CANADA

DEPARTMENT OF MINES

Hon. Charles Stewart, Minister; Charles Camsell, Deputy Minister

MINES BRANCH

John McLeish, Director

INVESTIGATIONS IN 1922

ORE DRESSING AND METALLURGY

(Testing and Research Laboratories)

(I) General review of investigations: by W. B. Timm.

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(Annual Summary Report of the Mines Branch, pp. 71-193)

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ORE DRESSING AND METALLURGICAL DIVISION

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm, *Chief of Division*

During the year 1922, the officers of the Ore Dressing and Metallurgical Division were engaged on investigations in connection with the treatment of Canadian ores, covering a wide variety of ore dressing and metallurgical problems. Some of these investigations were a continuation of the experimental work of the former year, and others were new investigations undertaken during the year The extent and scope of the experimental work can be seen from the table compiled of the ores and metallurgical products received for investigation. A brief review of the investigations is given below.

Test No. 128

THE RECOVERY OF PLATINUM IN BLACK SANDS, BY THE ZACHERT PROCESS, BY R. K. CARNOCHAN

Tests were made on the recovery of the platinum in the concentrates from the Bullion Mine, Quesnel district, B.C., by the use of the Zachert process. The results of the tests conducted were not favourable. By ordinary methods of amalgamation as good a recovery could be expected.

Test No. 156

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

This was a silver-lead ore carrying gold as its chief valuable mineral. Of the various methods tried that of table concentration followed by cyanidation gave the best results.

Test No. 161

THE CONCENTRATION OF THE FLIN FLON DISSEMINATED ORE, BY C. S. PARSONS

During the previous year, an investigation was conducted on the treatment of the sulphide ore of the Flin Flon ore body. During 1922 experimental work was conducted on the disseminated ore from the footwall and hanging wall portions of this ore body. The presence of a deleterious gangue mineral made the concentration of this ore a more difficult problem than first supposed from an examination of the ore. Methods of overcoming the difficulties are described in the report.

Test No. 162

THE ELIMINATION OF THE IMPURITIES FROM MALAGASH ROCK SALT

BY R. K. CARNOCHAN

This investigation covers the experimental work on the elimination of impurities in the Malagash rock salt. Due to the presence of iron and organic matter in the salt the products for the trade were slightly discoloured. Before embarking on an expensive process of dissolution and evaporating, other methods of purification were tried.

Test No. 163

THE CONCENTRATION OF THE GRAPHITE ORE FROM THE TIMMINS MINE, WESTPORT, ONT., BY C. S. PARSONS

The experimental work was conducted to determine the best method of producing a high grade flake with a maximum recovery of coarse flake and a high recovery of carbon values in the ore.

Test No. 164

SEPARATION OF FOSSIL RESIN FROM COAL, BY R. K. CARNOCHAN

This investigation covers the experimental work on the separation of fossil resin from the coal of the Coalmont Collieries, Ltd., Coalmont, B.C. Methods of recovering the resin from the coal are described in the report.

Test No. 165

THE WET SEPARATION OF ASBESTOS FROM ITS GANGUE,

BY R. K. CARNOCHAN

The results of this preliminary investigation show the possibility of the application of the wet method of milling asbestos rock for the recovery of the fibre. If such a process could be demonstrated to be feasible at a reduced cost over the dry methods in use, a decided advance in the practice of milling asbestos rock would be made.

Test No. 166

THE SEPARATION OF FLUORITE, CALCITE, AND BARITE IN THE FLUORSPAR FROM THE MADOC DISTRICT, ONT., BY R. K. CARNOCHAN

The experimental work conducted on this type of fluorspar has revealed a new method of separation applicable to the separation of the minerals of these deposits. By this new method described in the report, an exceptionally high grade fluorspar can be obtained.

Test No. 167

THE MILLING AND CONCENTRATION OF COPPER-GOLD ORE FROM KITSALAS MOUNTAIN COPPER CO., USK, B.C., BY C. L. DEWAR

The report of this investigation shows that a very high grade copper concentrate 50 per cent–55 per cent copper with recoveries of 90 per cent of the copper, gold, and silver values can be obtained from this ore. The ore is very amenable to the method described in the report.

Test No. 168

THE CONCENTRATION OF THE ANTIMONY ORE FROM LAKE GEORGE, N.B.,

BY C. S. PARSONS

The report of this investigation covers the most practicable method for the recovery of the antimony values in a high grade concentrate with a high recovery. From an ore assaying 11·65 per cent antimony a product is obtained assaying 60 per cent–65 per cent antimony with a recovery of 95 per cent of the antimony values. This was demonstrated by a tonnage check test confirming the results of the small scale tests.

Test No. 169

THE RECOVERY OF THE SILVER VALUES IN THE LEACHED CHLORIDIZED RESIDUES OF THE DOMINION REDUCTION CO., COBALT, ONT.,

BY C. S. PARSONS AND C. L. DEWAR

The experimental work on these residues shows that of the various methods tried for the recovery of the silver values, the one which gave the best results was an acid wash, and cyanidation of the washed residues. By this method, 75 per cent to 80 per cent of the silver values were extracted with a consumption of cyanide of about 10 pounds per ton of residues.

Test No. 170

GOLD ORE FROM BEAR RIVER, BEDWELL SOUND, VANCOUVER IS., B.C.,

BY R. K. CARNOCHAN

This investigation covers the methods employed in the treatment of a gold ore carrying metallic sulphides and arsenides of copper, iron, lead, and zinc. Conclusions are drawn as to those methods most applicable to this class of ore.

Test No. 171

THE CONCENTRATION OF THE RADIO ACTIVE MINERALS IN A SHIPMENT OF PEGMATITE FROM KEARNEY, ONT., BY R. K. CARNOCHAN

The report of this investigation shows that very small amounts of radio active minerals are present in this pegmatite, and that it could not be classed as a source of radium.

Test No. 172

THE PRECIOUS METALS AND OTHER VALUES IN THE COPPER-NICKEL ORES OF SHEBANDOWAN LAKE, ONT., BY H. C. MABEE

This is a report of analysis of five samples submitted by Dr. T. L. Tanton of the Geological Survey on the presence of a considerable quantity of the metals of the platinum group, especially palladium, in the copper-nickel-cobalt ores of Shebandowan lake west of Port Arthur, Ont.

Test No. 173

THE RECOVERY OF THE VALUES FROM A RICH SHIPMENT OF GOLD ORE FROM THE CARIBOU MINING DISTRICT, N.S., BY R. K. CARNOCHAN

This is a report on the methods employed in recovering the values from a gold ore, rich in metallics. Other than showing the spotty nature and characteristics of Nova Scotia gold ores, little of value in treating Nova Scotia gold ores can be obtained from the report.

Test No. 174

THE RECOVERY OF THE VALUES FROM THE ORE OF THE E. H. GLADWIN MINE, BEAVER DAM MINING DISTRICT, N.S., BY R. K. CARNOCHAN

This report is a description of the experimental work with the results obtained, on the ore from this mine, with conclusions as to the best methods for the recovery of the values.

Test No. 175

AN INVESTIGATION OF THE TAILING AND ROCK DUMPS AT THE GOLD MINES OF NOVA SCOTIA, WITH A REVIEW OF THE PAST MINING AND MILLING METHODS, THE PRESENT STATUS OF THE GOLD INDUSTRY IN THE PROVINCE, AND SOME SUGGESTIONS FOR THE REVIVAL OF THE INDUSTRY, BY C.S. PARSONS.

This report is the result of a field and laboratory investigation by Mr. C. S. Parsons, conducted during the latter part of the year. Due to the interest being taken in the development of the gold industry of the country, it was thought advisable to determine the possibility of treating the old tailing and rock dumps of the Nova Scotia gold mines, if they contained sufficient values, and at the same time revive interest in the re-opening of the more promising gold mines of the province. Very few of the dumps were found to contain sufficient values to be worked at a profit.

Test No. 176

CONCENTRATION TESTS ON TAILING SAMPLES FROM TWO MILLS OPERATING ON TYPICAL NOVA SCOTIA GOLD ORES, BY C. S. PARSONS AND R. K. CARNOCHAN

These tests were conducted to determine in what state the gold was present in the tailings from the two operating mills; whether it was free and could be amalgamated, or locked up in the sulphides. It was found that in both cases the mills were doing efficient amalgamation, and the gold in the tailings would have to be recovered by other means.

Test No. 177

THE SEPARATION OF DOLOMITE FROM MAGNESITE IN THE GRENVILLE MAGNESITES, BY R. K. CARNOCHAN

This is a report on some further experimental work on the separation of dolomite from magnesite by calcining, slacking, and classification of the slacked calcined product by washing and other means. The best magnesite product obtained contained 7 per cent lime.

Test No. 178

THE CONCENTRATION AND SEPARATION OF THE MINERALS IN THE ORE OF THE STIRLING MINE, RICHMOND CO., CAPE BRETON,

BY C. S. PARSONS AND R. J. TRAILL

The report of this investigation shows a method of concentration to obtain a marketable zinc product. Leaching tests showing the solubility of the zinc and copper in weak acid solution are also given.

Test No. 179

CONCENTRATION TESTS ON LE ROI NO. 2 DUMP ORE, BY C. L. DEWAR

This report covers the tests conducted on the concentration of this ore by tabling and flotation. The tests were made mainly to determine the results of using various flotation reagents on the concentration of Rossland copper-gold ores.

Test No. 180

THE USE OF FLOTATION REAGENTS MANUFACTURED IN CANADA,

BY C. S. PARSONS AND C. L. DEWAR

This investigation was undertaken at the request of Canadian manufacturers of flotation reagents in an endeavour to stimulate the use of Canadian products instead of imported ones. Tests were made on Canadian ores on which flotation was employed for the concentration and separation of the minerals, and a comparison of results was attempted using Canadian manufactured reagents as against the reagents in use. In many cases it was found that reagents manufactured in Canada could be used with equal results, but the chief difficulty lay in introducing a new reagent for one which had already given satisfaction and with which the operators were familiar.

Tests Nos. 181 and 182

THE METALLURGICAL TREATMENT OF CANADIAN PYRRHOTITE ORES BY LEACHING AND ELECTROLYTIC DEPOSITION AND OF CANADIAN PYRITE ORES BY ROASTING, LEACHING, AND ELECTRO-DEPOSITION, WITH THE PRODUCTION OF ELECTROLYTIC IRON AND RECOVERY OF BY-PRODUCTS SUCH AS COPPER, SULPHUR, ETC., BY R. J. TRAILL

The report on this investigation covers the general features of the Eustis and Estelle processes, and the preliminary experimental work on the Smith mine pyrrhotite from the west side of Lake Memphremagog, Que. (Test 181) and on the pyrite from the Eustis Mine, Que. (Test No. 182). After a preliminary study of the research carried out to date and the reports of engineers on the processes, the possibilities of the application of such processes to the large deposits of Canadian pyrrhotites and pyrites were recognized.

Tests Nos. 183 and 184

THE PRECIOUS METAL VALUES IN THE CONCENTRATION PRODUCTS OF THE NICKEL-COPPER ORES OF THE SUDBURY DISTRICT, BY H. C. MABEE

Test No. 183 is a report on the precious metal values in the concentration products from a low grade ore from the Murray mine. Test No. 184 is a report on the precious metal values in the concentration products from the lower grade ore from the No. 3 (Frood) mine. Both these reports show that the precious metal values are concentrated with the nickel and copper in the flotation of these ores.

II

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

W. B. Timm

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

Test No.	Ore or product	Source of shipment	Shipper and address	Weight pounds
128	Platinum sands...	Bullion mine, B.C.....	Geological Survey......................	465
156	Silver-lead........	Arrowhead, B.C......	Mansfield Mining Co., Mansfield, Wash., U.S.A.	220
161	Copper...........	Flin Flon mine........	Mining Corporation of Canada, Ltd., Toronto, Ont.	360
162	Rock salt.........	Malagash, N.S........	Chambers and MacKay, New Glasgow, N.S.	900
163	Graphite.........	Timmins mine, Westport, Ont.	H. P. H. Brumell, Buckingham, Que....	20
164	Fossil resin.......	Coalmont, B.C.......	Coalmont Collieries, Coalmont, B.C.....	400
165	Asbestos..........	Black Lake, Que......	Black Lake Asbestos and Chrome Co., Black Lake, Que.	200
166	Fluorite..........	Wallbridge mine, Madoc, Ont.	Gavin M. Wallbridge, Madoc, Ont.......	2,000
167	Copper-gold......	Kitsalas Mt...........	Kitsalas Mountain Copper Co., Usk, B.C.	100
168	Antimony........	Lake George, N.B....	North American Antimony Smelting Co., Lake George, N.B.	2,154
169	Silver............	Cobalt, Ont...........	Dominion Reduction Co., Cobalt, Ont..	1,500
170	Gold.............	Bear River, Bedwell Sound, Vancouver Is.	J. B. Woodworth, Vancouver, B.C......	200
171	Radium..........	Kearney, Ont.........	Ryan and Mann, Kearney, Ont..........	19,019
172	Nickel-copper.....	Shebandowan lake, Ont.	Dr. T. L. Tanton, Geological Survey....	100
173	Gold.............	Herman Hall mine, Caribou, N.S.	W. R. Hitchcock, Cornwall, Ont........	90
174	Gold.............	E. H. Gladwin mine, Beaver Dam, N.S.	W. R. Hitchcock, Cornwall, Ont........	575
175	Gold.............	Nova Scotia..........	C. S. Parsons, Mines Branch............	4,000
176	Gold.............	Nova Scotia..........	C. S. Parsons, Mines Branch............	20
177	Magnesite........	Grenville Co., Que....	North American Magnesite Co., Calumet Que.	30,000
178	Zinc..............	Stirling mine, Cape Breton Is.	F. M. Connell, Toronto, Ont............	400
179	Copper-gold......	Le Roi No. 2 mine, Rossland, B.C.	Douglas Lay, Rossland, B.C............	100
180	General investigation on flotation reagents—			
181	Iron (pyrrhotite).	Smith mine, Lake Memphremagog, Que.	F. A. Eustis, Boston, Mass..............	400
182	Iron (pyrite)......	Eustis mine, Eustis, Que.	F. A. Eustis, Boston, Mass..............	400
183	Nickel-copper....	Murray mine, Nickelton, Ont.	British America Nickel Corp., Nickelton, Ont.	200
184	Nickel-copper....	No. 3 mine (Frood)..	International Nickel Co., Copper Cliff, Ont.	100

III

REPORTS ON THE INVESTIGATIONS CONDUCTED

Test No. 128

TESTS ON THE RECOVERY OF PLATINUM IN BLACK SANDS, BY THE ZACHERT PROCESS

R. K. Carnochan

A shipment of three bags of black sand concentrate, gross weight 465 pounds, was received on November 14, 1919, at the Ore Dressing and Metallurgical Laboratories. This concentrate was from the Bullion mine on the south fork of the Quesnel river, Quesnel district, B.C., and was obtained from the sluice boxes in cleaning up. The concentrate contains gold and platinum and tests were desired to determine if these metals, especially the platinum, could be recovered.

A head sample was taken from the lot and this gave upon assay 0·41 oz. gold per ton and 0·032 oz. platinum per ton. The platinum is very low. To make a higher grade product for test purposes all the concentrate remaining after the removal of the head sample was ground dry in a small ball mill for two hours and then screened on 100 mesh. This gave:

Product	Weight lbs.	Au oz./ton	Pt. oz./ton
+100	107	0·90	0·105
−100	278	0·10	trace

The +100 is high enough in platinum to make experiments on. Two tests were made on this +100, using the Zachert process which consists of amalgamating in the presence of zinc amalgam, copper sulphate and sulphuric acid.

Test No. 1

A lot of 25 pounds 10 ounces of +100 was screened on 40 mesh and the oversize crushed to pass 40 mesh. In doing this some metallics were obtained on the 40 mesh screen. The −40 mesh was then fed to a small amalgamator, and the tailing from the amalgamator was allowed to flow over amalgamation plates. Zinc amalgam had been made in the proportion of 2 parts of zinc to 1 of mercury and then ground to −200 mesh. Twenty-five grams of this amalgam was mixed with the −40 before feeding to the amalgamator, 2½ grams were put into the amalgamator, and 2½ grams were sprinkled on at the top of the plates. Five hundred pounds of water, to which had been added ¼ pound of copper sulphate and ¼ pound of sulphuric acid, were used to wash the −40 through the amalgamator and over the plates.

The tailings from the plates were dried, weighed, and sampled for assay. The mercury was removed from the amalgamator and plates and put through a shammy to obtain amalgam. The amalgam was retorted to obtain the gold and platinum.

Product	Weight lbs.	Au oz./ton	Pt. oz./ton
Amalgam		15·58	0·006
Metallics		0·43	0·065
Tailing	25·19	0·70	0·070
Loss	0·43		
		16·71	0·141
Feed	25·62	0·90	0·105

The amount of gold in the products is much more than the amount that was in the feed. This is due to some gold remaining on the plates from a previous test on a gold ore. This does not affect the platinum, as no platinum ores had been put over the plates for a long time.

Test No. 2

A lot of 25½ pounds of +100 was treated in the same manner as described under test 1, except that zinc amalgam composed of 9 parts of zinc to 8 parts of mercury was used, and 1¼ pounds of copper sulphate and 1¼ pounds of sulphuric acid were added to the feed water.

Product	Weight lbs.	Au oz./ton	Pt. oz./ton
Amalgam		0·66	0·006
Metallics		0·34	0·056
Tailing	24·00	0·59	0·060
Loss	1·50		
		1·59	0·122
Feed	25·50	0·90	0·105

SUMMARY

The results given by the two tests are very similar.

The gold recovery in test 2 is 62·9 per cent.

The platinum recovery in test 1 is 50·3 per cent and in test 2 it is 50·8 per cent.

Of the platinum in the feed to the amalgamator there was amalgamated 7·9 per cent in test 1, and 9·1 per cent in test 2.

CONCLUSIONS

1. The Zachert process as used does not give good results on the black sand concentrate.

2. Better results might be obtained by grinding the concentrate finer, say to all through 200 mesh, and in such a way that the amalgamation feed would be scoured and the flakes of gold and platinum brightened up, which would make them easier to amalgamate. This scouring can be secured by using stamps, or by grinding wet in a ball mill before amalgamating.

Further tests under these conditions will be made later.

Test No. 156

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

R. K. Carnochan

A shipment consisting of two lots, one of twenty pounds representing what was designated as black sands; and the other of 200 pounds, representing what was designated as yellow sands, was received at the Ore Dressing and Metallurgical laboratories on November 11, 1921, from the Mansfield Mining Company's claims on Trout lake, near Arrowhead, B.C.

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

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

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

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

Experimental test work was conducted to determine what percentage of recovery of the valuable minerals could be obtained by table concentration; by flotation and table concentration; by table concentration and flotation of the tailings after sulphidizing; by table concentration and cyanidation of the table tailings; by amalgamation; by cyanidation and table concentration.

Test No. 1—Table concentration.
Test No. 2—Flotation and table concentration.
Test No. 3—Table concentration and cyanidation.
Test No. 4—Table concentration and cyanidation.
Test No. 5—Amalgamation.
Test No. 6—Cyanidation and table concentration.

Test No. 1—Tabling

A head sample of ore was ground to pass 20 mesh and screened on 40 and 100 mesh screens. This gave:

	Grams
−20+ 40	483
−40+100	340
−100	405
Total	1,228

Each size was tabled separately on a small laboratory Wilfley table. The slimes from the settling boxes produced in tabling all the sizes were allowed to flow into a large settling tank and were collected as a product. All the products from the table were dried, weighed, and sampled. The following table shows the results obtained:

Product	Weight grams	Analysis			Per cent of metal values		
		Au oz.	Ag oz.	Pb %	Gold	Silver	Lead
Concentrate—							
− 20+ 40	24	0·56	13·22	33·90	2·03	5·24	15·05
− 40+100	13	3·62	18·44	46·50	7·36	3·97	11·17
−100	12	9·80	55·24	47·20	18·46	10·97	10·47
Tails—							
− 20+ 40	431	0·22	2·48	1·80	14·87	17·68	14·35
− 40+100	289	0·28	3·68	2·70	12·68	17·45	14·42
−100	182	0·60	5·32	4·20	17·06	16·01	14·13
Slimes	88	0·64	6·70	5·80	8·76	9·76	·43
Slimes loss	189	0·63	6·05	3·14	18·78	18·92	10·98
Heads	1,228	0·52	4·92	4·40	100·00	100·00	100·00

Total recoveries—	Per cent
Gold	27·9
Silver	20·2
Lead	36·7

The results of this test showed that this treatment must be supplemented by other methods to recover more of the values in the ore.

Test No. 2—Flotation and tabling

A head sample of 1,000 grams at −20 mesh was ground with suitable oils in a small ball mill until about 75 per cent would pass 200 mesh. It was then floated in a small Ruth flotation machine, the concentrates being re-run to clean them up. The flotation tailing and middling were mixed and tabled in the manner described in test No. 1. All products were dried, weighed, and sampled.

Product	Weight grams	Analysis			Per cent of metal values		
		Au oz.	Ag oz.	Pb %	Gold	Silver	Lead
Flotation concentrate	56	4·50	19·96	23·60	48·46	22·73	30·00
Table "	15	3·81	4·58	25·10	10·96	1·40	8·64
" tailing	539	0·10	1·54	1·40	10·38	16·87	17·05
" slimes	362	0·08	6·96	5·40	5·58	51·22	44·31
Slime loss	28	4·57	13·68	....	24·62	7·78	....
Heads	1,000	0·52	4·92	4·40	100·00	100·00	100·00

Total recoveries—	Per cent
Gold	59·4
Silver	24·1
Lead	38·6

The results of this test showed that other methods of treatment were necessary to recover more of the valuable mineral in the ore.

Test No. 3—Tabling and cyanidation

A head sample was ground to pass 40 mesh and screened on 100. This gave

	Grams
− 40+100	2,334
−100	2,172

Each size was tabled separately and the slimes produced were put with the −100 tails. The tailings from each size were cyanided separately. The table shows the results obtained. In figuring the table out the assumption was made that the slime loss ran the same in gold and silver as the −100 tailing.

Product	Weight grams	Analysis			Per cent of metal values		
		Au oz.	Ag oz.	Pb %	Gold	Silver	Lead
+100 concentrate	113	4·32	17·04	43·20	20·83	8·68	24·61
−100 “	88	7·60	21·46	44·00	28·55	8·52	19·52
+100 tails	2,062	0·05	2·35	2·10	4·40	21·86	21·83
−100 “	1,673	0·06	3·74	3·90	4·27	28·22	32·88
Slime loss	570	0·06	3·74	0·40	1·45	9·62	1·16
Cyanided	......	......	......	......	40·50	23·10	......
Heads	4,506	0·52	4·92	4·40	100·00	100·00	100·00

Total recoveries—	Per cent
Gold	89·9
Silver	40·3
Lead	44·1

Test No. 4—Tabling and cyanidation

A head sample of 1782 grams −100 mesh was tabled and the tailing and slime produced were mixed and cyanided.

Product	Weight grams	Analysis			Per cent of metal values		
		Au oz.	Ag oz.	Pb %	Gold	Silver	Lead
−100 concentrate	75	4·72	21·38	33·70	38·19	18·28	32·27
−100 tails	1,344	0·03	1·90	2·28	4·31	29·13	39·03
Slime loss	363	0·03	1·90	6·20	1·19	7·78	28·70
Cyanided	......	......	......	......	56·31	44·72	......
Heads	1,782	0·52	4·92	4·40	100·00	100·00	100·00

Total recoveries—	Per cent
Gold	94·5
Silver	63·0
Lead	32·3

Test No. 5—Amalgamation

A head sample of 1,000 grams −40 mesh was amalgamated for 3 hours in a small pebble jar, after which it was panned to recover the mercury. The tailings were dried and sampled. An analysis of these tailings showed them to contain as much gold and silver as the heads, hence, amalgamation on this ore is not possible.

Test No. 6—Cyanidation and tabling

A head sample of 1,000 grams −100 mesh was cyanided and then tabled with the following results:

Tails from cyaniding—	Ozs. per ton
Gold	0·15
Silver	4·10

Recovery by cyaniding—	Per cent
Gold	71·1
Silver	16·7

Tabling—

Product	Weight grams	Analysis			Per cent of metal values		
		Au oz.	Ag oz.	Pb %	Gold	Silver	Lead
Table concentrate	20	3·28	11·60	40·70	51·56	6·62	20·84
Table tails	468	0·10	2·34	2·60	36·72	31·24	31·15
Table slime	245	0·04	4·48	5·40	7·81	31·32	33·86
Slime loss	122	0·04	8·85	4·53	3·91	30·82	14·15
Feed	855	0·15	4·10	4·57	100·00	100·00	100·00

Recoveries by tabling—		
Gold	51·6 per cent of feed to table;	14·9 per cent of heads
Silver	6·6 " " "	5·5 " "
Lead	20·8 " " "	20·8 " "

Total recoveries—	Per cent
Gold	86·0
Silver	22·2
Lead	20·8

Consumption of cyanide and lime.—In running test No. 6, tests were made of the cyanide solution at different times to determine the amount of chemicals consumed.

The original solution titrated 0·184 per cent KCN, and this fell to 0·168 per cent KCN at the end of the cyaniding.

In the test the ratio of solution to ore was six to one. This gives a consumption of 1·92 pounds of KCN, or its equivalent, per ton of ore.

The original solution titrated 0·086 per cent CaO and at the end of the test, 0·023 per cent CaO. This shows a consumption of 7·56 pounds CaO per ton of ore.

SUMMARY

The recoveries obtained from the various tests are given below:—

Test No.	Initial crushing mesh	Metallurgical treatment	Recoveries of metals, per cent		
			Gold	Silver	Lead
1	20	Table concentration	27·9	20·2	36·7
2	200	Flotation and table concentration	59·4	24·1	38·6
3	40	Table concentration and cyanidation	89·9	40·3	44·1
4	100	Table concentration and cyanidation	94·5	63·0	32·3
5	40	Amalgamation	nil	nil	nil
6	100	Cyanidation and table concentration	86·0	22·2	20·8

CONCLUSIONS

Of the various methods tried for the treatment of this ore, it would seem that the one most applicable is table concentration followed by cyanidation of the table tailings. Test No. 4 shows that by this method a concentrate is obtained assaying, gold, 4·72 ozs.; silver, 21·38 ozs.; lead, 33·7 per cent; and that by cyaniding the table tailings the greater proportion of the remaining gold values are extracted. The total recoveries from this operation are—gold, 94·5 per cent; silver, 63·0 per cent; lead, 32·3 per cent.

In practice, with careful manipulation of machinery, and the working out in detail, the treatment along the lines of procedure of test No. 4, which was conducted on a very small scale, improvement should be made in the grade of the concentrate and the recoveries, as indicated above. On similar ore, as submitted to the department for test purposes, it is safe to assume that 95 per cent of the gold, 65 per cent of the silver, and 45 per cent of the lead values would be recovered.

Test No. 161

THE CONCENTRATION OF THE FLIN FLON DISSEMINATED ORE

C. S. Parsons

In the Summary Report of the Mines Branch for 1921, the results of an investigation on the treatment of the sulphide ore of the Flin Flon ore body was given under test No. 142 of the Ore Dressing and Metallurgical Division. During 1922 an investigation was conducted on the concentration of the disseminated ore of this same ore body, and several small shipments were received from the Mining Corporation of Canada for this purpose.

Dates of shipments.—130 pounds of ore was received February, 1922, and a further shipment of 230 pounds in January 1923.

Analysis of Shipments

—	Cu per cent	Fe per cent	Zn per cent	Au oz.	Ag oz.
Shipment No. 1	2·55	20·05	....	trace	0·33
Shipment No. 2	2·72	....	0·49	0·01	0·65

Purpose of investigation.—The purpose of the experimental work was to determine if the ore was amenable to concentration, with the production of a concentrate in as coarse a form as possible, with a high recovery of the copper values. Pyritic smelting of the sulphide ore having been determined as the better practice, the concentrate from the disseminated ore could be mixed with it and bring up the grade of the furnace charge.

Lines of investigation.—On an ore of this class naturally the first line of investigation would be to determine if a copper concentrate could be produced by jigging, removing as much of the copper values as possible in a coarse form. This would be followed by tabling and flotation for the recovery of the remaining copper values more intimately disseminated in the gangue or floating off with the slimes.

If it were found that the copper values were not freed by crushing to a size suitable for jigging, the ore would be further reduced for table concentration, followed by regrinding and flotation for the concentration of finer particles.

