot No. 1 when preparing it for assay. No flake was obtained from the sample of lot No. 3. The samples showed the following assay values:—

——	Gold oz. per ton	Silver oz. per ton
Lot No. 1 (high grade)	4·75	0·66
Lot No. 2 (wall rock)	trace	trace
Lot No. 3 (low grade)	0·31	0·11

Considering lots Nos. 1 and 3 as belonging to the same shipment, this would give an assay value for the ore of—gold 0·525 oz./ton, silver 0·136 oz./ton.

Experimental tests.—Lot No. 2, consisting of wall rock, showed only traces of gold and silver values. No further work was necessary. Cyanide tests were conducted on lot No. 3 followed by tests on lot No. 1.

Test 1, Lot 3.—209·5 grammes of ore, −100 mesh, was agitated for 24 hours in 1000 c.c. water to which had been added 2 grammes of sodium cyanide and 2 grammes soda ash. After agitation the ore was dewatered and washed by means of a suction filter. The results of this test were as follows:—

——	Gold	Silver
Cyanide tailing	trace	trace
Recovered by cyanidation	100·0 per cent	100·0 per cent

Cyanide consumption	3·7 lbs. KCN per ton of ore
Lime "	6·0 lbs. CaO "

Test 2, Lot 3.—205·1 grammes of ore, −200 mesh, was treated in the same manner as described under test 1. The results were as follows:—

——	Gold	Silver
Cyanide tailing	trace	trace
Recovered by cyanidation	100·0 per cent	100·0 per cent

Cyanide consumption	5·0 lbs. KCN per ton of ore
Lime "	6·1 lbs. CaO "

Test 3, Lot 1.—215 grammes of ore, −100 mesh, were treated in the same manner as described under test 1. The results were as follows:—

	Gold	Silver
Cyanide tails	0·03 oz. per ton	0·31 oz. per ton
Recovered by cyanidation	99·4 per cent	53·0 per cent

Cyanide consumption	4·6 lbs. KCN per ton of ore
Lime "	5·7 lbs. CaO "

Test 4, Lot 1.—244 grammes of ore, −100 mesh, were treated in the same manner as described under test 1. The results were as follows:—

	Gold	Silver
Cyanide tailing	0·05 oz. per ton	0·51 oz. per ton
Recovered by cyanidation	98·9 per cent	22·7 per cent

Cyanide consumption	3·2 lbs. KCN per ton of ore
Lime "	5·7 lbs. CaO "

Test 5, Lot 1.—216 grammes of ore, −200 mesh, were treated in the same manner as described under test 1. The results were as follows:—

	Gold	Silver
Cyanide tailing	trace	trace
Recovered by cyanidation	100·0 per cent	100·0 per cent

Cyanide consumption	3·6 lbs. KCN per ton of ore	
Lime "	6·1 lbs. CaO	"

SUMMARY OF RESULTS

Test No.	Lot No.	Heads		Mesh	Cyanide tails		Recovery		Consumption	
		Gold oz. per ton	Silver oz. per ton		Gold oz. per ton	Silver oz. per ton	Gold per cent	Silver per cent	Cyanide lbs. per ton	Lime lbs. per ton
1	3	0·31	0·11	−100	tr.	tr.	100·0	100·0	3·7	6·0
2	3	0·31	0·11	−200	tr.	tr.	100·0	100·0	5·0	6·1
3	1	4·75	0·66	−100	0·03	0·31	99·4	53·0	4·6	5·7
4	1	4·75	0·66	−100	0·05	0·51	98·9	22·7	3·2	5·7
5	1	4·75	0·66	−200	tr.	tr.	100·0	100·0	3·6	6·1

CONCLUSIONS

The ore was very amenable to treatment by the cyanide process. Grinding to 100 mesh for the lower grade ore, and to 200 mesh for the higher grade, gave very low tailings with very high recoveries of the gold and silver values. The consumption of cyanide was high, due, very probably, to the weathered and oxidized condition of surface ore. This consumption should be materially lessened in treating fresh ore from below the zone of weathering and oxidization.

Report No. 198

CONCENTRATION TESTS ON GRAPHITE FROM THE CANADIAN GRAPHITE CORPORATION, GUENETTE, QUEBEC

C. S. Parsons and R. K. Carnochan

Shipment.—A shipment of six bags, containing 300 pounds of graphite ore, was received on June 22, 1923, from the Canadian Graphite Corporation, Guenette, Que. The head office of the company is at 425 Phillips place, Montreal.

Purpose of experimental work.—The purpose of the test work, on this shipment, was to check up the results being obtained in their milling operations, and to determine the percentages of recoverable graphite, in as coarse a form as possible, with high recoveries of the graphite content. The management was not satisfied with the results obtained in their milling operations, and requested the assistance of the Department in improving their practice. Their greatest concern was with their concentrating department, so that only this part of the treatment is considered in this report.

Synopsis of milling operations.—A milling plant had been erected on the property and was in operation for some time. The method first used for the concentration of the graphite was the Spearman film flotation process. This was later changed to the Callow oil flotation process. A flow sheet of the crushing and concentrating departments of the mill is given in Figure 7. The dried concentrates were sent to the finishing department, which is equipped with polishing rolls and screens.

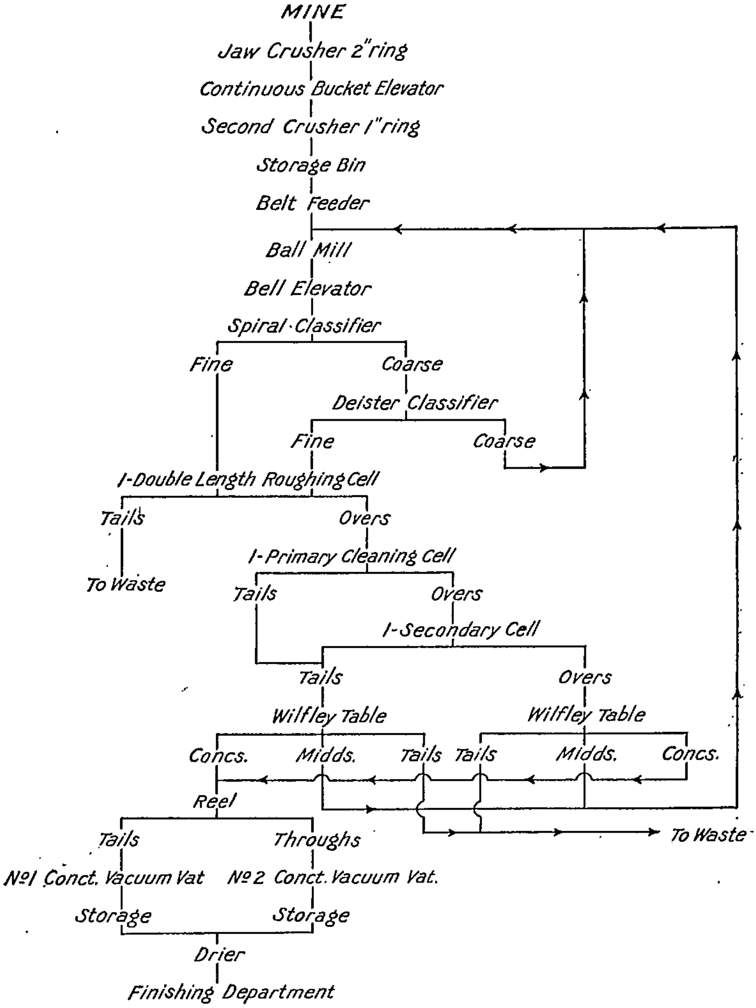


Fig. 7. Flow sheet graphite mill, Canadian Graphite Corporation Limited, Guenette, Que. Crushing and concentrating departments.

Arrangements for experimental work.—Arrangements were made with the company to have their resident manager, Mr. R. E. G. Burroughs, present while the tests were being conducted.

Characteristics of the shipment.—The graphite is of the flake variety, the flakes being of medium size but slightly smaller than those of the Buckingham deposits, the gangue was quite siliceous.

Sampling and analysis.—The entire shipment was sampled by stage crushing and cutting in a riffle sampler. The sample prepared for analysis showed it to contain 16·36 per cent carbon.

EXPERIMENTAL TESTS

Test No. 1

Object of test.—To obtain a high-grade concentrate and a clean tailing.

Procedure.—The flow sheet of Figure 8 was followed in conducting the test.

Reagent used.—To the ball mill while grinding the following reagents were added:—

Kerosene oil	1 pound per ton of ore
Pine oil (approx.)	¼ "

The grinding in the mill was done in a pulp dilution of 1:1. The flotation of the graphite was made in a Ruth laboratory flotation machine.

Results of test.—The results are given in the flow sheet of Figure 8 and in the following table:—

TABLE I

SCREEN ANALYSIS OF FINAL CONCENTRATE

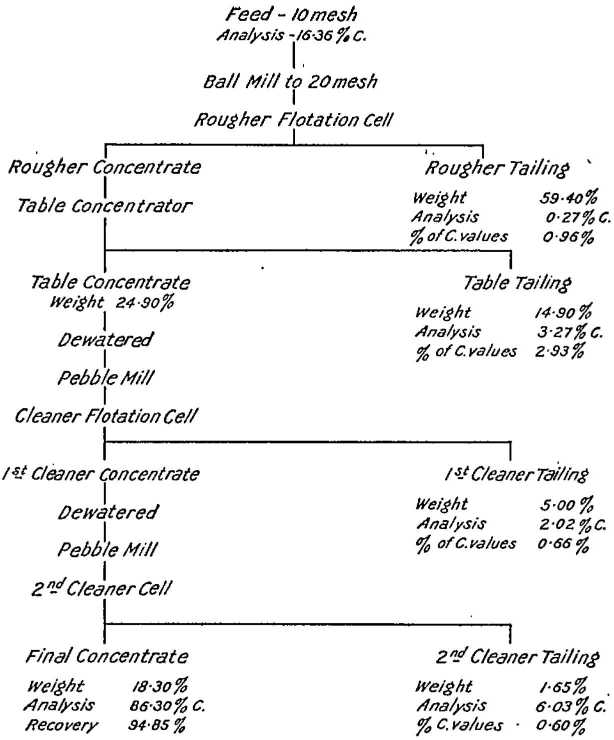
Mesh	Weight		Analysis per cent C.	Distribution of carbon content	Remarks
	Grammes	Per cent			
+100	53·2	32·4	93·55	35·5	Recovery of carbon in final concentrate, 94·85 per cent.
−100+150	34·5	21·0	93·72	23·0	
−150	76·5	46·6	76·05	41·5	
Total	164·2	100·0	84·22	100·0	

Discussion of results.—A high-grade concentrate was obtained, with an exceptionally high recovery, but it is very evident from the screen analysis of the final concentrate that the repeated grinding of the concentrate destroyed an excessive proportion of the large flake in the ore.

Test No. 2

Object of test.—To determine the highest grade concentrate which could be obtained and which would at the same time have the largest proportion of flake remaining on a 100-mesh screen.

Procedure.—The flow sheet of Figure 9 was followed. The flotation concentrate was reground only once and for a much shorter time than in the previous test.



Note - Table Tailing, 1st and 2nd Cleaner Tailing returned to Rougher Flotation Cell in practice

Fig. 8. Flow sheet test No. 1.

Results.—The results are given in the flow sheet of Figure 9, and in the following table:—

TABLE II

SCREEN ANALYSIS OF FINAL CONCENTRATE

Mesh	Weight		Analysis per cent C.	Distribution of carbon content
	Grammes	Per cent		
+100	98·5	63·3	74·05	62·1
−100+150	35·0	22·5	79·35	23·5
−150	22·3	14·2	76·65	14·4
Total	155·8	100·0	....	100·0

SUMMARY OF TESTS NOS. 1 AND 2

In test No. 2, the final concentrate contained a much larger percentage of the total carbon on a 100-mesh screen than in test No. 1. The figures are 61 per cent of the total in test No. 2 as compared to only 36 per cent in test No. 1. The grade of these two concentrates is, however, the reverse. The material on the 100-mesh screen in test No. 1 assayed 93·55 per cent C. as compared to 74·05 per cent C. in test No. 2. This means that although a lower grade concentrate on the 100-mesh screen was made in test No. 2, almost double the amount of graphitic carbon was obtained.

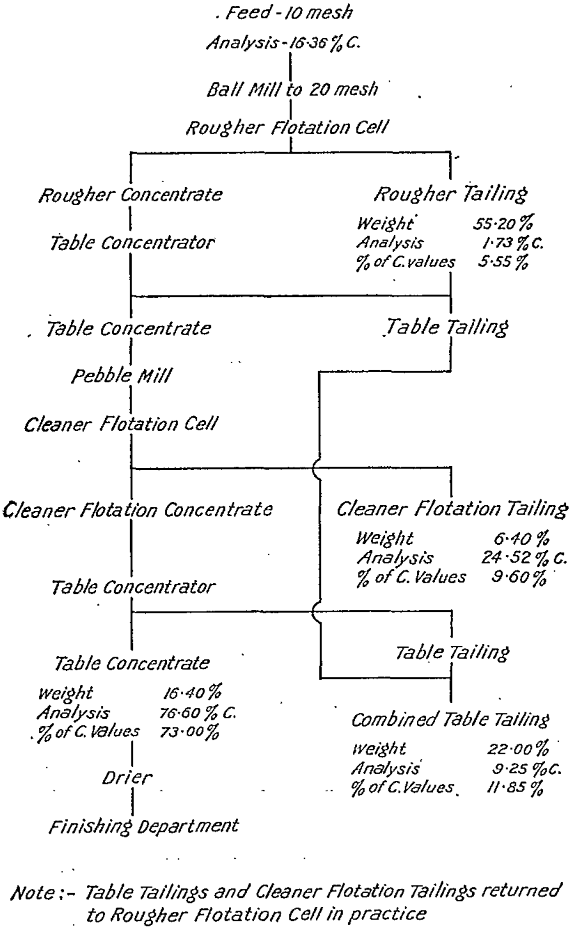


Fig. 9. Flow sheet test No. 2.

The 93·55 per cent carbon concentrate from test No. 1 still contains grit and must be refined before it can be marketed for lubricating purposes. As both the concentrates must be further refined, the essential point to be determined is where the concentrating should stop and the refining begin. The coarse flake on 100 mesh being worth three times as much as the finer flake, the efficiency of the milling will depend on the production of the largest possible amount of clean coarse flake that the ore will yield, or put another way, the efficiency will depend on the minimum destruction of the coarse flake during the grinding of the ore to free this flake.

It is quite evident that much more efficient work in this regard can be done in the refinery where the material is treated in batch lots and each operation is under control, than in the concentrator where the operation is continuous.

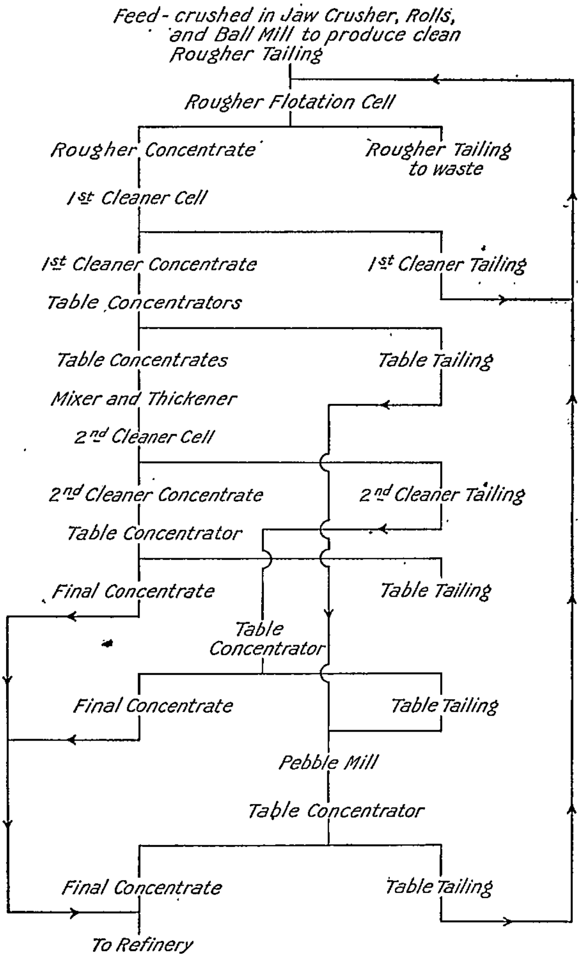


Fig. 10. Flow sheet suggested for treatment of Guenette graphite.

The writers are, therefore, of the opinion that the function of the concentrating department should be to crush the ore only fine enough to obtain a reasonably clean tailing from the flotation rougher cells rather than to produce a concentrate that contains a minimum of free gangue minerals. Middling products will be produced during the operations, and will consist of true middling, that is, particles of attached gangue and graphite, which the crushing has failed to free, and of particles of gangue which were mechanically entangled in the concentrate. The mill should be designed to divide the middlings into the two classes as far as possible. The class made up of particles in which the gangue is the largest proportion should be returned to the circuit and reground to free the graphite. The second class in which the graphite proportion is the largest, should be made to report in the concentrate, so that the work of freeing the attached pieces of gangue will be thrown on the refinery.

One flow sheet, Figure 10, is given, which it is believed, will produce these conditions, and which is suggested for the concentration of the graphite from the company's property at Guenette, Quebec.

Report No. 199

SELECTIVE FLOTATION OF THE COPPER-NICKEL ORE OF SHEBANDOWAN LAKE DISTRICT, ONTARIO

C. S. Parsons

Shipment.—A shipment of 400 pounds of copper-nickel ore was received January 30, 1923, from Messrs. Jamieson and Peacock, 509 Alworth Building, Duluth, Minn., U.S.A.

Location of deposit.—The Shebandowan nickel deposits are situated at the southwest end of lower Shebandowan lake in the Thunder Bay district, about 73 miles west of Port Arthur, and about 4½ miles south of the Canadian National railway, on mining claims TB 2192, 2204, 2219 and 2240.

Purpose of experimental tests.—Experimental tests were conducted to determine whether the ore could be concentrated and a high-grade copper-nickel concentrate obtained with a satisfactory recovery of the values. Previous work on the ore, the results of which are given in the Summary Report for 1922, page 157, Test No. 172, showed it to contain appreciable amounts of precious metal values, especially platinum and palladium. The tests were conducted to determine also whether the precious metal values were being concentrated with the copper, nickel, and cobalt values, by selective flotation.

Character of the ore.—The ore is mainly iron pyrite, chalcopyrite, and pyrrhotite, in a gangue of alumina, magnesia, and lime silicates. It carries variable percentages of nickel, copper, and cobalt, and precious metals of the platinum group.

Sampling and analysis.—The shipment was crushed to three-quarter inch and cut to 44 pounds. The 44-pound sample was crushed to 20 mesh and a sample of four pounds cut out for analysis. The analysis of this sample was as follows:—

Nickel	2·97 per cent	Alumina	10·15 per cent
Copper	1·65 “	Lime	2·50 “
Cobalt	0·15 “	Magnesia	7·64 “
Iron	24·30 “	Gold	0·01 oz. per ton
Sulphur	15·84 “	Platinum	0·03 “
Silica	26·55 “	Palladium	0·048 “

Experimental Tests

Tests Nos. 1 and 2 were made to determine whether a high-grade copper-nickel concentrate could be obtained by selective flotation. The procedure was as follows: 1,000 grammes of the ore crushed to 20 mesh was ground in a small ball mill with the flotation reagents for 30 minutes. The pulp density in the mill was one of solids to one of water. The reagents used for test No. 1 were:—

Lime	5·0 lbs. per ton
Barretts No. 634	0·5 "
TT mixture	0·15 "

The reagents used for test No. 2 were:—

Soda ash	5·0 lbs. per ton
Barretts No. 634	0·5 "
TT mixture	0·15 "

The results are given in the following table:—

Test No.	Product	Weight per cent	Analysis			Percentage of values			Total recoveries		
			Cu. per cent	Ni. + Co. per cent	Cu. + Ni. + Co. per cent	Cu.	Ni. + Co.	Cu. + Ni. + Co.	Cu. per cent	Ni. + Co. per cent	Cu. + Ni. + Co. per cent
1	Concentrate	25·7	6·35	9·88	16·23	83·0	92·1	88·3	83·0	92·1	88·3
	Tailing	74·3	0·45	0·24	......	17·0	7·9				
2	Concentrate	30·7	5·00	8·56	15·36	88·3	95·5	92·7	91·1	96·4	94·4
	Middling	17·0	0·58	0·30	......	5·7	1·9	......			
	Tailing	52·3	0·20	0·14	......	6·0	2·6				

Assuming the nickel mineral to be pentlandite, and the iron pyrrhotite, and taking the analysis of the ore and the tailing of test No. 2, the following percentages are arrived at:—

Mineral	Per cent ore	Per cent tailing
Pentlandite with cobalt	14·22	0·64
Chalcopyrite	5·71	0·57
Pyrrhotite	23·25	11·97
Gangue	56·72	86·82

These figures show that 36·1 per cent of the pyrrhotite has been eliminated in the tailing together with the gangue silicates.

CONCLUSIONS FROM EXPERIMENTAL TESTS

A high recovery of the copper, nickel and cobalt values was made in a good grade of concentrate. The recovery of copper is not as good as that obtained on the Sudbury ores, probably due to oxidization. Thirty-six per cent of the pyrrhotite has been eliminated in the tailing, together with the gangue silicates. This elimination of pyrrhotite, if practically barren of precious metal values, together with the gangue minerals, is very desirable for subsequent smelting operations.

Continuous grinding and flotation tests will be made on the ore, and the precious metal values in the flotation products determined, to prove whether these values are recovered in the concentrate.

Report No. 200

THE CONCENTRATION OF MOLYBDENITE ORE FROM THE PROPERTIES OF THE MOLYBDENITE REDUCTION COMPANY, NEAR AMOS, QUEBEC

C. S. Parsons

Shipments.—A shipment of molybdenite ore of about ten tons, in three separate lots, was received at the Ore Dressing and Metallurgical Laboratories, Mines Branch, Department of Mines, Ottawa, August 24, 1923. The shipment consisted of the following lots:—

Lot No.	Number bags	Net weight lbs.	Locality from which samples were taken
1	150	12,064	Shaft dumps from veins Nos. 4 and 4a, La Corne township.
2	75	6,070	Twelve other veins in La Corne township.
3	25	2,052	Two veins in Malartic township.
Total	250	20,186	

The three separate lots were supposed to be average and representative samples of the ore from the company's properties in La Corne and Malartic townships, Abitibi district, Quebec.

Purpose of experimental tests.—The object in conducting experimental tests was to determine whether the ore was amenable to treatment by flotation, with the production of a high-grade marketable product and a high recovery of the molybdenite values. Small scale preliminary tests previously made had shown very encouraging and satisfactory results. Tonnage check tests, or tests on a much larger scale were desired, and if satisfactory results were obtained, the installation of a treatment plant on the company's properties was contemplated.

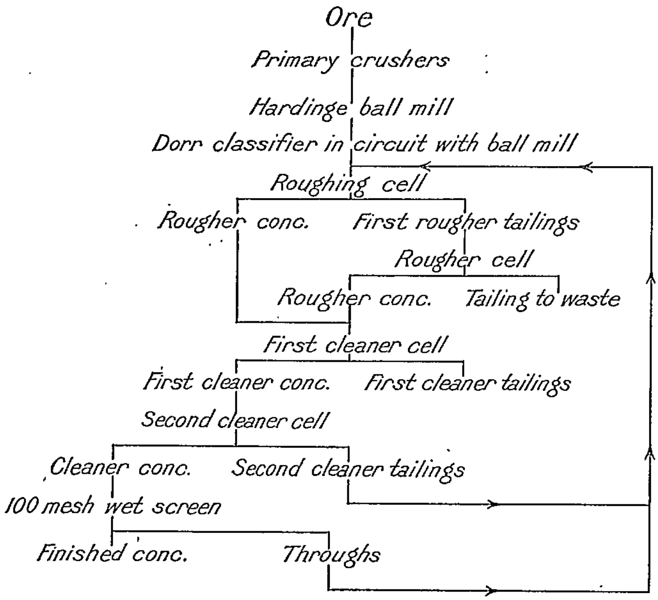
Arrangements for experimental tests.—Arrangements were made whereby the company's consulting engineer, Mr. H. H. Claudet, would co-operate with the officials of the Mines Branch in supervising the whole operations with respect to the experimental work on this ore, and who would submit to the company a separate report describing in detail the operations in conducting the mill runs and the results obtained. A copy of his report was to be filed with the Department for reference.

Characteristics of the ore.—A superficial examination shows the three lots of ore to possess very similar characteristics with respect to molybdenite content and gangue. The latter consists of quartz, a considerable amount of sericite (a thin flake mica), a little iron pyrites and tourmaline. Feldspar and fluorite are also present in small amounts. The molybdenite is associated mainly with the sericite, although it occurs with the quartz and other gangue minerals in smaller amounts. The molybdenite flake is quite heavy but not usually large. Molybdite—the yellow oxide of molybdenum and iron—occurs in such small amounts that it is of no commercial importance.

Sampling and analysis.—The three lots were sampled separately. They were reduced first in a Blake crusher set at three-quarter inch and in rolls set at three-sixteenth inch. A tenth part was cut out by a Vezin sampler. By stage grinding and cutting after each stage of reduction, samples of about 100 grammes ground to 150 mesh were submitted for analysis. The analysis of each lot was determined as follows:—

Lot No.	Total molybdenum (Mo) as molybdenite (MoS_2)	Molybdite MoO_3	Molybdenite MoS_2
	Per cent	Per cent	Per cent
1	2·33	0·014	2·316
2	1·52	0·027	1·493
3	1·88	0·023	1·857

Experimental tests.—The method used for the recovery of the molybdenite content in this ore was one that is considered to be the best practice for concentrating molybdenite ores. It consists of crushing the ore in primary crushers, grinding in ball mills in close circuit with classifiers to the required fineness to free the molybdenite, and floating the molybdenite from its gangue by the oil flotation process. The flow sheet used was as follows, with slight modifications for the various runs:—



Note:- It may be advisable to dispense with the 100 mesh wet screen as this ore contains very little impurities, such as iron pyrites and a high grade concentrate should be produced without the addition of the screen.

Fig. 11. Flow sheet of the Molybdenum Reduction Co's. mine, Amos, Que.

Lot No. 2, consisting of 5,942 pounds, was run first, followed by Lot No. 3, of 1,989 pounds. These two lots were considered as one run, as Lot No. 3 was too small to make a separate run. A partial clean-up of the plant was made before running Lot No. 1, so that the concentrates could be kept separate and an idea of the results on the smaller lot obtained in order to make any changes in operation considered necessary before running the larger lot. Lot No. 1 consisting of 11,826 pounds was run last.

It is a very difficult matter in a run of a few tons, to obtain ideal operating conditions at once. The nearest approach to actual mill conditions is the best results obtained over any one period of time during the operations. It is also difficult to obtain a full recovery of mineral, owing to a considerable portion of the pulp remaining as residue in the ball mill, in the elevators, in the classifiers, in the cells, pipes, etc., and losses which would not occur in practice under continual operation are inevitable in a short mill run. In order to keep these losses as low as possible, it was decided to consider all three lots together, and to make only one clean up at the end. This clean up was figured as a residue, and the molybdenite content in it determined.

The presence of considerable sericite, especially in Lot No. 1, presented another problem in the concentration of this ore. Consisting of thin flakes of mica, if recovered in the operations it would mean a valuable by-product worthy of consideration. In running Lot No. 1, an effort was made to produce a mica product from the second rougher cell, and this product is considered in the recapitulation of the results. Sufficient of this product was not produced to have any bearing on the results of the tests, and as the recovery of the molybdenite was of first importance, this problem was left for future consideration.

Results of experimental tests.—The following tables give the details of the results from the test operations:—

Lot No. 1—		
Net weight received	12,064 lbs.	
Deduct loss during sampling due to amount of sample, dust, floor sweepings, etc.	238	
Net weight to ball mill		11,826 lbs.
Lot No. 2—		
Net weight received	6,070	
Deduct loss as above	128	
Net weight to ball mill		5,942
Lot No. 3—		
Net weight received	2,052	
Deduct loss as above	63	
Net weight to ball mill		1,989
Total weight to ball mill		19,757

WEIGHTS, ASSAYS AND CONTENT OF ORE TREATED

Lot No.	Net weight to mill	*Assay MoS_2	Molybdenite content	
			Lbs. per ton	Total content
	Lbs.	per cent		
1	11,826	2·316	46·32	273·89
2	5,942	1·493	29·86	88·71
3	1,989	1·857	37·14	36·93
Totals and averages	19,757	2·02	40·40	399·52

*NOTE.—Molybdenum present as oxide deducted.

WEIGHTS, ASSAYS AND CONTENT OF MOLYBDENITE CONCENTRATES OBTAINED

Product	Run No.	Dry weight	Assay MoS_2	Content MoS_2
		Lbs.	Per cent	Lbs.
Concentrates produced from lots Nos. 2 and 3	1	33·0	84·40	27·85
	2	32·0	89·48	28·63
	3	34·0	91·35	31·06
		99·0	88·40	87·54
Concentrates produced from lot No. 1	4	275·5	91·80	236·39
Total concentrates produced	....	356·5	90·80	323·93

WEIGHTS, ASSAYS, CONTENT AND PERCENTAGES—TOTAL PRODUCTS OBTAINED

Product	Net weight	Assay MoS_2	Content MoS_2	Per cent of total MoS_2 in ore
	Lbs.	Per cent	Lbs.	
Total concentrates obtained	356·5	90·80	323·93	81·08
Mica product from rougher cell lot No. 1	164·0	2·37	3·89	0·97
Residue—clean-up of mill	1,089·5	3·75	40·82	10·21
Tailings, weight by difference	18,147·0	0·10	18·15	4·54
Loss, unaccounted for	....	....	12·74	3·20
Totals and averages	19,757·0	2·02	399·53	100·00

SUMMARY OF RESULTS

The results of the mill run show the following:—

With an average mill feed of 2·02 per cent molybdenite, a concentrate averaging 90·8 per cent molybdenite was obtained with an average tailing over the whole operations of 0·10 per cent molybdenite. This is equivalent to a recovery of about 95 per cent of the molybdenite content of the ore.

The grade of the concentrates produced was above the usual market requirements, which call for 85 per cent molybdenite. As the test proceeded the grade gradually increased from 84·4 per cent molybdenite in the first run, to 91·8 per cent in the fourth run, and there were no doubt intervals in the latter run where a higher grade was being produced, showing that with continuous operation under more ideal milling conditions, improvement on the grade produced in this test run could be expected.

Considering the grade of the ore, 2·02 per cent molybdenite, the average tailing of 0·10 molybdenite obtained was most satisfactory. In the first run, the tailing was higher than the average, being 0·14 per cent molybdenite, while as the test proceeded a lower tailing was being produced, and in the fourth run a tailing of 0·07 per cent molybdenite was being made, showing that with continuous operation a lower tailing assay could be expected.

The tables, giving the results from the mill run, show an unaccounted loss. This is due to unavoidable spills and leakages during operations on such a small quantity of ore. They also show a residue from clean-up operations, containing considerable molybdenite, that is not recovered in a test of this nature, but which would not occur in the operation of a molybdenite mill. In continuous operations there would be only two products, namely a marketable concentrate, and a tailing, so that the only way to arrive at the percentage recovery is to calculate it from the tailing assay. Taking 0·10 per cent molybdenite, as the average tailing obtained, this would give a recovery of 95·2 per cent of the molybdenite values in the ore.

CONCLUSIONS

The ore received for the mill run is of an exceptionally good grade' slightly higher than the average for molybdenite ores in Canada. It is of a different character from the majority of Canadian ores, in that it is associated mainly with sericite and quartz, and is quite free from other sulphides. In this respect it can be classed as a clean ore.

It is very amenable to concentration by flotation. Its physical character, and the absence of appreciable quantities of other sulphides such as copper and iron, make it an attractive milling ore. It is not difficult to grind, the molybdenite being freed at about 40 mesh. These characteristics permit of an exceptionally high-grade product being produced with a very high recovery of the molybdenite.

Report No. 201

THE CONCENTRATION AND TREATMENT OF ARSENICAL GOLD ORE FROM BATHURST, NEW BRUNSWICK

J. S. Godard

Shipment.—A shipment of 200 pounds of arsenical gold ore was received May 30, 1923, from Dr. L. D. Densmore, Bathurst, N.B. The ore occurs ten miles from Bathurst, on Stevens brook, in Beresford parish, Gloucester co., N.B.

Purpose of experimental tests.—The object, in conducting experimental tests, was to determine whether a marketable arsenopyrite concentrate could be obtained, and also whether the precious metal values could be recovered from the ore or concentrates before shipment to the smelter, or before subsequent refining operations for the recovery of the arsenic. As a general rule smelters and refiners of arsenical ores make no allowance for gold content under $10 per ton.

Characteristics of the ore.—The ore consists of arsenical pyrites with a considerable amount of iron pyrite in a siliceous gangue. Small amounts of gold and silver values are present.

Sampling and analysis.—The shipment was crushed to one-half inch in a laboratory jaw crusher and a sample of 23 pounds cut out in a Jones riffle sampler. The sample was reduced to 65 mesh and a small sample of 200 grammes cut out. This latter sample was ground to 150 mesh and assayed. Analysis showed the ore to contain:—

Arsenic	15·19 per cent
Gold	0·07 oz. per ton
Silver	1·75 "

EXPERIMENTAL TESTS

Test No. 1.—Cyanidation of the crude and roasted ore

Purpose.—To determine the amount of gold and silver values that could be recovered by cyaniding, first, the crude ore, and, second, the roasted ore.

First: Cyanidation of the crude ore

Procedure.—A sample of 543 grammes was cut from the 23 pounds of the ore crushed to 65 mesh. 100 grammes of this was taken and agitated for 24 hours with the following:—

500 c.c. of 0·25 per cent KCN solution
1·2 lb. per ton of lime

Results.—The solution at the end of agitation was tested for cyanide and lime. It showed a slight acid reaction and complete consumption of the cyanide. Assay of cyanide tailing:—

Gold	0·02 oz. per ton
Silver	1·05 "

The recoveries were:—

Gold	71·4 per cent
Silver	40·0 "

Second: Cyanidation of the roasted ore

Procedure.—A sample of 70 grammes of the roasted ore, containing arsenic 1·02 per cent, gold 0·10 oz. per ton, silver 2·46 oz. per ton, was agitated for 24 hours with the following:—

500 c.c. of 0·25 per cent KCN solution
1 lb. per ton of lime

Results.—On completion of the test the solution was found to be acidic and showed a complete consumption of the cyanide. Assay of cyanide tailing:—

Gold	0·04 oz. per ton
Silver	1·86 "

Recoveries were:—

Gold	60·0 per cent
Silver	24·4 "

Summary.—In both cases the recoveries are low. The solutions were found to be acidic at the completion of the tests. The acidity would destroy the free cyanide which accounts for the poor recoveries. Lower recoveries on the roasted ore are due to the more rapid consumption of the free cyanide by soluble sulphates formed during the roasting.

Test No. 2 — Table concentration

Purpose.—To determine whether the ore could be concentrated by tabling with the production of a high-grade arsenical product, carrying the gold and silver values.

Procedure.—1600 grammes of the ore previously crushed to one-half inch was further ground to 40 mesh and sized on 80 mesh, making two sizes, viz.: 720 grammes −40+80 and 880 grammes −80 mesh. Each size was tabled separately.

Results.—Are shown in following table:—

Product	Weight per cent	Analysis			Percentage of values		
		As. per cent	Au. oz.	Ag. oz.	As.	Au.	Ag.
Concentrate +80	27·8	24·00	0·10	1·70	43·5	41·5	28·0
Concentrate −80	26·7	24·71	0·09	1·83	43·8	35·9	29·0
Tailing +80	15·4	2·13	0·01	0·60	2·1	2·3	5·5
Tailing −80	18·8	3·94	0·03	1·52	4·8	8·4	16·9
Slimes	11·3	8·02	0·07	3·07	5·9	11·8	20·6

Ratio of concentration.. 1 : 1·8
Average grade of concentrate...As. 24·35 per cent Au. 0·10 oz. per ton Ag. 1·76 oz. per ton
Recoveries in concentrate......As. 87·1 " Au. 76·5 per cent Ag. 57·0 per cent

Summary.—Table concentration does not give a very high-grade concentrate, due to the presence of a considerable amount of iron pyrite which reports with the arsenopyrite. The gold content is slightly higher in the coarse concentrate, but the silver tends to go with the finer material. The slimes contain 20·6 per cent of the silver values.

Test No. 3.—Jig and table concentration

Purpose.—To determine the grade of the concentrate and recovery of the values by jig and table concentration of the sized products.

Procedure.—A sample of 40 pounds of the half-inch material was cut out and reduced to 4 mesh. It was closely sized, using Tyler standard screens, from 4 mesh to 100 mesh. Sizes coarser than 20 mesh were jigged, sizes finer than 20 mesh were tabled. Jig tailings were reground to 20 mesh, sized and tabled. Concentrates and tailings were made from each size.

Results.—The final results obtained on the combined jig and table concentrates and tailings are shown in the following table:—

Product	Weight per cent	Analysis			Percentage of values		
		As. per cent	Au. oz.	Ag. oz.	As.	Au.	Ag.
Combined concentrate	63·8	22·65	0·083	2·06	91·1	87·3	86·7
Combined tailings	31·9	2·40	0·024	0·62	8·9	12·7	13·3

Ratio of concentration .... 1 : 1·56

Summary.—Compared with test No. 2, jig and table concentration of the sized material gives a higher recovery of the arsenic, gold, and silver values, but the grade of the concentrate so produced is not as high as by straight table concentration.

Test No. 4—Cyanidation of the coarse concentrate

Purpose.—To determine whether the gold and silver values could be recovered by leaching with cyanide solution.

Procedure.—500 grammes of concentrates −28+35 mesh were leached with 0·25 per cent KCN solution. The first leach was for 24 hours. A second leach for 19 hours was given with partly used cyanide solution. The residue was washed twice, first with 0·05 per cent KCN solution and the second wash with 0·04 per cent KCN solution. The residue was then washed free from cyanide, dried and assayed.

Results.—The results of the test are shown below:—

Metal	Before leaching	After leaching	Recoveries
	oz. per ton	oz. per ton	per cent
Gold	0·083	0·046	44·5
Silver	2·06	1·21	41·3

Cyanide consumption	8	lbs. per ton
Lime "	9·5	"

Test No. 5—Cyanidation of the roasted concentrate

Purpose.—To determine whether the gold and silver values could be recovered from the roasted concentrate by cyanidation.

Procedure.—A series of preliminary tests were made on the roasted concentrate to determine the quantity of lime to be added to maintain a working protective alkalinity. It was found that lime equivalent to 18 lbs. per ton of roasted concentrate was required. A series of tests were made giving the roasted concentrate water washes to remove soluble sulphates and thus reduce the amount of lime required for cyanidation. Tests were made on the washed roasted concentrate using 0·05 per cent KCN solution and lime equivalent to 8, 10 and 12 lbs. per ton respectively. The time of agitation was 24 hours.

Results.—The concentrate previous to roasting contained: Gold, 0·083 oz. per ton, silver 2·06 oz. per ton. After roasting: Gold, 0·094 oz. per ton, silver 2·60 oz. per ton.

Sample No.	Before cyanidation		After cyanidation		Recoveries	
	Au. oz.	Ag. oz.	Au. oz.	Ag. oz.	Au. per cent	Ag. per cent
1	0·094	2·60	0·023	2·08	75·6	20·0
2	0·094	2·60	0·023	1·88	75·6	27·7
3	0·094	2·60	0·023	2·14	75·6	17·7

The cyanide and lime consumption was as follows:—

Sample No.	KCN			CaO		
	Per cent before	Per cent after	Lbs. per ton consumed	Lbs. per ton before	Per cent after	Lbs. per ton consumed
1	0·05	0·020	1·80	8	0·0065	7·63
2	0·05	0·020	1·80	10	0·025	8·51
3	0·05	0·016	2·04	12	0·024	10·57

Test No. 6—Cyanidation of the fine concentrate

Purpose.—To determine the recoveries of gold and silver values, using varying strengths of solution and lime equivalent to 15 lbs. per ton.

Procedure.—500 grammes concentrates were ground to pass 100-mesh screen. Three lots of 100 grammes each were agitated for 24 hours in 0·05, 0·10 and 0·20 per cent KCN solutions respectively, with lime equivalent to 15 lbs. per ton in each case.

Results.—The results were as follows:—

Sample No.	Before cyanidation		After cyanidation		Recoveries	
	Au. oz. per ton	Ag. oz. per ton	Au. oz. per ton	Ag. oz. per ton	Au. per cent	Ag. per cent
1	0·083	2·06	0·047	1·45	43·4	29·6
2	0·083	2·06	0·052	1·48	37·3	28·2
3	0·083	2·06	0·047	1·05	43·4	49·0

The cyanide and lime consumptions were as follows:—

Sample No.	KCN			CaO		
	Per cent before	Per cent after	Lbs. per ton consumed	Lbs. per ton before	Per cent after	Lbs. per ton consumed
1	0·05	Nil	3·0	15	neutral	15·0
2	0·10	0·01	5·4	15	0·0023	14·9
3	0·20	0·024	10·6	15	0·0024	14·8

Summary of cyanide tests on concentrate.—The best results were obtained by giving the roasted concentrate a water wash to remove soluble sulphates. Cyanidation of the washed concentrate gave a recovery of the gold of 75·6 per cent. Considering the low value of the gold content in the concentrate, it is questionable if it would pay for its recovery.

Flotation and Table Concentration Tests

Purpose.—To determine whether the ore could be concentrated by flotation and tabling with the production of a high-grade arsenical product carrying the gold and silver values.

Procedure:—

Test No. 9.—1,000 grammes of the ore ground wet in ball mill to 48 mesh with flotation reagents; floated in laboratory Ruth machine making concentrate and tailing. Tailing concentrated on laboratory Wilfley table.

Test No. 10.—Same procedure as test No. 9.

Test No. 12.—1,000 grammes of the ore ground wet in a ball mill to 48 mesh with flotation reagents; floated in laboratory Ruth machine making concentrate and tailing; rougher concentrate recleaned making concentrate and middling; pulp density 1:3.

Test No. 13.—1,500 grammes of the ore ground wet in ball mill to 48 mesh and screened on 100-mesh screen; the +100-mesh material was concentrated on laboratory Wilfley table making a clean concentrate and a tailing. The table tailing was mixed with the −100-mesh material and floated in laboratory Ruth machine making a concentrate and tailing. The rougher concentrate was recleaned making a concentrate and middling.

Test No. 14.—1,000 grammes of the ore ground wet in ball mill to 100 mesh with flotation reagents; floated in laboratory Ruth machine, the rougher concentrate being refloated making a concentrate, middling and tailing.

Test No. 15.—642 grammes of the ore ground wet in ball mill to 48 mesh with flotation reagents; floated in laboratory Ruth machine, the rougher concentrate being refloated making a concentrate, middling and tailing.

Reagents Used for Flotation Tests

Test No. 9.—	Tar oil	0·50	lb. per ton
	Fumol No. 6	2	drops
	H_2SO_4	18	lb. per ton
Test No. 10.—	Barretts No. 634	0·50	lb. per ton
	Fumol No. 6	2	drops
	H_2SO_4	18	lb. per ton
Test No. 12.—	Water-gas tar, 50 per cent; Coal tar creosote, 50 per cent	0·50	lb. per ton
	Fumol No. 6	2	drops
	H_2SO_4	15	lb. per ton
Test No. 13.—	Barretts No. 634	0·50	lb. per ton
	Fumol No. 6	2	drops
	H_2SO_4	18	lb. per ton
Test No. 14.—	Coal tar, 40 per cent; Coal tar creosote, 60 per cent	0·50	lb. per ton
	Pine oil No. 350	2	drops
	H_2SO_4	18	lb. per ton
Test No. 15.—	Coal tar	0·50	lb. per ton
	Fumol No. 6	2	drops
	H_2SO_4	18	lb. per ton

Results.—The results of the tests are shown in the following table:—

Test No.	Product	Weight p.c.	Analysis				Per cent of values		
			As. p.c.	Fe. p.c.	Au. oz.	Ag. oz.	As.	Au.	Ag.
9	Flotation concentrate	55·0	30·52	35·75	0·10	1·85	88·0	89·9	63·9
	Table "	2·8	27·09	32·93	0·07	3·38	4·0	3·2	5·9
	Tailing	42·2	3·63	......	0·01	1·14	8·0	6·9	30·2
10	Flotation concentrate	58·6	30·07	34·74	0·11	1·95	91·4	96·1	73·1
	Table "	0·9	29·16	32·93	0·07	2·62	1·3	0·9	1·5
	Tailing	40·5	3·53	......	0·005	0·98	7·3	3·0	25·4
12	Flotation concentrate	39·3	32·00	......	......	......	65·0	......	......
	" middling	22·5	25·40	......	......	......	29·5	......	......
	" tailing	38·2	2·79	......	......	......	5·5	......	......
13	Table concentrate	17·3	30·91	35·35	......	......	31·8	25·4 Fe.	
	Flotation concentrate	27·2	26·83	35·55	......	......	43·1	40·3 Fe.	
	" middling	15·9	19·76	30·70	......	......	18·7	20·3 Fe.	
	" tailing	39·6	2·69	8·48	......	......	6·4	14·0 Fe.	
14	Flotation concentrate	47·3	29·22	35·75	......	......	76·4	69·6 Fe.	
	" middling	16·3	14·45	21·21	......	......	13·0	14·7 Fe.	
	" tailing	36·4	5·41	10·20	......	......	10·6	15·7 Fe.	
15	Flotation concentrate	46·8	32·91	......	......	......	75·4	......	......
	" middling	17·0	18·65	......	......	......	15·5	......	......
	" tailing	36·2	5·12	......	......	......	9·1	......	......
								......	......

SUMMARY OF FLOTATION AND TABLE TESTS

The results of the above tests show that an arsenical concentrate, containing 30 per cent arsenic, 0·11 oz. gold and 1·95 oz. silver, can be obtained by flotation at 48 mesh and tabling of the flotation tailing. The recoveries of the values in this concentrate are about 92 per cent of the arsenic, 93 per cent of the gold, and 70 per cent of the silver values in the ore. The amount of sulphuric acid necessary to float the arsenopyrite is high, but no satisfactory flotation results were obtained without its use in liberal quantities.

Flotation at 48 mesh gives better results than when finer grinding is done.

GENERAL CONCLUSIONS

It does not seem advisable to cyanide the crude ore, or the concentrates either crude or roasted, as they do not contain sufficient values to pay for the cost of operation. The recoveries are low and the cyanide consumption high. The best results were obtained on the washed roasted concentrate where 76·5 per cent of the gold values were recovered.

The simplest and cheapest method of concentration is flotation at 48 mesh and tabling of the flotation tailing. An arsenical concentrate is produced, containing 30 per cent arsenic with a recovery of 92 per cent of the arsenic values. This concentrate would probably stand shipment to the arsenic smelters at present market prices for arsenic content in crude concentrate.

Report No. 202

THE TAILING DUMPS OF THE NAUGHTON GOLD MINE, NAUGHTON, ONTARIO

C. S. Parsons

The Naughton gold mine is situated 12 miles from Naughton station on the Soo branch of the Canadian Pacific railway. There is a good wagon road to the mine from the station, but at present a number of the bridges have been washed out, making the road impassible except in winter. The best method of reaching the mine is by wagon road from Sudbury or Naughton station to the Indian reserve on Black lake, and by canoe across Black lake, Round lake and Long lake, on which route there are two short portages. The mine is about a mile southeast of Long lake.

A visit was made to the mine on August 3, 1923, for the purpose of sampling the tailing dumps and estimating the available tonnage that could be reclaimed. It was reported that the dumps contained a large tonnage, assaying 10 per cent arsenic and 0·2 oz. per ton in gold.