If it were found that a table concentrate could not be obtained after crushing to a size suitable for table work, the next line of investigation would be to grind the ore to a fineness at which the copper minerals were free with the production of a copper concentrate by flotation.

Microscopic examination of the disseminated ore.—The microphotographs shown are from a microscopic examination of the ore by Prof. Alfred Wandke, whose report of his investigation brings out the following:—"The sulphide minerals are pyrite, chalcopyrite, and sphalerite. The gangue minerals are calcite, quartz, sericite, and chlorite. Pyrite is the chief sulphide. Chalcopyrite is the chief or only commercial copper mineral. Sphalerite is present, but in much smaller amounts than in the sulphide ore of the same ore body. Aluminous silicates such as chlorite, and sericite, are the chief gangue minerals. It would appear that the disseminated ore would be amenable to a simpler flotation process for the recovery of the copper." This was found to present a more intricate problem, as the results of the experimental test work following will show, due to the presence of deleterious gangue. This could not, however, be foreseen from the microscopic examination.

Experimental Tests

JIGGING

To determine if a coarse concentrate could be obtained from the ore by jigging, a test was made on $\frac{1}{4}''$ material. This test showed conclusively that jigging was not practicable. The concentrate obtained was very low in grade, consisting of all the sulphides. The tailing contained a large amount of unfreed copper mineral.

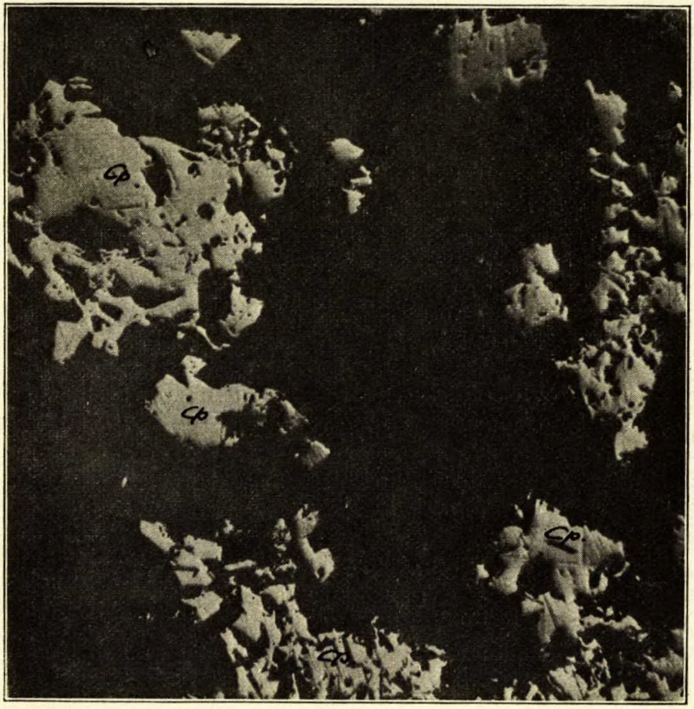
TABLE CONCENTRATION OF THE SIZED MATERIAL

Test No. 1

Object of test. The object of this test was to determine if tabling could be used in conjunction with flotation. It was of course recognized that flotation would have to be used on the fine sizes.

Procedure. 7,000 grams crushed to pass 20 mesh by graded crushing, so as not to produce an excessive proportion of fines. The material was then sized and each size tabled separately. The results are recorded in the accompanying table.

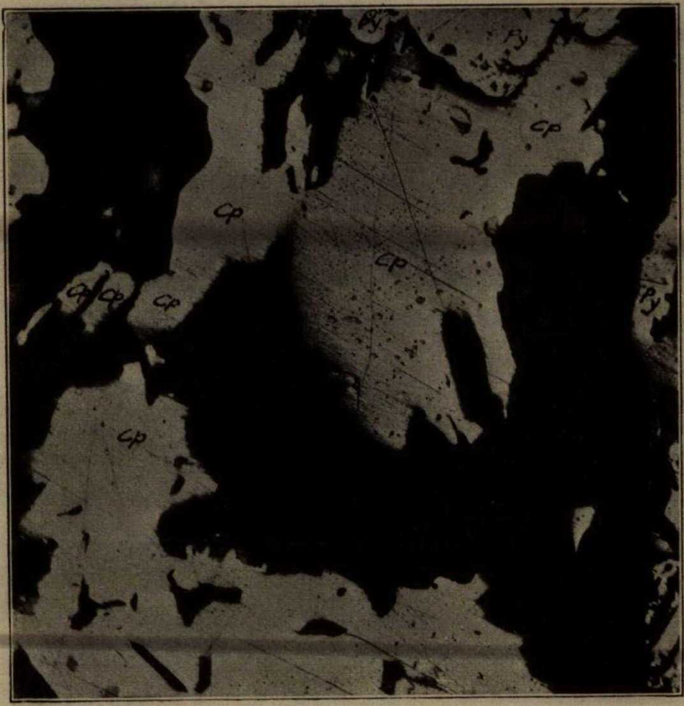
PLATE II.



Flin Flon disseminated ore, x 100

Black is chlorite gangue

PLATE III.



Flin Flon disseminated ore, x 100

☐ 200 mesh

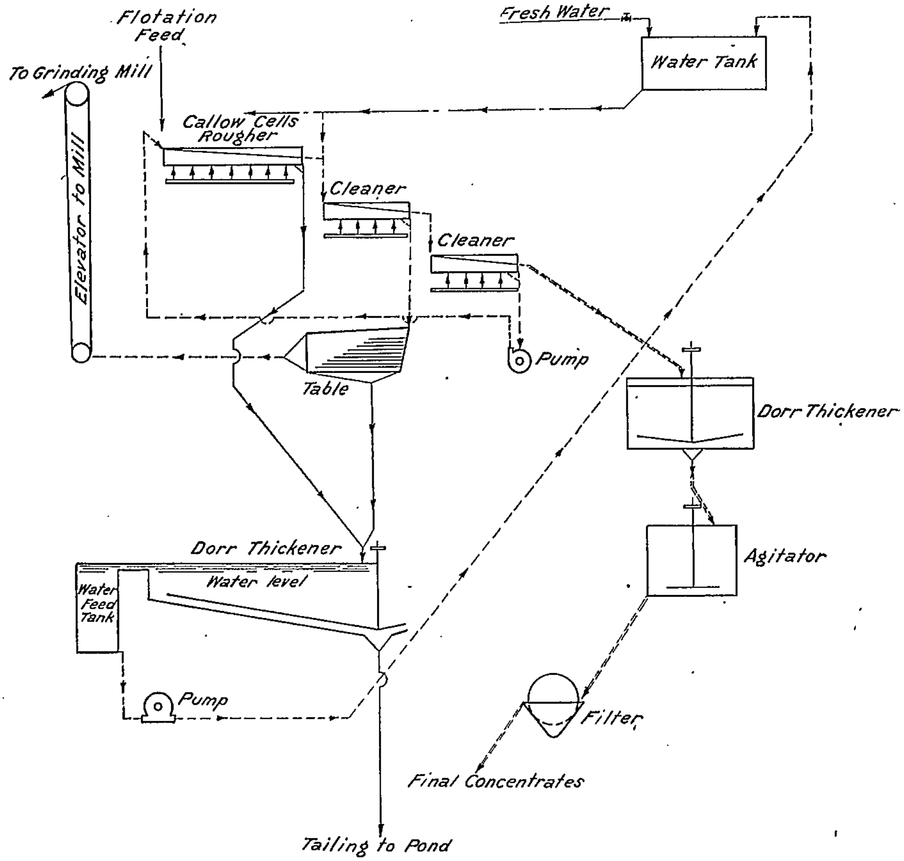


FIG. 4. Flin Flon disseminated ore.

Observations from table:—

−20 + 28. The separation was very poor, and no distinct line of sulphides was obtained.

−28 + 35. The separation was no better.

−35 + 48. The separation of the sulphides showed a marked improvement on this size. The proportion of middling obtained was smaller. The middling was re-run over the table but no separation could be obtained, showing the product was a true middling. The middling was added to the tailing.

−48 + 65. The separation of this size was good. Both the tailing and concentrate looked clean. Very little middling was obtained, and in order to have only two final products the middling was re-run, making a concentrate and tailing, which were added to their respective products obtained from the first run.

−65 + 100. The separation was better than in the previous size.

Test No. 1

Mesh	Heads				Concentrate			Tailing			Recovery per cent
	Weight		Analysis Cu per cent	Weight grams x assay per cent	Weight grams	Analysis Cu per cent	Weight grams x assay per cent	Weight grams	Analysis Cu per cent	Weight grams x assay per cent	
	Grams	Per cent									
+ 28	933	14·3	1·95	18·19	379	2·70	10·23	614	1·75	10·74	56·2
+ 35	1,293	18·6	2·35	30·38	325	3·55	11·54	976	1·85	18·06	37·98
+ 48	920	13·2	2·25	20·70	240	4·15	9·96	680	1·81	12·31	48·11
+ 65	665	9·5	2·55	16·96	252	4·60	11·59	413	1·35	5·57	68·33
+100	659	9·5	2·85	18·78	294	5·35	15·73	365	0·95	3·47	83·75
+150	498	7·1	2·90	14·44	235	5·55	13·04	263	0·70	1·84	90·30
+200	563	8·1	3·25	18·30	200	5·35	10·70	363	1·90	6·90	58·47
−200	1,369	19·7	2·85	39·02	....	....	....	....	....	....	....

$-100 + 150$. The separation looked extremely good, but considerable float material was observed going with the tailing. The tailing was, therefore, re-run in order to see what would table out. A distinct copper sulphide line was obtained which was practically pure chalcopyrite. The loss of copper by flotation in tabling this size would be considerable.

$-150 + 200$. The separation on the table was excellent. The sulphides separated into two lines. The sulphide band nearest the gangue line was distinctly higher in copper than the band of sulphides higher up on the table. The tailing was re-run with the same result as in the $+150$ mesh size. The tailing looked very clean and free from sulphide, but a glance at the table compiled on the test shows that the recovery was low, and that the tailing contained 1·90 per cent copper.

FLOTATION TESTS

The next series of tests made were by flotation. The first few tests were run using coal tar oils and fuel oils. The tests with these oils were unsuccessful. Nothing but a voluminous froth was obtained, consisting of fine flaky gangue slimes. All manner of reagents were tried to hold down these slimes and to coagulate them, but with no success. A series of experiments in the chemical laboratories showed that lime had the greatest effect on the coagulation and settlement of them. It was necessary, therefore, to turn our attention to some form of flotation reagent other than oil. The Callow alphabetical reagents were then tried, and the first test made with their thio-fizzan in an alkaline pulp with lime gave the result as indicated in test No. 2 (see table, flotation tests, page 103). This result was remarkably good. The copper floated so readily that it held down the slime. The middling, however, consisted chiefly of the deleterious gangue slimes.

GRAVITY TEST

It was apparent that the talc slime interfered with the flotation. A series of experiments was conducted in an endeavour to reduce the quantity present to a point where it would not interfere with flotation. This deleterious gangue slime seemed to be of a talcy nature or similar to ground up soapstone. It consisted of small flakes which had a greasy or soapy feeling between the fingers. A complete analysis showed these flakes to be of the following chemical composition:—

	Per cent		Per cent
SiO_2	50·0	FeO	11·8
Al_2O_3	3·9	CaO	5·5
MgO	18·8	H_2O	9·5

The gangue, due to the above characteristics, did not settle readily. The following series of gravity tests were run to determine if this property of the gangue could be utilized to separate it from the rest of the ore.

TABLE TESTS

Test No. 3

Object of test:—To determine if the deleterious gangue slimes could be washed out by classification on a Wilfley table and a sand product produced which would be reground and treated by flotation.

Procedure:—Crushing to 20 mesh and tabling without sizing on a small laboratory table. The results are tabulated in the following tables:—

Products	Weight grams	Per cent by weight	Assay per cent Cu	Weight x assay	Per cent of copper values		Remarks
					by feed assays	by product assays	
Concentrate..	885	22·1	4·15	36·73	37·0	36·02	These products re-treated by flotation.
Middling......	1,165	29·1	2·60	30·30	30·5	29·70	
Tailing.......	1,265	31·6	1·70	21·50	21·6	21·07	
Average....	3,315	82·8	2·67	88·53	....	86·79	
Froth.........	119	3·0	2·50	2·97	3·0	2·90	13·21 per cent of Cu lost in these products.
Slimes........	317	7·9	2·48	7·86	7·9	7·73	
Loss..........	249	6·3	1·06	2·64	....	2·58	

Flotation test on above three products, concentrate, middling and tailing, using 1,000 grams of the mixed products

Product	Weight grams	Weight per cent	Assay per cent Cu	Weight x assay	Per cent of Cu values	Remarks
Concentrate............	201	19·90	11·53	23·17	85·7	NOTE. This test was only run on 1,000 grams of the total 3,315 grams.
Middling...............	160	15·90	1·00	1·60	5·9	
Tailing.................	647	6·42	0·35	2·26	8·4	

Recapitulation

Product	Weight grams	Weight per cent	Assay per cent Cu	Weight x assay	Per cent of Cu values	Remarks
Concentrate...	662·5	16·5	11·53	76·5	75·0	The recovery is the total recovery of the test.
Tailing........	3,337·5	83·5	0·76	25·5	25·0	

A loss of 13·21 per cent of the total copper was accounted for in the froth, slimes, and loss in treatment. The froth was the collection of this talc gangue material which floated in the settling tank. The loss is the discrepancy between the amount of material fed, and that recovered in the products.

The recovery in the table sand products, namely, concentrate, middling, and tailing, in the above table was 86·79 per cent of the total copper.

Flotation.—These three sand products were mixed together and a sample weighing 1,000 grams cut out for a flotation test. The ore was ground wet in a ball mill and the test conducted as in test No. 2, using lime and thio-fizzan. A recovery of 85·7 per cent was obtained in the concentrate alone without considering the middling.

Recapitulation.—The recapitulation of the complete test shows that a total recovery of only 75 per cent of the copper was obtained in concentrate assaying 11·5 per cent copper and representing 16·5 per cent by weight of the original ore.

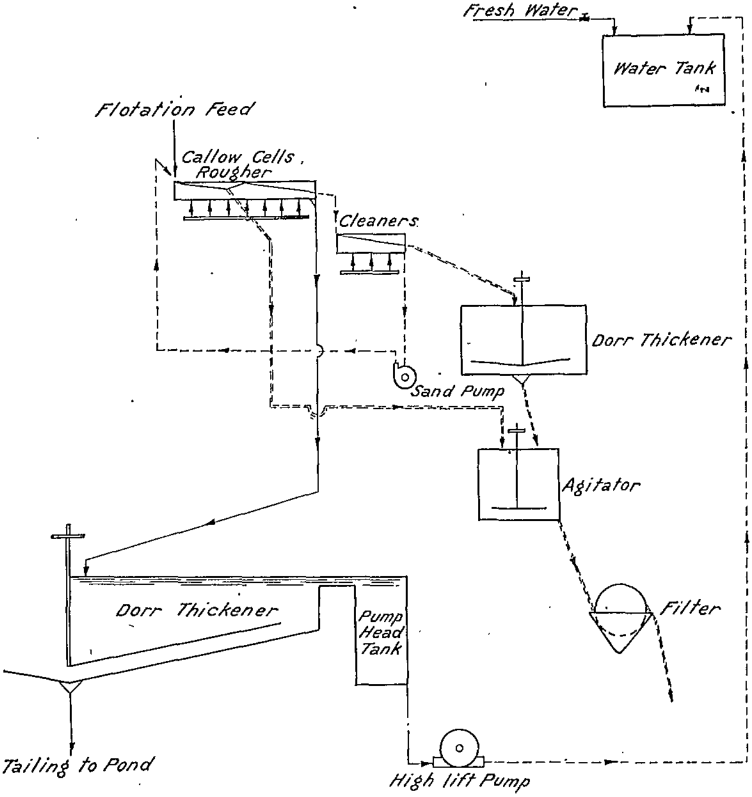


FIG. 5. Flin Flon disseminated ore.

Test No. 4

Object of test.—It was observed in test 3 that considerable fine chalcopyrite floated over with and reported in the sand tailing which otherwise looked clean. The sand tailing from this test was screened to determine if a portion of the sand could be eliminated at this stage as a final tailing.

Procedure.—A 1,000 gram sample was dry crushed in rolls to 20 mesh and tabled without further sizing. A concentrate, middling, and sand tailing was produced together with a slime product which consisted chiefly of the deleterious talcy gangue. The sand tailing was sized on 65 and 100 mesh screens and each product assayed for copper.

It is evident from a glance at the following table that all the sized products contained too high a content of copper to be discarded.

Test No. 4

Product	Weight grams	Weight per cent	Analysis Cu per cent	Weight grams x assay per cent	Per cent of total Cu in heads
Concentrate	255	25·5	4·15	10·58	40·5
Middling	330	33·0	2·35	7·76	29·7
Tailing − 20+ 65	148	14·8	1·30	1·92	7·3
" − 65+100	16	1·6	2·10	0·34	1·3
" −100	101	10·1	2·55	2·58	9·9
Slimes	135	13·5	2·18	2·94	11·3
Table froth	7	0·7	....	....	....
Loss	8	0·8	....	....	....

Test No. 5

Object of test.—A table test was made to determine if a low grade sand tailing could be produced which would not have to be retreated.

Procedure.—A sample weighing 4,000 grams was crushed to pass 20 mesh and sized as shown in the following table. Each size was tabled separately with the exception of the −200 mesh material which was not treated.

Conclusions.—The results show that a clean sand tailing cannot be produced by tabling.

Test No. 5—Run to make clean tailing

Mesh	Heads			Concentrate				Middling				Tailing			
	Weight		Calcul. analysis Cu per cent	Weight grams	Analysis Cu per cent	Weight x assay per cent	Re-covery per cent	Weight grams	Analysis Cu per cent	Weight x assay per cent	Per cent of Cu values	Weight grams	Analysis Cu per cent	Weight x assay per cent	Per cent of Cu values
	Grams	Per cent													
+ 28	731	14·91	2·51	193	3·78	7·3	39·8	374	2·43	9·1	49·5	164	1·20	2·0	10·7
+ 35	1,062	21·64	2·49	379	3·75	14·2	53·7	503	2·08	10·5	39·5	180	1·00	1·8	6·8
+ 48	653	13·32	2·64	147	4·15	6·1	35·3	331	2·80	9·3	53·7	175	1·08	1·9	11·0
+ 65	441	8·99	2·84	100	4·58	4·6	36·6	194	3·30	6·4	51·1	147	1·05	1·5	12·3
+100	415	8·05	3·12	95	4·85	4·6	35·5	172	3·60	6·2	47·7	148	1·47	2·2	16·8
+200	643	13·11	3·38	164	5·10	8·4	38·4	191	3·95	7·5	34·7	288	2·03	5·8	26·9
−200	906	18·48	....	....	....	....	....	....	....	....	....	....	....	....	....
Table slimes	52	1·06	2·60	....	....	....	....	....	....	....	....	....	....	....	....

Test No. 7

Object of test.—This test was to determine if the objectionable talcy material could be eliminated in a tailing without excessive loss of copper. It was thought that perhaps wet crushing would free this talcy gangue material better than dry crushing which had been used in the preceding tests.

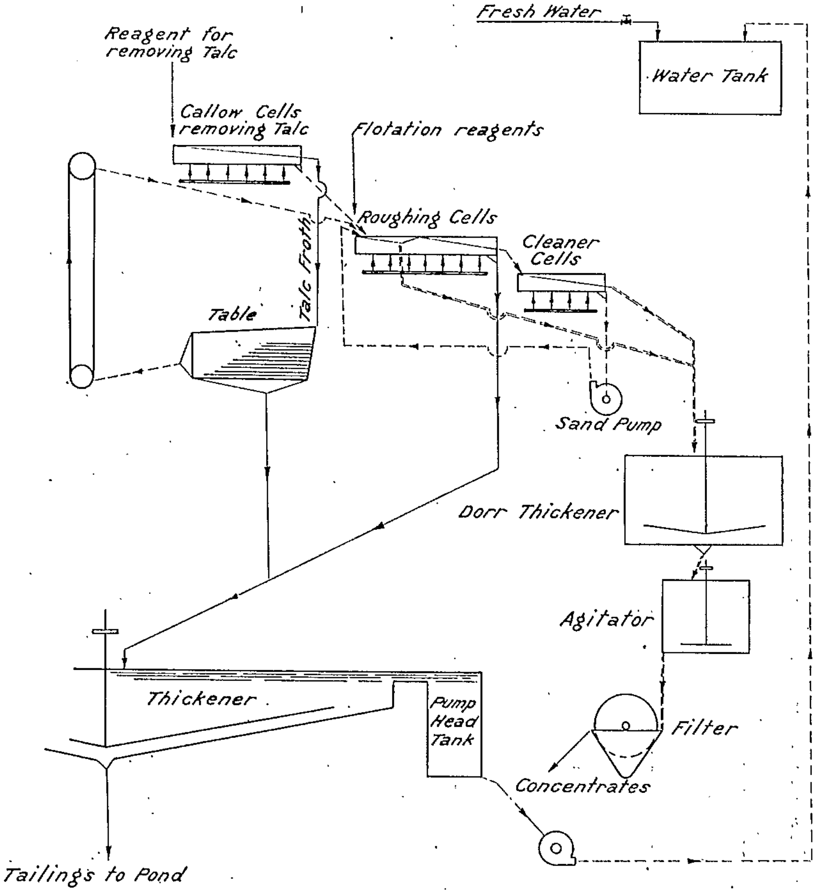


FIG. 6. Flin Flon disseminated ore.

Procedure.—A 1,000 gram sample of ore was prepared by dry crushing to pass 20 mesh and then placed in a wet ball mill for a five-minute period for further crushing. This crushed product was then tabled. The results are recorded in the accompanying table. The slimes contained 12·6 per cent of the total copper, and from observation it was noted that the sand tailing still contained considerable of the talcy material.

Product	Weight grams	Weight per cent	Analysis Cu per cent	Weight x assay per cent	Per cent of Cu values
Concentrate	268	26·8	5·10	13·7	51·1
Middling	145	14·5	4·10	5·9	22·2
Tailing	400	40·0	0·94	3·8	14·1
Slime	·187	18·7	1·80	3·4	12·6

Test No. 8

Object of test.—Same as in test No. 7.

Procedure.—The ore was not crushed as fine as in the preceding test in the hope that less copper would be lost in the slimes. The loss of copper in the slimes containing the deleterious talcy material was approximately 4 per cent less than in test No. 7.

Product	Weight grams	Weight per cent	Analysis Cu per cent	Weight x assay per cent	Per cent of Cu values
Concentrate	393	39·3	5·00	19·6	73·8
Tailing	461	46·1	1·00	4·6	17·3
Slime	131	13·1	1·78	2·3	8·8
Loss	15	1·5	....	....	....

ELIMINATION OF OBJECTIONABLE TALCY MATERIAL BY HYDRAULIC CLASSIFICATION

A series of tests, Nos. 10, 11, 12, 13, 18, 23, 24 and 25, were made using hydraulic classification after comparative coarse crushing to eliminate the slimes interfering with flotation.

Summary of results.—The products showing the results of these tests are tabulated in tables following. The variation in the amounts of slime removed is due to the difficulty in controlling the quantity of slime overflow from the Richards hydraulic classifier.

For instance, in test 10, too large a quantity of slime was taken off, approx. 44·1 per cent. The amount separated as slimes in the other tests varied between 12 per cent and 18 per cent of the total feed. The variations in the total recovery of copper in concentrate is due to erratic results from flotation. The reason for this variation has not been determined. The flotation results are apparently more erratic when the slimes are removed than when they are present in the flotation pulp.

Test No. 10

Object of test.—To determine if stage crushing with classification after each reduction would produce a sand product free from the objectionable slimes.

Procedure.—1,000 grams of ore $-\frac{3}{8}''$ crushed for 5 minutes in small ball mill and resulting product classified by panning. This was repeated twice. The sands were reground to 100 mesh for flotation and the separated slimes tabled. The reagents used for flotation were thio-fizzan and lime.

Test No. 11

Object of test.—The elimination of the deleterious slime by crushing and classification in order to produce a product suitable for flotation.

Procedure.—1,000 grams of ore $-\frac{1}{4}''$ crushed in wet ball mill for 5 minutes and the sands and slime of resulting product separated in Richards classifier. The sands reground to 100 mesh for flotation. Reagents used for flotation were thio-fizzan and lime.

Test No. 12

A duplicate of test No. 11, with the exception that the separated slimes were tabled making concentrate and tailing.

Test No. 13

A duplicate of tests 10 and 11, with the exception that the slimes from classification were treated by flotation in high pulp dilution. Note: The reagents used for tests 12 and 13 were thio-fizzan and lime.

Tests Nos. 18, 23, 24 and 25

These tests were run as duplicate check tests.

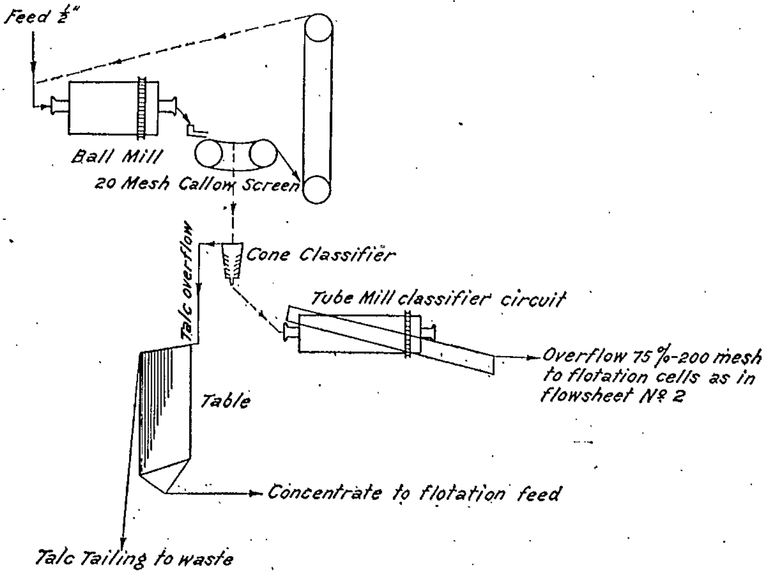


FIG. 7. Flin Flon disseminated ore.

Test No.	Product	Weight		Copper		Per cent of values	Remarks
		Grams	Per cent	Per cent	Weight x assay		
10	Table slimes....	208	20·8	1·68	3·49	13·7	
	“ tailg......	158	15·8	1·07	1·69	6·6	
	“ conc......	75	7·5	4·80	3·60	14·1	
	Flot. tailg......	259	25·9	0·52	1·34	5·3	
	“ midd.....	71	7·1	0·88	0·62	2·4	
	“ conc......	212	21·2	6·95	14·75	57·8	
	Loss............	12	....	....	....	....	
11	Flot. conc......	125	12·5	13·78	17·22	67·3	
	“ midd.....	98	9·8	2·00	1·96	7·7	
	“ tailg......	595	59·5	0·53	3·15	12·4	
	Slime tailg......	135	13·5	1·05	1·43	5·6	
	Loss............	47	4·7	....	....	6·9	(Estimated.)
12	Flot. conc......	163	16·3	11·16	18·19	76·8	Recovery in table concentrate and flotation concentrate, 82·4 per cent
	“ midd.....	147	14·7	1·35	1·98	8·4	
	“ tailg......	517	51·7	0·30	1·55	6·5	
	Table conc......	25	2·5	5·30	1·33	5·6	
	“ tailg......	84	8·4	0·75	0·63	2·7	
	Loss............	64	6·4	....	....	....	
13	Flot. conc. No. 2	214	21·4	9·92	21·33	81·7	Grade of concentrate from two concentrates, 8·71 per cent Cu.
	“ “ No. 1	45	4·5	2·74	1·23	4·7	
	Total conc......	259	25·9	8·71	22·56	86·4	
	Flot.Midd. No.2	174	17·4	1·12	1·94	7·4	
	“ tailg. No. 2	492	49·2	0·27	1·33	5·1	
	“ tailg No. 1	32	3·2	0·88	0·28	1·1	
	Loss............	43	4·3	....	....	....	
23c	Flot. conc. No. 1	103	10·3	12·07	12·44	49·0	Ore crushed dry ¾″; wet in ball mill to 6 mesh; after separating in classifier sands reground to 65 mesh and floated. Slimes floated in very dilute pulp. KK oil used in this test.
	“ “ No. 2	85	8·5	4·57	3·89	15·3	
	“ midd......	134	13·4	2·96	3·97	15·6	
	“ tailg. No. 1	474	47·4	0·75	3·55	14·0	
	“ “ No. 2	168	16·8	0·92	1·55	6·1	
	Loss............	36	3·6	....	....	....	
24c	Flot. conc. No. 1	141	14·1	13·55	19·25	74·0	} 81 per cent.
	“ “ No. 2	37	3·7	4·92	1·82	7·0	
	“ midd......	155	15·5	1·57	2·43	9·4	As in test 23c.
	“ tailg. No. 1	542	54·2	0·32	1·73	6·7	
	“ “ No. 2	101	10·1	0·75	0·76	2·9	
	Loss............	24	2·4	....	....	....	
25c	Flot. conc. No. 1	106	10·6	15·40	16·39	66·9	
	“ midd. No. 1	117	11·7	2·12	2·38	9·7	
	“ tailg. No. 1	563	56·3	0·40	2·25	8·8	As in test 23c.
	“ conc. No. 2 (Re-run classifier fines).....	38	3·8	5·66	2·44	10·0	
	Flot. tailg. No. 2	126	12·6	0·87	1·10	....	
	Loss............	50	5·0	....	....	....	
18	Flot. conc. No. 1	101	10·1	17·50	17·33	65·0	
	“ midd. No. 1	105	10·5	2·47	2·35	8·8	
	“ tailg. No. 1	654	65·4	0·77	5·04	18·9	As in test 23c.
	“ conc. No. 2 (Re-run classifier fines).....	19	1·9	6·86	1·30	4·9	
	Flot. tailg. No. 2	73	7·3	0·90	0·65	2·4	
	Loss............	48	4·8	....	....	....	