Observations.—The mine had been abandoned, the mill and other buildings torn down and the equipment removed. The tailings were in two main dumps, in a creek bed below the mill. They were held back by a retaining wall 100 feet below the mill, and by a dam one-half mile farther down the creek towards Long lake, and into which the creek entered. A certain amount of concentration had taken place in the dumps, the coarser sulphide minerals remaining in the upper dump, the sand, gangue minerals and slime being washed down by the creek to the lower dump.

Estimate of recoverable tonnage.—The upper dump covers an area of 13,800 square yards and in places would be 12 to 15 feet deep. Figuring on an average depth of 7 feet, this dump would contain approximately 40,000 tons that could be reclaimed. The lower dump covers a considerable area along the banks of the creek, and in the basin above the dam. It is estimated that the area is about 35,000 square yards, and figuring on an average depth of 3·5 feet, the dump would contain approximately 55,000 tons that could be reclaimed. The total reclaimable tonnage from the two dumps would be approximately 95,000 tons.

Samples and analysis.—Eight samples, numbers 1 to 8 inclusive, were taken from the upper dump, and four samples, numbers 9 to 12 inclusive, from the lower dump. These samples were so taken that they should be fairly representative. The analyses are as follows:—

TABLE I

Sample No.	Arsenic per cent	Iron per cent	Sulphur per cent	Gold oz. per ton	Silver oz. per ton
1	2·09	7·93	6·22	0·06	0·02
2	3·23	11·59	9·23	0·12	0·02
3	7·81	14·64	11·38	0·17	0·04
4	7·49	20·74	19·58	1·14	0·12
5	7·53	13·83	9·75	0·26	0·04
6 7	3·02	9·96	7·51	0·08	0·02
8	1·53	7·32	5·94	0·06	trace
9	1·06	4·88	4·64	0·04	trace
10	5·22	17·98	15·44	0·18	0·04
11	0·50	4·67	3·66	0·04	trace
12	1·74	5·69	3·24	0·04	trace

Composite samples for concentration tests.—Composite samples were made up from the samples from the two dumps, for the purpose of making concentration tests to determine the grade of concentrate that could be produced by a simple means of water concentration and to determine the recovery of the arsenic and gold values. These samples showed the following analyses:—

TABLE II

Description of sample	Arsenic per cent	Iron per cent	Sulphur per cent	Gold oz. per ton	Silver oz. per ton
Composite sample upper dump 1-8	3·92	8·90	12·02	0·15	trace
Composite sample lower dump 9-12	2·24	8·90	8·94	0·08	trace

Concentration tests.—The composite samples were run on a small Wilfley table without classification or sizing. The results are shown in the following table:—

TABLE III

Product	Weight per cent	Analysis					Per cent of values	
		As. per cent	S. per cent	Fe. per cent	Au. oz. per ton	Ag. oz. per ton	As.	Au.
Table test on combined samples Nos. 1–8 inclusive—								
Concentrate	14·4	15·28	31·27	41·68	0·48	trace	62·9	57·3
Tailings	85·6	1·52	4·46	6·51	0·06	trace	37·1	42·7
Table test on combined samples Nos. 9–12 inclusive—								
Concentrate	8·5	6·15	36·0	42·31	0·26	trace	22·7	28·8
Tailings	91·5	1·94	3·79	5·30	0·06	trace	77·3	71·2

CONCLUSIONS

The Naughton tailing dumps contain an estimated tonnage that could be reclaimed of 95,000 tons, 40,000 tons of which are in the upper dump, and 55,000 tons in the lower dump.

The upper dump shows an average analysis of about 4 per cent arsenic and 0·15 oz. per ton gold.

The lower dump shows an average analysis of about 2·25 per cent arsenic and 0·08 oz. per ton gold.

A concentration test on a composite sample from the upper dump shows a ratio of concentration of 1 : 7, a grade of concentrate 15·28 per cent arsenic and 0·48 oz. per ton gold, with a recovery of 62·9 per cent of the arsenic values and 57·3 per cent of the gold values.

A concentration test on a composite sample from the lower dump shows a ratio of concentration of 1 : 12, a grade of concentrate 6·15 per cent arsenic and 0·26 oz. per ton gold, with a recovery of 22·7 per cent of the arsenic values and 28·8 per cent of the gold values.

The results of the concentration tests show that a high-grade arsenical concentrate, with fair recoveries, cannot be obtained by simple gravity methods on account of the iron sulphides present in the tailings.

The analyses and results of the concentration tests indicate that it would not pay to reclaim and concentrate the dumps, especially the lower dump, as the grade of the arsenical concentrates produced would be too low to stand the cost of concentration, treatment charges, and transportation to the smelter.

Report No. 203

CONCENTRATION OF THE ZINC-IRON MIDDLING FROM THE DUMP AT NOTRE-DAME DES ANGES STATION, QUEBEC

C. S. Parsons

Shipment.—A shipment of one bag of zinc-iron middlings, weighing 109 pounds, was received at the Ore Testing Laboratories, October 5, 1923, from the British Metals Corporation, 263 St. James St., Montreal.

Location of dump.—The dump from which this sample was taken is situated at Notre-Dame des Anges station, Que., and is estimated to contain about 40,000 tons.

Character of sample.—The material, in this dump, is a zinc-iron middling product, produced by a former concentration mill in which the ore from the Notre Dame mine was treated. The material is a mixture of zinc, iron, and lead sulphides that have been freed to a large extent by the previous milling operations. There are small amounts of gold and silver associated with the sulphides.

Sampling and analysis.—A sample was taken of the shipment, and the report of the analysis is as follows:—

Zinc	24·54	per cent
Lead	2·51	"
Iron	25·02	"
Gold	0·10	oz. per ton
Silver	4·56	"

Purpose of experimental work.—A product of the above analysis is not marketable. The purpose of the experimental work was to determine whether the zinc, lead, gold, and silver values could be concentrated by the elimination of the iron sulphides and gangue minerals.

Experimental work.—Two methods of solving the problem were tried—flotation, and magnetic separation. The latter method was supplemented by a magnetic roast. One test only was run using magnetic separation, as this method was given an extensive trial in the past and met with very little success. Flotation processes offered the greatest possibility, and it was on these that the work was concentrated.

Magnetic separation.—The ore was roasted at approximately 800° F. in a small rotary kiln. The kiln was fed and discharged continuously. The results of this test are shown in the accompanying tables.

Conclusions from magnetic separation test.—A high recovery of the zinc can be obtained by this method if the roast is properly controlled.

The zinc concentrate, however, will contain all the siliceous material, which prevents a high-grade product from being obtained. It is possible that tabling would raise the grade of the zinc product without much loss.

Flotation.—A concentrate was desired which would contain over 40 per cent zinc. Twenty-five tests were conducted, using various combinations of reagents. The zinc concentrate in a few of the tests was tabled to remove the lead. The results are given in the accompanying tables.

Conclusions from flotation tests.—The results of these tests are not conclusive, they simply indicate that certain recoveries can be obtained under certain conditions. It is possible, and quite probable, that slightly better results may be obtained. The material is, however, difficult to float. The lead is partly oxidized, and there are considerable quantities of soluble salts present. There is no doubt that a mill starting operations on this dump material would have trouble obtaining and maintaining the correct flotation conditions. It would be extremely optimistic to assume that any better results can be obtained in practice than were obtained in tests Nos. 2, 23, 24, and 25. It is the writer's opinion that small scale batch tests on material of this kind are not reliable, and that tonnage check tests should be conducted before a mill is erected.

REAGENTS USED IN CONDUCTING FLOTATION TESTS

Test No. 1.—	Ore −60 mesh	1000 grammes (approx.)
	Lime	1 gramme
	Soda ash	2 grammes
	Copper sulphate	1 gramme
	Barretts No. 634	10 drops
	TT mixture to cell	4 “
	Pine oil to cell	1 drop
	Rougher concentrate recleaned.	
Test No. 2.—	Ore −80 mesh	1000 grammes (approx.)
	Lime	0·5 gramme
	Soda ash	2·0 grammes
	Copper sulphate	1·0 gramme
	Barretts No. 634	15 drops
	TT mixture to cell	6 “
	Pine oil to cell	1 drop
	Rougher concentrate recleaned.	
Test No. 3.—	Ore −90 mesh	1000 grammes (approx.)
	Lime	2·0 grammes
	Soda ash	2·0 “
	Copper sulphate	1·0 gramme
	Barretts No. 634	0·5 c.c.
	TT mixture to cell	6 drops
	Pine oil to cell	1 drop
	Rougher concentrate recleaned.	
Test No. 4.—	Ore −90 mesh	1000 grammes (approx.)
	Soda ash	2·5 grammes
	Copper sulphate	1·0 gramme
	Barretts No. 634	0·5 c.c.
	TT mixture to cell	6·0 drops
	Pine oil to cell	2·0 “
	Rougher concentrate recleaned.	

Test No. 5.—Ore −100 mesh	1500 grammes (approx.)
Lime	0·75 gramme
Soda ash	3·0 grammes
Copper sulphate	1·5 gramme
Barretts No. 634	20 drops
TT mixture to cell	6 "
Pine oil to cell	1 drop

Rougher concentrate recleaned.

Test No. 10.—1000 grammes ore, 10 minutes in ball mill.

Lime	0·8 gramme
Soda ash	2·5 grammes
Copper sulphate	1·0 gramme
Barretts No. 634	15 drops
TT mixture	6 "

Test No. 11.—1000 grammes ore, crushed for 15 minutes.

Sodium sulphide	5 lbs. per ton
Sodium sulphite	2 "
Barretts water-gas tar	6 drops=0·35 lb. per ton
YZ mixture	0·25 lb. per ton

YZ added in flotation machine, also sodium sulphide. It was allowed to mix for 2 minutes before floating.

Note.—Necessary to add 0·36 lb. more water-gas tar. Very hard to float anything.

Test No. 13.—1000 grammes ore, 15 minutes in ball mill.

Barretts water-gas tar	0·5 lb. per ton
50 per cent acid	5 c.c.
Pine oil	2 drops
XY mixture	6 "

Test No. 12.—1000 grammes ore, 15 minutes in ball mill.
No. 34 gravity fuel oil and
Salt cake added in Ruth machine.
Concentrate high in iron and tails high in zinc.

Test No. 14.—1000 grammes ore, 15 minutes in ball mill, −80 mesh.

Barretts water-gas tar to ball mill	0·5 drop
XY to cells	6 drops
Pine oil to cells	2 "

Floated rougher concentrate. In cleaning rougher concentrate very little floated. Added 3 drops more XY, very little additional float obtained.

Test No. 15.—1000 grammes, 15 minutes in ball mill.

Barretts water-gas tar to ball mill	½ lb. per ton
Na_2S to cell	7 "
YZ to cells	4 drops
Pine oil to cells	2 "
Coal tar creosote	6 "

Note.—Iron floated, apparently zinc in tails. No middling made.

Test No. 16.—1000 grammes ore, 15 minutes in ball mill.

$CuSO_4$ in ball mill	4 lb. per ton
Water-gas tar	18 drops
YZ to cells	6 "
Pine oil	2 "

Heavy froth obtained. Added 1 drop pine oil in recleaning rougher concentrate.

Test No. 17.—1000 grammes, 15 minutes in ball mill.

$CuSO_4$ in ball mill	4 lb. per ton
Water-gas tar	9 drops
XY in cells	6 “
Pine oil in cells	2 “

Test No. 18.—1000 grammes, 15 minutes in ball mill.

YZ mixture to ball mill	0·15 c.c.
Soda ash to ball mill	2·5 grammes
Copper sulphate	2·0 “
YZ to cleaner cell	1 drop

Note.—This test was run to determine effect of presence of soda ash in pulp compared to last two tests Nos. 16 and 17 when no soda ash was used.

Test No. 19.—1000 grammes, 20 minutes in ball mill.

Barretts water-gas tar to mill	6 drops
$CuSO_4$ to ball mill	2 grammes
Soda ash	2·5 grammes
YZ mixture to cells	4 drops
Pine oil to cells	1 drop

Made clean looking concentrate and tailing.

Test No. 20.—1000 grammes, 30 minutes in ball mill.

Barretts water-gas tar to ball mill	6 drops
$CuSO_4$ to ball mill	2 grammes
Soda ash to ball mill	2·5 grammes
XY mixture to cells	4 drops
Pine oil	1 drop

For comparison of YZ and XY mixtures.

Test No. 21.—1000 grammes ore, 20 minutes in ball mill.

Barretts water-gas tar to mill	6 drops
Soda ash to mill	2·5 grammes
YZ mixture to cells	4 drops
Pine oil to cells	1 drop

Note.—This test was run to determine effect of $CuSO_4$.

Test No. 22.—1000 grammes ore, 30 minutes in ball mill.

Barretts No. 634 to ball mill	6 drops
$CuSO_4$ to ball mill	2 grammes
Soda ash	2·5 grammes
YZ to cells	4 drops
Aldol to cells	1 drop

Note.—This test was run to determine effect of fine grinding. The pulp was not alkaline.

Test No. 23.—1000 grammes ore, 25 minutes in ball mill, −100 mesh.

Barretts No. 634 to ball mill	6 drops
$CuSO_4$ to ball mill	2 grammes
Lime	0·5 gramme
Soda ash	2·5 grammes
YZ to cells	4 drops
Aldol to cells	1 drop

Note.—Pulp was not alkaline, so added 1 gramme more soda ash to cell. Pulp turned alkaline. This looks the best test of all, probably due to alkalinity with soda ash. Other tests were not fully alkaline.

Test No. 24.—1000 grammes ore, 25 minutes in ball mill.

Water-gas tar to ball mill	3 drops
Soda ash to ball mill	3·5 grammes
$CuSO_4$ to ball mill	2·0 "
Lime to ball mill	0·5 gramme
YZ to cells	3 drops
Soda ash to cells	0·5 gramme
Aldol to cells	1 drop

Note.—Pulp was not alkaline, so added 0·5 gramme soda ash to make it so.

Test No. 25.—1000 grammes ore, 25 minutes in ball mill.

Water-gas tar to ball mill	3 drops
Soda ash to ball mill	4·5 grammes
$CuSO_4$ to ball mill	2·0 "
Lime to ball mill	0·5 gramme
YZ to cells	2 drops

Aldol added to cleaner cell only.

Note.—Pulp was alkaline from ball mill.

Results of Magnetic and Flotation Tests on Zinc-Iron Middlings from the British Metals Corporation

Head sample: Zn. 24·54 per cent. Pb. 2·51 per cent. Fe. 25·02 per cent. Au. 0·10 oz. per ton. Ag. 4·56 oz. per ton.

Test No.	Product	Weight		Analysis					Analysis per cent x weight per cent				Per cent of values			
		Grms	Per cent	Zn. per cent	Pb. per cent	Fe. per cent	Au. oz. per ton	Ag. oz. per ton	Zn.	Pb.	Au.	Ag.	Zn.	Pb.	Au.	Ag.
Magnetic	Zinc	1796·5	57·4	38·09	2·67	7·90	0·03	3·87	2185	153·4	1·72	222·0	86·7	68·6	50·3	49·7
	Iron	1332·5	42·6	7·87	1·64	48·60	0·04	5·28	335	70·0	1·70	225·0	13·3	31·5	49·7	50·3
1	Zn. concentrate table	338·2	33·8	46·73	2·80	9·54	0·04	6·14	1580	94·6	1·35	207·5	65·1	36·3	33·9	39·1
	Pb. " "	11·2	1·1	13·46	51·65	......	0·48	58·56	14·8	56·8	0·53	64·4	0·6	21·7	13·3	12·1
	Zn. middlings flotation	54·6	5·5	22·52	4·6	25·8	0·06	9·54	124	25·3	0·33	52·4	5·1	9·7	8·2	9·9
	Zn. tailings flotation	593·8	59·4	11·95	1·42	33·56	0·03	3·49	710	84·3	1·78	207·3	29·2	32·3	44·6	38·9
2	Concentrate	467·2	52·4	40·22	4·35	......	......	......	2120	228·0	......	......	81·0	81·8	......	......
	Middling	115·2	12·9	22·73	1·90	......	......	......	293	24·5	......	......	11·2	8·8	......	......
	Tailing	308·4	34·6	5·83	0·76	......	tr.	2·02	202	26·3	......	69·9	7·7	9·4	......	......
3	Concentrate	284	27·8	40·47	4·38	......	......	......	1125	122·0	......	......	46·2	49·4	......	......
	Middling	90·7	8·9	24·44	2·48	......	......	......	218	22·0	......	......	8·9	8·9	......	......
	Tailing	645·8	63·3	17·25	1·63	......	......	......	1080	103·0	......	......	44·9	41·7	......	......
4	Pb. concentrate	17·5	1·7	6·22	40·95	......	0·60	49·4	106	69·6	1·02	76·4	4·3	31·2	......	......
	Zn. "	448·7	45·2	37·02	2·37	......	......	......	1673	107	......	......	68·4	48·0	......	......
	Zn. middling	207·0	20·9	21·72	1·44	......	......	......	454·5	30·1	......	......	18·6	13·5	......	......
	Zn. tailing	318·5	32·2	6·51	0·50	......	......	......	209·5	16·1	......	......	8·6	7·3	......	......
5	Concentrate	604·4	40·0	34·28	5·1	16·58	0·10	8·78	1372	204·0	4·0	351·0	57·2	78·2	58·0	72·8
	Middling	253·2	16·8	25·34	1·7	24·20	0·07	3·60	426	28·5	1·2	60·5	17·8	11·0	17·4	12·5
	Tailing	652·0	43·2	13·88	0·65	27·97	0·04	1·64	600	28·2	1·7	70·8	25·0	10·8	24·6	14·7
10	Concentrate	364·4	......	47·61	3·74	10·68	......	......	1742	......	......	......	67·6	......	......	......
	Middling	89·9	......	25·48	4·72	26·07	......	......	229	......	......	......	8·9	......	......	......
	Tailing	541·2	......	11·15	1·17	37·53	......	......	607	......	......	......	23·5	......	......	......
11	Concentrate	237·7	23·8	37·41	......	......	......	......	890	......	......	......	36·6	......	......	......
	Middling	173·0	17·5	25·38	......	......	......	......	444	......	......	......	18·2	......	......	......
	Tailing	581·5	58·7	18·75	......	......	......	......	1101	......	......	......	45·2	......	......	......
13	Concentrate	431·0	43·5	29·22	......	......	......	......	1271	......	......	......	51·5	......	......	......
	Middling	221·2	22·3	28·64	......	......	......	......	639	......	......	......	25·9	......	......	......
	Tailing	339·7	34·2	16·34	......	......	......	......	559	......	......	......	22·6	......	......	......

14	Concentrate	194·5	19·4	28·32	.....	.....	.....	.....	550	.....	.....	.....	21·3	.....	.....	.....
	Middling	194·2	19·3	29·03	.....	.....	.....	.....	562	.....	.....	.....	21·8	.....	.....	.....
	Tailing	615·0	61·3	23·96	.....	.....	.....	.....	1470	.....	.....	.....	56·9	.....	.....	.....
15	Concentrate	280·2	27·9	10·64	.....	.....	.....	.....	294	.....	.....	.....	11·4	.....	.....	.....
	Tailing	723·7	72·1	31·57	.....	.....	.....	.....	2280	.....	.....	.....	88·5	.....	.....	.....
16	Concentrate	519·9	51·5	34·79	.....	.....	.....	.....	1792	.....	.....	.....	69·8	.....	.....	.....
	Middling	104·7	10·4	19·33	.....	.....	.....	.....	201	.....	.....	.....	7·8	.....	.....	.....
	Tailing	384·0	38·1	15·09	.....	.....	.....	.....	575	.....	.....	.....	22·4	.....	.....	.....
17	Concentrate	471·8	46·9	35·32	.....	.....	.....	.....	1655	.....	.....	.....	66·0	.....	.....	.....
	Middling	148·4	14·7	22·43	.....	.....	.....	.....	330	.....	.....	.....	13·2	.....	.....	.....
	Tailing	386·5	38·4	13·64	.....	.....	.....	.....	524	.....	.....	.....	20·8	.....	.....	.....
18	Concentrate	420·2	42·0	40·06	.....	.....	.....	.....	1682	.....	.....	.....	69·3	.....	.....	.....
	Middling	152·2	15·2	27·20	.....	.....	.....	.....	413	.....	.....	.....	17·0	.....	.....	.....
	Tailing	427·7	42·8	7·73	.....	.....	.....	.....	331	.....	.....	.....	13·6	.....	.....	.....
19	Concentrate	484·2	47·8	37·92	.....	.....	.....	.....	1810	.....	.....	.....	72·9	.....	.....	.....
	Middling	182·2	17·9	23·96	.....	.....	.....	.....	429	.....	.....	.....	17·3	.....	.....	.....
	Tailing	348·0	34·3	7·02	.....	.....	.....	.....	242	.....	.....	.....	9·7	.....	.....	.....
20	Concentrate	397·0	39·3	37·80	.....	.....	.....	.....	1485	.....	.....	.....	58·2	.....	.....	.....
	Middling	147·3	14·6	27·07	.....	.....	.....	.....	395	.....	.....	.....	15·5	.....	.....	.....
	Tailing	467·2	46·2	14·46	.....	.....	.....	.....	668	.....	.....	.....	26·2	.....	.....	.....
21	Concentrate	266·5	26·4	19·63	.....	.....	.....	.....	518	.....	.....	.....	20·2	.....	.....	.....
	Middling	185·2	18·4	25·50	.....	.....	.....	.....	470	.....	.....	.....	18·4	.....	.....	.....
	Tailing	557·9	55·2	28·41	.....	.....	.....	.....	1570	.....	.....	.....	61·4	.....	.....	.....
22	Concentrate	506·8	50·2	30·57	.....	.....	.....	.....	1535	.....	.....	.....	60·2	.....	.....	.....
	Middling	40·3	4·0	21·09	.....	.....	.....	.....	84	.....	.....	.....	3·3	.....	.....	.....
	Tailing	462·0	45·8	20·35	.....	.....	.....	.....	931	.....	.....	.....	36·5	.....	.....	.....
23	Concentrate	542·7	54·0	35·95	.....	.....	.....	.....	1940	.....	.....	.....	77·6	.....	.....	.....
	Middling	162·0	16·1	22·02	.....	.....	.....	.....	354	.....	.....	.....	14·2	.....	.....	.....
	Tailing	300·0	29·9	6·92	.....	.....	.....	.....	207	.....	.....	.....	8·3	.....	.....	.....
24	Concentrate	487·5	48·4	41·02	.....	.....	.....	.....	1984	.....	.....	.....	79·4	.....	.....	.....
	Middling	195·9	19·5	19·02	.....	.....	.....	.....	371	.....	.....	.....	14·8	.....	.....	.....
	Tailing	322·9	32·1	4·55	.....	.....	.....	.....	146	.....	.....	.....	5·8	.....	.....	.....
25	Concentrate	365·1	36·4	45·06	.....	.....	.....	.....	1640	.....	.....	.....	67·5	.....	.....	.....
	Middling	83·9	8·3	25·01	.....	.....	.....	.....	208	.....	.....	.....	8·5	.....	.....	.....
	Tailing	555·2	55·3	10·75	.....	.....	.....	.....	594	.....	.....	.....	24·3	.....	.....	.....

III

REPORT ON WORK AND INVESTIGATIONS OF THE CHEMICAL LABORATORY

H. C. Mabee

Chemist in Charge

The work covered by the staff of the chemical laboratory of the Division during the year 1923 may be included in the following classes:—

I. The regular routine analytical work in connexion with the investigations conducted on ores, etc., in the ore testing and research laboratories of the Division of Ore Dressing and Metallurgy.

II. Special investigations entailing preliminary chemical tests on ores and minerals with the object of detecting and regulating any interfering elements in the processes of concentration.

III. (a) A study of the results of the application of the flotation process to the treatment of low-grade nickeliferous pyrrhotite ores, with respect, more particularly, to the behaviour of the precious metal content and to the extent of the successful elimination of barren pyrrhotite in the tailings.

(b) A study of the application of leaching and electro-deposition processes for the treatment of Canadian pyrrhotite and pyrite ores.