CONTINUATION OF FLOTATION TESTS

It had already been found that oils could not be used. With oils, such large quantities of the talc froth are formed that it prevents the copper from floating.

Two methods of procedure were next tried. The first, straight flotation of the ore using alphabetical reagents, and the second, removing the talc by a preliminary flotation with a frothing reagent.

The first method is the simpler process, but the talc interferes seriously in the production of a high grade concentrate. The second method gives satisfactory results in the laboratory, but the possibility of certain difficulties in the application of this method to practice can be foreseen. One of the chief difficulties would be that of preventing this part of the flotation circuit from becoming contaminated by collecting oils, such as lubricating oils, etc., which would cause large losses of copper in the froth product. The froth produced is also sticky and difficult to handle.

Chemical reagents.—It has been previously mentioned that lime was found to be the most satisfactory reagent to hold down the talc slimes. A large number of salts were tested in the laboratory and their effect on the coagulation and settlement of this talc material compared. Lime was found to coagulate and settle these slimes more rapidly than any other reagent tried.

What actually happens when flotation of the copper is attempted in the presence of the talcy substances, is that the material being in the form of flat thin flakes, presents a large surface area, and is present in relatively large quantities compared to the copper, and so completely armours the bubbles and excludes the chalcopyrite from their interfacial boundaries.

Removal of talc by preliminary flotation.—Referring to the second scheme for removing the talc by the use of a frothing reagent which had no collecting action on the copper, we found that Fumol No. 6, a product of the Canadian Electro Products Co., Shawinigan Falls, Que., gave the most satisfactory results. It is possible that other similar reagents might give improved results. This phase of the problem was discussed with the chemists of the Canadian Electro Products Co., who undertook to prepare some separate samples. Nine samples of Aldol and mixtures of Aldol and Fumol products were prepared. We suggested the mixing of sulphur in some of their products and three such samples were prepared. Duplicates of all these samples with full particulars as to how they were made were kept by the company. Sample No. 22a is Aldol saturated with SO_2. No. 24a is Fumol saturated with SO_2, and No. 27 is Aldol saturated with H_2S. None of the nine samples gave any better results than the standard Fumol No. 6. For reference see tests Nos. 44 to 54.

Size of crushing.—The removal of this deleterious gangue by flotation introduced the question that perhaps the size of crushing had an important effect on the results. A series of tests were run, the ore being crushed to different sizes. When the ore was crushed through 100 mesh, and when approximately 75 per cent would pass 200 mesh, the best condition for flotation was obtained both as regards the removal of the deleterious gangue and the subsequent recovery of the copper.

General Procedure of Tests for Flotation.—The ore was crushed by dry crushing in rolls to 20 mesh. A 1,000 gram lot was then cut out for each test and placed in a small ball mill. If lime was used it was added in the ball mill. Experience has shown that this is the only satisfactory place to add the lime, as it is important for the lime to be in contact with the ore for considerable time previous to flotation. A pulp, one of solids to one of water, was made up and the charge ground for the required time to produce the desired size of product. The flotation tests unless otherwise mentioned were conducted in the Ruth laboratory machine. We find this the simplest machine to operate for test work. If the talc was to be removed by preliminary flotation, the frothing reagent was added in the cell, and after the bulk of the talc had been removed, the reagent for floating the copper was added.

TESTS BY GENERAL ENGINEERING CO.

A small sample of ore was sent to the General Engineering Co., who tried two methods of treatment to check our results. An abstract from their report reads as follows:—

> "Two methods of treatment were tried. The first by straight flotation with a frothing reagent, tabling the talc product so obtained, to remove any of the copper which floated with it, returning the table concentrate to the flotation circuit, and floating the pulp from which the bulk of the talc has now been removed for copper.
>
> "The first method is much the simpler flow sheet, but the concentrate produced is lower grade. The second method gives higher grade concentrate and should show about the same recovery, although I was not able to do this. I can offer no explanation as to why as much copper did not float after removing the talc as before removing it.
>
> "On the whole my tests by straight flotation show a slightly better tailing and recovery, and a lower grade concentrate than theirs (Mines Branch), but their recovery when removing the talc first is better than mine. About 8 pounds per ton lime seems to give the proper alkalinity for either method. In the straight flotation tests this is ground with the ore. When the talc is removed first, the lime is added to the pulp after the talc flotation. Grinding 1%+100 mesh and 80%−200 mesh gave as good results as 95% −200 mesh. T.T. mixture and thio-fizzan gave about the same results. Grinding thiocarbanilid with the ore gave no results."
>
> "The accompanying tables show the results of the best tests by each method. These tests were made in the Callow laboratory testing cells. In the operation of the cells, the middling from the cleaner cell is returned to the rougher cell and retreated."

RESULTS OF TESTS BY GENERAL ENGINEERING CO.

Ground with 8 lbs. lime, floated with ½ lb. of thio-fizzan.

Product	Per cent weight	Per cent copper	Per cent recovery
Concentrate	21·9	10·48	87·0
Tailing	78·1	0·44	13·0

Floated with $\frac{1}{20}$ lb. pine oil to remove talc, tabled talc product, added this table concentrate to the pulp remaining from talc flotation, floated with 8 lbs. lime, $\frac{4}{10}$ lb. TT mixture and $\frac{1}{12}$ lb. Aldol.

Product	Per cent weight	Per cent copper	Per cent recovery
Flotation concentrate	14·5	14·74	80·1
" tailing	66·5	0·67	16·7
Talc tailing	19·0	0·45	3·2
Combining the two tailings			
Flotation concentrate	14·5	14·75	80·1
Tailing	85·5	0·62	19·8

Same flow sheet and reagents as above except ¼ lb. thio-fizzan replaces TT and Aldol in floating copper. High grade concentrate obtained.

Product	Per cent weight	Per cent copper	Per cent recovery
Flotation concentrate	10·9	18·68	76·2
" tailing	65·5	0·81	19·8
Talc tailing	23·6	0·44	4·0
Combining the two tailings			
Flotation concentrate	10·9	18·68	76·2
Tabling	89·1	0·71	23·8

Tests made in the Callow laboratory cells.—After receiving the report from the General Engineering Co., tests Nos. 38 and 37b were run in our Callow testing unit to check their results. We found that we were still able to obtain a better recovery than they when removing the talc first, but that our recovery by straight flotation is lower than theirs.

Comparison and discussion of results in Ruth and Callow machines.—The recovery in the Ruth machine, under favourable conditions is higher than in the Callow. A 0·25 per cent copper tailing was consistently obtained in the Ruth against a 0·40 per cent or higher tailing in the Callow when using straight flotation. When the talc was removed by preliminary flotation we obtained approximately the same results in both machines.

In the Callow unit the middling from recleaning the rougher concentrate is returned continuously to the rougher cells, thus increasing the amount of talc present in that cell because a large proportion of the middling returned consists of the talc material. The operation of the Ruth is entirely different. A rougher concentrate is made and a final tailing in the first operation. The cell is then cleaned out, after which the rougher concentrate is placed in it and fresh water added to fill the cell. The rougher concentrate is then recleaned by a separate operation producing a final concentrate and a middling product. This middling product is kept separate and not added back to rougher cell, so that the circuit is not further contaminated by the presence of additional quantities of talc.

Accumulative middling test in Ruth machine.—An accumulative middling test was made to determine the effect of returning the middling to the rougher cell. This is test No. 36. The procedure followed was to start off with a lot of 1,000 grams of ore reduced to the required size. This lot was floated while a second lot was being prepared. The middling from the recleaning of the concentrate from the first lot was then added to the second lot and the flotation proceeded with as before. This was repeated until five lots of ore had been run. If the assays from the products resulting from this test are studied it will be evident that returning the middling to be mixed with the feed to the rougher cells is detrimental. The middling builds up in the circuit and loads the circuit up with froth, finally reducing the grade of the concentrate, and increasing the loss in the tailing.

Test No. 36

Product	Weight grams	Copper	
		Per cent	Per cent x grams
Concentrate No. 1	108	20·42	22·06
Tailing No. 1	820	0·45	3·69
			25·75
Concentrate No. 2	117	16·40	19·18
Tailing No. 2	822	0·85	6·99
			26·17
Concentrate No. 3	107	19·05	20·39
Tailing No. 3	833	0·60	5·00
			25·39
Concentrate No. 4	105	18·15	19·06
Tailing No. 4	1,011	1·15	11·63
			30·69
Concentrate No. 5	81	12·15	9·84
Tailing No. 5	910	1·75	15·93
			25·77

Explanation of lower tailing in Ruth machine.—The assumption is correct that lower tailings are obtained in the Ruth than in the Callow machine, for the reason that in the Callow, the middling is returned continuously to the rougher cell.

Separate treatment of middling.—The conclusion has been reached, based on the results from the above experiments that the middling from the cleaner cell should be either treated in a separate circuit or tabled to remove the talc before being returned to the flotation circuit. It may possibly be found more advantageous to regrind the table concentrate before returning to the flotation machines.

SUMMARY OF CONCLUSIONS BASED ON RESULTS OF WORK TO DATE

The work up to the present has shown that there are four possible flow sheets for the treatment of this ore, as shown by Figures 4, 5, 6 and 7.

1. Straight flotation, but middling tabled to remove the talc before being returned to the circuit as suggested under the above heading "Separate treatment of the middling."

2. Straight flotation with middling returned to circuit.

3. Removal of the talc by preliminary flotation with a reagent which will float the talc without having a collecting action on the copper, followed by flotation of the copper from the pulp after the removal of the bulk of the talc.

4. Removal of the bulk of the talc by hydraulic classification at different points in the grinding circuit and flotation of the modified pulp as in (1). (See Figs. 4, 5, 6 and 7.)

The first proposed flow sheet appears at present to be the most promising one. No work so far has been done on this. The second flow sheet the writer believes comes next in order. The results from the straight flotation tests have been erratic. The reason for this has so far not been determined. However, it is safe to assume that a 10 per cent copper concentrate with an 80-85 per cent recovery can be obtained. The third method is probably the next best, but it is hardly possible by laboratory experiments to determine how successfully the flow sheet could be used in practice. Our laboratory experiments show that a very high grade concentrate can be obtained after the talc is removed, containing 18-25 per cent copper with an 80 per cent or better recovery. What has been said about the third method applies also to the fourth.

PROPOSED LINES ON WHICH RESEARCH WILL BE CONTINUED

1. Straight flotation but with middling tabled to remove the talc before returning to the circuit.—Considerable detailed work will be done on this method of treatment as at present it appears to be the most promising flow sheet for the concentration of this ore. Careful attention will be given the regrinding of the middling before it is returned to the flotation circuit.

2. Straight flotation.—A number of experiments will be run using larger quantities of ore under conditions approaching as nearly as possible actual milling operations. Because the results of the straight flotation tests have been very erratic, a series of experiments will be run under careful temperature control. Both the temperature in the ball mill while grinding, and in the flotation machine will be considered. Other variables will be kept under close observation in an endeavour to determine the correct conditions governing the production of uniform results.

3. Removal of the talc by preliminary flotation.—Very little more work will be done on this method. It has been clearly demonstrated that this can be done successfully in the laboratory. A very high grade concentrate can be obtained, apparently higher than with any of the other proposed flow sheets. We find the recovery to be about the same as with the other methods, but the General Engineering Co. report that it is lower. Some further work will be done to clear up this point.

4. Removal of the bulk of the talc by hydraulic classification.—We consider it very difficult to obtain results in the laboratory applicable to milling operations, and, therefore, do not propose doing much further work on this flow sheet.

Flotation Tests

Test No.	Product	Weight		Copper		Per cent of Cu values	Remarks
		Grams	Per cent	Per cent	Per cent x weight		
2	Concentrate.....	127	12·7	16·90	21·46	85·6	0·10 lb./ton thio-fizzan, 4 lb. lime; 65 mesh, ground wet.
	Middling........	165	16·5	1·75	1·89	7·4	
	Tailing.........	708	70·8	0·25	1·77	7·0	
14	Concentrate.....	262	26·2	7·97	20·88	79·3	1 lb. KK oil and 2 lb. lime; 100 mesh, froth from gangue.
	Middling........	230	23·0	1·23	2·83	10·7	
	Tailing.........	507	50·7	0·52	2·64	10·0	
16	Concentrate.....	112	11·1	17·82	19·78	77·7	As in test 2.
	Middling........	135	13·3	1·13	1·52	6·0	
	Tailing.........	766	75·6	0·55	4·16	16·3	
19a	Concentrate.....	101	10·0	20·62	20·83	77·0	Thio-fizzan and 4 lb. lime.
	Middling........	108	10·6	1·85	1·98	7·3	
	Tailing.........	807	79·4	0·53	4·28	15·7	
20a	Concentrate.....	108	10·3	19·17	20·70	76·3	Thio-fizzan and 8 lb. lime.
	Middling........	227	21·6	1·10	2·29	8·5	
	Tailing.........	717	68·1	0·57	4·09	15·1	
26	Concentrate.....	156	15·5	11·60	18·10	64·6	—48 mesh; 44 per cent —200 mesh, thio-fizzan and 5 lb. lime.
	Middling........	177	17·6	2·27	5·08	18·1	
	Tailing.........	673	66·9	0·72	4·85	17·3	
27	Concentrate.....	145	14·5	13·62	19·75	76·8	—65 mesh; 53 per cent —200 mesh; thio-fizzan and 5 lb. lime.
	Middling........	146	14·6	1·50	2·19	7·5	
	Tailing.........	712	70·9	0·57	4·06	15·7	
28	Concentrate.....	121	11·9	18·67	22·60	86·4	—100 mesh; 70 per cent —200 mesh; thio-fizzan and 5 lb. lime.
	Middling........	136	13·4	0·40	0·54	2·0	
	Tailing.........	761	74·7	0·40	3·04	11·6	
29	Concentrate.....	141	14·0	14·92	21·04	80·1	—100 mesh; 82 per cent —200 mesh; thio-fizzan and 5 lb. lime.
	Middling........	151	15·0	1·57	2·37	9·0	
	Tailing.........	715	71·0	0·40	2·86	10·9	
31	Concentrate.....	109	10·8	20·22	22·04	83·0	—100 mesh; 82 per cent —200 mesh; TT mixture and 8 lb. lime.
	Middling........	90	9·0	2·15	1·94	7·3	
	Tailing.........	808	80·2	0·32	2·59	9·7	
33	Concentrate.....	113	11·2	18·02	20·36	77·3	YZ mixture and lime.
	Middling........	207	20·5	2·00	4·14	15·7	
	Tailing.........	692	68·3	0·25	1·73	6·6	
35	Concentrate.....	125	12·5	12·77	15·84	60·7	
	Middling........	195	19·5	2·20	4·29	16·5	
	Tailing.........	684	68·0	0·87	5·95	22·8	
37	Concentrate.....	149	14·9	12·80	19·07	74·3	
	Middling........	180	18·0	1·25	2·25	8·8	
	Tailing.........	671	67·1	0·65	4·36	16·9	
38	Concentrate.....	160	15·4	13·08	20·93	75·3	TT mixture and 6 lb. lime per ton. Callow cells.
	Clean up cells...	40	3·9	4·55	1·82	6·5	
	Tailing.........	840	80·7	0·60	5·04	18·2	
15	Concentrate.....	116	11·4	18·92	21·95	81·1	2 lb./ton lime; added Aldol to cell, took of froth, added thio-fizzan.
	Middling........	115	11·4	1·42	1·63	6·0	
	Tailing.........	587	58·0	0·33	1·94	7·2	
	Froth...........	194	19·2	0·80	1·55	5·7	

Flotation Tests—Continued

Test No.	Product	Weight		Copper		Per cent of Cu values	Remarks
		Grams	Per cent	Per cent	Per cent x weight		
21b	Concentrate.....	144	14·0	15·87	22·88	83·3	Same as in test 15.
	Middling........	116	11·3	1·10	1·28	4·7	
	Tailing.........	565	54·7	0·22	1·24	4·5	
	Froth..........	207	20·0	0·96	1·99	7·3	
22b	Concentrate.....	134	12·6	16·27	21·80	81·0	Same as in test 15.
	Middling........	125	11·8	1·25	1·56	5·8	
	Tailing.........	655	61·7	0·34	2·23	8·3	
	Froth..........	147	13·9	0·90	1·32	4·9	
37b	Concentrate.....	148	14·7	15·78	23·40	83·3	Same as in test 15. Run in small Callow.
	Clean up cells...	62	6·2	1·25	0·78	2·8	
	Tailing.........	576	57·2	0·35	2·02	7·2	
	Froth..........	221	21·9	0·85	1·88	6·7	
30	Concentrate.....	98	9·9	22·40	21·95	82·5	As in above tests. Z cake present during frothing.
	Middling........	117	11·8	1·10	1·29	4·8	
	Tailing.........	596	60·1	0·25	1·49	5·6	
	Froth..........	180	18·2	1·05	1·89	7·1	
17	Flot.concentrate	97	9·7	21·02	20·42	82·7	As above; froth was tabled.
	" middling..	75	7·5	1·47	1·10	4·5	
	" tailing.....	589	58·9	0·22	1·30	5·3	
	Re-run flotation froth—						
	Table concent'te	16	1·6	2·70	0·31	1·5	
	" tailing...	200	20·0	0·76	1·52	6·2	
	Loss...........	23	2·3	....	....	....	
40	Concentrate.....	177	17·7	10·00	17·70	64·6	Lime, 4 lb./ton; 0·1 c.c. TT; 1 drop thio-fizzan.
	Middling........	193	19·3	3·55	6·85	25·0	
	Tailing.........	630	63·0	0·45	2·83	10·4	
40a	Concentrate.....	75	15·1	12·75	9·50	71·9	As test 40, but 500g; 4 lb./ton lime; 0·05 c.c. TT; ½ drop thio-fizzan.
	Middling........	51	10·3	3·30	1·58	12·6	
	Tailing.........	371	74·6	0·55	2·06	15·5	
41	Concentrate.....	180	21·6	10·15	22·80	79·4	4 lb./ton lime; ½ drop c.t.c., 2 drops thio-fizzan—ground.
	Middling........	83	10·0	2·15	2·15	7·5	
	Tailing.........	570	68·4	0·55	3·76	13·1	
42	Concentrate.....	79	15·9	12·05	19·16	73·6	4 lb./ton lime; 2 drops thio-fizzan; ½ drop c.t.c. and c.t. mixture.
	Middling........	126	25·4	2·00	5·08	19·6	
	Tailing.........	291	58·7	0·30	1·76	6·8	
44	Froth..........	144	14·4	0·90	1·29	4·9	Gangue frothed with Fumol No. 21. Added 2 grams soda ash and 0·15 c.c. TT.
	Concentrate.....	106	10·6	18·55	19·67	74·5	
	Middling........	126	12·6	2·85	3·59	13·5	
	Tailing.........	623	62·4	0·30	1·87	7·1	
45	Froth..........	245	24·3	1·12	2·70	10·3	Lime, 6 lb./ton. Gangue frothed with Fumol No. 6,3 drops. Floated with TT 0·15 c.c.
	Concentrate.....	101	10·0	20·27	20·47	78·4	
	Middling........	138	13·7	1·75	2·43	9·3	
	Tailing.........	523	52·0	0·10	0·52	2·0	....
46	Froth..........	96	9·6	1·45	1·39	5·3	As No. 44, but Fumol No. 20.
	Concentrate.....	121	12·1	16·95	20·51	78·6	
	Middling........	143	14·2	1·68	2·40	0·2	
	Tailing.........	644	64·1	0·28	1·80	6·9	

Flotation Tests—Concluded

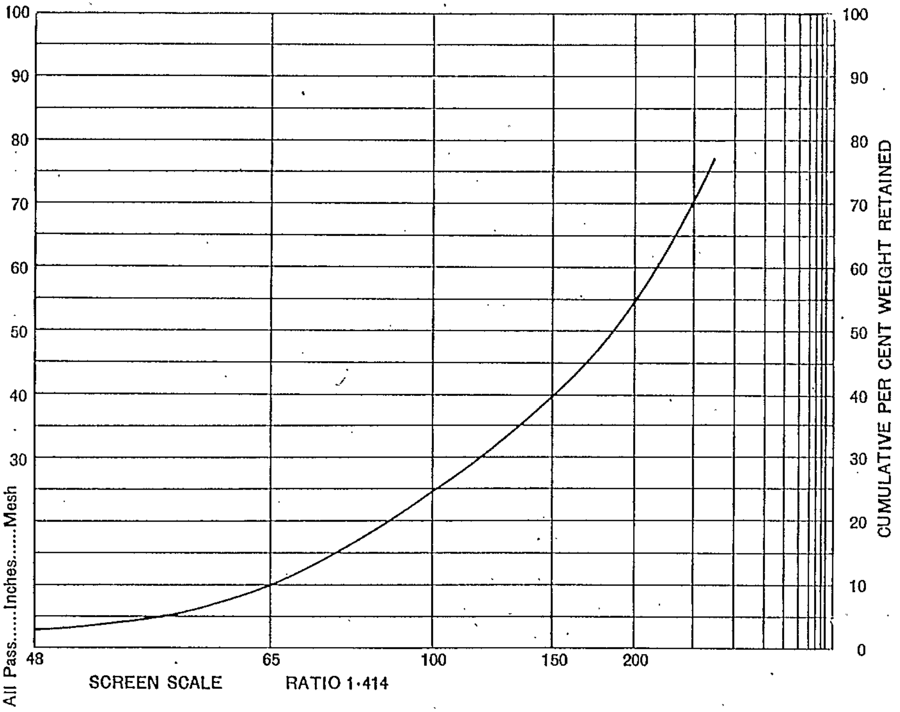
Test No.	Product	Weight		Copper		Per cent of Cu values	Remarks
		Grams	Per cent	Per cent	Per cent x weight		
47	Froth..........	318	31·7	3·80	12·08	46·1	As No. 45, but Fumol No. 21. This is very viscous and floats too much gangue.
	Concentrate.....	68	6·8	16·75	11·39	43·4	
	Middling........	84	8·4	1·95	1·64	6·2	
	Tailing.........	533	53·1	0·21	1·12	4·3	
48	Froth..........	308	30·9	1·60	4·94	18·7	Fumol No. 22a.
	Concentrate.....	73	7·3	24·43	17·83	68·1	
	Middling........	111	11·1	2·05	2·28	8·8	
	Tailing.........	506	50·7	0·23	1·17	4·4	
49	Froth..........	210	21·2	1·40	2·97	11·7	Fumol No. 23.
	Concentrate.....	85	8·5	20·93	17·80	70·0	
	Middling........	150	15·1	2·18	3·29	13·0	
	Tailing.........	548	55·2	0·23	1·27	5·3	
50	Froth..........	238	23·8	0·80	1·90	7·2	Fumol No. 24.
	Concentrate.....	113	11·3	18·80	21·24	80·9	
	Middling........	140	14·0	1·70	2·38	9·1	
	Tailing.........	500	50·0	0·15	0·75	2·8	
51	Froth..........	224	22·4	1·45	3·25	12·5	Fumol No. 24a.
	Concentrate.....	98	9·8	20·40	20·02	77·1	
	Middling........	130	13·0	1·48	1·92	7·7	
	Tailing.........	547	54·8	0·13	0·71	2·7	
52	Froth..........	138	13·8	1·05	1·45	5·4	Fumol No. 25.
	Concentrate.....	111	11·1	19·50	21·65	80·2	
	Middling........	144	14·5	1·57	2·26	8·4	
	Tailing.........	603	60·6	0·27	1·63	6·0	
53	Froth..........	206	20·3	2·12	4·37	16·0	Fumol No. 26.
	Concentrate.....	99	9·8	20·05	19·85	72·6	
	Middling........	154	15·1	1·32	2·03	7·4	
	Tailing.........	553	49·8	0·20	1·11	4·0	
54	Froth..........	100	9·9	6·09	6·09	21·8	Fumol No. 27.
	Concentrate.....	144	14·2	12·40	17·86	64·4	
	Middling........	158	15·6	1·52	2·40	8·7	
	Tailing.........	609	60·2	0·25	1·40	5·1	

Tests Nos. 45 to 54 inclusive were made to compare the froth removing properties of the various Fumols. For this reason all tests were as nearly as possible of the same duration.

Diagram IV

Cumulative Direct Diagram of Screen Analysis on Sample of Flin Flon Disseminated Ore

Flotation Tailings Test No. 26

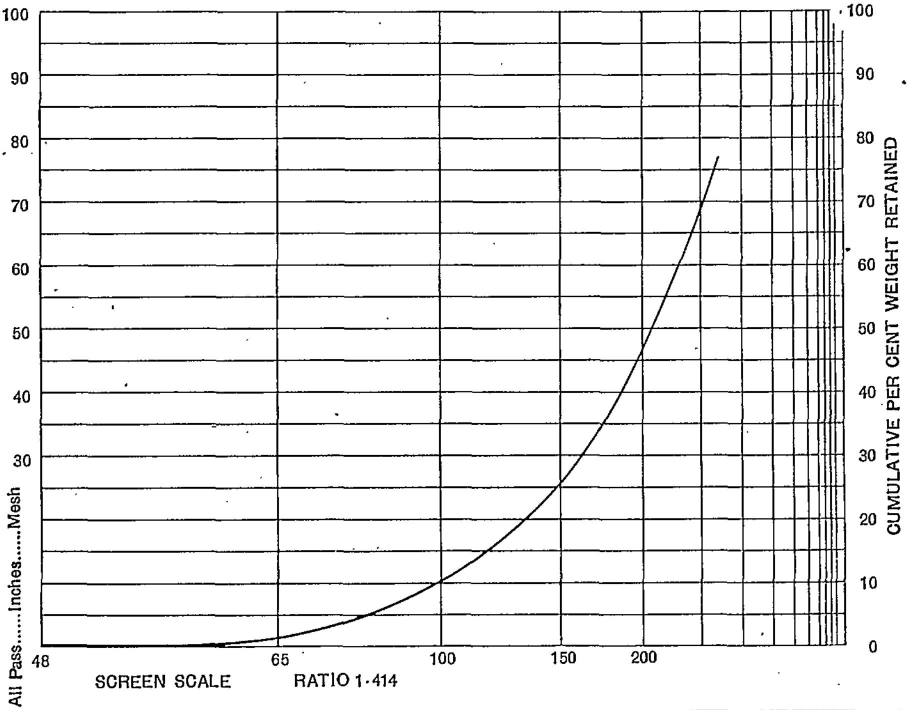


Indicate the Screen Crushed Through and also First Retaining Screen.	Screen Scale Ratio 1·414				Sample Weights	Per Cent.	Per Cent Cumulative Weights
	Openings.		Mesh.	Diameter Wire Inches.			
	Inches.	Millimeters.					
	1·050	26·67		·149			
	·742	18·85		·135			
	·525	13·33		·105			
	·371	9·423		·092			
	·263	6·680	3	·070			
	·185	4·699	4	·065			
	·131	3·327	6	·036			
	·093	2·362	8	·032			
	·065	1·651	10	·035			
	·046	1·168	14	·025			
	·0328	·833	20	·0172			
	·0232	·589	28	·0125			
	·0164	·417	35	·0122	3·0	0·52	
	·0116	·295	48	·0092	14·0	2·43	2·95
	·0082	·208	65	·0072	39·0	6·76	25·13
	·0058	·147	100	·0042	89·0	15·42	39·51
	·0041	·104	150	·0026	83·0	14·38	55·29
	·0029	·074	200	·0021	91·0	15·77	100·00
Pass	·0029	·074	200	·0021	258·0	44·71	
Totals					577·0		

Diagram V

Cumulative Direct Diagram of Screen Analysis on Sample of Flin Flon Disseminated Ore

Flotation Tailings Test No. 27

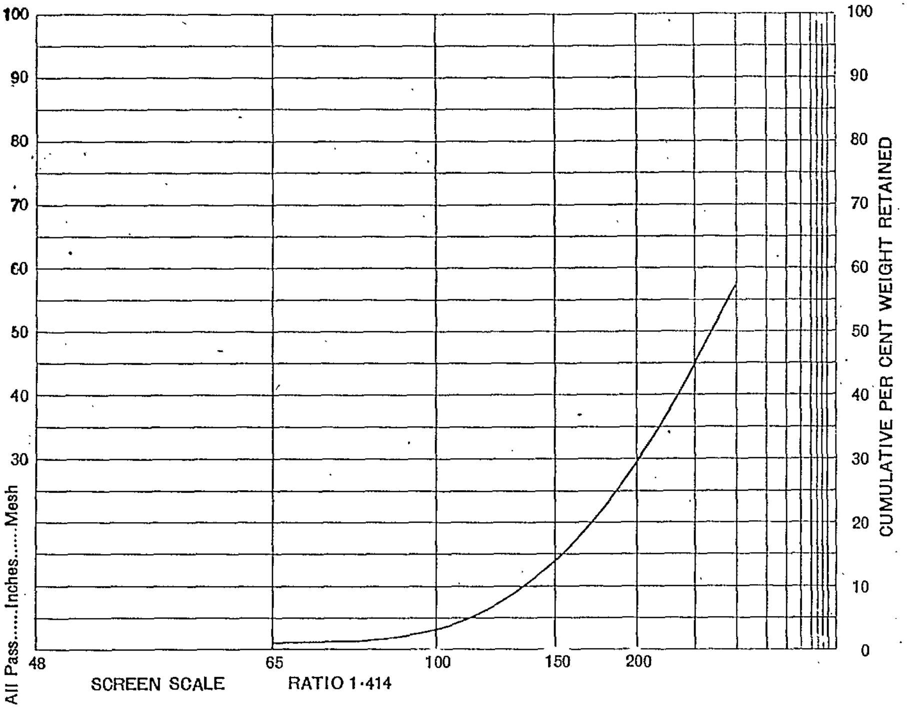


Indicate the Screen Crushed Through and also First Retaining Screen.	Screen Scale Ratio 1·414				Sample Weights	Per Cent.	Per Cent Cumulative Weights
	Openings.		Mesh.	Diameter Wire Inches.			
	Inches.	Millimeters.					
....	1·050	26·67	....	·149	....	....	....
....	·742	18·85	....	·135	....	....	....
....	·525	13·33	....	·105	....	....	....
....	·371	9·423	....	·092	....	....	....
....	·263	6·680	3	·070	....	....	....
....	·185	4·699	4	·065	....	....	....
....	·131	3·327	6	·036	....	....	....
....	·093	2·362	8	·032	....	....	....
....	·065	1·651	10	·035	....	....	....
....	·046	1·168	14	·025	....	....	....
....	·0328	·833	20	·0172	....	....	....
....	·0232	·589	28	·0125	....	....	....
....	·0164	·417	35	·0122	....	....	....
....	·0116	·295	48	·0092	3·0	0·49	....
....	·0082	·208	65	·0072	8·0	1·31	1·90
....	·0058	·147	100	·0042	50·0	8·17	9·97
....	·0041	·104	150	·0026	99·0	15·36	25·33
....	·0029	·074	200	·0021	132·0	21·57	47·00
Pass..........	·0029	·074	200	·0021	325·0	53·10	100·00
Totals..........	....	....	....	....	612·0	100·00	....

Diagram VI

Cumulative Direct Diagram of Screen Analysis on Sample of Flin Flon Disseminated Ore

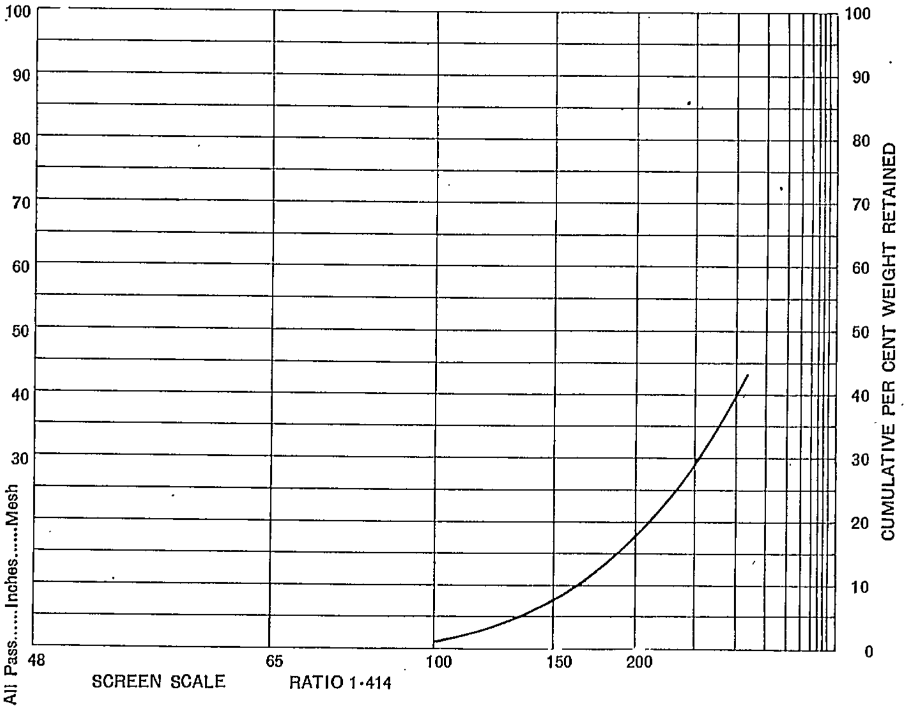
Flotation Tailings Test No. 28



Indicate the Screen Crushed Through and also First Retaining Screen.	Screen Scale Ratio 1·414				Sample Weights	Per Cent.	Per Cent Cumulative Weights
	Openings.		Mesh.	Diameter Wire Inches.			
	Inches.	Millimeters.					
....	1·050	26·67	....	·149	....	....	....
....	·742	18·85	....	·135	....	....	....
....	·525	13·33	....	·105	....	....	....
....	·371	9·423	....	·092	....	....	....
....	·263	6·680	3	·070	....	....	....
....	·185	4·699	4	·065	....	....	....
....	·131	3·327	6	·036	....	....	....
....	·093	2·362	8	·032	....	....	....
....	·065	1·651	10	·035	....	....	....
....	·046	1·168	14	·025	....	....	....
....	·0328	·833	20	·0172	....	....	....
....	·0232	·589	28	·0125	....	....	....
....	·0164	·417	35	·0122	....	....	....
....	·0116	·295	48	·0092	....	....	....
....	·0082	·208	65	·0072	2·0	0·31	....
....	·0058	·147	100	·0042	14·0	2·17	2·49
....	·0041	·104	150	·0026	72·0	11·23	13·73
....	·0029	·074	200	·0021	100·0	15·60	29·33
Pass........	·0029	·074	200	·0021	453·0	70·67	100·00
Totals........	....	....	....	....	641·0	99·99	....

DIAGRAM VII

Cumulative Direct Diagram of Screen Analysis on Sample of Flin Flon Disseminated Ore
Flotation Tailings Test No. 29

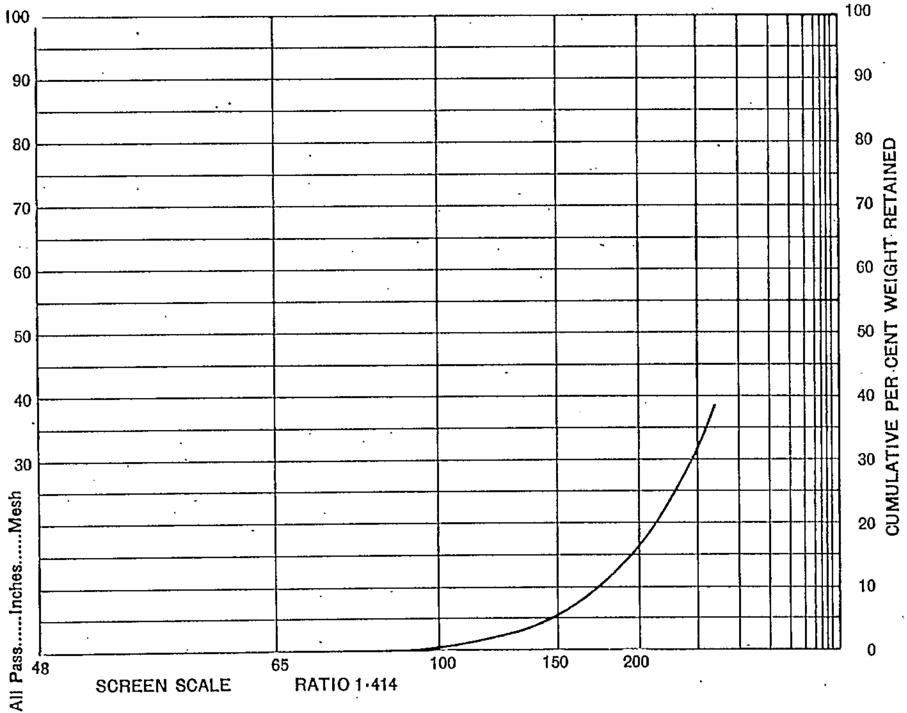


Indicate the Screen Crushed Through and also First Retaining Screen.	Screen Scale Ratio 1·414				Sample Weights	Per Cent.	Per Cent Cumulative Weights
	Openings.		Mesh.	Diameter Wire Inches.			
	Inches.	Millimeters.					
	1·050	26·67		·149			
	·742	18·85		·135			
	·525	13·33		·105			
	·371	9·423		·092			
	·263	6·680	3	·070			
	·185	4·699	4	·065			
	·131	3·327	6	·036			
	·093	2·362	8	·032			
	·065	1·651	10	·035			
	·046	1·168	14	·025			
	·0328	·833	20	·0172			
	·0232	·589	28	·0125			
	·0164	·417	35	·0122			
	·0116	·295	48	·0092			
	·0082	·208	65	·0072			
	·0058	·147	100	·0042	7	1·18	1·18
	·0041	·104	150	·0026	41	6·79	7·95
	·0029	·074	200	·0021	60	9·93	17·88
Pass	·0029	·074	200	·0021	496	82·12	100·00
Totals					604	100·00	

DIAGRAM VIII

Cumulative Direct Diagram of Screen Analysis on Sample of Flin Flon Disseminated Ore

Flotation Tailings Test No. 30



Indicate the Screen Crushed Through and also First Retaining Screen.	Screen Scale Ratio 1·414				Sample Weights	Per Cent.	Per Cent Cumulative Weights
	Openings.		Mesh.	Diameter Wire Inches.			
	Inches.	Millimeters.					
	1·050	26·67		·149			
	·742	18·85		·135			
	·525	13·33		·105			
	·371	9·423		·092			
	·263	6·680	3	·070			
	·185	4·699	4	·065			
	·131	3·327	6	·036			
	·093	2·362	8	·032			
	·065	1·651	10	·035			
	·046	1·168	14	·025			
	·0328	·833	20	·0172			
	·0232	·589	28	·0125			
	·0164	·417	35	·0122			
	·0116	·295	48	·0092			
	·0082	·208	65	·0072			
	·0058	·147	100	·0042	2	·39	·39
	·0041	·104	150	·0026	25	4·95	5·35
	·0029	·074	200	·0021	58	11·51	16·86
Pass	·0029	·074	200	·0021	419	83·13	100·00
Totals					504	100·00	

Test No. 162

THE ELIMINATION OF THE IMPURITIES FROM MALAGASH ROCK SALT

R. K. Carnochan

Samples of rock salt from Malagash, N.S., were received at the Ore Dressing Laboratories upon the following dates:—

May 8, 1922,	Sample No. 1	5·5	pounds net.
" 22, 1922,	" 2	9·5	"
" 22, 1922,	" 3	2·5	"
" 22, 1922,	" 4	5·0	"
" 22, 1922,	" 5	5·5	"
June 26, 1922,	" 6	871·0	"

The rock salt was sent in by Messrs. Chambers and McKay, New Glasgow, N.S., and was discoloured. Tests were desired on this material to see if a process could be devised to remove the colour and produce a clean white salt.

The discoloration in the salt occurs in different ways. Some pieces are very dark and must contain a large amount of the discolouring matter, while others show just a faint even tint, and can only have a very small percentage of the impurity.

Analytical work by the chemists of the laboratories shows the colouring to be due to iron and organic matter. The salt as received ran about 7 per cent water insoluble, of which 1·8 per cent was Fe_2O_3 and 0·7 per cent organic matter. This shows that the percentage of colouring in the salt is very small.

The discoloration is the chief objection to the salt, but the insoluble is also undesirable, for when used to pack fish it remains on them as a slime.

The following methods of treatment were tried:—

1. Dissolving in water, filtering, and evaporating the filtrate. This gives a pure clean white salt. Before considering the installation of such a process, it was desired that experimental work be conducted to determine if the impurities could be removed by other methods involving a less costly installation.

2. The rock salt was sized and each size fed to a revolving furnace, temperature about 1100° F. The furnace discharge was sized. It was expected that either the pure salt or the discoloured salt might decrepitate and go into the fines when the furnace discharge was screened. It was found that both decrepitated about the same amount, for when screening no separation was evident.

3. Tabling the salt in a saturated salt solution on a small Wilfley table. This gives no concentration whatever as the discoloured particles seem to be of the same specific gravity as the pure salt.

4. Dry magnetic separation in the Ullrich machine. This process removes only the very dirty pieces of salt. If the sample which is fed to the machines is quite dirty, it makes some difference, but when the sample is only slightly discoloured it does not remove anything.

5. The salt was sized and each size ground in a small pebble mill. The mill discharge was screened. It was thought that the discoloured particles would not grind as fast as the pure salt particles, and upon screening the mill product it would be found in the coarse product. It was found that this method would not work as the discoloured material grinds almost as easily as the pure salt.

6. Washing the salt −80 mesh in a log washer using a concentrated salt solution as a wash. The washed salt was no cleaner than the feed to the machine.

7. Melting the salt and allowing the dirt to settle, while the salt is molten. This gives some separation—a lot of the dirt settles, but the salt produced is still discoloured.

CONCLUSIONS

None of the methods investigated would be suitable for the treatment of the salt for the elimination of the impurities, except that of dissolving and evaporating, the common practice with the majority of producers.

If this method is not adopted, closer selection of clean faces underground, closer sorting on the surface, with the rejection of the discoloured material, will be necessary to produce a clean salt in the various sizes required by the trade.

Test No. 163

THE CONCENTRATION OF GRAPHITE ORE, FROM THE TIMMINS MINE, WESTPORT, ONTARIO

C. S. Parsons

A sample of graphite ore was received at the Ore Testing Laboratories, March 15, 1922, from the Timmins mine, Westport, Ont. This sample was submitted by Mr. H. P. H. Brumell.

It consisted of material which would pass a 6 mesh screen. The graphite was of the flake variety in a gangue of comparatively soft material, showing the effects of considerable weathering.

The sample was found to contain on analysis 11·50 per cent carbon.

The object of the test was to produce a high grade graphite concentrate with a minimum reduction in the sizes of flake compared to that contained in the original sample. The sample was crushed in rolls to 10 mesh. A sample of 1,000 grams was cut out and reduced to 20 mesh in a ball mill. The separation was made by flotation and table concentration. Coal oil and pine oil were used, both oils being added to the ball mill.

After grinding, the pulp was floated in a Ruth flotation machine. A rougher concentrate and tailing was made. The rougher concentrate was fed directly to a laboratory Wilfley table, and a considerable quantity of clean sand was eliminated which had been carried over in mechanical suspension in the heavy froth of graphite. This table operation seems feasible, as a clean product is obtained representing 21 per cent by weight of the rougher concentrate. The elimination of the sand throws less work on the regrinding mills and the subsequent separation and flotation

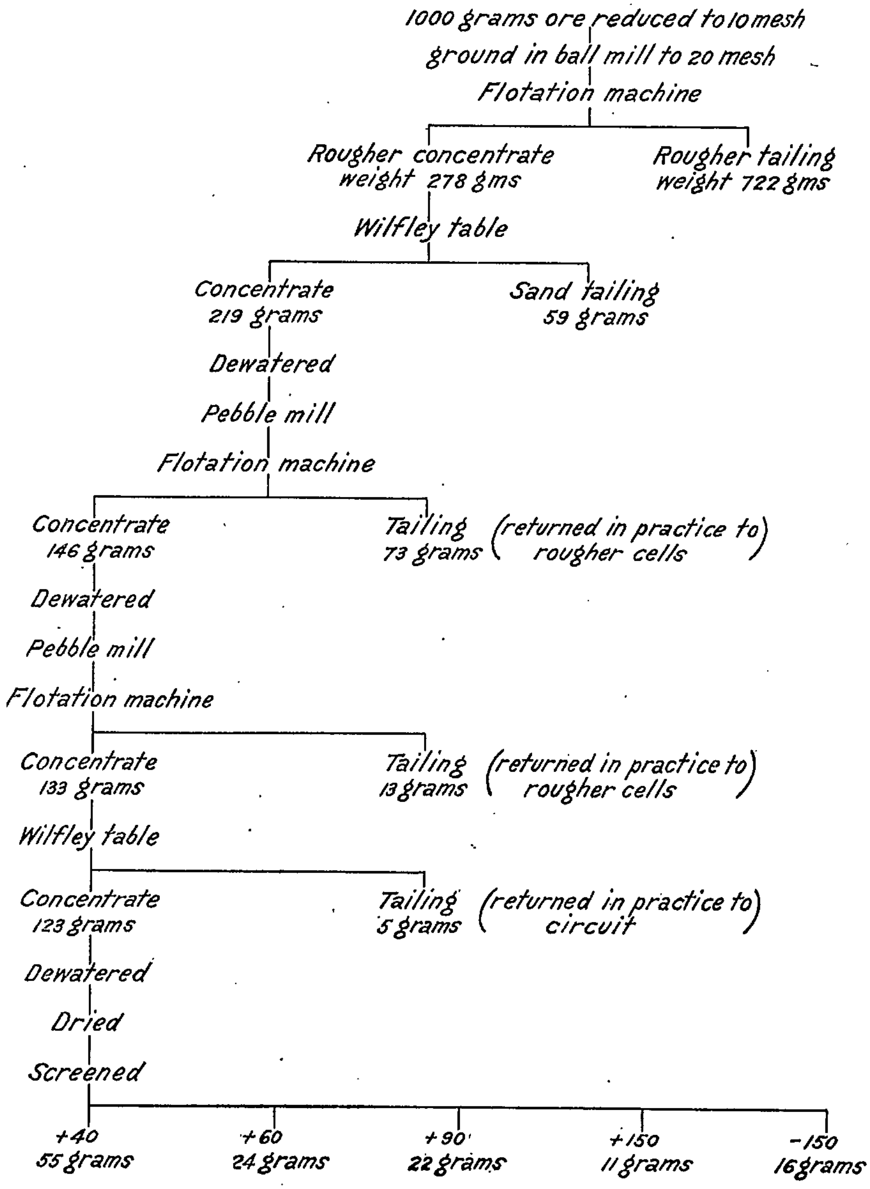


FIG. 8. Flow sheet of test made on 1,000 grams graphite ore from Timmins mine, Westport, Ontario.

machines. The concentrate from the table was dewatered, reground in a pebble mill, and floated, making a concentrate and tailing. This tailing would in practice be returned to the circuit. The concentrate was dewatered, reground in a pebble mill, and refloated, making a concentrate and tailing. This tailing would also be returned to the circuit. The concentrate was tabled, making a concentrate and tailing. It is doubtful if this second tabling is necessary. The concentrate was dried and screened. The results of the test are given in the following table:—

Analysis of sample	11·50 per cent C.
Weight used for test	1,000 grams.

Product	Weight		Analysis per cent C.	Content grams	Per cent of C. values in ore
	Grams	Per cent			
Concentrate + 40 mesh	55	5·5	91·20	50·16	44·6
" + 60 "	24	2·4	88·90	21·34	19·0
" + 90 "	22	2·2	85·50	18·81	16·7
" +150 "	11	1·1	78·50	8·64	7·7
" −150 "	16	1·6	51·55	8·25	7·3
Flotation tailing No. 1	722	72·2	0·25	1·81	1·6
" Nos. 2 and 3	86	8·6	0·81	0·70	0·6
Table tailing No. 1	59	5·9	1·80	1·06	1·0
" No. 2	5	0·5	34·00	1·70	1·5
Totals	1,000	100·0	11·25	112·47	100·0

Note.—The calculated analysis of the ore from the weights and analyses of concentration products checks very closely with the analysis of the ore. The above results show:—

A recovery of 80·3 per cent in concentrates + 90 mesh
" 7·7 " " +150 "
" 7·3 " " −150 "

A total of 95·5 "

and a total of 4·7 " of the C. values in all tailings.

The combined concentrate +90 mesh would have an analysis of 89·5 per cent C.

Test No. 164

SEPARATION OF FOSSIL RESIN FROM COAL

R. K. Carnochan

Six bags of resin in coal, shipping weight 460 pounds, were received April 19, 1922, at the Ore Dressing and Metallurgical Laboratories. The shipment was from the Coalmont Collieries, Coalmont, B.C., and consisted of yellow fossil resin in a bituminous coal. It has been proven that this fossil resin is not amber.

Tests were desired on the sample submitted to determine if it would be possible to recover the resin in the form of a high grade concentrate, and leave a tailing of clean coal. The heavy solution method was used to separate the resin from the coal. In this process the resin is floated on the surface of an aqueous solution of sodium chloride or other salt, and the coal sinks in the same solution.

Six tests in all were made, five being on a small scale, and one on a large scale. It was found that the coal had to be crushed to pass 4 mesh before the resin was freed sufficiently.

Test No. 1.—This test was made in a tub. The coal and resin were thoroughly mixed with solution and then allowed to stand until the resin collected on top of the solution and the coal settled to the bottom. The resin with some solution was then poured off gently onto a 100 mesh screen. The solution from the screen was returned to the tub, the contents of the tub stirred, and then allowed to stand, after which the resin was again decanted from the tub. This decantation was made three times and it was found that this gave a clean tailing but a rather dirty concentrate. The concentrate was cleaned by putting it along with some solution in a tub, and decanting it once in the same manner as before. The middlings were put with the tails, and then both concentrates and tails were washed three times with fresh water to remove the salt solution, and then dried and weighed.

Solution used—Sodium chloride, density 1·15.
Size of feed, $-\frac{1}{4}''$

Product	Weight pounds	Per cent of heads
Concentrate	0·18	1·29
Tailing	13·82	98·71
Heads	14·00	100·00

Test No. 2, Run No. 1.—This test was made in a small tank. The coal and resin were mixed with the solution by means of a horizontal revolving cross-arm at the bottom of the tank the arm being driven by a vertical shaft. A spray of solution was also used to wet the coal which, when dry, floated on the surface of the solution in the tank. After mixing, the contents of the tank were allowed to stand a while and then more solution was poured slowly into the centre of the tank and the floating resin was allowed to overflow into a circular launder running around the outside of the tank. By means of this launder the amber was conveyed to a 100 mesh screen to remove the solution. After overflowing, the contents of the tank were stirred and overflowed again, this process being repeated in all three times. The concentrates were then recleaned by putting them in the tank and overflowing in the same manner as before. The middlings were put with the tailing, and the concentrate and tailing were washed three times with fresh water and then dried and weighed.

Solution used—Sodium chloride, density 1·15.
Size of feed, $-\frac{1}{4}''$

Product	Weight pounds	Per cent of heads
Concentrate	0·67	1·56
Tailing	42·33	98·44
Heads	43·00	100·00

Thirty pounds of the tailing was screened on 20 mesh.

Product	Weight pounds	Per cent of feed	Per cent of heads
Tailing +20	29·00	80·55	79·30
Tailing −20	7·00	19·45	19·14
Heads	36·00	100·00	98·44

Test No. 2, Run No. 2.—15½ pounds of +20 tailing from run No. 1 were crushed to pass 20 mesh and treated in the same manner as described under run No. 1, the concentrates being re-run to clean them.

Solution used—Saturated sodium chloride, density 1·20.
Size of feed, −20 mesh.

Product	Weight pounds	Per cent of feed	Per cent of heads
Concentrate	0·16	1·03	0·82
Tailing	15·34	98·97	78·48
Feed	15·50	100·00	79·30

Concentrate run No. 1	1·56	per cent of heads.
" " 2	0·82	" "
Total concentrate	2·38	" "

Test No. 2, Run No. 3.—13½ pounds of +20 tailing from run No. 1 were crushed to pass 20 mesh and screened on 60.

Product	Weight pounds	Per cent of feed	Per cent of heads
Tailing −20 +60	10·50	77·78	61·68
Tailing −60	3·00	22·22	17·62
Feed	13·50	100·00	79·30

The 10½ pounds of −20 + 60 tailing were treated in the same manner as described under run No. 1, the concentrates being re-run to clean them.

Solution used—Saturated sodium chloride, density 1·20.
Size of feed, −20+60.

Product	Weight pounds	Per cent of feed	Per cent of heads
Concentrate	0·10	0·95	0·59
Tailing	10·40	99·05	61·09
Feed	10·50	100·00	61·68

Concentrate run No. 1	1·56	per cent of heads.
" " 3	0·59	" "
Total concentrate	2·15	" "

Test No. 3, Run No. 1.—20 pounds of heads −¼″ were treated in the same manner as described under test No. 2, run No. 1, the concentrates being re-run to clean them.

Solution used—Calcium chloride, density 1·16.
Size of feed, −¼″

Product	Weight pounds	Per cent of heads
Concentrate	0·31	1·55
Tailing	19·69	98·45
Heads	20·00	100·00

19¼ pounds of tailing from run No. 1 were screened on 20 mesh.

Product	Weight pounds	Per cent of feed	Per cent of heads
Tailing +20	13·50	70·13	69·04
Tailing −20	5·75	29·87	29·41
Feed	19·25	100·00	98·45

Test No. 3, Run No. 2.—13½ pounds of +20 tailing from run No. 1, were crushed to pass 20 mesh, and treated as described under test No. 2, run 1, the concentrate being re-run to clean them.

Solution used—Calcium chloride, density 1·16.
Size of feed, −20 mesh.

Product	Weight pounds	Per cent of feed	Per cent of heads
Concentrate	0·10	0·74	0·51
Tailing	13·40	99·26	68·53
Feed	13·50	100·00	69·04

Concentrate run No. 1	1·55	per cent of heads.
" " 2	0·51	" "
Total concentrate	2·06	" "

Test No. 4.—This test was made in exactly the same manner as test No. 3.

Run No. 1

Product	Weight pounds	Per cent of heads
Concentrate	0·30	1·50
Tailing	19·70	98·50
Heads	20·00	100·00

Product	Weight pounds	Per cent of feed	Per cent of heads
Tailing +20	13·20	67·00	66·00
Tailing −20	6·50	33·00	32·50
Feed	19·70	100·00	98·50

Run No. 2

Product	Weight pounds	Per cent of feed	Per cent of heads
Concentrate	0·08	0·60	0·41
Tailing	12·92	99·40	65·59
Feed	13·00	100·00	66·00

Concentrate run No. 1	1·50	per cent of heads.
" " 2	0·41	" "
Total concentrate	1·91	" "

Test No. 5.—This test was made in exactly the same manner as test No 3.