Various interruptions of the work of the chemical laboratory occurred during the year due to changes in the staff, rearrangements and additions to the present laboratory buildings, etc. Notwithstanding this, however, more work was accomplished than in any previous year.

I. ROUTINE ANALYTICAL WORK

H. C. Mabee

The regular routine work was of the usual wide variety, covering 1,295 samples, invloving about 4,800 chemical determinations, the results of which appear in the general reports on investigations conducted on ores and mineral products in the ore testing and research laboratories of the Division.

These samples may be included under the following classification:—

9	samples of antimony ores
58	“ arsenic ores
121	“ copper ores
500	“ gold ores
24	“ gold-cobalt ores
90	“ gold-copper ores
56	“ gold-silver-copper
1	“ iron ore
18	“ for identification
1	samples of lead-zinc-manganese ore
55	“ molybdenite ores
20	“ platinum-gold ores
22	“ silver ores
53	“ silver-lead-zinc ores
193	“ silver-lead-zinc-iron
55	“ silver residues
36	“ graphite ores

The renewed attention to gold mining continued during the year, and as a result, about seventy-five per cent of the total samples reported included results in gold and silver content. These ores were of various composition as the classification indicates, each variety requiring its special treatment. In order, therefore, to keep up with the work promptly, it was found necessary to purchase an additional muffle furnace and otherwise add to the assay laboratory equipment.

As the room provided for this work was too small to accommodate the additional equipment, and as it had become necessary to provide more space in which to carry on the work connected with the electro-deposition of iron, a request for an addition of sixty feet to the present temporary building was made. At the close of the year the work of construction was well under way.

II. SPECIAL INVESTIGATIONS PERTAINING TO THE INVESTIGATIONS CARRIED ON BY THE ENGINEERING STAFF

THE VALUE OF GIVING THE SILVER RESIDUES FROM THE DOMINION REDUCTION CO., COBALT, ONT., A WEAK SULPHURIC ACID WASH PREVIOUS TO CYANIDING, BY B. P. COYNE

Tests were made to find the value of a weak sulphuric acid wash previous to cyaniding. In each test, ore ground to 65 mesh was used.

No. 1, 50 grammes ore treated with 100 c.c. 1 per cent H_2SO_4 (approximately)
No. 2, 50 " " 100 c.c. 2 " " "
No. 3, 50 " " 100 c.c. 3 " " "

In each case, the sample was agitated for about 1 hour, the solution filtered off, 10 c.c. taken for analysis, the remainder made up to 100 c.c. with the original acid and then added to 50 grammes fresh ore, and the process repeated to find the amount of the acid consumed. The ore from the first treatment was washed until free from soluble iron and sulphate salts, the sample which required the least amount of washing was then cyanided.

	Grammes 1st	Grammes 2nd	Grammes 3rd
Iron dissolved (as Fe_2O_3)			
1 per cent acid, 50 grammes ore	0·0170	0·0255	0·0230
2 " " 50 "	0·0425	0·0580	0·0570
3 " " 50 "	0·0380	0·0670	0·0750
Acid consumption—	Per cent	Per cent	Per cent
1 per cent acid, 50 grammes ore	44·8	10·4	3·4
2 " " 50 "	37·9	25·4	2·9
3 " " 50 "	36·5	30·7	0·1

After washing five times, sample No. 1 was found to be free from soluble iron and sulphates, samples Nos. 2 and 3 still held these soluble salts. Sample No. 1 was cyanided, using 200 c.c. 0·125 per cent KCN solution and 0·1 gramme lime per 50 grammes ore. The cyanide consumption was 9·2 lbs. per ton. It was found that a 1 per cent sulphuric acid wash was as efficient as a stronger acid wash for removing the soluble salts from the ore; that it was easier to wash out the soluble salts when 1 per cent acid was used; that this treatment reduces the cyanide consumption.

DETERMINATION OF SOLUBLE CYANICIDES IN CRANBERRY HEAD GOLD ORE AND THE EFFECT OF CERTAIN REAGENTS ON CYANIDE CONSUMPTION, BY B. P. COYNE

Tests were made to determine whether the ore contained soluble cyanicides, and to determine the effect of certain reagents on the cyanide consumption.

Test No. 1.—To determine whether the ore contained soluble cyanicides 100 grammes ore (−65 mesh) were agitated for 24 hours with 600 c.c. water, 500 c.c. filtered off, 0·625 gramme KCN added to the filtrate and again agitated for 16 hours. Filtrate contained 0·124 per cent KCN.

Test No. 2.—Similar to No. 1 except that 1 gramme of lime was added to the ore.

Test No. 3.—100 grammes ore cyanided without the use of lime.

Test No. 4.—Similar to No. 2 except that 0·5 gramme NaOH was added instead of using the 1 gramme lime.

Test No. 5.—100 grammes ore cyanided with 0·125 per cent KCN and 0·15 gramme Na_2O_2.

It was found that the ore does not contain any cyanicides soluble under working conditions; that the use of Na_2O_2 reduced the cyanide consumption to 1·56 lbs. per ton ore.

DETERMINATION OF SOLUBLE SALTS IN EUSTIS COPPER ORE, AND THE EFFECT OF ADDING LIME FOR SELECTIVE FLOTATION OF THE CHALCOPYRITE, BY B. P. COYNE

A number of tests were made to determine the amount of soluble salts in the ore and the effect of adding lime. In each case 1,000 grammes ore were agitated with 1,000 c.c. distilled water. The time of contact varied from 35 to 45 minutes. In test No. 5, 5 grammes of lime were added to the pulp.

SOLUBLE SALTS

Test No.	35 minutes contact		45 minutes contact	
	Fe. grms. per litre	SO_3 grms. per litre	Fe. grms per litre	SO_3 grms. per litre
1	0·073	0·23	0·072	0·20
2	0·024	0·53	0·020	0·53
3	2·09	9·39	2·32	9·67
4	0·073	0·23	0·072	0·20
5	trace	0·43	trace	0·28

It was found that the oxidized ore (test No. 3) contained a much greater amount of soluble salts than the unoxidized ore; that the addition of lime kept the iron from going into solution.

III. A STUDY OF THE RESULTS OF TWO SMELTING TESTS ON SHEBANDOWAN LAKE ORE, AND OF THE ELIMINATION OF BARREN PYRRHOTITE IN THE FLOTATION TAILINGS FROM THE SUDBURY ORES.

(*a*) THE PRECIOUS METAL VALUES IN PRODUCTS FROM SMELTING TESTS ON COPPER-NICKEL ORE OF SHEBANDOWAN LAKE, ONT., BY H. C. MABEE

In the Summary Report of the Mines Branch for 1922 there is given, under test No. 172, the assay results of five samples of ore from the Shebandowan Lake nickel-copper deposits, collected by T. L. Tanton, and submitted to the Ore Dressing Laboratories through the Director of the Geological Survey. In January, 1923, a further shipment of 400 lbs. of this ore was received from Messrs. Jamieson and Peacock, Duluth, Minn., upon which the following experimental tests were conducted:—

Two lots of twenty pounds each were fluxed and smelted in large pots and a low-grade matte was obtained. Each of these lots was ground separately, the iron was partly eliminated by treatment with dilute hydrochloric acid and the residue carefully roasted was fluxed and assayed in the usual manner for the platinum metal content, with results as shown below:—

Analysis of Ore

Nickel	2·97 per cent	Lime	2·50 per cent
Copper	1·65 "	Magnesia	7·64 "
Cobalt	0·15 "	Sulphur	15·84 "
Iron	24·30 "	Gold	0·01 oz. per ton
Alumina	10·15 "	Platinum	0·03 "
Silica	26·55 "	Palladium	0·048 "

From the above analysis the content of the ore may be calculated as follows:—

Total sulphides—Pentlandite	14·22 per cent	
Pyrrhotite	23·24 "	
Chalcopyrite	4·71 "	
		42·18 per cent
Gangue		57·82 "

The average analysis of the slag from these two smelting tests on the ore is as follows:—

Nickel and cobalt	0·06 per cent
Copper	0·10 "
Iron	7·57 "
Sulphur	0·51 "

The weight of matte obtained averaged about 37·5 per cent of the original weight of ore in the charge. Taking into consideration the analysis of the slag as indicated above, it will be observed that only about 1 per cent of the total sulphides, including pyrrhotite, was eliminated in the slag, hence a yield of low-grade matte. This may be accounted for in this case, as these smelting operations were conducted almost out of contact with the air and consequently less opportunity for oxidation than would exist in regular smelting operations. The analysis of the matte gave the following result in platinum group metals:—

Melt No. 1.—Gold, 0·027 oz. per ton. Platinum, 0·080 oz. per ton. Palladium, 0·128 oz. per ton.
Melt No. 2. " 0·030 " " 0·085 " " 0·131 "

THE ELIMINATION OF BARREN PYRRHOTITE BY CONCENTRATION OF THE NICKELIFEROUS PYRRHOTITE ORES OF THE SUDBURY DISTRICT BY SELECTIVE FLOTATION, BY H. C. MABEE

In continuation of the experimental tests by flotation made during 1922 on the Sudbury pyrrhotite ores, further investigations were carried on with respect to the applications of this process, with regard more particularly to the nature of each flotation product, the behaviour of each of the sulphides contained, and the relative losses of precious metal values in the tailing product. In this work several analyses were made, especially of the tailing products, and from these the nature of their content was determined. The following shows the analyses of the two lots of ore upon which the tests were performed:—

Ore	Copper per cent	Nickel per cent	Iron per cent	Sulphur per cent	Silica per cent	Oz. per ton		
						Au.	Plat.	Palad.
No. 3 mine, International Nickel Company	1·25	1·35	23·7	10·7	37·85	0·008	0·014	0·017
Murray mine, British America Nickel Company	0·47	1·17	20·9	11·19	42·15	0·005	0·0067	0·009

From these analyses it may be calculated that the content of each of the above samples is made up as follows:—

Ore	Pentlandite per cent	Chalcopyrite per cent	Pyrrhotite per cent	Gangue per cent	Total sulphides
No. 3 mine, International Nickel Company......................	6·14	3·57	18·56	71·73	28·27
Murray mine, British America Nickel Company..............	4·59	1·34	24·44	69·63	30·37

In previous Summary Reports of the Mines Branch, a complete report of the results obtained from the application of flotation to the ores will be found, embracing percentages of recovery, assays, etc., of each of the flotation products. Upon further examination of the tailings derived from these tests, the content proved to be the following on analysis:—

Tailing from	Copper per cent	Nickel per cent	Iron per cent	Sulphur per cent	Silica per cent	Oz. per ton		
						Gold	Plat.	Palad.
No. 3 mine, International Nickel Company.........	0·05	0·16	8·14	3·23	46·75	0·002	0·0027	0·0038
Murray mine, British America Nickel Company......	0·05	0·21	9·49	5·52	41·30	0·0009	0·0011	0·0026

From these results the composition of the tailings may be assumed as follows:—

Tailing from	Pentlandite per cent	Chalcopyrite per cent	Pyrrhotite per cent	Gangue per cent	Total sulphides
No. 3 mine, International Nickel Company......................	0·73	0·14	7·62	91·51	8·49
Murray mine, British America Nickel Company..............	0·96	0·14	13·12	85·78	14·22

Considering the pyrrhotite composition of the ores, namely, 18·56 per cent and 24·44 per cent respectively, and of the tailings, namely 7·62 per cent and 13·12 per cent respectively, 30 per cent of the pyrrhotite content in the first ore, and 41 per cent of the pyrrhotite content of the second ore, has been eliminated in the tailings. This elimination of almost barren pyrrhotite, together with the gangue minerals, means considerable saving in several directions.

With regard to the particular sulphide minerals with which the platinum group metals are associated nothing definite has been determined. In studying the ratios between these metals and the copper, nickel, iron and sulphur content of the flotation products, there is no definite uniformity of values or proportion between these metals and the chalcopyrite, pentlandite and pyrrhotite content in the ores and flotation products. Whether this should be accepted in support of the opinion that the lower grade ores carry relatively higher values in the platinum group metals is questionable.

(b) THE HYDROMETALLURGICAL TREATMENT OF PYRRHOTITE AND PYRITE ORES SUCH AS THE LEACHING OF PYRRHOTITE AND THE ROASTING AND LEACHING OF PYRITE, WITH SUBSEQUENT PRODUCTION OF IRON BY ELECTRO-DEPOSITION AND THE RECOVERY OF SULPHUR AND OTHER METALS AS BY-PRODUCTS, BY R. J. TRAILL

Introductory.—The process being investigated was described in the Summary Report for 1922, and is known as the Eustis process.

Initial laboratory scale experiments were reported last year, and these have been continued, with the object of securing more complete data regarding the suitability of the process for the treatment of iron sulphide ores, preparatory to a study on a larger scale.

The satisfactory results obtained from these small laboratory tests show that the process gives promise of being commercially applied to the treatment of Canadian pyrrhotite and pyrite ores, and warrants the continuation of the investigation on a larger scale with a complete cycle of operations, including the electro-deposition of the iron. One of the greatest difficulties will be to obtain solutions from the ores, free from impurities detrimental to electro-deposition.

The production of iron by electrolytic methods from the leaching of sulphide ores is attracting increased interest, particularly in England and the United States. The Department has had various enquiries relative to the subject under investigation.

General Procedure—

Leaching Apparatus

An enamel pail of 5 to 6 litres capacity with a glass agitator and an asbestos slate cover was found best for leaching purposes. The solution was heated by means of a steam jacket around the pail.

Solutions for leaching were prepared from ordinary grade ferric chloride containing about 20 per cent iron, and the ferrous chloride, purified from copper and lead, obtained in previous leaching tests.

Treatment of Pyrrhotite

The ore used, in the following tests, was a shipment of pyrrhotite received from the Smith property on the west side of lake Memphremagog, Que., and contained about 80 per cent pyrrhotite and 2·5 per cent chalcopyrite. The general procedure was to treat, by continuous agitation, a certain quantity of the ore, ground to pass 200 mesh, with a solution containing known quantities of ferric and ferrous chlorides, at a temperature from 95° to 98° C. Samples of the solution were drawn off at various intervals to ascertain the extent of the reaction between the ore and the ferric chloride. When reaction was complete, or the set time limit had expired, the leach was cooled, filtered, and the volume noted or made up to the original bulk. The iron content (ferrous and ferric) was ascertained, and the residue dried, weighed and assayed for iron, copper, etc.

Calculation of Extraction

Iron extraction—

$$\underset{\text{ferric chloride}}{2FeCl_3} + \underset{\text{mono sulphide}}{FeS} = \underset{\text{ferrous chloride}}{3FeCl_2} + \underset{\text{sulphur}}{S}$$

From the amount of ferrous iron in grammes found after leaching, subtract the amount of ferrous iron in grammes originally present in the leach solution, plus twice the amount in grammes of copper, zinc or lead dissolved (obtained by subtracting amount in residue from amount in ore charge). The result gives the amount of ferrous iron formed in the reaction, and this divided by 3 gives amount of iron dissolved from the ore charge. Percentage extraction is arrived at by dividing the amount of dissolved iron by amount of iron treated and multiplying by 100.

Copper, zinc and lead, etc., extraction:—

$$\underset{\text{ferric chloride}}{2FeCl_3} + \underset{\text{metal sulphide}}{RS} = \underset{\text{ferrous chloride}}{2FeCl_2} + \underset{\text{metal chloride}}{RCl_2} + \underset{\text{sulphur}}{S}$$

Extractions of these metals are figured by subtracting the amount found in the residue from the amount contained in the ore charge. This amount divided by the amount in ore charge multipled by 100 equals percentage.

Test No. 181

ORE—SMITH PYRRHOTITE

Leaching test No. 5—

Purpose of test.—This test was made to study effect of leaching with a solution containing both ferric and ferrous chlorides, the condition that will occur in regular practice.

Quantities—

3·6 litres solution—82 grammes per litre ferrous iron = 295·2 grammes Fe.
71 " ferric " = 255·6 "
258 grammes ore —49·6 per cent iron = 128·0 "
0·85 " copper = 2·2 " Cu.
1·14 " zinc = 2·94 " Zn.

Excess of ore over theoretical required = 4·0 per cent.

The solution was heated to 70° C., the ore added, and temperature raised and maintained at 95° to 98° C.

Test taken after 30 minutes leaching showed 119 grammes per litre ferrous Fe.
" " 1 hour " " 136 " "
" " 3 hours " " 160 " "
" " 5 " " " 172 " "
" " 7 " " " 178 " "
" " 8 " " " 180 " "

Test stopped at this period, cooled and filtered.

Volume 3·3 litres at 180 grammes per litre ferrous Fe = 594 grammes Fe.
and 7 " ferric Fe. = 23·1 "

Residue weighed 140 grammes, and contained:—

Fe	25·8 per cent	=	36·1 grammes
Cu	0·35 "	=	0·49 "

Result of 8 hours extraction:—

Iron extraction	75·4 per cent
Copper "	77·7 "

This result shows that the presence of ferrous chloride in addition to the ferric chloride does not materially affect the leaching action.

Leaching test No. 8—

Purpose.—In depositing iron from a solution of ferrous chloride, a certain amount of ferric chloride is formed (compare Summary Report, 1922). In the process this ferric chloride, with some ferrous chloride, will be conveyed from the cell to the leaching apparatus. The leaching process on large scale operation may be conducted in two ways. (1) An excess of ore may be maintained in the leaching agitator, so that the ferric solution will be completely reduced in the one operation, and the residue obtained, treated for recovery of sulphur by distillation, or a non-oxidizing roast. The product from this roast would be available for further leaching treatment and recovery of iron. (2) Leaching may be done by a countercurrent extraction method. In this method two or more leaching agitators would be used, designated primary and secondary. The solution coming from the cell would be fed to the primary agitator, where the ore already partly treated in the secondary agitator and fed at intervals to the primary would be acted on, the solution becoming reduced to a certain degree, and finally being completely reduced in the secondary agitator by addition of fresh ore.

To study the feasibility of the latter method the following test was run, leaching fresh ore first with a solution considerably reduced and then leaching the residue with a solution of a composition similar to that coming from the electrolytic cell.

Quantities—

Solution No. 1 contained—
3·6 litres at 132 grammes per litre ferrous Fe. =475·2 grammes Fe.
20 " ferric Fe. = 72·0 "

Solution No. 2 contained—
3·6 litres at 71 grammes per litre ferrous Fe. =255·6 grammes Fe.
63 " ferric Fe. =226·8 "

Ore charge—
300 grammes at 49·6 per cent Fe. =148·8 grammes Fe.
0·85 " Cu. = 2·55 "
1·14 " Zn. = 3·42 "

Excess ore=3·5 per cent.

The ore was treated with solution No. 1 at 95° C. for one hour, after which time reduction was complete. Solution was cooled and filtered.

Volume 3·37 litres at 165 grammes per litre ferrous Fe=556 grammes Fe. The residue without drying was treated with solution No. 2 at 95 to 98° C. and the time test showed:—

After 1 hour leaching	113 grammes per litre ferrous Fe.
" 3 hours "	134 " "
" 6 " "	148 " "
" 8 " "	160 " "
" 10 " "	172 " "
" 12 " "	179 " "

The test was stopped at this period, cooled and filtered.

Volume 3·15 litres at 179 grammes per litre ferrous Fe. =563·8 grammes Fe.
and at 2 " ferric Fe. = 6·3 "

The residue weighed 145 grammes and contained:—

Fe	20 per cent	=29 grammes
Cu	0·48 "	= 0·696 "

Result, 13 hours extraction:—

Fe extraction	82·6 per cent
Cu. "	72·7 "

From this test it is evident that this method of treatment does not affect the percentage extraction. It may be observed, however, that the action is somewhat slower, due possibly to the sulphur that is set free, adhering to or coating particles of the ore.

The question whether method No. 2, or the straight single treatment method No. 1 would be the more satisfactory and economical in general practice will be more carefully investigated when equipment of the continuous circuit type has been installed.

Leaching tests Nos. 9, 10 and 11—

Purpose.—In various leaching tests it has been observed that the copper extraction has varied from 50 to 85 per cent, and an explanation has been sought for this difference in extraction.

In the leaching of chalcopyrite with ferric chloride, it is possible that both cupric and cuprous chloride may be formed. Cuprous chloride, it is claimed, when boiled with freshly precipitated sulphur, deposits the copper as cuprous sulphide, Cu_2S. It is quite possible that this precipitated sulphide of copper is insoluble in the conditions obtaining in the leaching. This explanation is borne out to some extent by the results obtained in the following tests:—

Leaching test No. 9—

Quantities—

4 litres solution at	70 grammes per litre	ferrous Fe.	= 280	grammes Fe.	
	68 "	ferric Fe.	= 272	"	
275 grammes ore at	49·6 per cent	Fe.	= 136·4	"	
	0·85 "	Cu.	= 2·34	"	Cu.
	1·14 "	Zn.	= 3·13	"	Zn.

Excess ore = 4·3 per cent.

The solution was heated to 75° C., the ore added, and the temperature increased and maintained at from 95 to 98° C. Time tests taken showed:—

After 1 hour	135 grammes per litre	ferrous Fe	
" 3 hours	156 "	"	
" 5 "	175 "	"	
" 6 "	180 "	"	(volume about 3·4 litres)

Test was stopped at this period, solution cooled rapidly and filtered immediately, the residue being slightly washed.

Volume 3·75 litres at	163 grammes per litre	ferrous Fe.	= 611·25 grammes Fe.
	13 "	ferric Fe.	= 48·75 "

Residue weighed 135 grammes and contained:—

Fe	21·7 per cent	= 29·3 grammes
Cu	0·25 "	= 0·34 "

Result 6 hours extraction:—

Fe extraction	79·4 per cent
Cu extraction	85·5 "

Leaching test No. 10—

Purpose.—To duplicate test No. 9.

Quantities—

4 litres solution at	69 grammes per litre	ferrous Fe.	= 276	grammes Fe.	
	73 "	ferric Fe.	= 292	"	"
280 grammes ore at	49·6 per cent	Fe.	= 138·8	"	"
	0·85 "	Cu.	= 2·38	"	Cu.
	1·14 "	Zn.	= 3·192	"	Zn.

No excess ore.

The solution was heated to 75° C., the ore added and temperature raised to 95–98° C. Time tests taken showed:—

After 1 hour	133 grammes per litre ferrous Fe.		
" 2 hours	156	" "	
" 3 "	163	" "	
" 4 "	172	" "	
" 5 "	179	" "	
" 6 "	195	" "	(volume about 3·2 litres)

Test stopped at this period, cooled rapidly and filtered immediately washing residue.

Volume 3·75 litres at 166 grammes per litre ferrous Fe. = 622·5 grammes Fe.
9 " ferric Fe. = 33·75 "

Residue weighed 140 grammes and contained:—

Fe	25·2 per cent	= 35·2 grammes
Cu	0·23 "	= 0·32 "

Result 6 hours extraction:—

Fe. extraction	80·7 per cent
Cu. extraction	86·5 "

This result agrees very closely with that obtained in test No. 9 in so far as extraction is concerned.

Leaching test No. 11—

Purpose.—To find copper extraction under conditions that will allow precipitated sulphur to act on any cuprous chloride formed in leaching reaction.

Quantities used—

4 litres at	70 grammes per litre	ferrous Fe.	= 280	grammes	Fe.
at	72 "	ferric Fe.	= 288	"	"
280 grammes ore at	49·6 per cent	Fe.	= 138·8	"	"
	0·85 "	Cu.	= 2·38	"	Cu.
	1·14 "	Zn.	= 3·19	"	Zn.

Excess ore = 0·25 per cent.

Solution heated to 75° C., ore added and temperature raised and maintained from 95 to 98° C. Time tests taken showed:—

After 1 hour	136 grammes per litre ferrous Fe.		
" 2 hours	154	" "	
" 3 "	166	" "	
" 4 "	176	" "	
" 6 "	192	" "	(volume about 3·1 litres)

Test stopped at this period, allowed to cool slowly with agitation and let stand overnight. Solution filtered and residue washed. Extra washing required on account of calcium sulphate crystallizing.