Run No. 1

Product	Weight pounds	Per cent of heads
Concentrate	0·30	1·52
Tailing	19·50	98·48
Heads	19·80	100·00

Product	Weight pounds	Per cent of feed	Per cent of heads
Tailing +20	16·00	82·05	80·80
Tailing −20	3·50	17·95	17·68
Feed	19·50	100·00	98·48

Run No. 2

Product	Weight pounds	Per cent of feed	Per cent of heads
Concentrate	0·06	0·40	0·32
Tailing	14·94	99·60	80·48
Feed	15·00	100·00	80·80

Concentrate run No. 1	1·52	per cent of heads.
" " 2	0·32	" "
Total concentrate	1·84	" "

Test No. 6.—This test was made on 303½ pounds of heads −20 mesh. A rough concentrate was made by decanting from a tub as described under test No. 1, about one-twelfth of the lot being treated at a time. The rough concentrate was mixed with solution in a large cone and allowed to stand. Any coal that settled out was drawn off from the bottom, and the contents of the cone were then stirred and allowed to settle and the coal drawn out from the bottom again. This procedure was continued until only the resin concentrate remained in the cone. The coal from the bottom of the cone was put into the cone again and rerun in the same manner to get any resin that might be in it. All products were washed three times in fresh water and dried and weighed. In washing the two concentrates produced, the wash water was decanted into a filter press and this gave a small amount of very fine concentrate which was kept separately.

Solution used—Sodium chloride, density 1·17.
Size of feed, −20 mesh.

Product	Weight pounds	Per cent of heads
1st concentrate (from cone)	6·50	2·14
2nd concentrate (from cone)	0·50	0·16
3rd concentrate (from press)	0·24	0·08
1st tailing (from tub)	230·00	75·78
2nd tailing (from cone)	57·50	18·95
Loss	8·76	2·89
Heads	303·50	100·00

Total concentrate.. 2·38 per cent of heads.

SUMMARY

Test No.	No. 1 conc. per cent heads	No. 2 conc. per cent heads	No. 3 conc. per cent heads	Total conc. per cent heads	Soln. used	Density first treatment	Density second treatment	Feed first treatment	Feed second treatment
1	1·29	....	....	1·29	NaCl	1·15	....	−¼″	....
2, run Nos. 1 and 2	1·56	0·82	....	2·38	NaCl	1·15	1·20	−¼″	−20
2, run Nos. 1 and 3	1·56	0·59	....	2·15	NaCl	1·15	1·20	−¼″	−20+60
3	1·55	0·51	....	2·06	CaCl	1·16	1·16	−¼″	−20
4	1·50	0·41	....	1·91	CaCl	1·16	1·16	−¼″	−20
5	1·52	0·32	....	1·84	CaCl	1·16	1·16	−¼″	−20
6	2·14	0·16	0·08	2·38	NaCl	1·17	1·17	−20	−20

CONCLUSIONS

1. It is possible to recover the resin from the coal. Although no analyses on the heads and products have been obtained, it is evident from a close inspection, that the concentrates are high grade, and the tails clean. This indicates a good recovery of the resin in a suitable form.

2. Sodium chloride is the best salt to make up the solution used in the separation, as it gives just as good results as calcium chloride, and is cheaper to use, in spite of the fact that more of it is needed to give a solution of a certain density.

3. The coal and resin should be ground to pass 20 mesh before separating, as this would give a much simpler flow sheet, use less equipment and power, and give just as good a product as separating at −¼", and regrinding and separating again.

4. A cone would be a suitable apparatus to use for separating, as it was found in the test work to work very nicely, and give the best separation.

5. In mixing the coal and resin with solution, some vigorous agitator will have to be used, as the coal seems to wet with difficulty, and when dry it floats on the surface of the solution.

Test No. 165

THE WET SEPARATION OF ASBESTOS FROM ITS GANGUE

R. K. Carnochan

In 1921 the Division of Ore Dressing and Metallurgy conducted dry milling tests on shipments of asbestos rock from two mines in the Quebec asbestos field. On the completion of these tests it was decided to investigate the wet milling of asbestos. This investigation was carried out as far as the means at the disposal of the division would allow. The asbestos rock with which the tests were made was the unused portion of a shipment received for dry milling from the Black Lake Asbestos and Chrome Company.

In all thirteen tests were made on this shipment, three large and one small scale dry milling tests, and nine small scale wet milling tests. To give an idea of the results obtained by wet milling, the last test will be given in detail.

Test No. 13

Run No. 1 (1st wet ball milling).—This test was made on 25 pounds of a head sample taken from the five tons of asbestos rock which was used in the first large scale dry milling test. The 25 pounds were crushed to pass ¾" and fed to a small wet ball mill, the mill discharge was screened on ½", and the −½" went to a small Richards pulsating jig. The jig separated the fibre from the rock. The fibre from the jig was run to a 4 mesh screen and the −4 mesh fell onto a 20 mesh screen. This gave a +4 mesh clean fibre and a +20 mesh fibre which contained some rock. The −20 fell into a tub, the overflow from the tub running to a series of two settling boxes. This gave a −20 sand product which contained some fibre and two float products from the settling boxes which are composed of very fine fibre. After the 25 pounds had been fed the +½″ was dried and crushed to −½″ and fed back to the mill.

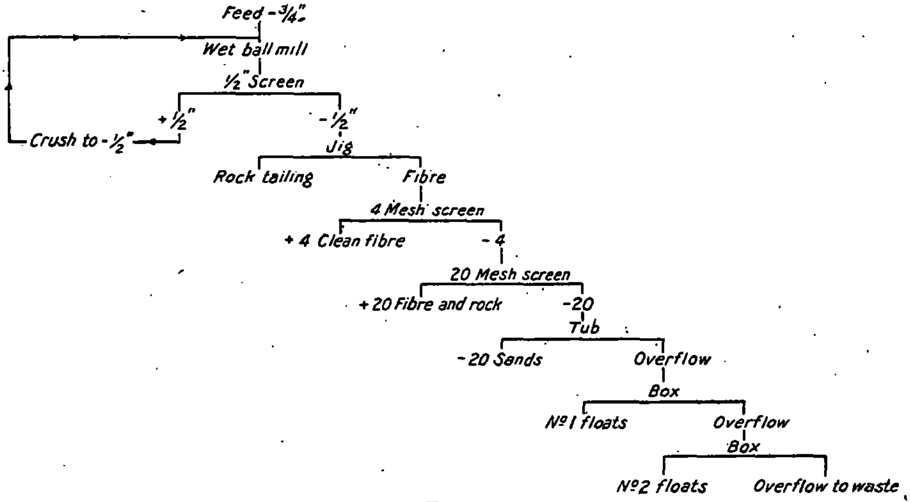

Flow-sheet for run No. 1

FIG. 9

Run No. 2 (cleaning up +20 fibre and rock from run No. 1).—The +20 fibre and rock from run No. 1 was fed to the jig making a rock tailing and a fibre product. The fibre was run to 10 and 20 mesh screens, giving a +10 clean fibre and a −20 clean fibre. The −20 was treated in the same way as the −20 of run No. 1. Flow sheet of run No. 2:—

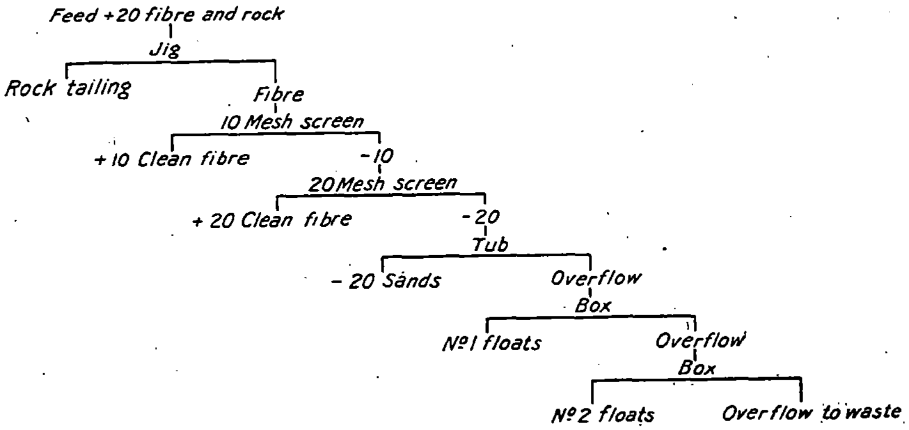

Flow-sheet for run No. 2

FIG. 10

Run No. 3 (2nd wet ball mill run).—The rock tailings from runs Nos. 1 and 2 were dried and crushed to pass 4 mesh and fed to the mill. The flow sheet for this run is the same as for run No. 1, except that no ½" screen was used, the mill discharge passing directly to the jig. Very little +4 mesh fibre was obtained in this run.

Run No. 4 (cleaning up +20 fibre and rock from run No. 3).—The +20 fibre and rock from run No. 3 was fed to the jig. The flow sheet for this run is the same as that of run No. 2. The +10 fibre obtained in this run was equal to about ¼ of the +10 fibre obtained in run No. 2, and the +20 fibre was equal to about ⅔ of the +20 of run No. 2.

Run No. 5 (3rd wet ball mill run).—The rock tailing from runs Nos. 3 and 4 were dried and crushed to pass 8 mesh and fed to the ball mill. The flow sheet for this run is the same as that for run No. 1, except that no ½" screen was used, the mill discharge passing directly to the jig. No +4 fibre was obtained from this run.

Run No. 6 (cleaning up +20 fibre and rock from run No. 5).—The +20 fibre and rock from run No. 5 was fed to the jig. The flow sheet for this run is the same as that of run No. 2. Very little fibre was obtained from this run.

Run No. 7 (tabling −20 sands).—The −20 sands from runs Nos. 1 to 6 inclusively were tabled on a small Wilfley table making a sand product, a middling, and a fibre product. The middling was re-run twice, and in the second re-run only two products were made. The overflow from the boxes used to collect the table products was led to the tub and two settling boxes that were used in the other runs.

Products Obtained, Runs Nos. 1 to 7 inclusive.—All the products were dried and weighed. The table fibre and No. 1 and No. 2 floats were screened on 65 mesh to remove sand. The following is a summary of all the products:—

Product	Weight grams
Rock tailing, run No. 5	5,221
Tailing, run No. 6, rock and fibre	113
+4 and +10 fibre	55
+20 fibre	17
Table fibre +65	116
Sand from table fibre −65	165
Table sands	4,086
No. 1 floats, fibre +65	15
No. 1 " sand −65	366
No. 2 " fibre +65	85
No. 2 " sand −65	142
Loss	969
Feed	11,350

Fibre Obtained, Runs Nos. 1 to 7 inclusive.—All the fibre obtained was rubbed between the hands to loosen it up as in drying it sticks together. Each lot of fibre was then tested on a standard testing machine with the following results:—

+4 and +10 fibre	+ 2	3 grams.	
	+ 4	27 "	
	+10	14 "	
	+10	10 "	
+20 fibre	+ 2	0 "	
	+ 4	2 "	
	+10	5 "	
	−10	10 "	
Table fibre	+ 2	0 "	
	+ 4	0 "	
	+10	22 "	
	−10	95 "	
No. 1 float fibre	+ 2	0 "	
	+ 4	0 "	
	+10	4 "	
	−10	11 "	
No. 2 float fibre	+ 2	0 "	
	+ 4	5 "	
	+10	33 "	
	−10	46 "	
Total fibre	+ 2	3 "	0·026 per cent of heads.
	+ 4	34 "	0·300 " "
	+10	78 "	0·687 " "
	−10	172 "	1·515 " "
		287 "	2·528 " "

CONCLUSIONS

1. As the test was on a small scale the results will only roughly approximate those that would be obtained in large scale work. In a large ball mill the fibre would be cut up more before being discharged and hence less coarse fibre would be obtained. In testing the fibre in the standard testing machine a pound of fibre should have been used, but as none of the fibre products weighed a pound, the tests had to be made on a much smaller amount. The result of this would be that the coarse fibre would have a better opportunity to work through the screens and the tests would indicate a lower grade of fibre than that really obtained. In the work on dry milling of asbestos rock it was found that large and small scale tests on similar feeds did not check. This is due to the great difficulty of getting a small representative sample.

2. The following table shows the total fibre obtained in test No. 13 compared with that obtained in the first large dry milling test. Both these tests were on similar rock, the feed to test No. 13 being part of a head sample cut out of the feed of the large dry milling test.

—	Total fibre. Test 13	Per cent of heads. Dry milling test
+ 2	0·026	0·048
+ 4	0·300	0·265
+10	0·687	0·346
−10	1·515	0·430
Total	2·528	1·089

The results of the wet milling test compare favourably with those of the dry milling test. As to which is the best milling method, it would be difficult to say with the present data. This question could only be decided by comparative large scale tests which it is hoped the Division of Ore Dressing and Metallurgy will be able to carry out in the near future.

Test No. 166

THE SEPARATION OF FLUORITE, CALCITE, AND BARITE IN THE FLUORSPAR FROM THE MADOC DISTRICT, ONT.

R. K. Carnochan

Three shipments of barite, fluorite, and calcite were received at the Ore Dressing and Metallurgical Laboratories from Gavin M. Wallbridge, manager of the Wallbridge mine, Madoc, Ont. These shipments were obtained from a vein on the west half of lot 1, con. I, Madoc township, Hastings county, Ont.

Shipment	Date received	Net weight	$BaSO_4$ per cent	CaF_2 per cent	$CaCO_3$ per cent	SiO_2 per cent
		pounds				
Lot No. 1	Feb. 12, 1919	287	4·40	48·35	40·00	3·10
Lot No. 2	April 3, 1922	505	6·00	72·80	15·60	1·55
Lot No. 3	June 16, 1922	1,368	6·00	76·40	15·50	....

The three lots consisted of a natural mixture of barite, fluorite and calcite, the size of the individual pieces of mineral ranging from about 2 inches to very small grains. Tests were desired to determine if it would be possible to separate the three minerals into products pure enough to be marketable.

TABLING

Lot No. 1 was used in making table tests. A full report covering these tests was published in the Summary Report of the Mines Branch for 1919 (see test No. 115, page 71).

The conclusions derived from these table tests were:—

1. To secure good work by tabling, the feed will have to be ground to −80 mesh.

2. The slime and dust loss in the tests made on −80 mesh material was very high, but with better equipment this loss could be lowered.

3. If the loss could be kept down to 10 per cent, the following grades of products and recoveries would be obtained by tabling at −80 mesh:—

Product	Recovery per cent	Grade per cent
Barite	75·5	81·85
Fluorite	65·2	74·80
Calcite	63·7	59·20

4. Lot No. 1 is lower in barite and fluorite than either lots No. 2 or No. 3. Better barite and fluorite products, and better recoveries of barite and fluorite would have been obtained by tabling material similar to lot No. 2 or lot No. 3.

DECREPITATION

The Rock Candy Fluorspar Mill, Grand Forks, B.C., is successfully concentrating a fluorite by decrepitation. The fluorite is crushed, dried, and sized. Each size, excepting the fines, is fed into a separate rotary kiln, temperature 1200°F. In passing through the kiln, the fluorite decrepitates while the gangue does not, and the discharges from the different kilns are screened on suitable screens, the throughs being finished concentrate and the oversize tailings. The fines from the first sizing are concentrated by tabling.

This decrepitation process was tried on part of lot No. 2, a portion of which was crushed to 4 mesh and then carefully sized down to 20 mesh. Each size was fed separately to a small revolving furnace heated to about 1200°F. Upon entering the furnace the fluorite was seen to glow and then decrepitate. The furnace discharge from each size fed to the furnace was screened into a suitable number of sizes and each of these sizes was analysed for fluorite. It was found that the different sizes all ran about the same in fluorite, the fines in most cases running a little higher than the coarse. Decrepitation tests made by placing small pieces of fluorite and calcite picked from lot No. 2 on a heated wire gauze, show that the calcite when crystalline decrepitates just as readily as the fluorite, but when the calcite is not crystalline, it does not decrepitate. This explains the poor results obtained by decrepitation.

The conclusion derived from these decrepitation tests was that the method is unsuitable for the concentration of the fluorite in the Madoc deposits.

JIGGING

A large scale jig test was made on 381 pounds of lot No. 2 −4 mesh. The products of this test, after sampling for analysis, were mixed together and used to make another large scale jig test.

Analyses on the products of these jig tests show very little separation of the minerals.

From these tests it was concluded that jigging was not a suitable method for use in treating the material submitted.

CALCINING, SLACKING, WASHING AND TABLING

A method of removing lime from magnesite had been developed by the Mines Branch when working on magnesite from Grenville township, Que. This method consists of calcining the magnesite, slacking in boiling water, upon which the lime goes into a thin slurry while the magnesia stays coarse, and then washing out the lime.

It was decided to try and adapt this process to remove the calcite from the barite and fluorite, with the subsequent separation of the barite and fluorite by tabling.

Two small scale tests were made on lot No. 2, and one medium scale test on lot No. 3. These tests gave similar results.

In the medium scale test, 29 pounds of lot No. 3 -4 mesh were calcined at 1400°F. The calcined material ran 0·70 per cent CO_2 and weighed 27 pounds. This 27 pounds was sampled to 26·5 pounds, slacked in boiling water, and washed by agitating in pails and decanting. The washed material when dried weighed 20 pounds and analysed $BaSO_4$ 5·9 per cent, CaF_2 87·4 per cent, CaO 2·38 per cent. The washings weighed 6·5 pounds and analysed $BaSO_4$ 8·7 per cent, CaF_2 51·3 per cent, CaO 26·4 per cent. A 9·75 pound sample of the washed material was crushed to pass 40 mesh and tabled.

Table concentrate............	0·29 pounds	$BaSO_4$	52·4 per cent.
Table tailing..................	8·75 "	CaF_2	91·7 "
Table slime....................	0·17 "	CaF_2	71·5 "

A 10 pound portion of washed material was crushed to pass 80 mesh and tabled.

Table concentrate............	0·26 pounds	$BaSO_4$	52·5 per cent.
Table tailing..................	8·50 "	CaF_2	93·5 "
Table slime....................	0·52 "	CaF_2	86·4 "

Tabling at −40 mesh gives the following recoveries and grades of products:—

Product	Recovery per cent	Grade per cent
Barite—table concentrate..	18·3	52·4 $BaSO_4$
Fluorite—table tail and slime..	76·8	91·3 CaF_2
Lime—washings..	69·4	26·4 CaO

Tabling at −80 mesh gives the following recoveries and grades of products:—

Product	Recovery per cent	Grade per cent
Barite—table concentrate..	16·0	52·5 $BaSO_4$
Fluorite—table tail and slime..	77·2	93·1 CaF_2
Lime—washings..	69·4	26·4 CaO

From the results of the tests it is seen that—

1. The grade of fluorite produced is very high, and the fluorite recovery is good.

2. The barite product is poor in grade and is not pure white in colour. The barite recovery is very low, but there is no reason why this should not be as high as that obtained in tabling lot No. 1.

3. The lime product is poor in grade, but the recovery is fair.

CONCLUSIONS

The process as outlined above has promise of being commercially adaptable to the fluorspar deposits of the Madoc district, Ontario.

As the fluorite is the most valuable constituent, the process which produces the highest recovery of this mineral in a marketable grade would naturally be the one to use. The small scale tests show that a good recovery is made, with the production of a high grade fluorite product by calcining, slaking, washing, and tabling. Slaking and washing eliminates the lime content while the barite is removed by tabling.

The small scale laboratory work already performed should be followed up by large scale or tonnage check tests to prove more conclusively the commercial possibilities of this process for the separation of the minerals in the Madoc fluorspar deposits. The tests should be conducted in a rotary kiln furnace in preference to the vertical stack type of kiln, as the former will take a mixed feed of both coarse and fine, whereas the latter will only work successfully on the coarse material.

Before any commercial enterprise is undertaken, the large scale experimental tests should be carried out.

Test No. 167

THE MILLING AND CONCENTRATION OF COPPER-GOLD ORE FROM KITSALAS MOUNTAIN COPPER COMPANY, USK, B.C.

C. L. Dewar

A shipment of approximately 100 pounds of ore was received in July 1922, at the Ore Dressing and Metallurgical Laboratories, from the Kitsalas Mountain Copper Company, Usk, B.C.

The ore consisted of chalcocite, a little malachite (the copper carbonate), and some free gold, in a quartz gangue. A sample cut out from the shipment gave the following analysis:—

Copper	Cu	5·92 per cent
Iron	Fe	2·15 "
Sulphur	S	1·50 "
Silica	SiO_2	81·75 "
Lead	Pb	Trace.
Zinc	Zn	"
Gold	Au	0·46 oz. per ton.
Silver	Ag	2·17 " "

The company has a small mill on the property consisting of 8 x 10 Blake crusher, Gibson mill, amalgamation plates, and Wilfley concentrator. The ore is crushed in the Blake crusher, ground to 25 mesh in the Gibson mill with mercury, run over 5' x 6' amalgam plate, and concentrated on a Wilfley table. The concentrate has an assay value of—

Copper	Cu	26·88 per cent.
Gold	Au	0·65 oz.
Silver	Ag	16·45 oz.

By this treatment the free gold is amalgamated and the copper, silver, and remaining gold values recovered in the form of a copper concentrate.

Tests were conducted to determine methods of treatment by which a high recovery of the values in the ore could be obtained, with the production of a high grade copper concentrate which could be readily disposed of to the copper smelters.

Test No. 1.—Concentration by flotation.—1,000 grams of the ore previously crushed to 18 mesh were ground wet in a laboratory ball mill for 20 minutes, with 0·8 pound per ton of oil mixture—coal tar creosote 60 per cent, coal tar 40 per cent—and 0·2 pound per ton of Barrett No. 4. It was then floated in a laboratory Ruth flotation machine with the addition of 0·1 pound per ton frothing oil—P. T. and T. Co.'s No. 350. A concentrate and tailing were made.

86·9 per cent of the copper values, 82·1 per cent of the silver, but only 54·2 per cent of the gold values were recovered in a concentrate assaying 46·60 per cent copper, 17·75 ozs. silver, and 2·30 ozs. gold. The results of this test are shown in the table following, as is also the degree of crushing in table of screen analyses.

Test No. 2.—Table concentration followed by flotation of the table tailing.—1,500 grams of the ore previously crushed to 18 mesh were concentrated on a laboratory Wilfley table, making a concentrate, a middling, and a tailing. The middling product was re-run over the table, the concentrate from which was added to the first concentrate, and the tailing to the first tailing. As the pulp was passing over the table some small flakes of free gold (about 1 mm. diameter) were noticed. 51·0 per cent of the copper, 51·6 per cent of the gold, and 50·8 per cent of the silver was recovered in a concentrate assaying 53·80 per cent copper, 3·90 ozs. gold, and 20·60 ozs. silver.

The table tailing was reground for 40 minutes with 1 pound per ton of oil mixture—hardwood oil 50 per cent, wood tar 50 per cent—and floated in a laboratory Ruth machine. This further concentration of the values recovered 33·8 per cent of the copper, 23·0 per cent of the gold, and 33·6 per cent of the silver, in a concentrate assaying 34·45 per cent copper, 1·68 ozs. gold, 13·20 ozs. silver.

The combined table and flotation concentrates would give the following assay values and recoveries:—

Ratio of concentration—12 per cent of original feed, or 1 : 8·33. Assays—copper 43·9 per cent; gold, 2·77 ozs.; silver, 16·85 ozs. Recoveries—copper 84·8 per cent; gold 74·6 per cent; silver 84·4 per cent.

Test No. 3.—Amalgamation followed by flotation.—1,000 grams of the ore previously crushed to 18 mesh were ground wet for 30 minutes in a small ball mill, using a small ball charge, to which was added 100 grams of mercury. The mercury was recovered by panning the pulp away from it, and the pulp was reground for 15 minutes with 0·8 pound per ton coal tar mixture and 0·2 pound per ton Barrett No. 4. It was then floated in a laboratory Ruth flotation machine.

——	Copper per cent	Gold per cent	Silver per cent
Recoveries by amalgamation	....	32·8	5·7
Recoveries in flotation concentrate	80·4	36·6	74·7
Total recoveries	80·4	69·4	80·4
Grade of flotation concentrate	28·8	oz. 1·07	oz. 10·34

Test No. 4.—Flotation followed by table concentration.—1,000 grams of the ore previously crushed to 18 mesh were ground wet for 15 minutes with 0·8 pound per ton coal tar mixture and 0·2 pound per ton Barrett No. 4, and floated in a laboratory Ruth flotation machine. The tailing was dried, weighed and sampled, the remainder being tabled on a laboratory Wilfley table.

Product	Per cent by weight	Assay			Recoveries		
		Per cent Cu	Ozs. Au	Ozs. Ag	Per cent Cu	Per cent Au	Per cent Ag
Flot. conc.	10·6	40·30	1·85	14·40	75·1	46·7	70·8
Table conc.	3·3	25·35	3·95	11·60	14·5	38·2	18·7
Combined conc.	13·9	36·50	2·35	13·75	89·6	84·9	89·5

A screen analysis showing the degree of crushing is given in the table following:—

Test No. 5.—Table concentration followed by flotation.—1,000 grams of the ore, previously crushed to 18 mesh was ground for 15 minutes and tabled on a laboratory Wilfley table, making a concentrate and a tailing. The tailing was floated in a laboratory Janney machine with 0·87 pound per ton coal tar mixture, 0·22 pound per ton Barrett No. 4, 0·11 pound per ton P. T. and T. Co.'s No. 350. The concentrate was recleaned in a laboratory Ruth machine with the addition of 0·1 pound per ton No. 5 pine oil.

Product	Per cent by weight	Assay			Recoveries		
		Per cent Cu	Ozs-Au	Ozs-Ag	Per cent Cu	Per cent Au	Per cent Ag
Table conc.	8·8	49·33	5·25	18·80	72·4	83·3	71·3
Flot. conc.	2·1	39·80	2·40	17·00	14·0	9·1	15·4
Combined conc.	10·9	47·40	4·72	18·40	86·4	92·4	86·7

These recoveries do not include what would be recovered by the retreatment of the middling product, which would be returned to the circuit in practice. A screen analysis showing the degree of crushing, is given in the table following.

Test No. 6.—Table concentration followed by flotation.—1,200 grams of the ore, previously crushed to 24 mesh was tabled on a laboratory Wilfley table, making a concentrate, middling and tailing. The middling and tailing products were ground for 10 minutes with 0·9 pound per ton coal tar mixture, 0·2 pound per ton Barrett No. 4, and floated in a laboratory Ruth machine with the addition of 0·1 pound per ton P. T. and T. Co.'s No 350. A concentrate, middling, and tailing were made:—

Product	Per cent by weight	Assay			Recoveries		
		Per cent Cu	Ozs-Au	Ozs-Ag	Per cent Cu	Per cent Au	Per cent Ag
Table conc............	8·9	48·95	3·06	23·94	74·0	72·4	78·5
Flot. conc............	3·4	26·33	2·38	11·78	14·8	20·7	14·5
Combined conc.......	12·3	42·85	2·86	20·60	88·8	93·1	93·0

By retreating the flotation middling product, the recoveries could be increased slightly. The high copper tailing obtained in this test, namely 0·5 per cent Cu., and in the preceding one, 0·55 per cent Cu. is chiefly due to the presence of copper carbonate in the ore.

Test No. 7.—Table concentration of sized material followed by flotation.—While the recoveries obtained in tests Nos. 5 and 6 were as high as could be expected, test No. 7 was run to determine if the grade of the concentrate could be improved by sizing before table concentration, and thus produce a more suitable shipping product.

4,300 grams of the ore previously crushed to 18 mesh was sized on 35 and 70 mesh screens, as follows:—

−18+35..	1,975 grams
−35+70..	1,135 "
−70...	1,190 "
Total...	4,300 grams

The −18 +35 mesh material was tabled on a laboratory Wilfley table, making a concentrate, middling, and tailing. A clean concentrate was produced, but the material was a little coarse to obtain a clean tailing.

The −35 +70 mesh material was tabled in the same manner and the middlings combined. The amount of middling from the two lots was about the same, 13·7 per cent of the weight of the −18 + 35 mesh material and 14·5 per cent of the weight of the −35 +70 mesh material.

The −70 mesh material was ground in a ball mill for 10 minutes with 2 pounds per ton coal tar mixture and floated in a laboratory Ruth machine, making a concentrate and tailing.

The middling products from the table concentration of the −18 +35 and the −35 +70 mesh lots were reground and floated, using 1·8 pounds per ton of coal tar mixture, making a concentrate and tailing. The tailing products from the table concentration of the −18 +35 and the −35 +70 mesh lots were reground and floated, making a concentrate and tailing. A summary of the concentrates, assay values, and recoveries made in this test follows:—

Product	Per cent by weight	Assay			Recoveries		
		Per cent Cu	Ozs-Au	Ozs-Ag	Per cent Cu	Per cent Au	Per cent Ag
Table conc. −18+35..	3·07	61·14	4·84	22·86	30·42	34·15	30·96
Table conc..−35+70..	2·23	60·50	4·90	23·12	21·85	25·08	22·71
Flot. conc. −70.......	3·44	45·53	1·56	14·08	25·40	12·30	21·41
Flot. conc. (re-run middlings)..........	0·60	60·44	6·57	23·50	5·82	9·02	6·70
Combined conc........	9·34	55·20	3·76	19·70	83·50	80·60	81·80
Flot. conc. (re-run tailing).............	1·19	22·82	3·67	15·75	4·23	9·93	8·40
Combined conc.......	10·53	51·50	3·75	19·25	87·70	90·50	90·20

If the middling products from table concentration are only treated by flotation a combined concentrate is made assaying 55·2 per cent copper, 3·76 ozs. gold, 19·7 ozs. silver, with recoveries of 83·5 per cent of the copper, 80·6 per cent of the gold, and 81·8 per cent of the silver.

If the tailing products from table concentration are also reground and floated, the grade of the combined concentrate is lowered and the recoveries increased. A combined concentrate is made assaying 51·5 per cent copper, 3·75 ozs. gold, 19·25 ozs. silver, with recoveries of 87·7 per cent of the copper, 90·5 per cent of the gold, and 90·2 per cent of the silver.

The re-run middling and tailing and the −70 flotation tailing are high in copper, due to the carbonate content in the ore. A determination was made for carbonate content in the flotation tailing from the re-run of the middling product, which assayed 2·05 per cent copper, and was found to contain 1·70 per cent copper as carbonate.

The tables following give the results obtained from each test. A table of screen analyses is also given for tests Nos. 1, 4, and 5.

Test No.	Product	Weight grams	Per cent weight	Per cent total weight	Cu per cent	Cu % assay x % weight	Au-oz.	Au per cent x oz.	Ag-oz.	Ag per cent x oz.	Per cent of values		
											Copper	Gold	Silver
1	Concentrate.....	116	....	11·4	46·60	532·0	2·30	26·2	17·75	202·0	86·9	54·2	82·1
	Tailing.........	885	....	88·6	0·90	80·0	0·25	22·1	0·50	44·0	13·1	45·8	17·9
		1,001		100·0		612·0		48·3		246·0	100·0	100·0	100·0
2	Table conc......	87	....	5·9	53·80	317·2	3·90	23·0	20·60	121·5	51·0	51·6	50·8
	Flot. conc.......	90	....	6·1	34·45	210·3	1·68	10·24	13·20	80·5	33·8	23·0	33·6
	Flot. tailing.....	1,290	....	80·0	1·07	94·2	0·13	11·44	0·43	37·8	15·2	25·4	15·6
	Loss...........	33	....	....	....	....	....	....	....	....	....	....	....
		1,500		100·0		621·7		44·64		239·8	100·0	100·0	100·0
								mg.		mg.			
3	Concentrate.....	167	....	16·7	28·80	481·0	1·07	6·13	10·30	59·00	80·4	36·6	74·7
	Tailing.........	838	....	83·3	1·40	117·2	0·18	5·14	0·54	15·45	9·6	30·6	19·6
	Amalgam.......	....	....	....	....	....	....	5·50	....	4·50	....	32·8	5·7
		1,005		100·0		598·2	(0·48)	16·77	(2·30)	78·95	100·0	100·0	100·0
4	Flot. conc.......	106	....	10·6	40·30	423·8	1·85	19·60	14·40	152·7	75·1	46·7	70·8
	Flot. tailing.....	890	....	89·4	1·58	140·3	0·25	22·40	0·70	62·6	24·9	53·3	29·2
		996		100·0		564·1		42·00		215·3	100·0	100·0	100·0
	Table conc......	28	3·65	3·3	25·35	83·6	3·95	13·03	11·60	38·30	14·5	38·2	18·7
	Table tailing....	732	96·35	86·1	0·70	60·2	0·06	5·17	0·25	21·60	10·3	15·1	10·5
		760	100·00	89·4		143·8		18·20		59·90	24·9	53·3	29·2
5	Table conc......	88	8·8	....	49·33	434·0	5·25	46·20	18·8	165·4	72·4	83·3	71·3
	Flot. conc.......	21	2·1	....	39·80	83·6	2·40	5·04	17·0	35·7	14·0	9·1	15·4
	Flot. midd......	61	6·1	....	5·70	34·8	0·42	2·56	1·7	10·4	5·9	4·6	4·5
	Flot. tailing.....	823	83·0	....	0·55	45·7	0·02	1·64	0·25	20·5	7·7	3·0	8·8
		993	100·0			598·1		55·44		232·0	100·0	100·0	100·0
6	Table conc......	107	8·9	....	48·95	435·7	3·06	27·25	23·94	213·0	74·0	72·4	78·5
	Flot. conc.......	41	3·4	....	26·33	89·5	2·38	8·00	11·78	40·0	14·8	20·7	14·5
	Flot. midd......	77	6·4	....	3·93	25·2	0·40	2·56	1·88	12·0	4·3	6·7	4·4
	Flot. tailing.....	958	81·3	....	0·50	40·7	0·01	0·81	0·07	6·7	6·9	0·2	2·6
		1,183	100·0			588·5		38·62		271·5	100·0	100·0	100·0

Test No.	Product	Weight grams	Per cent weight	Per cent total weight	Cu per cent	Cu % assay x % weight	Au-oz.	Au per cent x oz.	Ag-oz.	Ag per cent x oz.	Per cent of values		
											Copper	Gold	Silver
7	*−18 +35*												
	Table conc...	132	6·7	3·07	61·14	187·7	4·84	14·87	22·86	70·13	30·42	34·15	30·96
	Table tailing.	1,569	79·6	36·54	0·95	34·7	0·12	4·38	0·54	19·73	5·62	10·01	8·73
		1,701	86·3	39·61		222·4		19·25		89·86	36·04	44·16	39·69
	−35 +70												
	Table conc...	96	8·5	2·23	60·50	134·8	4·90	10·93	23·12	51·39	21·85	25·08	22·71
	Table tailing	872	77·0	20·29	0·77	15·6	0·08	1·62	0·32	6·49	2·53	3·72	2·87
		968	85·5	22·52		150·4		12·55		57·88	24·38	28·80	25·58
	−18 +70		13·7										
	Middling......	434	14·5	10·11	5·47	55·3	0·54	5·46	2·16	21·84	8·96	12·52	9·66
	−70												
	Flot. conc.....	148	12·4	3·44	45·53	156·7	1·56	5·36	14·08	48·45	25·40	12·30	21·41
	Flot. tailing..	1,045	87·6	24·32	1·32	32·2	0·04	0·97	0·34	8·27	5·22	2·22	3·666
		1,193	100·0	27·76		188·9		6·35		56·72	30·62	14·52	25·07
	Totals.........	4·296		100·00		617·0		43·59		226·30	100·00	100·00	100·00
	Feed...........	4,300	....	....	5·92	592·0	0·46	46·00	2·17	217·00	....	....	....
	Re-run midd.												
	Flot. conc.....	23	5·9	0·60	60·44	36·2	6·57	3·94	23·50	14·10	5·82	9·02	6·70
	Flot. tailing..	367	94·1	9·51	2·05	19·5	0·16	1·52	0·65	6·18	3·14	3·50	2·94
		390	100·0	10·11		55·7		5·46		20·28	8·96	12·52	9·64
	Re-run table tailing.												
	Flot. conc.....	21	2·1	1·19	22·82	27·1	3·67	4·37	15·75	18·77	4·23	9·93	8·40
	Flot. tailing.	978	97·9	55·64	0·45	25·0	0·03	1·67	0·14	7·79	3·92	3·80	3·20
		999	100·0	56·83		52·1		6·04		26·56	8·15	13·73	11·60

SCREEN TESTS

Test No. 1—Screen Analysis of Flotation Tailing

Mesh	Weight grams	Per cent	Cum. per cent
+ 48	13	2·6	2·6
+ 65	17	3·4	6·0
+100	114	22·8	28·8
+150	93	18·6	47·4
+200	98	19·6	67·0
−200	165	33·0	100·0

Test No. 4—Screen Analysis of Table Tailing

Mesh	Weight grams	Per cent	Cum. per cent
+ 48	30	6·1	6·1
+ 65	77	15·7	21·8
+100	109	22·2	44·0
+150	63	12·9	56·9
+200	73	14·9	71·8
−200	138	28·2	100·0

Test No. 5—Screen Analysis of Flotation Tailing

Mesh	Weight grams	Per cent	Cum. per cent
+ 48	38	5·7	5·7
+ 65	118	17·6	23·3
+100	152	22·6	45·9
+150	82	12·2	58·1
+200	103	15·3	63·4
−200	179	26·6	100·0

CONCLUSIONS FROM RESULTS OF EXPERIMENTAL TESTS

The ore as submitted to the Testing Laboratories is amenable to treatment.

The presence of free gold would indicate that amalgamation should be the first step for the recovery of the values. While the free gold can be recovered in the table concentrate, its presence in the concentrate makes sampling difficult. It is always practicable to recover it by amalgamation as the cost is not excessive, and it provides a quick return of a portion of the values in the ore. Test No. 3 shows that 30 per cent of the gold values can be recovered in this manner.

The copper values are present chiefly as chalcocite with small amounts of the carbonate malachite.

By table concentration and flotation the greater part of the carbonate content is lost in the slime tailing, but the chalcocite content is recovered in a high grade concentrate, a most desirable product for the smelters.

The silver values seem to be closely associated with the copper, and are recovered with the copper in the concentrate. The recoveries of copper and silver are practically identical in the concentrate.

The sizing of the pulp before table concentration results in a higher grade copper concentrate and is worthy of consideration, from a shipping standpoint, in the disposal of the concentrate to the smelters.

The results of the tests show that it is possible to recover 90 per cent of each of the copper, gold, and silver values in the ore. A shipping concentrate assaying 50 per cent to 55 per cent copper is also obtained.

The following flow sheet is suggested for the treatment of the ore:—

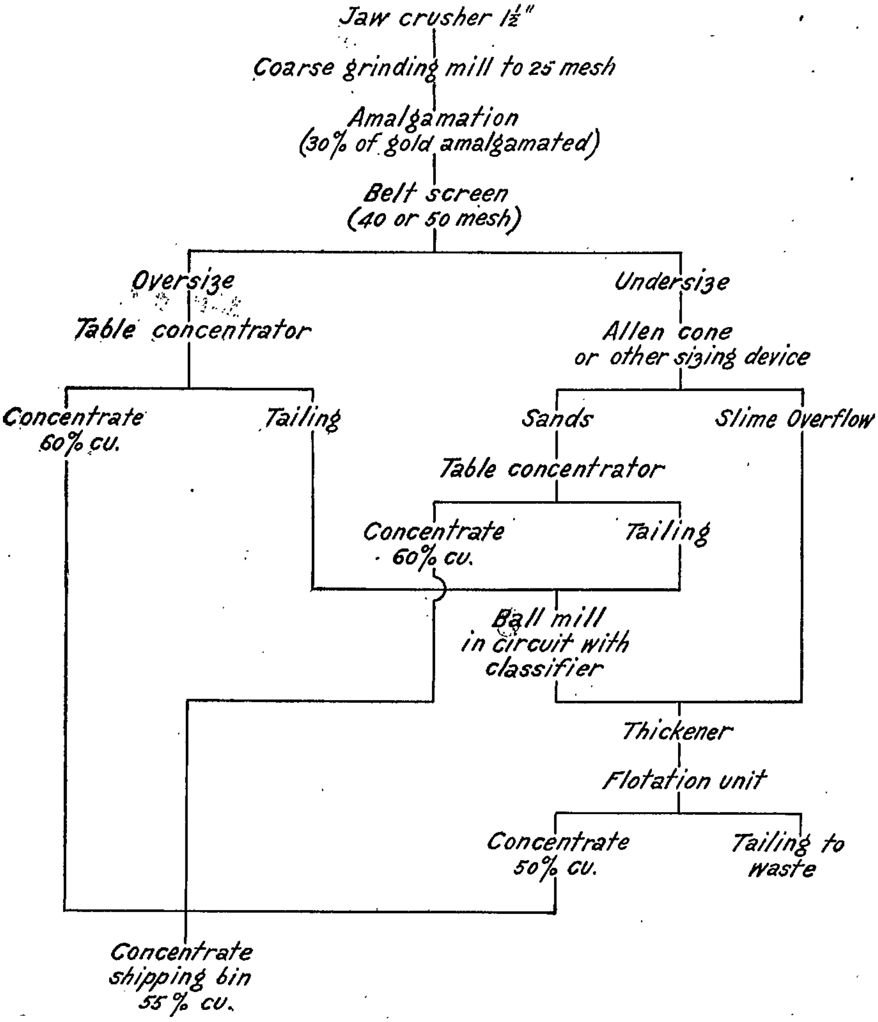


FIG. 11. Flow sheet suggested for treatment of copper-gold ore from Usk, B.C.

Test No. 168

THE CONCENTRATION OF THE ANTIMONY ORE FROM LAKE GEORGE, N.B.

C. S. Parsons

A shipment of 2,154 pounds of antimony ore was received in August 1922 from the North America Antimony Smelting Company, Limited, Lake George, N.B. This shipment was supposed to be a true and representative sample of the milling ore from the company's mine.

Experimental test work was desired to determine a concentration process, suitable for the treatment of the ore on the basis of 50 to 100 tons daily capacity. The company contemplated the erection of a mill on the results of the experimental work.

The whole shipment was crushed in a jaw crusher to 1¼" size and in rolls to ½" and sampled. The head sample cut out gave an analysis as follows:—

Antimony	11·65 per cent
Arsenic	0·37 "

HAND SORTING

The sample received did not contain sufficient lumps of pure stibnite to warrant hand picking.

JIGGING TESTS

Considerable stibnite was freed at ¼" size, and it is possible to obtain a jig product at this size, or finer, by careful manipulation. This concentrate would average about 50 per cent Sb. Jigging, however, would hardly be practicable owing to the fact that a clean tailing could not be produced, and that a middling product would always be present, which would prevent a high grade concentrate being made. This is based on the results of a test made in a small laboratory Richards jig.

Considering jigging from the stand-point of operation and costs in a small mill, it is quite evident that the use of jigs would not be advisable owing to the necessity of introducing re-crushing machinery for the jig tailing and middling, and would mean a more complexed flow sheet.

TABLE CONCENTRATION

Crushing in ball mill to tabling size (−14 mesh).—The following results were obtained by crushing to 14 mesh and sizing on 24, 35, and 50 mesh. The −50 mesh was not tabled, it being assumed that it would be sent to the flotation machines. The following sizes and weights were obtained:

Product	Weight grams	Per cent
−14+24 mesh	3,610	40·94
−24+35 "	1,817	20·61
−35+50 "	647	7·34
−50 "	2,742	31·10

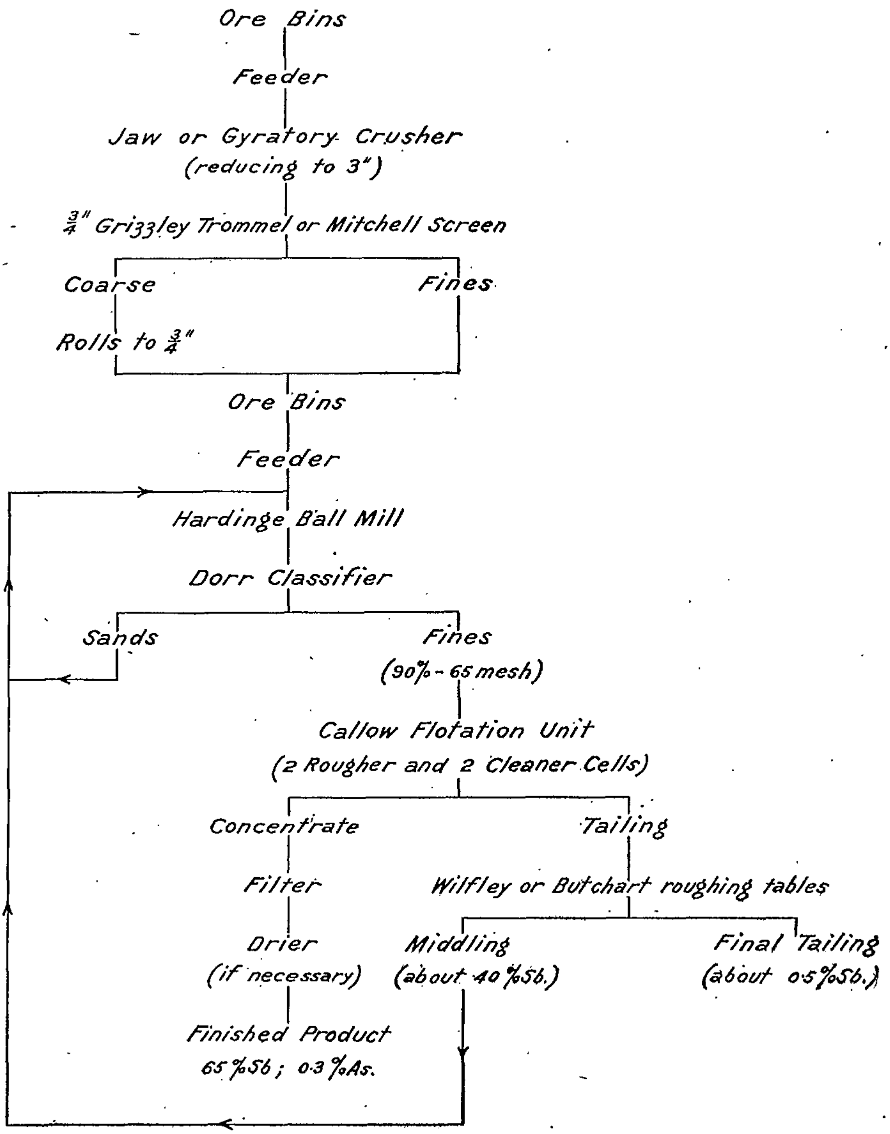


Fig. 12. Flow sheet Lake George antimony.

TABLE No. I

Tabling —14+24 mesh

Product	Weight gram	Per cent weight	Per cent Sb	Content Sb	Per cent Sb Recvy.	Per cent As	Content As	Per cent As. Recvy.
Concentrate..........	337	9·36	54·80	184·60	62·5	0·20	0·67	6·82
Mids from re-run.....	120	3·33	7·15	8·58	2·9	0·48	0·58	5·91
Middling.............	509	14·13	7·15	36·40	12·3	0·44	2·24	22·81
Tailing..............	2,636	73·16	2·50	65·90	22·3	0·24	6·33	64·46

Procedure and notes on above test.—A concentrate, middling, and tailing were first made. The concentrate was recleaned, using a special cleaner deck on the small table. A second middling product was obtained from the recleaning of the concentrate. The concentrate represents about as high grade a product as could be expected from this size. The middling is composed chiefly of unfreed mineral, and therefore, would have to be reground in order to recover the antimony content. The tailing contains most of the stibnite in an unfreed condition. These conclusions were arrived at by the examination of the products under the microscope.

TABLE No. II

Tabling —24+35 mesh

Product	Weight grams	Per cent weight	Per cent Sb	Content Sb	Per cent Sb Recvy.	Per cent As	Content As	Per cent As Recvy.
Concentrate..........	165	9·11	57·86	95·37	59·90	0·31	0·51	10·62
Mids from re-run.....	62	3·42	13·31	8·25	5·17	0·79	0·49	10·21
Tailing..............	1,584	87·46	3·52	55·76	34·93	0·24	3·80	79·16

Procedure and notes on above test.—This size tabled more readily than the preceding one. A concentrate and tailing were made, the middling being returned with the feed, giving two products only from the first pass. The concentrate was recleaned on a special cleaner deck. The recleaning of the concentrate produced a middling product. The concentrate represents practically as high grade a product as can be obtained on this size. Considerable float antimony mineral was observed passing over the table into the tailing. The middling is made up chiefly of unfreed mineral and is, therefore, a true middling, which would have to be recrushed in order to recover the antimony content. The tailing is very dirty, and on examination under the microscope, shows that about half the antimony is there as freed stibnite.

TABLE No. III

Tabling —35+50 mesh

Product	Weight grams	Per cent weight	Per cent Sb	Content Sb	Per cent Sb Recvy.	Per cent As	Content As	Per cent As Recvy.
Concentrate..........	63	9·53	58·02	36·55	57·14	0·28	0·18	8·41
Mids from re-run.....	36	5·45	10·45	3·76	5·88	0·53	0·19	8·88
Tailing..............	545	82·45	3·74	20·38	31·86	0·31	1·69	78·97
Slimes..............	17	2·57	19·25	3·27	5·11	0·47	0·08	3·74

Notes on above test.—A concentrate and tailing were made from the first pass of the ore, and the concentrate was recleaned on a special cleaner deck.

CONCLUSIONS FROM TABLE TESTS

It is possible to obtain a table concentrate with an analysis of 55 per cent Sb. if careful sizing is resorted to after crushing to 14 mesh, with a recovery of 55 per cent to 60 per cent of the stibnite in the ore. Here again the middling and tailing products would require regrinding to recover the unfreed antimony. Half the loss in the tailing from the −24 +35 and −35 +50 mesh products was due to flotation of the antimony when tabling. The finer the ore is crushed for tabling, the greater will be the loss from the antimony mineral floating off on the surface of the water, and reporting in the tailing.

Laboratory Flotation Tests on Crude Ore

See Table No. IV

Part of the original head sample was used for these tests. The procedure adopted was to use 1,000 grams of ore crushed dry to 20 mesh and then to approximately 65 mesh in a small ball mill wet.

SCREEN TEST OF FLOTATION FEED

Screen	Weight	Weight per cent	Weight Acc. per cent
+ 65	50	5·2	5·2
− 65+100	204	21·1	26·3
−100+150	137	14·1	40·4
−150+200	168	17·4	57·8
−200	410	42·2	100·0

Test No. 1.—1,000 grams of ore; mixture of 40 per cent coal tar and 60 per cent coal tar creosote, and pine oil for frothing; neutral pulp.

Test No. 2.—1,000 grams of ore; No. 1 K.K. oil from Southwestern Engineering Co., and pine oil for frothing; neutral pulp.

Test No. 3.—1,000 grams of ore; No. 1 K.K. oil and pine oil; pulp alkaline with 4 pounds per ton of soda ash.

Test No. 4.—1,000 grams of ore; No. 1 K.K. oil and pine oil; pulp alkaline with 4 pounds per ton of lime. Note, no stibnite would float in presence of lime.

Test No. 5.—1,000 grams of ore; No. 1 K.K. oil and pine oil; pulp acid with 10 pounds per ton sulphuric acid. Note, this test gave better results.

TABLE No. IV

Test No.	Product	Weight per cent	Analysis		Per cent recovery		Reagents
			Per cent Sb	Per cent As	Sb	As	
1	Concentrate	16·0	64·24	0·28	86·7	....	Coal tar mixture
	Middling	2·8	6·38	1·62	1·4	....	Pine oil
	Tailing	81·2	1·75	0·35	11·9	....	
2	Concentrate	17·7	61·80	0·62	81·4	35·6	K.K. oil
	Middling	4·0	4·50	1·04	13·4	13·5	Pine oil
	Tailing	78·3	0·88	0·27	5·2	50·9	
3	Concentrate	15·3	62·70	1·00	83·6	40·0	Soda ash
	Middling	4·1	13·10	1·44	4·7	15·4	K.K. oil
	Tailing	80·6	1·65	0·21	11·7	44·5	Pine oil
4	No flotation	....	....	....	....	....	Lime, K.K. oil, Pine oil
5	Concentrate	17·4	63·36	0·31	95·6	16·4	K.K. oil
	Middling	4·3	5·94	1·29	2·2	16·7	H_2SO_4
	Tailing	78·3	0·33	0·28	2·2	66·9	Pine oil

CONCLUSIONS

Flotation tests on the crude ore show that by crushing to about 65 mesh, a high grade concentrate 60 per cent to 65 per cent Sb. can be obtained with a recovery up to 95 per cent of the antimony values in the ore. The arsenic content in this concentrate is about 0·3 per cent. If no objectionable feature arises in the handling of this fine concentrate in the subsequent treatment, this would seem to be the simplest and most feasible method of treating the ore. An outline of such a flow sheet would be as follows:

Coarse crushing in breaker and rolls to ¾″ to ½″, and ball mill in closed circuit with classifier of Dorr type, to approximately 65 mesh. The overflow of the classifier would go to flotation machines, say of the Callow type, making a rougher concentrate which would be recleaned on a series of cleaner cells, and a tailing, which would go to roughing tables. These roughing tables are recommended on account of the high grade feed going to cells. They would act as a guard on the operation of the cells. The concentrate from the tables would be fed back to the circuit at the ball mill.

These deductions are based on the results obtained from the sample of ore received, and do not take into consideration any local conditions at the mine, or special market conditions regarding the physical character of the concentrate required for subsequent treatment. The process which gave the highest grade concentrate, the highest recovery, with simplicity of operation is recommended

In the construction of a small concentrator of 100 tons daily capacity, simplicity of design is important. Jigging and tabling followed by flotation would mean extra regrinding, classification, sizing, and thickening units, giving a lower grade final product, but a product much coarser than that obtained by straight flotation, and in this respect may be more suitable for subsequent refining. The final recovery of antimony values would be about the same in both cases. (See flow sheet, page 137.)

LARGE SCALE OR TONNAGE CHECK TESTS

A large scale, or tonnage check test, was made in a small pilot Callow flotation unit consisting of two rougher cells of the new flat bottom type, and two cleaner cells of the same design. The ore was reduced to ½" in a jaw crusher and rolls, and fed to a Hardinge conical ball mill in closed circuit with a Dorr classifier. The classifier was adjusted to give a −50 mesh product to the flotation cells. The test was made on 1,800 pounds of ore.

The concentrate assayed	58·21 per cent antimony
The tailing assayed	0·94 " "

The recovery is figured from the formula $R = \frac{(H - t)\ C}{(C - T)\ H} \times 100 = 93 \cdot 5\%$.

That is, 93·5 per cent of the antimony was recovered in a concentrate assaying 58·2 per cent antimony.

H = Assay of feed
T = Assay of tailing
C = Assay of concentrate

The reagents used for flotation were a mixture of 20 per cent coal tar and 80 per cent coal tar creosote, from the Dominion Tar and Chemical Co., Sault Ste. Marie, Ont., and sufficient steam distilled pine oil to maintain a good froth. Five pounds of sulphuric acid was added per ton of ore just before entering the cell.

CONCLUSIONS

The results from this test confirm the results obtained from the small scale laboratory tests. The grade of the concentrate is slightly lower, but this does not mean that a concentrate of higher grade, similar to that obtained in the small tests, cannot be obtained. After the operator becomes familiar with the operation of the flotation unit, better work can be expected. It is difficult to gauge the grade of the concentrate being produced, but with training and efficient manipulation, similar results to those of the small tests should easily be obtained.

Test No. 169

THE RECOVERY OF THE SILVER VALUES IN THE LEACHED CHLORIDIZED RESIDUES OF THE DOMINION REDUCTION COMPANY, COBALT, ONT.

C. S. Parsons and C. L. Dewar

A shipment of 15 bags of silver residues was received July 25, 1922, at the Ore Dressing and Metallurgical Laboratories, from the Dominion Reduction Company, Cobalt, Ont.

The material is the residue from the chloridizing, roasting and leaching of flotation and table concentrates produced by Cobalt mills, and contains silver to the extent of 20 to 30 ozs. per ton. It was desired that tests be conducted to determine, if possible, some satisfactory process for the recovery of the silver.

HEAD SAMPLES

Head samples from bags 1 and 2 were analysed with the following results:—

—	Bag 1	Bag 2
	Oz./ton	Oz./ton
Silver	29·0	21·68
	per cent	per cent
Copper	0·75	0·77
Sulphur	1·36	0·73
Arsenic	0·82	0·94
Cobalt	0·40	0·60
Silica	41·70	41·45
Alumina		23·43
Lime		2.60
Magnesia		3·42
Carbon		0·47
Chlorides	trace	trace

The following approximately complete analysis has been calculated upon the assumption that the metals are present as oxides:—

	Bag 2	
Fe_2O_3	17·16	per cent
Al_2O_3	23·43	“
SiO_2	41·45	“
CaO	2·60	“
MgO	3·42	“
SO_3	1·83	“
As_2O_3	1·24	“
Co_3O_4	0·78	“
CuO	0·96	“
C	0·47	“
Ag	0·07	“
Total	93·41	“

(Alkalies not determined)

A screen test on a sample from one bag of residues gave the following results:—

Mesh	Weight grams	Weight per cent	Weight cum. per cent
3	12	2·1	2·1
6	92	16·3	18·4
10	122	21·6	40·0
20	73	12·9	52·9
35	56	9·9	62·8
65	50	8·9	71·7
150	44	7·8	79·5
−150	116	20·5	100·0

Test 1.—This test was made to determine if any separation could be effected by classification. A grab sample was taken from bag 1, and screened on 10 mesh, as the +10 mesh material would be too coarse for the small Richards pulsator classifier used.