Volume 4 litres at 152 grammes per litre ferrous Fe. = 608 grammes Fe.
at 13 " ferric Fe. = 52 "

Residue weighed 142 grammes and contained:—

Fe	24·1 per cent	= 34·2 grammes
Cu	0·5 "	= 0·71 "

Result 6 hours extraction:—

Fe. extraction	76·3 per cent
Cu. extraction	70·1 "

Comparing the results of extraction on these tests it will be observed that test No. 11 shows a slightly lower iron extraction (due in part possibly to slight oxidation occurring in standing overnight) and it shows a very much lower extraction in copper amounting to 15 or 16 per cent. It would appear therefore that some such action, as suggested under purpose of tests Nos. 9, 10, and 11, has taken place.

COPPER RECOVERY TESTS

The solutions obtained from leaching tests 9 and 10 were treated for recovery of copper by the iron precipitation method, the procedure being as follows:—

The solution, contained in the enamel pail, was heated by means of a steam jacket to 90° C., and steel in the form of short turnings added. The amount of steel used was about 50 per cent in excess of that required to reduce the ferric iron present in the solution, and to precipitate the copper. After treatment the residue was filtered off, dried, and assayed for copper content.

Copper recovery from solution test No. 9.—3·75 litres solution containing 2 grammes copper was treated for 1½ hours at 90°C. with 40 grammes steel turnings. A residue weighing 25 grammes (mostly steel) was obtained containing 1·94 grammes copper. This would be equivalent to 97 per cent of the copper present in solution. Result:—

Copper recovery from solution or	97·0 per cent
Copper recovery from ore	83·0 “

Copper recovery from solution test No. 10.—Solution from test No. 10 was treated in the same manner for 4 hours and showed a

Copper recovery from solution or	98·7 per cent
Copper recovery from ore	85·4 “

No attempt was made to rid the above solutions of the last trace of copper, as it seems reasonable to assume that this would be accomplished in a second treatment with iron or steel.

RECOVERY OF SULPHUR

The process as described in the Summary Report for 1922 provides for the recovery of the sulphur. The reaction between the ore and the ferric chloride is shown as—

$$\underset{\text{ferric chloride}}{2FeCl_3} + \underset{\text{mono sulphide}}{FeS} = \underset{\text{ferrous chloride}}{3FeCl_2} + \underset{\text{sulphur}}{S}$$

the sulphur set free being found in the gangue or insoluble material. It is proposed to recover this sulphur by distillation.

In addition to this, where the single extraction method, as suggested in leaching test No. 8, is used, a product suitable for further leaching may be obtained after the sulphur distillation, thereby recovering the iron and other metals in the residue.

For the purpose of determining the feasibility of this idea the following test was made.

The iron content of these residues runs from 22 to 25 per cent, consisting of unattacked pyrrhotite or pyrite (if present in ore), iron as oxychloride precipitated by hydrolysis, and perhaps iron precipitated by lime or magnesia in the ore.

The copper content ranges from 0·23 to 0·5 per cent, present as unaltered chalcopyrite or chalcocite and possibly precipitated cuprous sulphide. A retort type of container was made from a piece of iron pipe 8 inches long by 2 inches inside diameter, closed at one end with a screw cap, and the other end fitted with a screw cap having a three-eighth-inch diameter pipe as outlet, bent at right angles.

The retort containing the charge was placed in a small electric muffle furnace at 750°C. for 2½ hours. The retort was then removed from the furnace, allowed to cool, and the loss in weight determined. No special precautions were taken to assure the complete condensation of the gaseous sulphur, the outlet simply delivering into a loosely covered receiver and the pipe being heated by means of a Bunsen burner to prevent sulphur condensing in it.

The residues obtained from tests 9, 10 and 11 were thus treated and showed volatile losses respectively as follows:—

No. 9	55 per cent loss
No. 10	53 " "
No. 11	52 " "

From residues Nos. 10 and 11, 42 grammes and 35 grammes of sulphur were recovered, equivalent to about 60 and 52 per cent respectively. Much sulphur was lost because of the unsuitability of the condensing apparatus, and some sulphur would be oxidized to sulphur dioxide (SO_2) in the initial stage of heating. The sulphur recovered appeared to be of quite good grade, judging from its colour and general appearance.

The test satisfactorily demonstrated the possibility of recovering the sulphur from these residues. The material left, after the sulphur distillation, had the following composition:—

	Iron	Copper	Sulphur	SO_3
Residue test No. 9	47·30 per cent	0·54 per cent	28·16 per cent	1·61 per cent
" No. 10	52·70 "	0·52 "	30·90 "	0·15 "
" No. 11	52·00 "	1·00 "		

There was insufficient of this material to make a standard comparative leaching test, but a treatment with ferric chloride proved that this material was as easily leached as the natural pyrrhotite, and that recoveries equivalent to those obtained on the raw ore could be expected.

Test No. 182

TREATMENT OF PYRITE

Pyrite is much less soluble in ferric chloride than is pyrrhotite, but it is readily made soluble by a simple non-oxidizing roast. By this treatment an artificial pyrrhotite is formed, one atom of sulphur being driven off, which, by a suitable apparatus, may be recovered as a by-product.

A test illustrating the roasting of pyrite and subsequent leaching was described in Summary Report for 1922 under test No. 182.

Further tests have been carried out on this method of treatment and are described below. The retort type of container described as used in sulphur recovery from pyrrhotite residue treatment tests Nos. 9, 10 and 11, was used in the roasting process.

Two distinct methods were used and compared in effecting the elimination of the sulphur atom. (A) The first method was in the nature of a reducing roast. The charge, placed in the 8 by 2-inch tube, closed at one end, was subjected to a temperature of 750°C. in an electric muffle furnace. Almost immediately sulphur was set free, which combined with the oxygen in the container forming a reducing gas, namely, sulphur dioxide (SO_2). The gaseous sulphur gradually replaced the SO_2 in the container and burned at the outlet. This feature served as an indication of the progress of the roasting, the flame gradually diminishing as the roasting progressed. When the flame died out the container was withdrawn from the furnace and placed on end, the open end being loosely covered to prevent undue oxidation. It was observed that the product was strongly magnetic. Several samples weighing about 200 grammes were roasted in this manner for a period of 5 to 7 hours and results were fairly concurrent, the magnetic portion (using a horseshoe magnet) running from 88 to 92 per cent, and the total iron 50 to 52 per cent.

(B) The second method employed was by distillation, and is identical with that described in the treatment of residues from pyrrhotite leaching tests Nos. 9, 10 and 11, the container being similar to the one used in the previous method. This container was charged with 200 grammes ore and placed in the muffle furnace at 750°C. It was found that the time necessary to accomplish the elimination of the sulphur atom required from 12 to 14 hours. This time was reduced to about 7 to 9 hours by increasing the temperature to 840°C. No special precautions were taken to condense the gaseous sulphur, a simple air-cooled receiver with a loose cover being employed. This served to recover only 30 to 45 per cent of the sulphur in the ore charge. The sulphur so obtained was of a deep yellow colour, and there was also a very small quantity of dark brown oily fluid which finally solidified with a deep yellow colour.

No satisfactory method of observing the progress of distillation was available so that it was necessary to examine the residue for unaltered pyrite, and if any was found the whole was re-distilled. A number of samples were thus treated, and on examination the residues were non-magnetic, and of a slightly lighter colour than the magnetic material.

To determine the difference between the magnetic and non-magnetic products an analysis was made. The samples were first passed over an Ullrich magnetic separator. From the material obtained by method A, 98 per cent was separated as magnetic, the 2 per cent non-magnetic being discarded. From the material obtained by method B, 14 per cent was separated as magnetic and discarded. Analysis of the magnetic and non-magnetic resulted as follows:—

	Magnetic	Non-magnetic
Total iron	53·50 per cent	51·85 per cent
Ferrous iron	52·25 "	51·85 "
Sulphur	32·90 "	33·40 "
Copper	1·95 "	1·90 "
Lead	0·76 "	0·74 "

It will be observed that the magnetic product contains a small percentage, namely, 1·25 per cent ferric iron, possibly due to magnetite having been formed in the roasting.

Leaching roasted pyrite.—Leaching tests were made on the magnetic and non-magnetic products to determine which product gave the best extractions, and to find how the extractions compared with those of pyrrhotite under the same conditions as in the pyrrhotite leaching tests Nos. 9 and 10.

Leaching test No. 13 on magnetic material—

Quantities:—

4 litres solution at	71	grammes per litre ferrous Fe.	= 284	grammes Fe.	
	73	" ferric Fe.	= 292	"	
265 grammes ore at	52·25	per cent available Fe.	= 138·5	"	
	1·95	" copper	= 5·17	"	Cu.
	0·76	" lead	= 2·0	"	Pb.

No excess ore.

Solution heated to 75°C., ore added, and temperature raised to 95 to 98°C. Time tests taken showed:—

After 2 hours	127 grammes per litre ferrous Fe.
" 4 "	160 " "
" 6 "	178 " "

Test stopped at this period, rapidly cooled and filtered, washing residue and making up to original volume of 4 litres.

Volume 4 litres at	152	grammes per litre ferrous Fe.	= 608	grammes Fe.
	17	" ferric Fe.	= 68	"

Residue weighed 135 grammes and contained:—

Iron	22·2 per cent	= 30	grammes
Copper	0·35 "	= 0·47	"
Lead	Trace		

Result 6 hours extraction:—

Iron extraction	75·0 per cent
Copper extraction	90·9 "
Lead extraction	95 to 100 "

Leaching test No. 13B:—

Purpose.—To duplicate test No. 13.

Quantities:—

4 litres solution at	73	grammes per litre ferrous Fe.	= 292	grammes Fe.	
	70	" ferric Fe.	= 280	"	
265 grammes ore at	52·25	per cent available Fe.	= 138·5	"	
	1·95	" copper	= 5·17	"	Cu.
	0·76	" lead	= 2·00	"	Pb.

Excess ore, 4·0 per cent.

Solution heated to 75°C., ore added, and temperature raised to 95 to 98°C. Time tests taken showed:—

After 2 hours	125 grammes per litre ferrous Fe.
" 4 "	140 " "
" 6 "	163 " "

Test stopped at this period, cooled rapidly and filtered, washing residue and making up to 4 litres.

Volume 4 litres at	157	grammes per litre ferrous Fe.	= 628	grammes Fe.
	15	" ferric Fe.	= 60	"

Residue weighed 134·5 grammes and contained:—

Iron	26·45 per cent	= 35·6 grammes (90 per cent in ferrous condition)
Copper	0·45 "	= 0·6 "
Lead	Trace	

Result of 6 hours extraction:—

Iron extraction	77·7 per cent
Copper "	88·4 "
Lead "	95 to 100 "

Leaching test No. 14—Non-magnetic material—

Purpose.—Comparison with test No. 13.

Quantities:—

4 litres solution at	73	grammes per litre ferrous Fe.	= 292	grammes Fe.	
	70	" ferric Fe.	= 280	"	
270 grammes ore at	51·85	per cent iron	= 140	"	
	1·90	" copper	= 5·13	"	Cu.
	0·74	" lead	= 2·00	"	Pb.

Excess ore, 5 per cent.

Solution heated to 75°C., ore added and temperature raised to 95 to 98°C. Time tests taken showed:—

After 2 hours	134 grammes per litre ferrous Fe.
" 4 "	156 " "
" 5 "	186 " "

Test stopped at this period, rapidly cooled and filtered, washing residue and making up to 4 litres.

Volume 4 litres at 168 grammes per litre ferrous Fe. = 672 grammes Fe.
3 " ferric Fe. = 12 "

Residue weighed 142·1 grammes and contained:—

Iron	21·15 per cent	= 30 grammes (88 per cent ferrous)
Copper	1·23 "	= 1·75 "
Lead	Trace	

Result 6 hours extraction:—

Iron extraction	88 per cent
Copper "	66 "
Lead "	95 to 100 "

Leaching test No. 14B—

Purpose.—Duplicate of 14.

Quantities:—

4 litres solution at	72	grammes per litre ferrous Fe.	= 288	grammes Fe.	
	71	" ferric Fe.	= 284	"	
270 grammes ore at	51·85	per cent iron	= 140	"	
	1·90	" copper	= 5·13	"	Cu.
	0·74	" lead	= 2·00	"	Pb.

Excess ore, 3·6 per cent.

Solution heated to 75°C., ore added and temperature raised to 95 to 98°C. Time tests taken showed:—

After 2 hours	143 grammes per litre ferrous Fe.
" 4 "	186 " "
" 6 "	196 " "

Test stopped at this period, rapidly cooled and filtered, washing residue and making up to 4 litres.

Volume 4 litres at 166 grammes per litre ferrous Fe. = 664 grammes Fe.
8 " ferric Fe. = 32 "

Residue weighed 136·6 grammes and contained:—

Iron	18·3 per cent=25 grammes (91 per cent ferrous)
Copper	0·75 " = 1·02 "
Lead	Trace

Result 6 hours extraction:—

Iron extraction	86·6 per cent
Copper "	80·0 "
Lead "	95 to 100 "

Comparing these results it would appear that the magnetic product gives the higher copper extraction and the non-magnetic gives the higher iron extraction. No explanation can be given for the difference in copper extraction on duplicate tests Nos. 14 and 14B. The iron and copper extractions obtained on the magnetic material compare very closely with those obtained on the natural pyrrhotite, and the iron extraction from non-magnetic material is slightly better than from the natural pyrrhotite.

Copper Recovery

95 to 98 per cent of the copper was recovered from the solutions by precipitating on steel turnings, but no lead was found to be precipitated with the copper.

Test on Complex Sulphide Ore

In an endeavour to obtain some idea as to how other metallic sulphides, and particularly gold and silver, would be acted upon in this process, a preliminary test was run on a complex sulphide ore. It was thought advisable to use a sample with high gold and silver content, as more accurate data would probably be obtained than by using low-grade ore.

The sample, selected for this test, was from a concentrate obtained from the Woodsworth ore which was shipped to this laboratory for ore dressing tests. An analysis of the concentrate is as follows:—

Gold	20·2 ozs. per ton
Silver	15·68 "
Copper	0·92 per cent
Lead	2·50 "
Zinc	15·20 "
Iron	28·90 "
Sulphur	39·28 "
Silica	8·00 "
Arsenic	1·57 "

The iron was present mostly as pyrite, and the sample was therefore roasted in a non-oxidizing atmosphere as outlined in pyrite roasting treatment A, the temperature not exceeding 750°C. The roasted product had the following analysis:—

Gold	22·2 ozs. per ton
Silver	17·2 "
Copper	1·08 per cent
Lead	2·94 "
Zinc	17·70 "
Iron	34·0 "
Sulphur	29·20 "
Arsenic	0·57 "

The loss in weight was 14·6 per cent.

From these figures it would appear that the volatiles consisted only of sulphur and arsenic, 70 per cent of the latter being volatilized. Theoretically, the product contains 55·7 per cent of metallics capable of reacting with ferric chloride (excluding arsenic).

Leaching test.—A leaching test was made on this roasted material, observing the regular procedure.

Quantities:—

3·3 litres solution at	30	grammes per litre ferrous Fe.	= 99	grammes Fe.
	71	" ferric Fe.	=234·3	"
210 grammes ore at	55·7	per cent reacting metals	=117	"
	34	" iron	= 71·4	"
	17·7	" zinc	= 37·17	grammes Zn.
	2·94	" lead	= 6·17	" Pb.
	1·08	" copper	= 2·27	" Cu.

No excess ore.

Solution heated to 75°C., ore added and temperature raised to 95 to 98°C.

Time tests taken showed:—

After 2 hours	91 grammes per litre ferrous Fe.
" 4 "	134 " "
" 5 "	143 " "
" 6 "	148 " "

Test was stopped at this period, cooled and filtered, and residue washed.

Volume 3 litres at 132·33 grammes per litre ferrous Fe. =397 grammes Fe. and trace ferric Fe.

Residue weighed 96 grammes and contained:—

Iron	6·46 per cent	=6·2	grammes
Copper	0·35 "	=0·336	"
Lead	Trace		
Zinc	1·97 "	=1·89	"
Gold	50·79 ozs. per ton	=0·1672	"
Silver	6·88 "	=0·02265	"

Result 6 hours extraction:—

Iron extraction	98·5 per cent
Copper "	85·2 "
Lead "	99·0 "
Zinc "	94·9 "
Gold "	None
Silver "	81·7 "

Arsenic was present in both residue and solution, but was not determined.

The extraction is remarkable in that it is about 98 per cent of the theoretical. The iron extraction is outstanding and it is perhaps possible that some of the other metals act in the manner of catalysts and increase the extraction. The gold is left almost free from other metals and from the residue it should easily be recovered.

RECOVERY OF METALS

The recovery of these metals from solution presents an interesting problem, upon which the success of this process will very much depend. It is essential that the method employed should not introduce any substances that would interfere in any way with the subsequent recovery of the iron by electro-deposition. Scrap iron or steel, hydrogen sulphide, and sodium sulphide are probably the most commonly used precipitants. These precipitants were tried with the following results:—

Recovery using steel turnings

A quantity of the solution from the above test was treated with steel turnings for 2½ hours with stirring, at a temperature of 75°C., the precipitate filtered off and assayed. The following recoveries were made, based on the amount of the metal in solution:—

Copper	85·7 per cent
Lead	84·9 "
Silver	77·8 "
Zinc	None

It is quite within reason to expect that a second treatment would have resulted in a more complete recovery. The recoveries obtained above would represent a total recovery of these metals from the ore of:—

Copper	73·0 per cent
Lead	84·9 "
Silver	63·6 "

The silver recovery would of course be increased by the amount of silver recovered with the gold values in the subsequent treatment of the residue.

Recovery using sodium sulphide

The amount of sodium sulphide required to precipitate the reacting metals in solution, plus a small excess, was added to the solution at about 70°C. and treated with occasional shaking for 30 minutes. The precipitate obtained was filtered off and assayed, showing the following recoveries of dissolved metals:—

Copper	64·5 per cent
Lead	25·0 "
Silver	48·5 "
Zinc	3·0 "

The filtrate was treated with a further quantity of sodium sulphide, using half as much as in the first treatment, in the same manner and time. This second treatment resulted in additional recoveries that brought the total recoveries up to:—

Copper	85·2 per cent
Lead	66·9 "
Silver	79·7 "
Zinc	10·5 "
Iron precipitated	1·7 "

A third treatment in the same manner showed a further increase in metals precipitated, resulting in total recoveries from solution of:—

Copper	88·5 per cent
Lead	88·0 "
Silver	87·0 "
Zinc	12·0 "

representing total recoveries from the ore of:—

Copper	75·4 per cent
Lead	88·0 "
Silver	71·0 "
Zinc	11·4 "

Recoveries using hydrogen sulphide

This test was conducted in a large flask, using hydrogen sulphide as precipitant under a slight pressure for 5 minutes. Precipitate was filtered off and an assay showed recoveries from solution of:—

Copper	90·0 per cent
Lead	8·6 "

Amount taken for this test, namely 200 c.c., was too small to permit accurate silver determinations, and therefore no recovery in silver could be figured. A second treatment with hydrogen sulphide for 20 minutes showed an increase that gave a total recovery from solution of:—

Copper	92·0 per cent
Lead	91·3 "

A third treatment with hydrogen sulphide for 20 minutes brought the total recovery from solution to:—

Copper	96·2 per cent
Lead	94·1 "

representing a total recovery from the ore of:—

Copper	82·0 per cent
Lead	94·1 "

This test clearly demonstrates the suitability of this process for the treatment of this type of ore, and it is evident that quite simple methods can be employed in the recovery of metals contained in the leach from this ore.

CONCLUSIONS

The tests were carried out on a laboratory experiment scale, anticipating, as far as possible, problems and conditions that would arise in actual practice, before attempting to experiment on a larger scale.

These tests have demonstrated that on an average 80 per cent of the iron and copper can be extracted with ferric chloride from pyrrhotite ores. Pyrite, with a preliminary treatment, has also been shown to be amenable to this treatment, with almost similar recoveries. Lead, zinc and silver are also easily extracted, gold is separated in a more concentrated condition, and methods for recovery of these metals have been described, with the exception of zinc, for which no satisfactory method of separation has yet been found.

The separation of iron has, so far, not been attempted, as no suitable apparatus is at hand. However, a special laboratory is being equipped with the necessary apparatus for electro-deposition of metals, and also with leaching and precipitating apparatus, so that a more complete study of the application of this process to Canadian ores can be made under conditions more closely approximating those of actual practice.

IV

SELECTIVE FLOTATION AS APPLIED TO CANADIAN ORES

C. S. Parsons

SELECTIVE FLOTATION PROBLEMS

For the purpose of this article, selective flotation is taken to mean the separation by flotation of one mineral from another, where two or more flotative minerals are present in an ore. The separations generally required are on complex ores, consisting of combinations of lead, zinc and iron sulphides, copper, zinc and iron sulphides, molybdenum and iron sulphides, antimony and arsenical iron sulphides, graphite and iron sulphides, and other similar combinations.

The logical way to separate these minerals by flotation is to float first the mineral which has the highest floating properties. A list of the more common minerals met with in selective flotation, in order of the ease with which they are floated, from the writer's own experience is: graphite, molybdenite, stibnite, galena, chalcopyrite, sphalerite, pentlandite, pyrite, arsenides and antimonides, etc., pyrrhotite, arsenical pyrite. For instance, in the selective flotation of a lead-zinc sulphide ore, the lead should be floated first, as it is the more easily floated mineral of the two, in a copper-zinc ore, the copper should be floated first, and so on.

OILS USED FOR SELECTIVE FLOTATION*

During the early stages in the development of the flotation process much more weight was given to the function of the oil than is given at present, as it was supposed that every ore required special oils. Dozens of different oils and oil mixtures were tested on each ore. Some mills were operated in which oil mixtures containing four or five different oils were used, whereas in all probability only one or two were essential. The Department has over 300 samples of oils, but of this number 90 per cent are never used in the regular test work. It has been found that satisfactory results can be obtained on the majority of ores from a group containing a few varieties. A list of the oils in this group is given, together with the selective properties of each as found by actual experience.

Reagents used for the flotation of lead sulphides in the presence of zinc and iron sulphides:—

Mixture of coal tar and coal tar creosote.
Cresylic acid.
Z cake or thiocarbanilide.
Steam distilled pine oil.
Light hardwood creosote oil.

Reagents used for the flotation of zinc sulphides from iron sulphide after lead or copper sulphides have been removed:—

KK oil No. 2—Southwestern Engineering Co.
No. 634 creosote oil—Barrett Co.
Sodium creosote from hardwood.
YZ mixture, or xylidine and thiocarbanilide.

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*Oil, in flotation terminology, is a generic term used to designate the organic substance that is used to produce a froth or effect selection of metalliferous minerals (Taggart).

No. 34 gravity fuel oil—G. W. Oil Co.
TT mixture.
Water-gas tar—Barretts.

Reagents used for the flotation of copper sulphides from zinc and iron sulphides:—

Mixture of coal tar and coal tar creosote.
XY mixture or xylidine and alpha-naphthylamine.
Neutral fractions of hardwood oils.

Reagents used for the flotation of copper sulphides from iron sulphides:—

Barretts No. 634.
XY mixture, TT mixture, Thiofizzan, ZY mixture — Reagents of the General Engineering Co.
Sodium resinate—use very limited.
Fuel oils (paraffin base).

Reagents used for the flotation of copper-nickel sulphides from iron sulphides:—

Thiofizzan.
TT mixture.
XY mixture.
Fuel oil (paraffin base).
Coal tar and coal tar creosote mixtures.

Reagents used for the flotation of molybdenite and graphite from iron sulphides:—

Kerosene oil.

Frothing reagents used with the above:—

Steam distilled pine oil.
Fumol—manufactured by Canadian Electro Products Co.
Ketone oil—manufactured by the Standard Chemical Co.