Weight of sample: 2500 grams   Moisture 16 p.c.
Dry weight: 2100 "   Silver 33·0 oz./ton.

Product	Weight grams	Weight per cent	Silver oz./ton	Per cent of total
+10 mesh	570	27·2	18·43	15·1
Class 1	375	17·9	21·14	11·4
" 2	177	8·4	23·16	5·9
Overflow	665	31·7	43·35	41·5
Slime	114	5·4	53·10	8·7
Loss (calculated)	199	9·4	60·8	17·3

The loss was very fine slime that would not settle in the slime box. The test shows that no separation could be made by classification.

FLOTATION

Test 2.—A sample of 1,175 grams (15 per cent moisture) was taken and ground in a ball mill for half an hour with 0·7 lb. per ton of a coal tar mixture (coal tar 40 per cent, coal tar creosote 60 per cent) and 0·2 lb. per ton of Barrett No. 4, then floated in a Ruth machine. In this test there was too much coarse material, so for subsequent tests the residues were crushed to −14 mesh and then ground half an hour in the ball mill. In all nine flotation tests were made. The conditions of tests and data desired therefrom are given in tabular form in the accompanying table. (Page 146.) As a whole the results were not very satisfactory, for in all tests the recovery in concentrate was less than 66 per cent, and in the majority it did not exceed 50 per cent. It is possible that the recovery might be increased by the retreatment of the middling, but in any case the tailing would still carry 11·0 or more ozs. per ton.

From the results obtained it is deemed impossible to satisfactorily treat these residues by flotation alone.

AMALGAMATION

Test 14.—A 1,000 gram sample of −35 mesh residues was ground in the pebble mill with 100 grams of mercury and 5 grams of salt (NaCl) for 2 hours. The pulp was panned and the mercury recovered.

Silver amalgamated.. 13·4 per cent

RUSSELL PROCESS

Test 8.—A 200 gram sample of −14 mesh residues was ground for half an hour in the ball mill, then agitated with 1,000 c.c. of solution (0·75 per cent $CuSO_4$ and 1·25 per cent $Na_2S_2O_3$) for 18 hours.

Feed.................. Ag 29·0 oz./ton
Tailing.............. Ag 23·9 " .... Extraction 17·6 per cent

Cyanidation Tests

Test 13.—A 200 gram sample of tailing from flotation test 7 was agitated with 100 c.c. water, 5 grams lime, and 5 grams cyanide (25 per cent NaCN) for 26 hours.

Feed	Ag 11·7 oz./ton		
Tailing	Ag 8·26 "	Extraction	29·4 per cent
Recovery by flotation (conc.—mids.)			69·2 "
Recovery by cyanidation			9·1 "
Total recovery			78·3 "

Test 15.—A 500 gram sample of −200 mesh residues from bag 2 was agitated with 3,000 c.c. of 3 per cent sulphuric acid solution for 24 hours. It was then washed, made alkaline, cyanide added and agitated for 48 hours.

Solution at beginning of agitation	0·40 per cent CaO, 0·14 per cent KCN		
Feed	29·0 oz./ton Ag		
Tailing	5·96 oz./ton Ag	Extraction	79 per cent

Test 17.—A 750 gram sample of −200 mesh residues was agitated with 2,200 c.c. of sulphuric acid solution (3 per cent H_2SO_4) for 5 hours, washed, made alkaline, cyanide added and agitated for 27 hours. The agitation was for 8 hours each day, and the total time of contact was 80 hours.

Solution at beginning of agitation	0·01 per cent CaO, 0·05 per cent KCN		
Feed	29·0 oz./ton Ag		
Tailing	9·5 oz./ton Ag	Extraction	67 per cent

Test 19.—A 500 gram sample of −200 mesh residues was agitated with sulphuric acid solution (2,250 c.c. of 6 per cent H_2SO_4) for 6 hours, washed, made alkaline, cyanide added and agitated for 85 hours.

Feed	21·68 oz./ton Ag		
Tailing	5·72 oz./ton Ag	Extraction	73·6 per cent

Test 20.—A 500 gram sample of −200 mesh residues was agitated in cyanide solution for 85 hours.

Feed	21·68 oz./ton Ag		
Tailing	6·94 oz./ton Ag	Extraction	68 per cent

Test 21.—A 500 gram sample of −200 mesh residues was agitated with 4,000 c.c. solution containing 20 grams bleaching powder (commercial) washed and decanted, then agitated with cyanide solution for 85 hours:—

Feed	21·68 oz./ton Ag		
Tailing	6·44 oz./ton Ag	Extraction	70 per cent

Test 22.—It was thought that the coal in the residues was precipitating part of the silver dissolved. To determine whether this was the case or not, the coal was floated with pine oil No. 5 from 500 grams residues.

The cleaned residues were then agitated with sulphuric acid solution (3½ per cent H_2SO_4) for 40 hours, then cyanided for 95 hours:—

Feed.................. 21·68 oz./ton Ag
Tailing............... 6·98 oz./ton Ag............. Extraction 68 per cent

1,000 c.c. of the pregnant solution was divided into two parts A and B. 400 c.c. of A was put through a filter containing very finely ground coal. The filtrate was evaporated and the residue assayed. The B part was merely assayed.

	A	B
Silver gm/litre	66·0	74·0

These results showed that the effect of the small amount of coal present in the residues is negligible.

SUMMARY AND CONCLUSIONS

(1) No separation was effected by classification.

(2) Flotation alone does not give very satisfactory results.

(3) Amalgamation gives the very low recovery of 13 per cent.

(4) Hyposulphite lixiviation was not successful in recovering the silver.

(5) Cyanidation alone gives a fair recovery—68 per cent—which can be increased by a preliminary acid wash (70-79 per cent).

(6) The recovery by flotation and cyanidation of the flotation tailing was the same as that obtained by acid washing and cyaniding for about twice the length of time. It should be noted, though, that the flotation concentrate would require further treatment while the silver dissolved by the cyanide is readily recovered as precipitate. This is a point in favour of all cyanidation.

(7) Bleaching powder is not so efficacious as the acid for a preliminary wash, but it requires much less lime or soda ash to produce the required protective alkalinity for subsequent cyanidation.

(8) The small amount of coal in the residues has little effect upon the recovery of the silver by cyanidation.

This concludes the test work completed in 1922, but as the recoveries obtained were not sufficiently high, other tests will be made.

Flotation Tests

Silver residues. Dominion Reduction Co.

Test No.	Product	Weight per cent	Silver oz./ton	Silver per cent x oz.	Re-coveries	Remarks
2	Concentrate... Tailing........	18·9 81·1	105·1 12·8	1,985 1,053	65·9 34·1	Coal tar and coal tar creosote (40 per cent and 60 per cent) 0·7 lb./ton. Barrett No. 4, 0·2 lb./ton.
3	Concentrate... Middling...... Tailing........	4·8 19·5 75·7	76·8 46·6 21·6	383 937 1,728	12·6 30·7 56·7	−14 mesh material ground for half hour with coal tar mixture 0·7 lb./ton. Barrett No. 4, 0·2 lb./ton. Floated with $Na_2S, 9H_2O$, 8 lb./ton and P.T.T. No. 350, 0·1 lb./ton.
4	Concentrate... Middling...... Tailing........	5·5 14·0 80·5	145·5 36·6 19·8	800 512 1,594	27·5 17·6 54·9	Coal tar mixture 0·7 lb./ton. Barrett No. 4, 0·25 lb./ton. Caustic soda 4·0 lb./ton. P.T.T. No. 350, 0·2 lb./ton.
5	Concentrate... Middling...... Tailing........	4·3 14·9 80·8	81·5 42·4 23·2	351 632 1,875	12·3 22·2 65·5	Ground 25 min. with coal tar mixture 0·7 lb./ton. Barrett No. 4, 0·25 lb./ton. Then 5 min. with 8·0 lb./ton Na_2S. $9H_2O$. Floated with P.T.T No. 350, 0·1 lb./ton. Concentrate cleaned with Barrett No. 4 and W.G. tar.
7	Concentrate... Middling...... Tailing........	5·1 16·8 78·1	287·0 35·0 11·7	1,463 588 914	49·4 19·8 30·8	Coal tar mixture 1·8 lb./ton. Pine oil No. 5, 0·15 lb./ton. Concentrate cleaned with Barrett No. 4, 0·15 lb./ton and Pine oil No. 4.
9	Concentrate 1. Concentrate 2. Tailing........	21·0 10·3 68·7	78·4 40·6 11·7	1,646 419 804	57·3 14·6 28·1	Ground half hour with coal tar mixture 1·15 lb./ton, No. 28 F.P.L. Ketone to froth, 4 lb./ton caustic soda. Made first concentrate then added water and 4 lb./ton $CuSO_4$ and little Ketone.
10	Concentrate... Tailing........	16·9 83·1	50·0 22·8	845 1,899	30·8 69·2	Coal tar mixture 1·15 lb./ton and NaCl 4·0 lb./ton (both to mill). Pine oil No. 5, 0·15 lb./ton to cell.
11	Concentrate... Tailing........	18·0 82·0	49·0 21·2	882 1,738	33·7 66·3	Coal tar mixture 1·15 lb./ton and NaCl 4·0 lb./ton (both to mill). 10 c.c. 40 per cent solution $Na_2S\ 9H_2O$ (agitated in cell 7 minutes). Pine oil No. 5, 0·15 lb./ton. Pulp neutral to litmus.
12	Concentrate... Tailing........	17·9 82·1	74·8 15·3	1,339 1,256	51·6 48·4	Coal tar mixture. Pine oil No. 5. NaCl 4·0 lb./ton. 10 c.c. 40 per cent solution of $Na_2S\ 9H_2O$.

Test No. 170

GOLD ORE FROM BEAR RIVER, BEDWELL SOUND, VANCOUVER I.

R. K. Carnochan

A shipment of 198·5 pounds of gold ore was received at the Ore Dressing and Metallurgical Laboratories on September 14, 1922, from J. B. Woodworth, Vancouver, B.C. The ore was obtained from a mining property near Bear river, Bedwell sound, Vancouver island, and consisted of mineralized quartz. Sphalerite, galena, pyrite, chalcopyrite, and arsenopyrite are present in varying amounts. Silver and gold are present, and while these are closely associated with the metallic sulphides, free gold is noticeable on close examination.

The presence of metallic gold in the ore makes sampling somewhat difficult. An average of four samples gave the following analysis:—

Gold	Au	5·63 ounces.
Silver	Ag	4·28 "
Arsenic	As	0·41 per cent.
Copper	Cu	0·42 "
Lead	Pb	0·78 "
Zinc	Zn	4·70 "
Iron	Fe	9·33 "
Sulphur	S	10·00 "
Silica	SiO_2	69·00 "
Alumina	Al_2O_3	2·26 "
Lime and magnesia	CaO, MgO	trace

Experimental test work was desired to determine a suitable method of treatment, and as the chief value is in its gold content, the recovery of the gold is of primary importance and the other values secondary.

The test work conducted on the ore was as follows:—

First: Amalgamation
Second: Cyanidation.
Third: Table concentration and cyanidation of the table tailing.
Fourth: Table concentration, flotation, and cyanidation of the tailing.

AMALGAMATION

Test No. 1.—1,000 grams of heads −40 mesh were ground in a small pebble jar for 3 hours with 100 grams of mercury and 400 c.c. of water. The charge was then panned and 83·7 grams of mercury was recovered. The tailing was dried and sampled for assay.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Tailing	2·70	3·34	48·0	78·0
Amalgamated	2·93	0·94	52·0	22·0
Heads −40	5·63	4·28	100·0	100·0

Test No. 2.—This test was made similar to test No. 1, with 5 grams of sodium hydroxide added to the mill charge. The mercury recovered weighed 92·2 grams.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Tailing	1·96	3·04	34·8	71·0
Amalgamated	3·67	1·24	65·2	29·0
Heads −40	5·63	4·28	100·0	100·0

Test No. 3.—500 grams of heads, out of which some metallics had been removed, were reduced to −200 mesh and then ground for 3 hours in a small pebble jar with 50 grams of mercury, 200 c.c. of water, and 10 grams of sodium hydroxide. The charge was then panned and 47·9 grams of mercury were recovered. The tailing was dried and sampled for assay.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Tailing	1·92	3·06	34·1	71·5
Metallics	0·17	0·04	3·0	0·9
Amalgamated	3·54	1·18	62·9	27·6
Heads −200	5·63	4·28	100·0	100·0

Total recovery Au	65·9 per cent.
Total recovery Ag	28·5 "

SUMMARY AND CONCLUSIONS FROM RESULTS OF AMALGAMATION TESTS

The above results show that 50 per cent to 65 per cent of the gold values can be recovered by amalgamation, crushing the ore to 40 mesh. Finer grinding as in test No. 3 does not seem to improve the results. The consumption of mercury was high in all these tests, and would be prohibitive in practice, but this may be due to some foreign matter, such as oil, getting into the sample, as the addition of sodium hydroxide reduces the mercury loss and improves the recovery. Amalgamation for the recovery of 50 per cent of the gold values in the ore may, therefore, be feasible, as it entails very little additional cost of installation, and provides an early return of one-half of the values in the ore.

CYANIDATION

Test No. 4.—1,142 grams of the ore −100 mesh was agitated for 82 hours in 6,000 c.c. of water to which 15 grams of sodium cyanide and 10 grams of lime had been added.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Tailing	1·36	2·53	24·2	59·1
Cyanided	4·27	1·75	75·8	40·9
Heads −100	5·63	4·28	100·0	100·0

SUMMARY AND CONCLUSIONS FROM RESULTS OF CYANIDATION TEST

Seventy-five per cent of the gold and 40 per cent of the silver was recovered by straight cyanidation of the ore after grinding to 100 mesh. The consumption of cyanide was high, and the recoveries would be considered low on such a high grade ore. Grinding to 200 mesh would improve the extraction. As a large proportion of the precious metals values is very closely associated with the sulphides in the ore, a long time of agitation is necessary for the extraction of these values.

Table Concentration and Cyanidation of the Table Tailing

Test No. 5.—1,252 grams of heads −40 mesh were tabled on a small Wilfley table, making a concentrate and a tailing.

Product	Weight grams	Au oz./ton	Ag oz./ton	Au grams x oz./ton	Ag grams x oz./ton	Per cent Au value	Per cent Ag value
Table concentrate	259	22·34	15·57	5,786	4,033	82·1	75·3
Table tailing	975	0·70	1·32	682	1,287	9·7	24·0
Loss	18	32·28	2·17	581	39	8·2	0·7
Heads −40	1,252	5·63	4·28	7,049	5,359	100·0	100·0

Eight hundred and fifty grams of the table tailing were ground to pass 100 mesh and agitated for 8 hours with 6,000 c.c. of water to which had been added 15 grams of sodium cyanide and 10 grams of lime.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Cyanide tails	0·42	0·66	5·8	12·0
Cyanided	0·28	0·66	3·9	12·0
Table tail −100	0·70	1·32	9·7	24·0

Product	Per cent Au value	Per cent Ag value
Table concentrate	82·1	75·3
Cyanided	3·9	12·0
Total recovery	86·0	87·3

Test No. 6.—1,307 grams of heads −80 mesh were tabled on a small Wilfley table making a concentrate and a tailing.

Product	Weight grams	Au oz./ton	Ag oz./ton	Au grams x oz./ton	Ag grams x oz./ton	Per cent Au value	Per cent Ag value
Table concentrate	302	20·60	13·39	6,221	4,044	84·5	72·3
Table tailing	922	0·66	1·20	608	1,106	8·3	19·8
Loss	83	6·37	5·35	529	444	7·2	7·9
Heads −80	1,307	5·63	4·28	7,358	5,594	100·0	100·0

Eight hundred and seven grams of the table tailing was ground to pass 100 mesh and agitated for 8 hours with 6,000 c.c. of water to which had been added 15 grams of sodium cyanide and 10 grams of lime.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Cyanide tailing	0·46	0·84	5·8	13·9
Cyanided	0·20	0·36	2·5	5·9
Table tailing −100	0·66	1·20	8·3	19·8

Product	Per cent Au value	Per cent Ag value
Table concentrate	84·5	72·3
Cyanided	2·5	5·9
Total recovery	87·0	78·2

SUMMARY AND CONCLUSIONS FROM RESULTS OF TABLE CONCENTRATION AND CYANIDATION OF THE TABLE TAILING

By table concentration 80 per cent to 85 per cent of the gold values and 70 per cent to 75 per cent of the silver values are recovered in a high grade concentrate, assaying over 20 ozs. in gold and about 15 ozs. in silver. This is a marketable concentrate that could be disposed of to the smelter. By cyanidation of the table tailing after grinding to 100 mesh, 2·5 per cent to 4 per cent additional gold values are recovered. Finer grinding to 200 mesh for cyanidation would improve the extraction, but these results show the difficulty of obtaining good extractions by cyanidation, due to the close association of the precious metal values with the sulphides in the ore.

TABLE CONCENTRATION, FLOTATION AND CYANIDATION OF THE TAILING

Test No. 7.—2,322 grams of heads −40 mesh out of which some metallics had been removed, were tabled on a small Wilfley table, making a concentrate and a tailing. The table tailing was then floated in a small Ruth machine, making a concentrate, a middling, and a tailing.

Product	Weight grams	Au oz./ton	Ag oz./ton	Au grams x oz./ton	Ag grams x oz./ton	Per cent Au value	Per cent Ag value
Metallics	....	·17	·04	295	93	3·0	0·9
Table conc	562	20·20	15·68	11,352	8,812	86·3	82·7
Flot. conc	172	7·94	9·94	1,366	1,710	10·4	16·0
Flot. midd	103	·46	·44	47	45	0·3	0·4
Flot. tail	1,452	trace	trace	....	....	....	....
Loss	33	....	....	....	....	....	....
	....	....	....	13,160	10,660	....	....
Heads −40	2,322	5·63	4·28	13,073	9,938	100·0	100·0

1,303 grams of the flotation tailing was ground to pass 200 mesh and agitated for 15 hours with 6,000 c.c. of water to which had been added 15 grams of sodium cyanide and 10 grams of lime. The tailing from this operation gave only a faint trace of gold and silver upon assay.

Product	Per cent Au value	Per cent Ag value
Metallics	3·0	0·9
Table concentrate	86·3	82·7
Flotation concentrate	10·4	16·0
Total recovery	99·7	99·6

The flotation middling should have been put with the flotation tailing and both cyanided together. This would have increased the recovery.

Test No. 8.—2,292 grams of heads −80 mesh, out of which some metallics had been removed, were tabled on a small Wilfley table, making a concentrate and a tailing. The table tailing was then floated in a small Ruth machine, making a concentrate, a middling, and a tailing.

Product	Weight grams	Au oz./ton	Ag oz./ton	Au grams x oz./ton	Ag grams x oz./ton	Per cent Au value	Per cent Ag value
Metallics	....	0·17	0·04	390	92	3·3	0·9
Table conc	571	18·22	14·86	10,404	8,485	86·9	84·3
Flot. conc	248	4·52	5·78	1,121	1,433	9·4	14·2
Flot. midd	114	0·26	0·32	30	36	0·2	0·4
Flot. tail	1,175	0·02	0·02	23	23	0·2	0·2
Loss	184	....	....	....	....	....	....
	....	....	....	11,968	10,069	....	....
Heads −80	2,292	5·63	4·28	12,904	9,810	100·0	100·0

One thousand and forty-eight grams of the flotation tailing was ground to pass 200 mesh and agitated for 8 hours with 6,000 c.c. of water to which had been added 15 grams of sodium cyanide and 10 grams of lime.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Cyanide tailing	trace	trace		
Cyanided	0·02	0·02	0·2	0·2
Flotation tailing −200	0·02	0·02	0·2	0·2

Product	Per cent Au value	Per cent Ag value
Metallics	3·3	0·9
Table concentrate	86·9	84·3
Flotation concentrate	9·4	14·2
Cyanided	0·2	0·2
Total recovery	99·8	99·6

The flotation middling should have been put with the flotation tailing, and both cyanided together. This would have given a better recovery.

Test No. 9.—160 pounds of ore, −40 mesh, out of which some metallics had been removed, was tabled on a large Wilfley table, making a concentrate and a tailing. The tailing was then floated in one compartment of a large Ruth machine, making a concentrate and a tailing. The flotation concentrate was then re-run in a small Ruth machine, making a clean concentrate and a middling.

Product	Weight pounds	Au oz./ton	Ag x oz./ton	Au pounds x oz./ton	Ag pounds x oz./ton	Per cent Au value	Per cent Ag value
Metallics		0·17	0·04	27·2	6·4	3·0	0·9
Table conc	40·25	21·16	11·83	851·7	476·2	94·5	69·5
Flot. conc	2·90	3·52	6·02	10·2	17·5	1·1	2·6
Flot. midd	6·10	0·26	0·34	1·6	2·1	0·2	0·3
Flot. tail	95·75	0·08	0·22	7·7	21·1	0·9	3·1
Loss	15·00	0·16	10·77	2·4	161·5	0·3	23·6
Heads −40	160·00	5·63	4·28	900·8	684·8	100·0	100·0

Fifteen hundred grams of flotation middling and flotation tailing mixed in proper proportion were ground to pass 200 mesh and agitated for 22 hours with 6000 c.c. of water to which had been added 15 grams of sodium cyanide and 10 grams of lime.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Cyanide tailing	0·00	0·00	0·0	0·0
Cyanided	0·09	0·23	1·1	3·4
Flot. middling and tailing −200	0·09	0·23	1·1	3·4

Product	Per cent Au values	Per cent Ag values
Metallics	3·0	0·9
Table concentrate	94·5	69·5
Flot. concentrate	1·1	2·6
Cyanided	1·1	3·4
Total recovery	99·7	76·4

SUMMARY AND CONCLUSIONS FROM RESULTS OF TABLE CONCENTRATION, FLOTATION, AND CYANIDATION OF THE TAILING

The results of the above three tests show a complete recovery of the precious metal values in the ore. Over 99 per cent of the values are recovered by table concentration and flotation in a high grade concentrate, assaying over 20 ozs. in gold, and about 15 ozs. in silver. By cyanidation of the tailing the remaining precious metals values are extracted. Cyanidation would probably not be necessary, as practically all the values are recovered by concentration. In test No. 9 there is shown a recovery of 69·5 per cent of the silver values in the table concentrate. This does not check with the two former tests, where 82·7 per cent and 84·3 per cent are recovered in similar concentrates, and is probably due to volatilization of some of the silver in making the determination.

Treatment of Concentrate

A complete analysis of the table concentrate obtained from test No 7, was as follows:—

Gold	Au	20·20	ozs.
Silver	Ag	15·68	"
Arsenic	As	1·57	per cent
Copper	Cu	0·92	"
Lead	Pb	2·50	"
Zinc	Zn	15·20	"
Iron	Fe	31·67	"
Sulphur	S	39·28	"
Silica	SiO_2	8·00	"
Alumina	Al_2O_3	0·72	"
Lime and magnesia	MgO, CaO	trace	

The above analysis would be typical of the concentrates obtained by table concentration and flotation on the grade of ore submitted for test purposes. If amalgamation was introduced before table concentration and flotation, the gold and silver values in the concentrate would be reduced by the amount of these values recovered by amalgamation. Otherwise the grade would be about the same.

Test No. 10—Cyanidation of concentrates.—1,500 grams of table and flotation concentrate from test No. 9 mixed in proper proportions, were ground to pass 200 mesh and agitated for 174 hours in 6,000 c.c. of water, to which were added 45 grams of sodium cyanide and 40 grams of lime. The tailing from this treatment assayed: gold, 2·00 ozs. per ton; silver, 4·58 ozs. per ton. This tailing was agitated again for 119 hours in 6,000 c.c. of water, to which were added 15 grams of sodium cyanide and 10 grams of lime. The tailing from this second agitation assayed: gold, 0·25 ozs. per ton; silver, 2·48 ozs. per ton. The consumption of cyanide was approximately 31 pounds per ton of concentrate.

Product	Au oz./ton	Ag oz./ton	Per cent Au values	Per cent Ag values
Cyanided	19·72	8·96	98·7	78·3
Tailing	0·25	2·48	1·3	21·7
Concentrate	19·97	11·44	100·0	100·0

A test was conducted by cyaniding the roasted concentrate, but the results were not encouraging.

SUMMARY AND CONCLUSIONS FROM RESULTS OF TREATMENT OF CONCENTRATES

The gold values in the concentrates can be recovered by cyanidation. Fine grinding to 200 mesh is necessary. Better results are obtained by cyaniding the raw concentrates than by roasting before cyanidation. The results show a recovery of 98·7 per cent of the gold values and 78·3 per cent of the silver values in the concentrates, which is considered a high recovery for this class of material. The time of agitation is excessive and the consumption of cyanide is fairly high, but both these could no doubt be considerably reduced in practice by better agitation and the addition of the proper amounts of reagents.

CONCLUSIONS ARRIVED AT FROM RESULTS OF EXPERIMENTAL TESTS

The ore as submitted to the testing laboratories is amenable to treatment.

By amalgamation after crushing to 40 mesh, 50 per cent of gold values and 25 per cent of the silver can be recovered.

By table concentration and flotation, 99 per cent of the gold values and 99 per cent of the silver values can be recovered in the form of a concentrate of metallic sulphides assaying 20 ozs. gold, 15 ozs. silver.

The concentrate can be disposed of to the smelters without further treatment.

The concentrate can be cyanided with a high recovery of the gold content.

In the treatment of this ore, two methods have to be considered, depending largely on local conditions, such as transportation and freight rates, favourable smelter treatment charges, etc.

First:—The reduction of the ore to 40 mesh, table concentration, and flotation of the table tailing. Whether it is more profitable to dispose of the high grade concentrate thus produced to the smelters, or to grind it to 200 mesh and extract the precious metal values by cyaniding.

Second:—The reduction of the ore to 40 mesh, amalgamation to recover the free gold, table concentration and flotation of the table tailing. The disposal of the high grade concentrate thus produced, which will contain about 10 ozs. of gold after amalgamation of the ore, or to regrind it to 200 mesh, and extract the precious metal values by cyaniding.

Test No. 171

THE CONCENTRATION OF THE RADIO-ACTIVE MINERALS IN A SHIPMENT OF PEGMATITE FROM KEARNEY, ONT.

R. K. Carnochan

A shipment of 251 bags of pegmatite supposed to contain radio-active minerals, gross weight 19,019 pounds, was received on October 9, 1922, at the Ore Dressing and Metallurgical Laboratories, from Messrs. Ryan and Mann, Kearney, Ont.

The pegmatite is composed of coarsely crystallized red feldspar quartz and black mica. No radio-active mineral could be seen in hand specimens of the pegmatite.

Tests were desired to determine if the pegmatite contained sufficient radio-active minerals to make it a commercial source of supply of radium, and to determine if the radio-active minerals could be recovered by concentration with the production of a marketable concentrate.

The shipment as received was divided into two lots marked No. 1 and No. 2. Lot No. 1 consisted of 220 bags, net weight 16,449 pounds, and Lot No. 2 contained 31 bags, net weight 2,372 pounds.

Lot No. 1

The whole lot was crushed by a breaker and rolls to ½″ and a 177 pound sample cut out by means of a Vezin sampler. The 177 pounds was crushed to −40 mesh by means of a small breaker and a small set of rolls. In screening on 40 mesh a lot of mica was obtained as oversize. The −40 mesh and the +40 mica products were cut by means of a Jones riffle into the following:—

Head sample—	−40	·672	pound
	+40 mica	·012	"
		·684	"
Test portion—	−40	20·5	pounds
	+40 mica	·375	"
		20·875	"
Remainder—	−40	151·5	pounds
	+40 mica	2·75	"
		154·25	"

The 20·5 pounds of −40 of the test portion was run on a small Wilfley table making a concentrate, middling, and tailing. The middling was re-run 3 times. In the last re-run any middling produced was put with the tailing. This gave only three products from the tabling—a concentrate, a tailing, and a slime product. These products were dried. The table concentrate was treated by a hand magnet to remove any very magnetic material, and then put through an Ullrich magnetic separator, making a concentrate and tailing. The table tailing was put through the Ullrich magnetic separator also making a concentrate and tailing. Samples of the feed and all products of the test work were tested for radio-activity, and the following table shows the results obtained:—

Product	Weight pound	Radio-activity
+40 mica	·375	nil
Magnetic from table concentrate	·009	"
Ullrich conc. from table conc	·030	1·52
Ullrich tailing from table conc	·062	0·46
Ullrich conc. from table tailing	·187	nil
Ullrich tailing from table tailing	17·000	"
Table slime	1·718	"
Loss	1·494	....
Feed	20·875	nil

In testing the radio-activity of the above products, the standard used was a sample of low grade carnotite ore containing about 2 per cent uranium oxide. This standard has a radio-activity of 6·90.