ADDITION REAGENTS

The use of addition reagents is as important as the use of oils for selective flotation. If the conditions in the pulp are correct there is generally found to be considerable latitude with regard to the kind of oil used. In the writer's experience, the chief function of the more common addition reagents used in practice for selective flotation are:—

Lime.—The use of lime for the selective flotation of certain minerals was first brought to the attention of the Department in 1917, when operating a customs plant for the concentration of molybdenite ores from all points in the Dominion. A few carloads of very badly oxidized ore were received and trouble was encountered in floating the molybdenite. It was found that large quantities of soluble salts were contaminating the pulp. Lime was added to the ball mill with the object of preventing these salts from going into solution, and it was found that the lime, besides eliminating the trouble with soluble salts, also prevented the iron sulphides from floating, resulting in a much higher grade concentrate. In this connexion a very interesting phenomenon was observed. When the pulp was contaminated with soluble sulphates, as much as 100 pounds per ton of kerosene oil could be added without causing any raw oil effect on the cells, a voluminous but absolutely barren froth being obtained. On ordinary clean ores the addition of one pound of kerosene oil would produce a raw oil effect on the cells and kill the froth. The only explanation that could be advanced was that the presence of soluble sulphates in the pulp caused the oil to form a stable emulsion. It was found that if the soluble salts were subsequently washed from the pulp, the emulsion was broken down and the oil appeared on the surface of the cells.

When lime is used it should always be added to the ball mill feed, as long contact with the pulp is essential owing to the time required for the lime to go into solution. Lime is used in the selective flotation of copper sulphide ores to prevent the iron sulphides, also the zinc sulphides, if present, from floating. In the selective flotation of lead-zinc sulphide ores, it prevents the zinc sulphide, as well as the iron sulphides, from floating. In the flotation of antimony ores it has been found to absolutely prevent stibnite from floating. It is used in the flotation of molybdenite and graphite, where iron sulphides are present, for the same purpose. The quantity added varies from one to twenty pounds per ton of ore.

Soda ash.—The addition of soda ash to the flotation pulp will increase the flotation of zinc sulphides, copper and nickel sulphides and stibnite, and has a tendency to prevent iron sulphides from floating. The quantity used varies from one to ten pounds per ton of ore.

Sodium sulphite.—This reagent is used in the selective flotation of lead-zinc sulphide ores. It prevents zinc sulphide from floating but has a slight tendency to float iron sulphides, so that if these are present in any quantities, lime must be used also.

Copper sulphate.—The addition of one to two pounds of copper sulphate per ton of ore, when an alkaline pulp is used, will generally assist the flotation of zinc sulphide.

CONTROL OF FLOTATION PULP

The control of the pulp in a flotation plant is very important and should be given as careful attention as in a cyanide plant. Commencing at the discharge of the primary, wet-grinding mills, control samples should be taken of each succeeding step in the operations at various intervals of time. The correct condition of the pulp for flotation must be obtained before it enters the flotation cells. Characteristics displayed by the pulp in the ball mill discharge and in the classifier circuits will in most cases indicate to the operator how the cells are working and whether the pulp is in the right condition for flotation.

Density control.—Careful control must be maintained of the density of flotation pulps. The densities should be kept as uniform as possible and only vary between very narrow limits. The reason for this is obvious, as any variation in the dilution of the pulp will change the concentration of the modifying reagents used to increase the selective flotation. It is often found necessary to sacrifice efficiency in the grinding circuits to obtain efficiency in the flotation circuit. For instance, in order to obtain a low tailing in the selective flotation of a copper-iron sulphide ore, it has been necessary to maintain a pulp density in the flotation cells of 1 : 2·5. Dewatering of the flotation feed could not be practised because a definite protective alkalinity with lime must be maintained in the cells to prevent the iron sulphides from floating.

Temperature control.—The control of temperature in the grinding mills is very important in the case of easily oxidized ores, such as the copper-nickel ores of the Sudbury district and copper ores where the chalcopyrite is associated with large quantities of iron sulphides. Trouble in this connexion is seldom met with in conducting the small scale tests. The temperature can be controlled by pulp dilution, but to do so generally means sacrificing the grinding efficiency.

The control of temperatures in the flotation circuits for selective flotation is practised in certain plants where large tonnages are handled to raise the grade of the concentrate being produced. It introduces an extra item of cost and should only be done when absolutely essential, and where a profit can be shown as against flotation at ordinary temperatures.

Control of soluble salts.—In conducting selective flotation tests, the greatest care should be taken to obtain samples which will truly represent the ore to be concentrated. The samples should be of freshly mined ore. In an ore consisting of a large percentage of sulphides, soluble salts form very rapidly. The writer has found that the lead and nickel sulphides in samples of lead and copper-nickel ores, crushed to 10 mesh, will oxidize so rapidly that, in as short a time as ten days, the flotation properties of the minerals will be seriously affected. All ores contain soluble salts which are generally sulphates of iron, aluminium, calcium and magnesium. These salts play a very important part in selective flotation and no definite rules can be given as to how they will act. Large amounts of them will interfere with and even prevent flotation. Small amounts either interfere or improve the selective flotation of one mineral from another. In conducting selective flotation tests, the amount and nature of the salts should be determined for every ore. The subject of overcoming or taking advantage of their effect is too large a one to consider in detail in this article. Further reference is made to them in dealing with the type of testing apparatus.

Control of modifying reagents.—Samples of the water from the flotation cell tailing should be taken at regular intervals and analysed for protective amounts of modifying reagents.

EXAMPLES OF SELECTIVE FLOTATION PROBLEMS

A copper-zinc-iron sulphide ore.—This particular ore was a complex association of copper, zinc and iron sulphides on which grinding to 200 mesh was necessary to free the minerals. It contained chalcopyrite 7 per cent, zinc blende 7 per cent, iron sulphides 73 per cent, and gangue minerals 13 per cent.

It was found that a high-grade copper concentrate could be obtained by using lime to prevent the pyrite and zinc blende from floating. The pulp, after flotation of the chalcopyrite, was dewatered to eliminate the lime. Fresh water and soda ash were added for the flotation of the zinc blende. The governing condition for the separation was in the control of the lime added for the selective flotation of the chalcopyrite. It was found that lime, in excess of the requirements to produce a high-grade copper concentrate, was necessary to prevent the pyrite from floating with the zinc blende. The addition of lime to the zinc heads did not produce the same effect, nor did running the pulp direct from the copper cell without dewatering. In both cases so much lime was present that the zinc blende could not be floated. The lime had to be added to the grinding circuit before flotation of the chalcopyrite. Investigating the reason for this it was found that, apparently, a skin formed around the grains of pyrite resulting in a greater concentration of lime around the pyrite than in the remainder of the pulp. The concentration of the lime around the grains would vary according to the amount of lime in the pulp, so that it was necessary, in the first place, to add enough lime to obtain a concentration in the film that would maintain the wetting effect on the pyrite after dewatering for the selective flotation of the zinc blende.

A copper-iron sulphide ore.—This was a heavy sulphide ore of copper containing 3 per cent copper as chalcopyrite, 39 per cent iron as pyrite, and 8 per cent insoluble. Grinding to 100 mesh was necessary to free the minerals. An alkaline pulp was maintained by the addition of 10 pounds of lime to the ball mill. The most important point was found to be the control of the pulp density in the flotation cells. A low tailing could only be obtained when a density of 1 : 2·5 to 1 : 3 was maintained in the cells. Under these conditions a concentrate containing 22 per cent copper with a recovery of 95 per cent of the copper values was obtained from this ore without the least difficulty.

Lead-zinc-iron sulphide ores.—The following method has been proved to have the most general application. Lime and sodium sulphite are added and ground in contact with the ore in the ball mills. The lime is added to keep the iron down, and the sodium sulphite to keep the zinc from floating. After the flotation of the lead, the tailing pulp from the lead cells is dewatered, eliminating the bulk of the lime and sodium sulphite. The pulp is then made up to the required density with fresh water, and soda ash and copper sulphate added for the flotation of the zinc. This method was used to obtain the separations given in examples Nos. 1 and 2 in the accompanying tables. The method used by the Consolidated Mining and Smelting Co., at Trail, to concentrate the ore from the Sullivan mine has also been used with success. An ore containing 1·96 per cent lead, 5·52 per cent zinc and 7·42 per cent iron, all as sulphides, was treated by this method. Example No. 3 in the accompanying tables gives the results of a tonnage test on the ore. In this particular case the ore was ground to 65 mesh to free the minerals. The oils used on the Sullivan ore were found to be unsatisfactory, but no change in the addition reagents was necessary. Soda ash at the rate of 5 pounds per ton was fed to the ball mill together with a mixture of coal tar and coal tar creosote. A little cresylic acid was added to the lead cells together with 0·1 pound per ton of sodium cyanide. The tailing from the lead cell went to a mixer where the Barretts No. 634 oil, TT mixture, and 1 pound copper sulphate per ton were added for the flotation of the zinc in the zinc cells.

A zinc-iron sulphide ore.—The selective flotation of the zinc can generally be obtained by the use of an alkaline soda ash pulp, but in some cases the soda ash will not hold down the iron and it is necessary to use lime. When lime is used it should be added in the ball mills and the pulp dewatered before the soda ash is added. Copper sulphate will often increase the recovery of zinc.

The oils used in the above methods of separation have been purposely omitted. The reason for this is that while the addition reagents have been found to give consistent results on a variety of ores, the oils have not.

EXAMPLES OF SELECTIVE FLOTATION

The following tables contain examples of separations obtained in the laboratory of the Department of Mines by selective flotation:—

Separation of copper-zinc-iron sulphide ore:—

Approximate analysis: Chalcopyrite 7 per cent, zinc blende 7 per cent, iron sulphides 73 per cent, gangue minerals 13 per cent.

Product	Analysis				Distribution of values		
	Cu. per cent	Zn. per cent	Fe. per cent	Au. oz.	Cu. per cent	Zn. per cent	Au. per cent
Copper concentrate	16·7	4·4	24·3	0·10	87·9	10·6	18·2
Copper middling	1·8	8·7	28·3	0·10	1·89	4·17	3·6
Zinc concentrate	0·45	40·7	17·9	0·11	1·27	52·52	10·8
Zinc middling	0·5	8·9	33·7	0·07	1·34	10·90	6·6
Tailing	0·3	1·9	35·4	0·07	7·57	21·84	60·8

Separation of copper-iron sulphide ore:—

Analysis: Copper 2·98 per cent, Fe. 39·0 per cent, insoluble 8·0 per cent.

Product	Analysis Cu. per cent	Recovery of copper per cent
Copper concentrate	20·20	93·3
Tailing	0·12	

Separation of lead-zinc-iron ores:—

Analysis: Lead 17·6 per cent, zinc 28·88 per cent, silver 13·45 oz. per ton.

EXAMPLE No. 1

Product	Analysis			Distribution of values		
	Pb. per cent	Zn. per cent	Ag. oz.	Pb.	Zn.	Ag.
Lead concentrate	60·0	8·5	42·0	76·2	6·5	69·6
Zinc concentrate	7·0	45·9	6·6	21·6	86·5	26·7
Tailing	1·6	8·5	2·1	2·1	6·8	3·6

EXAMPLE No. 2

Analysis: Lead 11·40 per cent, zinc 11·40 per cent, iron 32·25 per cent.

Product	Analysis		Distribution of values	
	Pb. per cent	Zn. per cent	Pb.	Zn.
Lead concentrate	52·60	6·10	63·0	7·5
Lead middling	21·9	10·0	9·4	4·3
Zinc concentrate	8·3	37·4	17·1	77·3
Zinc middling	9·8	11·1	2·1	2·4
Tailing	1·67	1·7	8·4	8·5

EXAMPLE No. 3

Analysis: Lead 1·96 per cent, zinc 5·52 per cent, iron 7·42 per cent, gold 0·02 oz. per ton silver 3·23 oz. per ton.

Product	Analysis						Distribution of values	
	Pb. per cent	Zn. per cent	Cu. per cent	Insol. per cent	Au. oz.	Ag. oz.	Pb.	Zn.
Lead concentrate	51·09	4·17	2·60	16·13	0·30	66·20	89·4	
Zinc concentrate	1·09	46·19	....	....	....	....	....	94·8
Tailing	0·15	0·32						

Separation of copper-nickel pyrrhotite ore:—

Analysis: Copper 1·56 per cent, nickel 1·24 per cent.

Product	Analysis			Distribution of values	
	Cu. per cent	Ni. per cent	Cu.+Ni. per cent	Cu.	Ni.
Concentrate	12·5	9·6	22·10	97·2	85·6
Tailing	0·05	0·20	....	2·8	14·4

SMALL SCALE TESTING APPARATUS FOR SELECTIVE FLOTATION

The majority of testing laboratories are equipped with the small batch type of laboratory apparatus. The writer has found that the results obtained in this type of machine are, in many cases, unreliable and cannot be obtained in practice. This particularly applies to selective flotation tests and to tests on ores containing appreciable amounts of soluble salts. The batch machine is very satisfactory for preliminary work, but a continuous feed type of small testing unit with its own grinding circuit should be used to check the results. In this latter type of testing apparatus, the reagents can be added continuously with the feed, as in practice, and not all at once which is generally the case in conducting batch tests. The batch method will usually give a better separation than can be obtained in practice. If a small scale flotation machine, with a continuous grinding circuit, is not available, the following procedure should be tried out to check the batch methods of adding reagents. During the grinding of the sample the ball mill should be stopped, opened up, and portions of the total amount of reagents to be used added from time to time instead of all at once, and in this way conditions more nearly approaching mill practice are obtained. In conducting flotation tests, the ore should be prepared by wet grinding. Dry ore should never be fed to the flotation machine unless dry crushing is to be adopted in the milling practice.

CONCLUSIONS

The application of selective flotation has increased rapidly in recent years. Although it is a new field in ore-dressing, the scope of its usefulness is gradually being enlarged. It will play a very important part in the economic treatment of complex ores and will permit the commercial development of many ore-bodies which have hitherto been thought unprofitable and will prove to be an important factor in the development of our mineral resources.

The purpose of this paper is to introduce the subject of selective flotation in the hope that other investigators will publish the results of their work and the methods used both in the laboratory and in practice. As yet, very little information has been published. It would be to the mutual advantage and benefit of all concerned if it were more openly and freely discussed.

V

EXPERIMENTAL TESTS ON THE BENEFICIATION OF CANADIAN IRON ORES

W. B. Timm

The Ore Dressing and Metallurgical Laboratories of the Mines Branch, Department of Mines, Ottawa, were first equipped in 1909 for experimental work on iron ores, and previous to the war a large amount of test work was conducted on the beneficiation of Canadian iron ores. During the war the laboratories were used almost entirely for work on war minerals, and since the war for experimental test and research on non-ferrous ores. The progress made on the beneficiation of iron ores has been closely watched, and the laboratory equipment for this work kept up to date by the purchase of improved apparatus, so that the pre-war work can be supplemented by further investigations, when, by prospecting and development work, representative ore is made available for test.

Experimental work has been conducted on shipments of a few hundred pounds to forty-five tons, from various localities, such as the Bristol mines, Bristol, Que., the Bathurst mine, Bathurst, N.B., the Wilbur mine, Wilbur, Ont., the Culham mine, Lanark, Ont., Nictaux-Torbrook mine, Torbrook, N.S., Goulais river, Algoma district, Ont., Natashkwan river, lower St. Lawrence, Que., the Bessemer mine, Bessemer, Ont., the Childs mine, Bessemer, Ont., Riviere des Rapids, Saguenay, Que., Groundhog river, Sudbury dist., Ont., Lavant mine, Lanark, Ont., Kaministikwia, Ont., and Clarendon, Ont. Results of these tests have been fully described in the Summary Reports of the Mines Branch.

The following is a brief summary of the more important results of the experimental tests on the majority of the above ores:—

Wilbur Mine Ore—Shipment 10 Tons

Location of property.—Lots 3 and 4, con. IX, XII, XIII, Lavant tp., Lanark co., Ont.

Character of ore.—Moderately fine crystalline magnetite, the gangue consisting for the most part of quartz, calcite, and chlorite in seams and stringers throughout the ore. Other gangue minerals present in smaller amounts are hornblende and muscovite. Both sulphur and phosphorus are low.

Results of preliminary cobbing tests.—These tests showed the magnetite fairly free at about 40 mesh and a 60 per cent iron concentrate with a low tailing was made at this mesh.

Results of wet magnetic separation tests.—Screen analysis of ball mill discharge was—on 100 mesh, 24 per cent; on 200 mesh, 67 per cent;

through 200 mesh, 33 per cent. Analyses of crude ore, concentrate, and tailing, were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron	48·5	66·10	7·2
Insoluble residue	13·08	3·30	
Sulphur	0·105	0·028	
Phosphorus	0·011	0·004	
Lime	3·00	0·30	
Magnesia	6·40	1·40	

Ratio of concentration	1 : 1·43
Recovery	95·3 per cent

Wilbur Mine Waste Dump—Shipment 5 Tons

Character of ore.—Similar to run-of-mine. Major portion of gangue granitic gneiss; calcite and chlorite present in about the same proportion as in run-of-mine.

Results of wet magnetic separation tests.—Screen analysis of ball mill discharge was—on 100 mesh, 32 per cent; on 200 mesh, 63·5 per cent; through 200 mesh, 36·5 per cent. Analyses of crude ore, concentrate, and tailing, were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron	38·2	64·6	5·0
Insoluble residue	22·18	6·11	
Sulphur	0·101	0·023	
Phosphorus	0·011	0·006	
Lime	3·20	0·14	
Magnesia	7·0	1·64	

Ratio of concentration	1 : 1·8
Recovery	94·5 per cent

General conclusions.—A high-grade iron concentrate can be made from both the run-of-mine and the lower grade waste ore at fairly coarse grinding. The concentrate would have to be sintered, but the resultant product would be very satisfactory for blast furnace use.

Robertsville Mine Waste Dump—Shipment 5 Tons

Location of property.—Lots 3 and 4, con. IX, Palmerston tp., Frontenac co., Ont.

Character of ore.—Fairly coarse crystalline magnetite in a gangue of diorite country rock. Black hornblende and pink calcite with smaller amounts of white quartz is also associated with the magnetite. Sulphur and phosphorus present in small amounts.

Results of wet magnetic separation tests.—Analyses of crude ore, concentrate and tailing:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron	28·00	70·5	3·6
Insoluble residue	50·70	3·1	
Sulphur	0·004	0·00	
Phosphorus	0·028	0·014	
Lime	3·30	0·00	
Magnesia	1·90	0·06	

Ratio of concentration .... 1 : 2·74
Recovery .... 91·8 per cent

General conclusions.—A high-grade iron concentrate was made with a high recovery. The concentrate would have to be sintered, but the resultant product would be very satisfactory for blast furnace use.

Culham Mine Ore—Shipment 3 Tons

Location of property.—N. ½ lot 21, con. VII, Bagot tp., Renfrew co., Ont.

Character of ore.—Moderately fine-grained crystalline magnetite, the gangue being schistose material, calcite and iron pyrites. Both sulphur and phosphorus are present in objectionable amounts.

Results of wet magnetic separation tests.—Analyses of the crude ore, concentrate and tailing were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron	47·7	67·0	6·6
Insoluble residue	9·3	2·5	
Sulphur	1·65	0·357	
Phosphorus	0·179	0·074	
Lime	4·20	0·10	
Magnesia	0·66	0·05	

Ratio of concentration .... 1 : 1·47
Recovery .... 95·5 per cent

General conclusions.—A high-grade iron concentrate was made with a high recovery. The phosphorus content is above the Bessemer limit, but low enough for mixing purposes. Sintering would be required, and this would reduce the sulphur content and make the resultant product satisfactory for blast furnace use.

Bathurst Mine Ore—Shipments 15 and 10 Tons

Location of property.—Lot 12, range XVII, Bathurst tp., Gloucester co., N.B.

Character of ore.—A compact cryptocrystalline mixture of hematite, magnetite and quartz. The ore possesses a slightly laminated structure, although an alternate banding of hematite, magnetite and quartz is not apparent to the naked eye. Hematite and magnetite appear to be intimately associated in about equal proportions, the colour and streak of the ore varying from red to black, according to the proportion of ferric iron.

Results of wet magnetic separation tests.—Analyses of crude ore, concentrate and tailing were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron	48·7	60·3	38·2
Ferrous oxide	18·1	23·8	9·6
Ferric oxide	49·4	59·7	43·8
Insoluble residue	16·9	7·7	
Sulphur	0·127	0·046	
Phosphorus	0·760	0·350	

Ratio of concentration	1 : 2·1
Recovery	58·8 per cent

General conclusions.—A good grade of concentrate was produced, but with a low recovery of the iron. The loss in iron was due to the hematite in the crude reporting in the tailing. This method of concentration is not adaptable to the Bathurst ore. Magnetic cobbing tests and jig tests were made on carefully sized ore, but by both these methods it was found that the grade could not be raised any appreciable extent without a high loss of the iron content. It would seem that concentration methods beyond a rough cobbing to remove any waste material broken with the ore are not applicable to Bathurst ore.

Torbrook Mine Ore, "Hematite Vein"—Shipment 15 Tons

Location of property.—Nictaux-Torbrook area, Annapolis co., N.S.

Character of ore.—The Torbrook hematite vein is composed of hard compact ore of finely crystalline magnetite and hematite of the specular variety. The gangue is predominantly siliceous, although lime and magnesia are present in small amounts. Hematite and magnetite exist in about the proportion of 1 to 2·6 respectively.

Results of wet magnetic separation tests.—Analyses of the crude ore, concentrate and tailing were:—

	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron	47·2	61·0	33·0
Ferrous oxide	14·9	21·6	9·4
Ferric oxide	50·8	63·7	36·7
Insoluble residue	21·7	9·9	
Sulphur	0·011	0·005	
Phosphorus	1·41	0·54	
Magnesia	0·86	0·28	

Ratio of concentration	1 : 1·97
Recovery	65·6 per cent

Torbrook Mine Ore, "Shell Vein"—Shipment 15 Tons

Character of ore.—Very similar in physical characteristics to the hematite vein, although the former is more magnetic. The ore is siliceous, but contains considerably more lime than the hematite vein.

Results of wet magnetic separation tests.—Analyses of the crude ore, concentrate and tailing were:—

—	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron	42·5	60·5	22·8
Ferrous oxide	19·0	21·8	8·1
Ferric oxide	39·6	62·1	23·6
Insoluble residue	18·4	8·5	
Sulphur	0·033	0·005	
Phosphorus	1·03	0·62	
Lime	5·50	0·58	
Magnesia	0·37	trace	

Ratio of concentration	1 : 1·91
Recovery	74·5 per cent

General conclusions.—The Torbrook ores being a mixture of magnetite and hematite, the greater proportion of the hematite is lost in the tailing by fine grinding and magnetic concentration. A concentrate over 60 per cent iron was made in both cases, with low sulphur content but with phosphorus over one-half per cent. Recoveries of 65 per cent to 75 per cent were made depending on the proportion of magnetite in the ores. Hematite slimes badly and it is, therefore, doubtful if tabling of the tailings from magnetic concentration would pay.

Goulais River Ore—Shipment 15 Tons

Location of property.—Tp. 22, range XII, Algoma district, Ont.

Character of ore.—The ore is extremely fine-grained. It consists almost entirely of a mixture of silica and magnetite alternating in narrow bands, the two constituents varying widely in their proportions. The leaner bands of quartz contain considerable iron and the richer bands of magnetite appreciable amounts of silica. The average iron content of the shipment was about 35 per cent. Sulphur is low, being about 0·05 per cent.

Result of wet magnetic separation tests.—Analyses of the crude ore, concentrate and tailing were:—

—	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron	33·9	50·5	4·3
Insoluble residue	52·1	30·9	
Sulphur	0·056	traces	
Phosphorus	0·090	0·046	
Lime	0·20		
Magnesia	0·10		

Ratio of concentration	1 : 1·57
Recovery	94·8 per cent

A screen analysis of the concentrate shows 35 per cent on 200 mesh, 65 per cent through 200 mesh. The −200-mesh concentrate contained 61·4 per cent iron, showing that final grinding to 200 mesh is necessary to produce a product over 60 per cent iron.

General conclusions.—From the nature of the ore it would seem that the Babbitt practice of stage crushing, sizing and cobbing, for the elimination of waste and final reduction and separation at 200 mesh would be applicable, resulting in a product, on being sintered, better than 60 per cent iron, low sulphur and phosphorus within the Bessemer limits, highly desirable for blast furnace use.

Natashkwan Sands—Shipment 45 Tons

Location of property.—Natashkwan river, lower St. Lawrence, Saguenay district, Que.

The shipments made consisted of the samples obtained from an accurate survey and sampling of the deposits. The sands occur at the mouth of the Natashkwan river on the north side of the lower St. Lawrence, for a distance of 6 miles east of the present mouth of the river. The sands consist of rounded particles of titaniferous magnetite and silica, washed down by the river and deposited at its mouth. Concentration of the sands has taken place by wave and wind action along the coast.

Results of wet magnetic separation tests.—Average analyses of crude ore, 1st and 2nd concentrates, and tailings:—

—	Crude ore	1st concentrate	2nd concentrate	Tailing
	per cent	per cent	per cent	per cent
Iron	9·60	64·61	68·37	5·61
Titanic oxide	2·26	2·36	1·61	2·69
Silica	....	6·35	2·27	84·58
Sulphur	....	....	trace	
Phosphorus	....	....	0·023	
Manganese	....	....	0·19	
Lime	....	....	0·13	
Magnesia	....	....	0·145	

Ratio of 1st concentration	1 : 14·8	Recovery	45·5 per cent
" 2nd "	1 : 1·06	"	96·0 "
Final ratio of "	1 : 16·2	Final recovery	43·7 "

NOTE.—Iron analysis given above is total iron, and not magnetic iron, hence the low recovery by magnetic separation.

General conclusions.—The magnetic iron sands of the lower St. Lawrence can be concentrated up to a high-grade iron product, low in sulphur and phosphorus, but containing up to 2 per cent titanic oxide. The concentrate would require sintering for blast furnace use.

Bessemer Mine Ore—Shipment 2 Tons

Location of property.—Lot 4, con. VI, Mayo tp., Hastings co., Ont.

Character of ore.—Fine-grained magnetite, with an associated gangue of calcite, garnet, epidote and other silicates. The magnetite and gangue are, however, intimately mixed, and it is possible to cob out a large proportion of the gangue before final separation.

Result of wet magnetic separation tests.—Analyses of crude ore, concentrate and tailing were:—

—	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron	36·5	67·4	4·5
Insoluble residue	35·37	5·87	
Sulphur	0·314	0·185	
Phosphorus	0·026	0·007	
Lime	5·68		
Magnesia	0·30		

Ratio of concentration	1 : 1·96
Recovery	94·0 per cent

A screen analysis of the ball mill discharge shows:—On 100 mesh, 22 per cent; on 200 mesh, 50 per cent; through 200 mesh, 50 per cent.

General conclusions.—The screen analysis shows that very fine grinding is not necessary to produce a high grade concentrate. The concentrate would require sintering, but the resultant product would be high grade, low in sulphur and phosphorus, and highly desirable for blast furnace use.

Bessemer Mine Ore—Shipment 5 Tons

Results of wet magnetic separation, and sintering tests.—Analyses of crude ore, concentrate and sinter were:—

—	Crude ore	Concentrate	Sinter
	per cent	per cent	per cent
Iron	43·9	67·5	66·2
Insoluble	30·88	5·98	
Sulphur	0·3	0·11	0·023
Phosphorus	0·028	0·019	0·016

The sinter produced was hard and extremely porous, an ideal product for blast furnace use.

Childs Mine—Shipment 2 Tons

Location of property.—Lots 11 and 12, con. IX, Mayo tp., Hastings co., Ont.

Character of ore.—Granular magnetite with an associated gangue of calcite, pyroxene, chlorite, etc. Although the ore is quite soft, the gangue minerals are so intimately associated with the magnetite that fine grinding is necessary to effect a satisfactory separation of the magnetite from the gangue.

Results of wet magnetic separation tests.—Analyses of crude ore, concentrate, and tailing were:—

—	Crude ore	Concentrate	Tailing
	per cent	per cent	per cent
Iron	35·0	66·4	5·7
Insoluble residue	36·8	6·09	
Sulphur	0·045	0·022	
Phosphorus	0·083	0·016	
Lime	5·83		
Magnesia	0·41		

Ratio of concentration	1 : 2
Recovery	92 per cent

A screen analysis of the ball mill discharge showed:—On 100 mesh, 33 per cent; on 200 mesh, 63 per cent; through 200 mesh, 37 per cent.

General conclusions.—The screen analysis shows that very fine grinding is not necessary to produce a high-grade concentrate. The concentrate would require sintering but the resultant product would be high grade, very low in sulphur and phosphorus, and very desirable for blast furnace use.

Groundhog Ore—Shipment 15 Tons

Location of property.—Groundhog river, tps. of Keith and Penhorwood, Sudbury district, Ont.

Character of ore.—Banded structure, consisting of bands of magnetite, hematite and jasper, the bands ranging from one-half inch in width down to a very small fraction of an inch. The magnetite occurs almost pure in some cases, in others mixed with the jasper, and in other cases finely disseminated throughout the jasper. Coarse crushing will not free the thinner bands of magnetite nor the hematite, as the hematite is in most cases firmly attached to the jasper. The shipment showed an average analysis of 35 per cent iron, of which 26·5 per cent was magnetite, and 8·5 per cent hematite, with only traces of sulphur, phosphorus and other impurities. The problem was therefore simply one of raising the iron content.

Results of experimental tests.—A large number of tests were conducted, including magnetic cobbing, jig and table concentration, and wet magnetic separation tests. Magnetic cobbing tests were made on the sized material from 1 inch down. These tests showed that no appreciable amount of concentrate approaching 55 per cent metallic iron could be recovered in sizes coarser than 1/16 inch. By crushing to 1/16 inch, magnetic cobbing, jig and table concentration of the sized 1/16-inch tailings, a concentrate was produced, assaying 53·5 per cent iron, with a recovery of 73·2 per cent and with a ratio of concentration 1: 2.

Wet magnetic separation tests on this ore, followed by tabling of the tailings, after grinding to 69 per cent through 200 mesh, gave a magnetic product assaying 64 per cent iron, and a hematite product assaying from 50 per cent to 55 per cent iron, with a recovery of total iron up to 85 per cent. The ratio of concentration was 1: 1·8.

General conclusions.—The results of the experimental tests show that a high recovery of the iron content cannot be expected on this class of ore by any simple method of concentration. A fair grade of concentrate was made after grinding to 1/16 inch, and a much better grade on finer grinding. The concentrate would require sintering for blast furnace use. The sinter would be high in silica and low in sulphur and phosphorus.

Kaministikwia Ore—Shipment 240 Pounds

Location of property.—Marks-Wiley property, Loon lake, Thunder Bay district, Ont.

Character of ore.—Banded structure, very similar to Groundhog ore.

Results of experimental tests.—Crushing to 4 mesh, jig and table concentration of the sized material gave:—

Analysis of crude ore:—	Total iron (Fe.)	37·19	per cent
	Hematite (Fe_2O_3)	17·98	"
	Magnetite (Fe_2O_4)	33·97	"
Analysis of concentrate: Iron		55	"
Analysis of tailing: Iron		20	"
Recovery of iron content		70 to 75	"
Ratio of concentration		1 : 2	

General conclusions.—Concentrate too fine for blast furnace use. Would require sintering. Product high in silica, low in sulphur and phosphorus.

The equipment of the Mines Branch laboratories consists of small and large scale apparatus and machinery permitting of experimental tests on a few hundred pounds, and on carload lots. With the present equipment, a flow sheet duplicating, on a small scale, the Babbitt practice can be followed, and various combinations can be set up to permit of deviations therefrom. Roasting and sintering tests can be made on a small scale. With very little additional equipment the laboratory investigations could be extended to the point where the details of operation might be worked out in a pilot plant. The chemical laboratories of the Division are equipped for making all necessary determinations in connexion with the experimental work by means of the Davis testing tube.

The small scale equipment of the laboratories consists of jaw crusher, rolls, pulverizer, screen frame for sizing, jigs, Wilfley table, classifiers, magnetic separators, rotating kiln, roasting furnace, and a Davis magnetic testing machine for the determination of magnetic iron.

The larger scale equipment consists of Blake crusher, picking belt, two sets of rolls, Hardinge ball mill, Traylor ball mill, automatic samplers, two sets of jigs, standard size Wilfley table, Ferraris screen, Hummer screen, Callow duplex screen, Dorr simplex classifier with bowl attachment, launder classifier, Grondal magnetic cobber, Grondal magnetic separator, Davis magnetic log washer, Ullrich magnetic separator, roasting furnaces, duplex Dwight Lloyd sintering pans.

VI

THE SELECTIVE FLOTATION OF THE LOWER GRADE NICKELIFEROUS PYRRHOTITE ORES OF ONTARIO

W. B. Timm

INTRODUCTORY

The practice of smelting the copper-nickel ores of the Sudbury district direct in the blast furnace, or by mixing the green ore with a certain portion of roasted product, or by smelting the roasted fines in a reverberatory furnace, has given very favourable results. This practice has been especially adaptable to the higher grade ores when consideration is given to the favourable market conditions that have prevailed to within the last few years.

In the treatment of the lower grade ores of the district, of which there are enormous reserves, it is a question whether it would not be more economical to apply certain methods of concentration to produce a product with a fairly high copper-nickel content prior to smelting operations. One of the operating companies has for several years been treating a portion of their lower grade material by concentration on tables and by flotation. By sintering these concentrates with flue dust and mine fines they obtain a roasted product very desirable in many ways for mixing with green ore for the blast furnace charge.

Experimental tests were conducted in the Mines Branch laboratories at Ottawa to determine whether a large portion of the barren pyrrhotite, as well as the siliceous gangue, could be eliminated, with the production of a concentrate having a copper-nickel content similar to that of the matte now being produced. Further, as the success of any such process would depend to a large extent on the recovery of the precious metal values, as well as the copper-nickel values, the experimental work included a study of the concentration products, to determine whether the precious metal values were reporting in the copper-nickel concentrate.

THE SELECTIVE FLOTATION OF THE LOWER GRADE COPPER-NICKEL ORES OF THE SUDBURY DISTRICT, ONTARIO

Experimental tests.—The first series of tests was made on the lower grade ores from two mines, to determine whether a high-grade concentrate could be made by selective flotation with high recoveries of the copper and nickel values, by the elimination of the gangue minerals and a large portion of the barren pyrrhotite. No attention was paid to where the precious metal values were reporting in this series of tests.

Table I gives the results of tests Nos. 1, 2, 4 and 5, on an ore of the following analysis:—

Copper	1·25 per cent
Nickel	1·35 "
Iron	23·70 "
Sulphur	10·70 "
Silica	37·85 "

Recoveries indicated include the actual recovery made in the concentrate plus 50 per cent of the values in the middling that would be recovered by returning this product to the head of the circuit. This percentage was proven to be a conservative estimate.

Table II gives the results of tests Nos. 6 and 8 on the same ore. The middling was re-run without further grinding. By regrinding the middling product, higher recoveries could be expected.

Table III gives the results of tests Nos. 1, 2, 3, 4, 5, 6 and 8 on a still lower grade ore, of the following analysis:—

Copper	0·47 per cent
Nickel	1·17 "
Iron	20·90 "
Sulphur	11·19 "
Silica	42·15 "

TABLE I

Test No.	Concentration products	Weight grms.	Analysis			Percentage of values		Total recoveries		
			Cu. per cent	Ni. per cent	Cu. +Ni. per cent	Cu.	Ni.	Cu. per cent	Ni. per cent	Cu. +Ni. per cent
1	Concentrate	106	12·50	9·60	22·10	93·2	76·2	95·3	82·4	89·1
	Middling	147	0·40	1·13		4·1	12·4			
	Tailing	758	0·05	0·20		2·7	11·3			
2	Concentrate	75	16·50	10·20	26·70	88·3	58·7	91·9	68·8	80·8
	Middling	155	0·65	1·70		7·2	20·2			
	Tailing	785	0·08	0·35		4·5	21·1			
4	Concentrate	95	13·66	9·75	23·41	93·6	69·8	95·4	78·3	87·1
	Middling	146	0·35	1·55		3·7	17·0			
	Tailing	758	0·05	0·23		2·7	13·1			
5	Concentrate	170	7·40	6·20	13·60	92·0	80·0	93·9	84·5	89·2
	Middling	94	0·55	1·27		3·8	9·0			
	Tailing	730	0·08	0·20		4·2	11·0			

TABLE II

Test No.	Concentration products	Weight grms.	Analysis			Percentage of values		
			Cu. per cent	Ni. per cent	Cu. +Ni. per cent	Cu.	Ni.	Cu. +Ni.
6	Concentrate	376	7·60	5·70	13·30	91·4	86·5	89·2
	Tailing	1,594	0·17	0·21	0·38	8·6	13·5	10·8
8	Concentrate	292	9·10	7·50	16·60	96·7	83·4	90·2
	Tailing	1,675	0·055	0·26	0·31	3·3	16·6	9·8

TABLE III

Test No.	Concentration products	Weight grms.	Analysis			Percentage of values			Total recoveries		
			Cu. per cent	Ni. per cent	Cu. +Ni. per cent	Cu.	Ni.	Cu. +Ni.	Cu. per cent	Ni. per cent	Cu. +Ni. per cent
1	Concentrate..........	155	2·6	6·2	8·8	87·4	82·6	84·0	90·0	86·8	87·8
	Middling............	159	0·15	0·62	......	5·2	8·4				
	Tailing..............	694	0·05	0·15	.....	7·4	8·9				
2	Concentrate..........	175	2·30	5·75	8·05	89·3	84·7	86·0	90·7	87·0	88·0
	Middling............	118	0·10	0·45	......	2·7	4·5				
	Tailing..............	715	0·05	0·18	......	8·0	10·8				
3	Concentrate..........	194	2·15	5·26	7·41	89·7	87·1	82·4	90·9	81·6	84·1
	Middling............	82	0·15	0·57	......	2·6	3·6				
	Tailing..............	730	0·05	0·29	......	7·7	10·6				
4	Concentrate..........	220	1·90	4·80	6·70	91·5	88·5	89·3	92·3	90·3	90·8
	Middling...........	144	0·05	0·30	......	1·5	3·6				
	Tailing.............	630	0·05	0·15	.....	7·0	7·9				
5	Concentrate.........	64	3·05	7·12	10·17	83·7	76·6	78·6	86·3	80·0	81·8
	Middling............	69	0·17	0·56	......	5·1	6·5				
	Tailing..............	369	0·07	0·27	......	11·2	16·8				
6	Concentrate..........	66	6·65	13·36	20·01	82·1	73·0	75·7	84·9	79·6	81·2
	Middling............	122	0·25	1·33	......	5·6	13·4				
	Tailing..............	825	0·08	0·20	......	12·3	13·6				
8	Concentrate..........	168	2·70	5·83	8·53	93·2	80·4	84·0	94·6	85·5	88·1
	Middling............	205	0·07	0·61	......	2·9	10·3				
	Tailing..............	632	0·03	0·18	......	3·9	9·3				

The procedure followed in conducting the tests was:—

The ore was crushed to 20 mesh; 1,000 grammes was ground wet in a small ball mill with the reagents for about 40 minutes; a rougher concentrate and tailing were first made; the rougher concentrate was recleaned, making a concentrate and a middling. In tests Nos. 6 and 8 of Table II, two 1,000-gramme lots were used in order to obtain sufficient middling for refloating. The concentrate from refloating the middling was added to the final concentrate, and the tailing to the rougher tailing, so as to have only two products, concentrate and tailing.

No test work was done to determine the degree of fineness to which the ores should be ground to obtain the best results. For the tests, approximately 90 per cent would pass a 200-mesh screen. For test No. 8 of Table III, it was ground to pass a 100-mesh screen, 67 per cent through 200 mesh.

Conclusions from experimental tests.—There is no apparent difficulty in making a fairly high-grade concentrate, with a good recovery of the copper-nickel values. A product was made, in certain cases, with a higher copper-nickel content than that contained in the blast furnace mattes and certainly higher than the mattes that would be produced by present smelting practice on the same grades of ores.

For the selective flotation of the ores, the alphabetical reagents were found to be the most suitable. In using these reagents the essential point is to maintain an alkaline pulp by the addition of lime, soda ash or caustic soda. The chalcopyrite floats with remarkable ease, but the nickel minerals are more difficult to float.

Examination for the amounts of pyrrhotite being eliminated in the tailing was made on the flotation products of the tests run for the determination of precious metal values as given in Tables IV and VI, where the concentration was not nearly so good as in some of the tests given in Tables I and III. This was found to be 30 per cent and 41 per cent respectively, of the amounts in the ore. This elimination of practically barren pyrrhotite, together with the gangue minerals, means considerable savings in several directions. Taking the flotation products of tests Nos. 1, 2 and 4 of Table I and of test No. 6 of Table III, where high-grade concentrates were made, the percentage of pyrrhotite eliminated in the tailings would be much higher than that given above.

THE PRECIOUS METAL VALUES IN THE FLOTATION PRODUCTS OF THE LOWER GRADE COPPER-NICKEL ORES OF THE SUDBURY DISTRICT

Experimental tests.—To determine if the precious metal values in the ores were being concentrated with the copper-nickel values, flotation tests in one kilogram lots were conducted on 13 kilograms of each ore. The products from the thirteen tests on each ore were combined to obtain sufficient quantities of the flotation products for accurate determination of the precious metal contents. In conducting these tests attention was not paid to obtaining the best flotation results, as this was demonstrated by former tests, the results of which are given in Tables I, II and III.

Tables IV and V give the results on the first ore, the flotation results on which are given in more detail in Tables I and II.

Tables VI and VII give the results on the second ore, which was of still lower grade, the flotation results on which are given in more detail in Table III.

TABLE IV

Product	Weight grms.	Assays						
		Nickel per cent	Copper per cent	Gold oz. per ton	Plat. oz. per ton	Palladm. oz. per ton	Rhod., Irid., etc. oz. per ton	Precious metal values content
							*	**
Concentrate..........	2,050	5·88	7·55	0·036	0·062	0·072	0·006	$13 18
Middling.............	1,490	0·93	0·20	0·012	0·019	0·022	0·004	4 47
Tailing..............	9,460	0·16	0·05	0·002	0·0027	0·0038	trace	0 62
Ore..................	13,000	1·15	1·25	0·008	0·014	0·017	0·0014	3 04

*Results are no doubt low due to oxidation and volatilization in fusing and scorifying.

**Figures from December, 1922, quotations on these metals.

TABLE V

Product	Percentages in products						
	Weight	Nickel	Copper	Gold	Plat.	Palladm.	Precious metal values
Concentrate..........	15·8	80·6	95·5	66·8	70·3	68·5	68·4
Middling.............	11·5	9·3	1·8	16·1	15·7	16·3	16·8
Tailing..............	72·7	10·1	2·9	17·1	14·0	15·2	14·8

TABLE VI

Product	Weight grms.	Assays						Precious metal values content
		Copper per cent	Nickel per cent	Gold oz. per ton	Plat. oz. per ton	Palladm. oz. per ton	Rhod., Irid., etc. oz. per ton	
							*	**
Concentrate..........	1,635	2·90	6·40	0·027	0·032	0·043	0·005	$7 40
Middling.............	1,418	0·27	1·65	0·013	0·016	0·015	0·003	3 54
Tailing..............	9,947	0·05	0·21	0·0009	0·0011	0·0026	....	0 30
	13,000	0·43	1·14	0·0055	0·0067	0·0009	0·0009	1 55

*Results are no doubt low due to oxidation and volatilization in fusing and scorifying.
**Figures from December, 1922, quotations on these metals.

TABLE VII

Product	Percentages in products						
	Weight	Copper	Nickel	Gold	Plat.	Palladm.	Precious metal values
Concentrate..........	12·6	84·4	70·3	61·8	60·9	60·0	60·3
Middling.............	10·9	6·8	15·7	25·7	26·4	18·0	24·9
Tailing..............	76·5	8·8	14·0	12·5	12·7	22·0	14·8

Conclusions from examination of flotation products.—The results show, in the case of the first ore, that 68 per cent of the total precious metal values is contained in the concentrate, 17 per cent in the middling product and 15 per cent in the tailing. By regrinding and floating the middling, which represents 11·5 per cent of the original feed, a total recovery of 80 per cent of the precious metal values would be obtained in the copper-nickel concentrate by selective flotation.

The results show, in the case of the second ore of lower grade, that 60 per cent of the precious metal values is contained in the concentrate, 25 per cent in the middling product, and 15 per cent in the tailing. By regrinding and floating the middling, which represents 10·9 per cent of the original feed, a total recovery of 75 per cent of the precious metal values would be obtained in the copper-nickel concentrate by selective flotation.

The results of the above tests show that the gold and the platinum group metals are reporting to a large extent in the concentrate with the copper and nickel minerals.

In the determination of the precious metal contents, the total flotation products were used, and the assay results obtained figured to the per ton basis. The assays so obtained should be reliable and accurate.

No definite conclusions were arrived at with respect to the particular mineral or minerals with which the platinum group metals are associated. In studying the ratios between these metals and the copper, nickel, iron and sulphur content in the flotation products, there is no definite uniformity of values or proportions between these metals and the chalcopyrite, pentlandite, pyrrhotite, content in the ores and flotation products. Whether this should be accepted in support of the opinion that the lower grade ores carry relatively higher values in the platinum group metals is questionable.

THE SELECTIVE FLOTATION OF THE COPPER-NICKEL ORES OF SHEBANDOWAN LAKE DISTRICT, ONTARIO

Experimental tests.—Tests were conducted on a shipment of ore from this district to determine whether a high-grade copper-nickel concentrate could be obtained with a satisfactory recovery of the copper-nickel and precious metal values. The tests were made on an ore of the following analysis:—

Nickel	2·97 per cent	Lime	2·50 per cent
Copper	1·65 "	Magnesia	7·64 "
Cobalt	0·15 "	Sulphur	15·84 "
Iron	24·30 "	Gold	0·01 ozs. per ton
Alumina	10·15 "	Platinum	0·03 "
Silica	26·55 "	Palladium	0·048 "

Table VIII gives the results of the tests. In test No. 1 a concentrate and tailing were made. In test No. 2 a concentrate, middling and tailing were made.

TABLE VIII

Test No.	Product	Weight per cent	Analysis			Percentage of values			Total recoveries		
			Cu. per cent	Ni.+ Co. per cent	Cu.+ Ni.+ Co. per cent	Cu. per cent.	Ni.+ Co.	Cu.+ Ni.+ Co.	Cu. per cent	Ni.+ Co. per cent	Cu.+ Ni.+ Co. per cent
1	Concentrate	25·7	6·35	9·88	16·23	83·0	92·1	88·3	83·0	92·1	88·3
	Tailing	74·3	0·45	0·24	......	17·0	7·9				
2	Concentrate	30·7	5·00	8·56	13·56	88·3	95·5	92·7	91·1	96·4	94·4
	Middling	17·0	0·58	0·30	......	5·7	1·9	.....			
	Tailing	52·3	0·20	0·14	......	6·0	2·6	.....			

Conclusions from experimental tests.—A high recovery of the copper, nickel and cobalt values was made in a good grade of concentrate. The recovery of the copper is not as good as on the Sudbury ores, probably due to oxidation. Continuous grinding and flotation tests will be made, and the precious metal values in the flotation products determined, to prove whether these values are reporting in the concentrate. Taking the results obtained in test No. 2 of Table VIII, determinations made, show that 36 per cent of the pyrrhotite in the ore is eliminated in the tailing, and this elimination of pyrrhotite, if practically barren as is the case in the tests on the Sudbury ores, together with the gangue silicates, is very desirable for subsequent smelting operations.

VII

DESCRIPTION OF ORE CONCENTRATION PLANTS IN CANADA

THE NEW SULLIVAN CONCENTRATOR AT KIMBERLEY, B.C.

On August 24, 1923, the Consolidated Mining and Smelting Co. of Canada put into operation its new concentrator at Kimberley, B.C., for the treatment of the Sullivan mine ore. It is the result of years of experimentation in the laboratory and in pilot plants on the treatment of this complex sulphide ore.

The Sullivan ore is a valuable mixture of zinc, lead and iron sulphides with relatively small amounts of gangue, principally calcite and quartz. The zinc content varies from 12 to 15 per cent; the lead content from 10 to 12 per cent; the iron from 32 to 35 per cent; the sulphur from 30 to 33 per cent; the insoluble from 4 to 6 per cent; the silver is about 3 ozs. per ton. The chief minerals are marmatite, galena, pyrrhotite, pyrite, calcite and quartz. A polished section indicates the intimate association of the minerals and fine grinding to at least 200 mesh is necessary to free them.

The concentrator is favourably situated on the slope of a hill, 22½° to the horizontal, and advantage is taken of gravity for the flow of the ore and pulp. Below the concentrator is a large area of 40 acres for tailing disposal. The concentrator is connected with the coarse crushing plant, situated below the portal of the lower adit, by a standard gauge railway, 3½ miles long, and is also connected with the main railway for the shipping of concentrate to the company's smelter at Trail, B.C.

Coarse crushing plant.—In this plant the ore from the mine is reduced to 3 inches in a 36 by 42-inch type C Buchanan jaw crusher and in two No. 8 Gates gyratory crushers, and conveyed to one 2,500-ton railway bin.

Roll plant.—This plant is situated to one side and above the concentrator. It receives its feed from a 1,000-ton ore bin. The ore is here reduced to three-fourths inch in two sets of 72 by 20-inch Alaska type rolls.

Sampling plant.—From the roll plant the ore is conveyed to a 2,500-ton fine ore bin, directly above the concentrator. In passing to this bin a one ten-thousandth portion of the feed is cut out by a system of automatic sampling and mixing. Flood automatic samplers are also used for control samples at various points in the mill.

Fine grinding plant.—In this plant the ore is reduced to the required fineness for flotation, namely 95 per cent through 200 mesh. To obtain this reduction, two 8-foot by 48-inch Hardinge ball mills are used for primary grinding and four 8-foot by 48-inch Hardinge ball mills for secondary grinding. Ten 6 by 25-foot rake Dorr classifiers are used for primary classification and two 10-foot bowl, 3-foot rake Dorr classifiers for secondary classification.

Lead recovery plant.—For the recovery of the lead sulphide, three primary 18-cell, 24-inch modified Minerals Separation flotation machines are used, followed by three steps of cleaner cells each consisting of 8-cell 24-inch modified Minerals Separation flotation machines.

Zinc recovery plant.—For the recovery of the zinc sulphide, four primary 18-cell, 24-inch modified Minerals Separation flotation machines are used. Four products are made, namely a lead-zinc concentrate, a clean zinc concentrate, a zinc middling, and an iron tailing, The lead-zinc concentrate is tabled on 32 Plat-O and Wilfley tables, a lead-iron concentrate being returned to the rake classifiers of the fine grinding plant; a clean zinc concentrate combined with that from the flotation cells, and a zinc middling to be re-treated. The zinc middling from the flotation cells and the zinc middling from the tables are reground in an 8-foot by 48-inch Hardinge ball mill. The iron tailing from the flotation cells is classified in a 10-foot bowl, 3-foot rake Dorr classifier, the sands being reground in the ball mill with the zinc middling, the slimes going to the dewatering plant. Zinc is recovered from the reground zinc middling and the sand tailing in an 8-cell 24-inch modified Minerals Separation flotation machine.

Dewatering plant.—The lead concentrate is dewatered in two modified Genter vacuum thickeners and in two 6-foot diameter, 6-disk, 2-valve American filters. The filter cake is conveyed to the lead concentrate loading bins. The zinc concentrate is dewatered in two 50 by 8-foot Dorr thickeners, and in three 6-foot diameter, 6-disk, 2-valve American filters. The filter cake is conveyed to the zinc loading bins. The iron tailing is dewatered in a modified Genter vacuum thickener and sluiced to the tailing pond.

Reagents used.—For the separation of the minerals, selective flotation is employed. The water used in the fine grinding plant is made alkaline by sodium carbonate. For the flotation of the lead sulphide, a mixture of water-gas tar and coal tar creosote is used with small amounts of sodium cyanide and cresylic acid. Copper sulphate and water-gas tar are added to the zinc cells for the flotation of the zinc sulphide. The consumption of reagents is considerably reduced by the return of solutions from the dewatering plant to storage tanks above the mill.

Special features of construction.—Ample room is provided on all floors. Floors are provided with overhead travelling cranes and are served by an inclined tramway running from the repair shops to the bottom of the mill. Individual direct connected motor drives are used on all machinery where practical. All circulating solutions and pulps are handled by Wilfley pumps. The arrangement for the pumps in the tunnel below the main floors in the centre of the building provides for taking care of spills, overflows caused by temporary shut downs, for keeping the mill clean, and delivering the material to the proper compartment of the mill. Railway track scales are provided for the weighing in of the ore as received from the coarse crushing plant, and the weighing out of the lead and zinc concentrates to the company's smelter.

Results being obtained.—In October, 1923, W. B. Timm, of the Department of Mines, visited the concentrator, and although it was not operating at capacity, due to shortage of power, 2,000 tons of ore were being treated daily. The full equipment was not being utilized; in the fine grinding plant, the bowl classifiers were not in operation; in the lead recovery plant, only one cleaner was being used; in the zinc recovery plant, the re-treatment portion for the zinc middlings, and the tables, were not in operation, yet a better grade of concentrate was being produced and a higher recovery obtained than in the concentrator at Trail, formerly treating this ore.

Working at full capacity, and with all units in operation, 3,000 tons of ore daily will be treated. Final concentrates made on November 17, 1923, showed the lead concentrate to contain—lead 70·1 per cent, zinc 5·4 per cent, and the zinc concentrate to contain—lead 4·0 per cent, zinc, 43·3 per cent.

THE SILVERSMITH CONCENTRATOR AT SANDON, B.C.

The Silversmith mill, of the Silversmith Mines, Ltd., Sandon, B.C., represents the latest design in modern mill practice for the concentration of the silver-lead-zinc ores of the Slocan district, by a combination of gravity methods and the oil flotation process.

The principal minerals of the Slocan ores are argentiferous galena, zinc blende, siderite (spathic iron) and iron pyrite. Tetrahedrite and chalcopyrite are found in small quantities associated, as a general rule, with the galena. A general assay of the Silversmith milling ore shows, silver 25 ounces per ton, lead 9 per cent, and zinc 7 per cent. The silver values being of greatest importance, the process that is followed recovers the greatest percentage of these values in a concentrate, having the greatest marketable value. The galena slimes badly, therefore the main object is to remove the greatest possible amount of it, by the use of jigs and tables, with as little grinding as possible. No attention is paid to the zinc blende in the gravity stages of concentration, as a clean zinc concentrate cannot be made by this means on account of the spathic iron having a similar specific gravity to the blende. A clean zinc concentrate is made by the flotation process after the galena is removed.

The concentrator has a daily capacity of 125 tons. The ore is supplied to the mill bins by an aerial tramway. It is crushed in a 10 by 20-inch Blake crusher, passed by a conveyer belt, which is also used as a sorting belt, to the mill feed bins. Clean ore is picked off the sorting belt and shipped direct to the smelter. From the mill bins the ore goes to a set of rolls, thence to jigs, and from there through successive stages of regrinding, classification, sizing, jig and table concentration, until finally reduced to a fineness (minus 80 mesh) suitable for treatment by flotation. Regrinding is done in two sets of rolls, a 4 by 5-foot cylindrical ball mill, and in a 6-foot by 16-inch Hardinge mill. Classification is done in two sets of 6-cell hydraulic classifiers, and in a Dorr classifier. Sizing is done in a set of 3 mm. trommels. By this method the galena is removed as soon as it is freed, without excessive loss in slimes from fine grinding. The jig and table concentrate is easily dewatered for shipment by a simple method of drainage.

The classifier overflows are thickened in Dorr settling tanks for flotation. The flotation plant is of the Callow pneumatic type, and consists of two lead rougher cells and one lead cleaner cell; one zinc rougher cell, and two zinc cleaner cells. Lead concentrate made from the two lead roughers goes to the lead cleaner, where a high-grade argentiferous lead concentrate is made, the tailing from the cleaner being returned to the head of the rougher cells. The tailing from the lead rougher cells goes to the zinc rougher cell, the tailing from which goes to waste and the concentrate passes on to the first cleaner cell. The concentrate from this cell is further cleaned in a second cleaner cell. The tailing from the cleaner cells is returned to the head of the zinc rougher cell. The final zinc concentrate made is about 48 per cent zinc. The lead and zinc flotation concentrates are thickened in Dorr tanks and dewatered in American disk filters.

A visit was made to the mill by W. B. Timm, of the Department of Mines, in July, 1922. Since that time the concentrator has been in continual operation and regular shipments are made to the smelter of the Consolidated Mining and Smelting Co. of Canada, at Trail, B.C.

THE NEW BRITANNIA CONCENTRATOR AT BRITANNIA BEACH, B.C.

In January, 1923, the new concentrator for the concentration of the Britannia mine copper ores was completed. It was built with an initial capacity of 2,000 tons daily, but this tonnage can readily be increased to 3,000 tons by slight alterations in the fine grinding equipment. It replaces the former mill, destroyed by fire in 1921, and is of fireproof construction throughout. It is situated on a steep hillside, and has seven main floors for the various operations.

The Britannia ores are iron pyrite and chalcopyrite in a highly siliceous gangue. The metallic sulphides constitute about 17 per cent of the total weight of the ore.

Coarse crushing is done at the mine. The mine ore is reduced to 3-inch size in one 24 by 35-inch Blake crusher and in two Allis Chalmers No. 7½ gyratory crushers. From the coarse crushing plant, the ore is transported to bins of 2,500 tons capacity, situated above the concentrator at the beach.

The ore, as received from the mine, contains from 5 to 15 per cent moisture, mainly in the fines. A novel method of reducing the ore to three-sixteenths of an inch, for ball mill feed is practised without resorting to wet crushing, thus avoiding the difficulties generally experienced in dry crushing ore containing this percentage of moisture. The fines are washed out by spraying, the ore passing over Hummer screens having three-sixteenth-inch openings. Compressed air sprays are used on the coarse material which goes direct to the rolls containing only 2 per cent moisture. The fines, representing 20 per cent of the total ore, plus the wash water, go direct to the ball mills.

The roll plant consists of one 72-inch by 20-inch Traylor roll, crushing to three-quarters of an inch in closed circuit with 8 Hummer screens delivering oversize plus three-sixteenths of an inch to four 54-inch by 20-inch Traylor rolls in circuit with the screens. The ore reduced to three-sixteenths of an inch is conveyed to the fine storage bins, 3,600 tons capacity.

Fine grinding is done in eighteen 7 by 10-foot ball mills, each in closed circuit with a 19-foot simplex Dorr classifier. The feed from the fine storage bins is conveyed and divided by switch distributors to the classifiers. The ore is reduced to a fineness, 18 per cent on 65 mesh. The Dorr classifier overflows contain 20 per cent solids and are laundered direct to the flotation machines.

The flotation plant consists of six 14-cell Minerals Separation standard machines. The feed is distributed to the head of four of these units. Clean concentrate is obtained from the first four cells of each unit, a middling product is taken off from the remainder, and is conveyed by air lifts to the head of the other two units for recleaning, and the tailing is sent to waste. From the two units used for recleaning the middling, a clean concentrate is obtained, and a middling which is returned to the head of the first four units.

The flotation concentrate is dewatered and settled in three 40-foot two tray Dorr tanks, the thickened concentrate being pumped to two 8-foot six disk American filters. The filter cake is conveyed a quarter of a mile to a 10,000-ton storage bin at the wharf. From this bin it is removed by cranes and conveyer belt to the steamers for shipment to the smelter.

A visit was made by W. B. Timm, of the Department of Mines, to the concentrator in October, 1923. At that time the mill feed was averaging 2·0 per cent copper, the copper concentrate produced contained 20 per cent copper, and the tailing contained 0·16 per cent copper. These results were being obtained by flotation of the chalcopyrite from the pulp, made alkaline by the addition of two pounds of lime per ton of ore. The flotation reagents used were a mixture of coal tar creosote and pine oil, and small quantities of sodium resinate. The original flow sheet of the concentrator included Hancock jigs ahead of the fine grinding mills for the removal of a portion of the copper as coarse concentrate. Danish flint pebbles were used in the fine grinding mills at the start. The flotation plant was installed for roughing only. Laboratory experiments showed that by changing from an acid to an alkaline pulp, the grade of the concentrate could be doubled by selective flotation of the chalcopyrite from the iron pyrite, without any appreciable loss of copper. The jigs were eliminated from the flow sheet; the pebble mills were changed over to ball mills for greater capacity, and the flotation units converted into rougher and cleaner units for selective flotation.

A NEW COPPER CONCENTRATOR AT ANYOX, B.C.

The Granby Consolidated Mining, Smelting, and Power Company is erecting a concentrator at Anyox, B.C., to concentrate the greenstone copper ores of the Hidden Creek ore-bodies. The initial capacity will be 1,500 tons daily. Crushing capacity of 3,000 tons is being installed to take care of increased capacity later.

The Hidden Creek ores are of two types, namely, a heavy pyritic ore, and a greenstone ore, impregnated with chalcopyrite and pyrrhotite. Smelting operations are semi-pyritic, so that in the course of mining operations a large tonnage of greenstone ore was left behind, it being too expensive to smelt this type of ore.

A typical analysis of the greenstone ore will show:—

Copper	1·85	per cent	Sulphur	16·0	per cent
Insoluble	42·0	"	Lime	4·0	"
Silica	35·0	"	Magnesia	5·0	"
Iron	24·0	"	Alumina	9·0	"

The chief sulphide minerals are chalcopyrite, pyrrhotite, pyrite, and sphalerite. Small amounts of precious metal values are also present in the ore.

The erection of the concentrator is the result of experimental work in the laboratory and in a pilot plant, in which 50,000 tons of ore were concentrated. It was proven that a copper concentrate, assaying from 12 to 15 per cent copper, with a concentration ratio of 8 to 1, and with a good recovery of the copper and precious metal values could be produced by selective flotation.

A visit was made to the works of the Granby Consolidated Mining, Smelting and Power Co., at Anyox, B.C., by W. B. Timm of the Department of Mines in November, 1923. At that time the foundations and floors of the concentrator were completed, the steel frame work was erected, and the machinery was being set in place. It was hoped to have the concentrator completed and ready for operation in the early spring. It is situated on the slope of a hillside between the mine and smelter. The mine ore will be reduced to 3-inch size in gyratory crushers and to three-eighth-inch size in two sets of Alaska type rolls. Fine grinding will be done in rod mills in closed circuit with drag classifiers, grinding to a fineness of 70 per cent through 200 mesh. The use of rod mills is new to fine grinding practice in this country, their use being confined to comparatively coarse grinding. The flotation plant will consist of Cole rougher and cleaner cells, the concentrates thickened in Dorr tanks and dewatered by Oliver filters.

THE COPPER CONCENTRATOR, ALLENBY, B.C.

The Allenby concentrator, built by the Canada Copper Corporation, was completed in 1919, and operated for a short time during the latter part of that year, when operations ceased, due to the decline in the copper market.

The 2,000-ton concentrator is situated five and a half miles south of the town of Princeton, B.C., and seven and a half miles farther south is the Copper Mountain mine, both connected by the Copper Mountain branch of the Kettle Valley railway from Princeton. Development work at the mine is reported to have blocked out 10,000,000 tons of copper ore of an average grade of 1·74 per cent copper.

The ore is chalcopyrite and bornite, disseminated through the granodiorite. Associated with it are small amounts of hematite, magnetite and iron pyrite. An average analysis of the ore is:

Copper	1·77	per cent	Sulphur	1·1	per cent
Silica	50·0	"	Alumina	19·9	"
Iron	5·8	"	Gold	0·005	oz./ton
Lime	9·1	"	Silver	0·20	"

The ore lenses have been developed from three adits, the lowest one being used as the main haulage level. It is connected with the intermediate level by an ore pass 804 feet long. Below the lowest adit is the coarse crushing plant, consisting of one 30 by 42-inch jaw crusher in which reduction is made to 8-inch size, and two No. 7½ gyratory crushers reducing the ore to 2½ inch, from which it is delivered to storage bins of 2,000 tons capacity.

From the mine bins the ore is hauled in railway cars to concentrator bins of 1,800 tons capacity. From these bins it is conveyed and distributed to six 3 by 4-foot impact screens with five-eighth-inch openings, the oversize from the screens being crushed in one set of 72 by 20-inch Garfield rolls, three-quarter-inch opening, and conveyed to the five-eighth-inch impact screens. The undersize from the screens is conveyed to a 2,000-ton storage bin, from which it is drawn to two sets of 54 by 20-inch rolls set close. After reduction by these rolls it is screened on eight Colorado Iron Works impact screens, 5-mesh openings, the oversize from the screens returned to the rolls, and the undersize going to the primary fine grinding units. Primary fine grinding is accomplished in four 7 by 10-foot ball mills, the discharge of the mills classified in four Dorr model D classifiers, the oversize ground in six 5 by 20-foot tube mills in closed circuit with six Akins classifiers. The overflow from the Dorr and Akins classifiers, which will practically all pass 80 mesh, a large proportion being finer than 200 mesh, goes to the flotation units.

From the flotation units, a concentrate, a middling that is returned to the head of the unit, and a tailing which goes to waste, are made. The concentrate is thickened in two 40-foot Dorr thickeners and dewatered in two continuous American filters, and the cake conveyed to a concentrate bin of 200 tons capacity.

The copper minerals, in the ore, being chalcopyrite and bornite, with only small amounts of iron sulphides present, permit of a high-grade copper concentrate, about 30 per cent copper being made by flotation. Such a concentrate is highly desirable for mixing purposes with lower grade ores and concentrates. The concentrate will be shipped to the Trail smelter of the Consolidated Mining and Smelting Co. of Canada.

A large pumping plant supplies water to the concentrator and other buildings from the Similkameen river. Electric power is furnished by the West Kootenay Power Co. from their transmission lines from Greenwood to Copper Mountain.

During 1923 the Granby Consolidated Mining, Smelting and Power Co. acquired the holdings of the Allenby Copper Co., until recently known as the Canada Copper Corporation, and took immediate steps towards putting the mine, surface plant and concentrator in condition for operation. Due to the unfavourable conditions of the copper market this work was suspended.

THE CONCENTRATOR FOR ROSSLAND ORES, TRAIL, B.C.

In November, 1923, the Consolidated Mining and Smelting Company of Canada, Ltd., commenced treating their Rossland ores in the concentrator at Trail, B.C., that had formerly been used for the treatment of the lead-zinc ores from the Sullivan mine. The new concentrator at Kimberley, B.C., having been put into successful operation on this ore, the Trail concentrator became available for the treatment of Rossland ores.

The original concentrator was built as a pilot plant for Rossland ores, and after running for some time, demonstrating the feasibility of treating these ores by flotation, it was re-adjusted and used as a pilot plant for the Sullivan ore. The successful treatment of the Sullivan ore having been demonstrated, it was enlarged to a daily capacity of 1,000 tons and treated Sullivan ore until the new concentrator at Kimberley was completed.

The Rossland ores consist of the sulphides of iron, pyrrhotite and pyrite, and chalcopyrite in a siliceous gangue. The average milling grade of the ore contains up to one per cent copper, with four to five dollars in gold values. Smelting operations on the crude ore being no longer profitable, a method of concentration prior to smelting, was developed after much experimental work.

The ore is transported from the mine bins at Rossland to Trail in standard railway ore cars. It is crushed in jaw crushers and rolls, and delivered to the concentrator bins. The concentrator has a capacity of 700 tons per day. Primary fine grinding is done in one 8-foot by 48-inch Hardinge ball mill, and in one 5 by 10-foot Marcy rod mill. Secondary fine grinding is accomplished in three 5 by 18-foot ball tube mills, each one being operated in closed circuit with a 6 by 25-foot Dorr rake classifier. A fineness 85 per cent through 200 mesh is obtained in the grinding circuit.

The pulp density for flotation is one of solids to three of water. A warm neutral circuit is maintained, the temperature being between 20° and 25°C. Two and one-half to three pounds per ton of coal tar creosote, and 0·05 per ton of pine oil or wood creosote are used.

Flotation is accomplished in two standard 24-inch Minerals Separation modified machines. Sub-aeration machines will later be used as cleaners. The final concentrate is thickened in two 40-foot Dorr thickeners and dewatered in two 4-foot 6-disk American continuous filters. The moisture in the cake is 14 to 15 per cent.

The feed averages 1 per cent copper and 0·25 oz. per ton gold. The concentrate made will average 8 per cent copper, 1·8 ozs. per ton gold. The tailing will average 0·06 per cent copper, 0·045 oz. per ton gold. The ratio of concentration is 1 : 8·5 and the recoveries are 93 per cent of the copper values, and 82·5 per cent of the gold values in the ore.

NEW CONCENTRATOR FOR THE EUSTIS MINING CO., EUSTIS, QUE.

Alterations to the concentrator of the Eustis Mining Company were completed and the concentrator put into operation in the early part of October, 1923.

The Eustis mine is in the Eastern Townships of Quebec, about 7 miles south of Sherbrooke. The first discovery of copper was made in 1865, and since then about a million tons of ore have been mined. The present shaft is now at a depth of over 4,000 feet.

The milling ore is a heavy sulphide, containing chalcopyrite associated with pyrite and small quantities of calcite, quartz, and other gangue minerals. It is free from arsenic and is an excellent ore for acid-making. The ore is now mined principally for the copper content. A typical analysis is copper 3 per cent, iron 40 per cent, sulphur 45 per cent, insoluble 7·5 per cent, other gangue 4·6 per cent.

Concentration by selective flotation.—Research carried on for a period of over a year, by the company in co-operation with the Department of Mines, was successful in working out a flotation process by which the copper-bearing mineral could be separated from the iron sulphides (pyrites) and gangue. The old concentrator, which had been able to do little more than eliminate the gangue minerals, was completely remodelled, and the new process installed.

A brief description of the new process is as follows. The ore is taken by electric haulage and gravity incline from the mine to the mill, where ample storage is provided. The mill has a capacity of 200 tons per day. Standard practice is used in the coarse crushing or breaking department of the mill, where the ore is reduced to one and one-half inches. A 300-

ton circular ore-bin provides storage capacity for the ball mill feed. The ore is fed by a belt conveyer to an 8-foot cylindrical ball mill, the discharge of this mill is divided between two 6-foot ball mills that operate in closed circuit with two duplex standard Dorr classifiers. The classifier overflows, which will pass a 100-mesh screen, are maintained at a high density (30 per cent solids) and elevated by Wilfley sand pumps to a Callow, flat bottom, flotation cell, which is 36 feet long by 24 inches wide and 18 inches deep. In this cell, the copper mineral is floated and the iron and gangue minerals dropped. A rougher concentrate, containing the copper, and an iron tailing, which goes to a separate section of the mill for the recovery of the iron sulphides (pyrites) is made. The rougher copper concentrate is recleaned in a second cell, from which a final copper concentrate and a middling product, which is returned to the first cell for re-treatment, are obtained. The final copper concentrate contains over 20 per cent copper and the recovery made is over 90 per cent.

Reagents for copper flotation.—The separation between the copper (chalcopyrite) and the iron sulphides is controlled by the use of an alkaline pulp. Lime at the rate of 5 to 10 pounds per ton is fed to the ball mills with the ore. The flotation oils used at present are neutral coal tar creosote and a mixture of 25 per cent thio-carbanilide and 75 per cent ortho-toluodine. A little, steam-distilled, pine oil is also used.

Pyrite recovery plant.—The pyrite in the ore is recovered for sale to the Nichols Chemical Co., at Capelton, one mile distant. The tailing from the copper flotation plant is dewatered in a large Dorr thickener to eliminate the bulk of the lime. The thickened pulp is mixed with fresh water and reduced to a pulp containing 25 per cent solids. This diluted pulp is acidified and distributed to two half-size Callow flotation cells, where the pyrite is recovered by flotation. The pyrite concentrate contains 50 to 55 per cent sulphur.

Reagents used for pyrite flotation.—The diluted pulp from the Dorr thickener is acidified by the addition of sulphuric acid. The quantity used is about 10 pounds per ton, original feed. The flotation oils used are fuel oil and crude pine oil.

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