Lot No. 2

This lot, after the removal of a few specimens, was crushed to ½" and a sample of 240·5 pounds cut out by means of a Vezin sampler. The sample was crushed to −40 mesh and a test portion of 15 pounds was cut by a Jones riffle. This test portion was tabled in the same way as described under lot No. 1. The table concentrate and tailing were each run through the Ullrich magnetic separator. Samples of the feed and all products of the test work were tested for radio-activity. The results are shown in the following table:—

Product	Weight pound	Radio-activity
Ullrich concentrate from table concentrate	0·995	0·56
Ullrich tailing from table concentrate	0·086	0·23
Ullrich concentrate from table tailing	0·123	0·04
Ullrich tailing from table tailing	12·500	0·01
Table slime	1·312	0·08
Loss	0·884	....
Feed	15·000	0·01

The same standard was used to test the radio-activity of the products of the test work on lot No. 2 as was used in testing the products of the test work on lot No. 1.

CONCLUSIONS

The best product obtained in testing lot No. 1 is less than one-fourth as radio-active as a low grade carnotite ore containing 2 per cent uranium oxide, and represents 0·14 per cent of the sample treated.

The best product obtained in treating lot No. 2 is less than one-twelfth as radio-active as a low grade carnotite ore containing 2 per cent uranium oxide, and represents 0·63 per cent of the sample treated.

From the above it is evident that neither of the lots submitted can be considered as ores of radium.

Test No. 172

THE PRECIOUS METALS AND OTHER VALUES IN THE COPPER-NICKEL ORES OF SHEBANDOWAN LAKE, ONT.

In the twenty-ninth Annual Report of the Ontario Bureau of Mines, vol. XXIX, part I, will be found a full report as to location and occurrence of the Shebandowan Lake nickel deposits.

On August 17th, 1922, five bags were forwarded to the Ore Testing Laboratories, containing five samples of this ore, collected by Dr. T. L. Tanton, and submitted by the Director of the Geological Survey, with a detailed memorandum concerning these samples, as follows:—

Sample No.	Locality	Description	Nature of work required
1	Pit No. 3................	Channel sample across 2 feet of ore.	Complete analyses of the samples, and it is requested that concentration and recovery of the platinum groups metals be attempted in those which warrant the treatment.
2	Pit No. 5, east side.......	Channel sample across 6 feet of ore.	
3	Northerly part of pit No. 10.	Channel sample across 3½ feet of ore.	
4	Southerly part of pit No. 10.	Channel sample across 4½ feet of ore.	
5	Bottom pit No. 10........	Picked sample representative of the ore.	

These samples were crushed, carefully sampled down, and were given complete analyses, with the results shown below.

Sample No.	Insol. per cent	Iron per cent	Arsenic per cent	Copper per cent	Nickel per cent	Cobalt per cent	Sulphur per cent	Gold oz./ton	Plat. oz./ton	Pallad. oz./ton
1.........	16·40	35·65	trace	1·50	5·95	0·20	29·25	0·01	0·04	0·12
2.........	31·60	26·45	"	3·00	2·10	0·25	18·50	0·007	0·03	0·07
3.........	38·30	20·70	"	3·10	0·04	0·21	9·10	0·007	0·03	0·10
4.........	35·90	22·50	"	2·15	0·04	0·21	10·25	0·006	0·02	0·08
5.........	18·60	33·20	"	5·95	4·10	0·25	27·40	0·083	0·04	0·11

Samples 3 and 4 being very much oxidized and weathered, and furthermore low in nickel, were temporarily discarded. Samples 1, 2, and 5 were mixed, making a combined sample of 40·1 pounds. This was crushed and sampled down and 1,000 gram samples cut out, upon which flotation tests were made, yielding a concentrate, middling, and tailing product. These products, together with a sample cut from the combined head sample yielded the following on analysis:—

Product	Copper per cent	Nickel per cent	Cobalt per cent	Gold oz./ton	Plat. oz./ton	Pallad. oz./ton
Combined head sample....................	4·78	4·30	0·18	0·06	0·035	0·07
Flotation concentrate......................	10·77	8·96	0·35	0·10	0·053	0·12
Flotation middling........................	3·55	3·71	0·21	0·05	0·050	0·09
Flotation tailing..........................	1·77	1·07	0·07	0·02	0·01	0·10

The results from the above concentration tests are not promising. The samples were taken close to the surface where the ore was subject to more or less oxidation, and this may account for the unfavourable results obtained from flotation.

A further shipment of 400 pounds has been received, and it is proposed to conduct further concentration tests, and also to carry out a series of smelting tests to obtain as full information as possible as to the behaviour of the precious metal content.

Test No. 173

THE RECOVERY OF THE VALUES IN A RICH SHIPMENT OF GOLD ORE FROM THE CARIBOU MINING DISTRICT, NOVA SCOTIA

R. K. Carnochan

A shipment of 89½ pounds of gold ore was received at the Ore Dressing and Metallurgical Laboratories from the Herman Hall mine, Caribou, Nova Scotia. This shipment had been sent in by Mr. W. R. Hitchcock, Cornwall, Ont.

The ore consisted of white quartz carrying a very large amount of free gold and small amounts of pyrite and galena. A few specimens were selected from the shipment and the balance, 86 pounds, was crushed in a breaker and a set of rolls to pass 40 mesh. A great deal of metallics was obtained on the screen. The −40 mesh was cut into quarters by means of a Jones riffle and a sample was cut from each quarter for assay. These samples gave the following values:—

—	Au oz./ton	Ag oz./ton
−40 mesh, 1st quarter	3·30	0·55
" 2nd "	2·08	0·60
" 3rd "	3·57	0·63
" 4th "	2·71	0·59
Average of four quarters	2·92	0·59

The −40 metallics were cleaned by fluxing in an assay crucible. This gave 2·057 oz. of gold and silver from 86 pounds of ore, which is equal to 47·84 oz. gold and silver per ton. A test on this bullion gave 935·6 parts of gold per 1,000. Therefore the metallics correspond to 44·76 ozs. per ton gold and 3·08 ozs. per ton silver. This added to the average value of the −40 mesh, makes the heads run:—

Gold	47·68 ozs. per ton
Silver	3·67 "

AMALGAMATION AND CYANIDATION

Test No. 1.—1,127 grams of the −40 mesh material were mixed in a small pebble jar for 3 hours with 100 grams of mercury and 400 c.c. of water. The contents of the mill were then panned to recover the mercury. The tailing was dried and sampled for assay.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent. Ag value
Metallics	44·76	3·08	93·9	83·9
Amalgamated	2·56	0·47	5·4	12·8
Tailing	0·36	0·12	0·7	3·3
Heads	47·68	3·67	100·0	100·0

957 grams of tailing from amalgamation were ground to pass 200 mesh, and agitated for 18 hours with 6,000 c.c. of water to which had been added 15 grams of sodium cyanide and 10 grams of lime.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Cyanided	0·28	0·09	0·5	2·5
Tailing	0·08	0·03	0·2	0·8
Feed	0·36	0·12	0·7	3·3

AMALGAMATION, TABLING, FLOTATION, AND CYANIDATION

Test No. 2.—1,069 grams of −40 mesh material was ground for 3 hours in a small pebble jar with 100 grams of mercury and 400 c.c. of water. The contents of the mill were then panned to recover the mercury. The amalgamation tailing was tabled, making a concentrate and a tailing. The table tailing was floated in a small Ruth machine, making a concentrate and a tailing.

Product	Weight grams	Au oz./ton	Ag oz./ton	Au grams x oz./ton	Ag grams x oz./ton	Per cent Au value	Per cent Ag value
Metallics	....	44·76	3·08	47,848	3,293	93·87	83·94
Amalgamated	....	....	....	3,006	448	5·90	11·42
Table conc.	28	0·84	3·50	24	98	0·05	2·50
Flot. conc.	63	0·52	1·02	33	64	0·06	1·63
Flot. tailing	844	0·06	0·02	51	17	0·10	0·43
Loss	134	0·06	0·02	8	3	0·02	0·08
Heads	1,069	47·68	3·67	50,970	3,923	100·00	100·00

655 grams of flotation tailing were ground to pass 200 mesh and agitated for 33 hours with 6,000 c.c. of water to which had been added 45 grams of sodium cyanide and 10 grams soda ash.

Product	Au oz./ton	Ag oz./ton	Per cent Au value	Per cent Ag value
Cyanided	0·06	0·02	0·10	0·43
Tailing	0·00	0·00	0·00	0·00
Feed	0·06	0·02	0·10	0·43

CONCLUSIONS

1. Amalgamation alone on this ore gives a very high recovery. The metallics would be amalgamated in regular milling, so they should be added to the values amalgamated to give the proper recovery by amalgamation. If this is done the recoveries by amalgamation in test No. 1 are gold 99·3 per cent, silver 96·7 per cent, and in test No. 2, gold 99·77 per cent, silver 95·36 per cent.

2. Although the recovery by amalgamation is very high, the amalgamation tailings are too high to discard and some further treatment should be given them.

3. In test No. 1, the amalgamation tailing was cyanided, but this failed to produce a low tailing.

4. In test No. 2, the amalgamation tailing was tabled, floated, and then cyanided. This produced a very low tailing.

5. It is possible that it would be sufficient to table and cyanide the amalgamation tailing.

Test No. 174

THE RECOVERY OF THE VALUES FROM THE ORE OF THE E. H. GLADWIN MINE, BEAVER DAM MINING DISTRICT, NOVA SCOTIA

R. K. Carnochan

A shipment of 5 bags of gold ore, weighing 575 pounds, was received at the Ore Dressing and Metallurgical Laboratories November 13, 1922. This ore was sent in by Mr. W. R. Hitchcock, Cornwall, Ont., and came from the E. H. Gladwin Mine, near Upper Musquodoboit, Beaver Dam mining district, Halifax county, N.S.

The shipment consisted of vein quartz carrying a fair amount of sulphides and some free gold in large flakes. Tests were desired on the ore to determine the best method of extracting the gold.

HEAD SAMPLE

The ore is very difficult to sample due to the presence of metallics. Four samples of about 30 pounds each were taken and crushed through 40 mesh, some metallics being obtained on the screen. The results from these four samples were not satisfactory, so the remainder of the shipment was cut into halves, making two more samples, and these were crushed through 40 mesh, metallics being obtained on the screen. The following table shows the results obtained from the six head samples:—

Sample	Weight lbs.	Metallics				—40 mesh		Total	
		Au mgms	Ag mgms	Au oz./ton	Ag oz./ton	Au oz./ton	Ag oz./ton	Au oz./ton	Ag oz./ton
1......	33·25	11·04	0·40	0·02	0·00	0·27	0·04	0·29	0·04
2......	35·00	25·00	1·50	0·05	0·00	0·08	0·00	0·13	0·00
3......	30·25	338·47	23·33	0·72	0·05	0·08	0·02	0·80	0·07
4......	30·75	36·46	2·48	0·08	0·01	0·11	0·03	0·19	0·04
5.....	208·00	304·22	20·96	0·10	0·01	0·08	0·04	0·19	0·05
6......	210·00	298·50	20·59	0·09	0·01	0·01	0·06	0·19	0·07
Total	547·25	1,013·69	69·17	0·12	0·01	0·12	0·03	0·24	0·04

As the table shows, the shipment of ore assays 0·24 oz. per ton gold and 0·04 oz. per ton silver, this being equal to $4.96 in gold and 3c. in silver, making a total of $4.99.

AMALGAMATION AND CYANIDATION

A 684 gram portion of the −40 mesh material of No. 1 head sample was amalgamated:—

Product	Au oz./ton	Per cent Au value
Metallics	0·02	6·9
Amalgamated	0·05	17·2
Tailing	0·22	75·9
Heads	0·29	100·0

The amalgamation tailing was ground to pass 200 mesh, and cyanided:

Product	Au oz./ton	Per cent Au value
Cyanided	0·22	75·9
Tailing	0·00	....
Feed	0·22	75·9

AMALGAMATION, TABLING AND FLOTATION

A 1,090 gram portion of the −40 mesh material of No. 2 sample was amalgamated, the amalgamation tailing being tabled, and then the table tailing was floated. The flotation concentrate was re-run to clean it.

Product	Weight grams	Au oz./ton	Assay x Wt. gms	Per cent of Au value
Metallics	....	0·05	54·50	38·5
Amalgamated	....	....	34·06	24·1
Table concentrate	34	0·46	15·64	11·0
Flotation concentrate	120	0·26	31·20	22·0
" middling	126	0·05	6·30	4·4
" tailing	681	0·00	0·00	....
Loss	129	0·00	0·00	....
Heads	1,090	0·13	141·70	100·0

TABLING AND FLOTATION

A 1,018 gram portion of the −40 mesh material of head sample No. 5 was run over a small Wilfley table, making a concentrate and tailing. The table tailing was then floated in a small Ruth machine, the flotation concentrate being re-run to clean it.

Product	Weight grams	Au oz./ton	Assay x Wt. gms.	Per cent of Au value
Metallics		0·09	91·6	57·8
Table concentrate	26·6	2·52	67·0	42·2
Flotation concentrate	78·0	trace		
" middling	121·0	0·00		
" tailing	677·4	0·00		
Loss	115·0	0·00		
			158·6	
Heads	1,018·0	0·19	193·4	100·0

In this test the table recovers practically all the gold and leaves only a trace in the table tailing. In the amalgamation, tabling, and flotation test, the table did not remove all the gold. These different results are due to the spotty nature of the ore, and the difficulty of getting a representative sample.

Amalgamation and Flotation

Three tests were made on the ore using amalgamation followed by flotation, about 1,000 grams of ore being used in each test. In every one of them more gold was obtained in the products than the amount which assays showed to be in the feed. This is due to the very spotty nature of the ore. These tests show the flotation concentrate to run about 1·25 oz. ton gold, and the flotation tailing about 0·08 oz. ton gold.

CONCLUSIONS

The tests show that the ore does not amalgamate very readily, and to make a good recovery of the gold, amalgamation would have to be followed by some other process.

Amalgamation followed by cyanidation gives a very high recovery of the gold and this seems the best method of treatment.

The ore is low grade, and a considerable tonnage of it would need to be available before a mill to treat it is built.

Test No. 175

AN INVESTIGATION OF THE TAILING AND ROCK DUMPS AT THE GOLD MINES OF NOVA SCOTIA—WITH A REVIEW OF THE PAST MINING AND MILLING METHODS, THE PRESENT STATUS OF THE GOLD INDUSTRY IN THE PROVINCE, AND SOME SUGGESTIONS FOR THE REVIVAL OF THE INDUSTRY

C. S. Parsons

INTRODUCTION

It has been frequently estimated that in the early days of gold mining in Nova Scotia, the mills did not recover over 70 per cent of the gold, and in some cases, not over 50 per cent. This has led to the general belief that the old tailing dumps contain high values in gold. A table was compiled by F. H. Mason in 1898 giving assays of tailings and concentrates from different districts. This table shows that large amounts of gold remained in the tailings. Old operators also report high losses, especially when treating ores containing an appreciable quantity of arsenical pyrite.

The rock or waste dumps are also believed to carry considerable amounts of gold. There would seem to be some foundation for this belief, as the early returns to the Department of Mines show a yield of approximately ¾ oz. of gold per ton of ore milled, while since 1882 there has been a general falling off from this figure. This indicates that in the early days selective mining or sorting was done to obtain only high grade ore. Consequently much material that was then considered too low grade was thrown on the rock dumps, and the large waste dumps that are found about the old mines bear witness to the fact.

At times attempts have been made to re-treat the old tailing dumps, but they have met with failure, except where operating mills were able to re-run their own tailings.

The recent demand for arsenic, with the consequent increase in its price has revived the hope that some of the old tailing dumps could be profitably re-worked for their gold and arsenic content.

To determine the value of the rock and tailing dumps, it was planned to examine and sample as many of them as possible. This would include an approximation of the tonnages available for re-treatment and the assay values in gold, silver, and arsenic. Information was also to be collected on the mining and milling methods, in order to suggest, if possible, some way in which the gold mining industry could be encouraged. Test work was to be done at the laboratories of the Ore Dressing and Metallurgical Division upon samples from the dumps if the preliminary investigation showed that they contained enough gold or arsenic to warrant it.

Through the co-operation of Mr. Faribault of the Geological Survey, the help was obtained of Mr. Cruickshanks, for many years his assistant. Mr. Faribault also provided a list of mines that he considered the most important to visit. This saved much time and was of great assistance.

Visits were made to twenty-one gold districts, where fifty tailing dumps and numerous rock dumps were sampled. The tailing dumps are generally fan-shaped, grading into a swamp, the sands remaining on the slope, the slimes running into the swamp. The depth as a rule does not exceed five feet. They were sampled with a three inch flange auger, but where it could not be used because of the holes caving in too rapidly, a split pipe auger with a removable solid core was used.

To properly sample the rock dumps they should have been trenched, but it was impossible to spare the time necessary for this. All that could be done was to take grab samples from the surface of some of them. Of course, these rough samples cannot be very representative and too much reliance should not be placed on the assay results from them.

It was hoped that the data derived from the examination and sampling of the dumps would shed some light on the past milling practice in the province.

Past Mining and Milling Practice

The mines were small, and the majority of them had scarcely passed the prospect stage. Even the largest do not seem to have milled more than 75 tons per day averaged over a period of a year. Old mining methods have been frequently blamed for helping to cause the decline in the industry, but disparaging the efforts of the old time miners will not revive it. However, it may be well to keep in mind the possibility that some of the mines which are now closed might be reopened and profitably worked.

With a few exceptions very little systematic development was carried on and no assay plans or underground records were kept. Underhand stoping was largely used and but little ore was developed ahead of stoping. The ore shoots were followed closely, but sooner or later lean areas were bound to be encountered. The operators were then forced to abandon work either on account of insufficient funds or because of lack of the necessary underground records by means of which they could possibly have located the extension of the ore shoot.

It may be well to remark that although there has been much useful and instructive structural geological work done on the gold areas, the economic geology based on assays and tonnages has been comparatively neglected.

The mills generally consisted of five, ten and sometimes twenty stamps, and relied wholly upon plate amalgamation for the recovery of the gold. Later, tables were introduced, and installed in most of the mills. The gold in the table concentrate was associated with arsenopyrite and pyrite, and the recovery made was poor. A number of mills stacked their concentrates without attempting to treat them. (A list of table concentrates at some of the mines is given in table I).

At two of the operating mills, samples were taken of the tailings from the amalgamation plates. One of these mills was working on $3 to $4 feed, and five samples taken over a period of five shifts showed 0·02 to 0·05 oz. per ton (average 0·03) of gold in the tailings (see table No. IV). This was a loss of about 60 c. per ton.

The tailings from the other mill were much higher, 0·22 oz. per ton or about $4.40, but it was treating $12 ore. These two mills were working on typical Nova Scotia ores and following the practice that has been in vogue for the past 40 years. Their recoveries should, therefore, approximate those of the older mills.

THE TAILING DUMPS

The tailing dumps are, with few exceptions, small, so they will have to contain relatively high values of gold and arsenic before it would pay to re-treat them. A list of the dumps sampled is given in table No. II, together with the number of samples taken, the gold, silver, and arsenic content, and approximate tonnages. The tonnages were obtained by a rough measurement of the dumps by compass bearings and pacing. This list shows that none of the dumps sampled contain values worth considering.

Comparing these results with those in the table compiled by Mason, the very surprising point is noticed that the gold that was in the tailings is not there now. If it ever was there, and the tailings from the mills operating at the present time would indicate that it was, it is an interesting problem to determine what has become of it, and how it was taken out.

In volume V (1899-1900) of the Transactions of the Mining Society of Nova Scotia, is a paper by Mr. B. C. Wilson on this very problem. He claims that lavish use of cyanide for dressing the plates gradually leached the gold from the tailing dumps. This explanation is very plausible, as cyanide was used freely, and the dumps being small would be pretty thoroughly saturated by the dilute solvent solution.

A screen analysis of the tailing from one of the mills now operating shows that 50 per cent of the gold is in the −200 mesh product. This fine gold would be readily dissolved, especially when the solution was in contact for such a long time.

There is generally a large amount of decayed vegetable matter at the bottom of the dumps, and it might be expected that part of the dissolved gold would be precipitated there. Some of the samples taken contained considerable amounts of this vegetable matter, but the gold content was not noticeably higher.

The tailing dumps at Montague were not sampled as the Metals Recovery Company of Canada had obtained control of them. It is expected they will be re-treated for their gold and arsenic content, as it is said they assay 2-3 per cent arsenic, and about $3 in gold.

ROCK DUMPS

A list of samples from a few rock dumps is given in table No. III. As mentioned previously these samples cannot be relied upon to accurately represent the values in the dumps. They are published merely to give some idea of the content. They show that the dumps do not contain much gold or arsenic.

WEST GORE ANTIMONY DUMPS

Samples were taken of the sand tailing and rock dumps of the old antimony mines of West Gore. Table No. V shows the value of these dumps in gold, arsenic, and antimony from which it is seen that the sand dumps contain a fair amount of gold. The slimes from the mill have been run into the river at the foot of the hill. From personal experience in concentrating antimony ore, it is judged that they carried away large losses in gold and antimony. Stibnite ores can be readily concentrated by froth flotation, with the production of a concentrate carrying 50-60 per cent metallic antimony with a recovery of better than 90 per cent. If the gold in the West Gore ores is associated with the stibnite, it would also be recovered by the same treatment.

Flotation tests will be made on the sand tailings to determine if the gold follows the antimony into the concentrate. The successful solution of the problem of concentrating these ores by flotation should encourage prospecting for other antimony veins in that locality.

INCREASE IN GOLD MINING DURING THE PAST YEAR

The interest in gold mining is becoming greater, and there has been more work done during the past year than for some years.

Prospecting is being done in the western section of the province at Cranberry Head, Whiteburn, Malaga, Leipsigate, and Brookfield. At Brookfield the King mine is doing some development work and is experimenting with cyanidation under the direction of Mr. Badger. The Malaga Mining Company at Malaga is trying to operate its property but the lack of power is a serious handicap. The use of coal is out of the question and wood would cost $5-$6 a cord, and at best is unsatisfactory.

Mr. S. A. Hister of Halifax is doing work on the Parker-Douglas property in the same district. At Mt. Uniacke Mr. James Crease is running a ten stamp mill using plate amalgamation only. A sample of the tailing running to waste from the plates assayed 0·22 oz. in gold, about $4.50 per ton. This is a very high tailing and indicates that amalgamation will not always save all the gold. A table test on this tailing was made at the Ore Dressing and Metallurgical Laboratories, and a concentrate produced which ran 4·12 oz. per ton with a tailing of 0·06 oz. per ton.

There is a little prospecting going on at South Uniacke and Oldham.

The Bradbury Mines Ltd. at Tangiers were keeping their mine pumped, and in November were waiting for an engineer to make an examination. Since then it is reported that the diamond drilling and other development work recommended by him will be immediately carried out.

At Caribou Mr. Hermon Hall was doing some prospecting and was operating a five stamp mill and a Wilfley table. He had uncovered a number of leads which showed considerable quantities of free gold, but he had done very little underground development and was endeavouring to sell the property as it stood. The concentrate from the table contained 0·85 ozs. gold and 18·43 per cent arsenic, and the tailing contained 0·08 oz. gold and 1·19 per cent arsenic.

Mr. Matthew Higgins was doing a little prospecting at Moose River. A sample of the mineralized slate from the walls of his working ran 0·44 oz. per ton in gold and carried considerable arsenic.

There was some work done at Beaver Dam during the summer, but most of the time seems to have been spent in trying a secret process for the recovery of the gold.

At Goldenville, Mr. J. A. Warner is operating for the Sherbrooke Mining Company and is doing development work and running a small mill of twenty stamps. He is working on low grade ore, but by careful management is making expenses and hopes to put the mine on a paying basis. He is working under ideal conditions as regards power.

At Cochrane Hill work is being done by two separate parties. Mr. Grant McDonald is doing some development work on the Mitchell belt and is blocking out a fair tonnage of ore. This is one of the large belts or series of veins and mineralized slate and quartzite which is worthy of attention as it carries gold values across its width of 75 ft. Mr. Brown has opened up a mine on a large slate belt at the foot of Cochrane Hill. He has also built a mill of his own design for the treatment of the ore.

Suggestions for the Revival of the Industry

The gold industry in Nova Scotia can best be encouraged by giving more attention to improving conditions in the gold areas which limit the successful development and profitable operation of the ore bodies.

The gold is not evenly disseminated through the quartz veins but is segregated in shoots or lenses. This characteristic makes mining uncertain, and unless a systematic geological study is made of underground conditions and a wise policy of development based on geological information, systematic sampling and assaying is carried out, so that ore shoots will be developed ahead of stoping operations, there are small chances of success. The mine operators now in the field should be induced to embark on such a policy.

Very little systematic sampling was done in the past, either underground or in the mill. Where the values are so erratic as is the case with the Nova Scotia gold ore, more of such work should be done to obtain an intelligent idea of the values of the ore being mined, of the mill feed, and of the pulp from each of the milling operations. By encouraging the operators in this respect, it will enable them to detect at once the weak points in their operations requiring attention, and their efforts can be directed to their improvement. It will enable them to ascertain the tonnage of milling ore, the values in such ore, the values in their mill feed, in the pulp from each milling operation, in their tailings. It will also enable them to ascertain their recoveries and the value of the ore to be milled, so that a profit can be made. Such systematic work is essential to successful operation.

The mining laws of the province should be studied to determine if improvements can be made which would encourage more prospecting.

Cheap power for operating purposes should be given consideration. In certain of the mining districts, especially in the western portion of the Province, the cost of fuel, either coal or wood, is a great handicap. The question of operating with oil engines of the Diesel type, or of obtaining cheap hydro-electric power should be investigated to determine the most suitable source of power for the operator.

Research and experimental test work should be carried out on the milling and metallurgical treatment of the ores, to improve the milling practice. Attention should be given to developing a simple and economical method to increase extraction in the small mills, recovering more of the values in the tailings from the plates after amalgamation.

This can be accomplished by cooperation between the provincial and federal mining departments.

The present, when so much attention is being paid to gold mining in Canada, is an opportune time to encourage the industry in Nova Scotia. If the above suggestion can be carried out, the existing mines and other ore bodies which may be discovered in the future, will have a much better chance of being worked as profitable enterprises than has been the case in the past.

TABLE No. I

Concentrates Stacked at Old Mills

Locality	No. of samples	Gold ozs.	Silver ozs.	Arsenic per cent	Approx. tonnage	Remarks
Caribou, H. Hall, conc..........	1	0·85	....	18·43	10	
" " tailing........	1	0·08	....	1·19	....	
" Lake mine, conc........	1	0·42	....	trace	100	Cyanided
Cochran Hill, conc...............	1	0·60	....	18·65	10	
Harrington Cove conc............	1	0·05	....	26·56	few	
Forest Hill, Bendego mine conc..	1	0·20	....	18·96	20	Cyanided
Dufferin mines, cyanided conc...	1	0·30	....	17·70	200	
Dufferin mine, conc.............	1	0·93	....	18·96	200	

TABLE No. II

Samples and Assays of Old Tailing Dumps

Locality	No. of samples	Gold ozs.	Silver ozs.	Arsenic per cent	Approx. tonnage	Remarks
Goldboro, Richardson—						
“ “ No. 1 dump...	2	0·10	....	4·58	10,000	Carried considerable sulphides
“ “ No. 1 “ ...	6	0·15	....	1·30		
“ “ No. 2 “ ...						
“ Holes 1-6..............	6	0·01	....	0·16	62,000	
“ Holes 7-9..............	3	0·013	....	0·095		
Goldboro, Skunk den dump......	1	0·03	0·01	trace	1,000	
“ Gold finch dump.....	1	0·02	trace	trace	1,500	
Forest Hill, Bendego dump......	3	0·01	trace	0·28	25,000	
Wine Harbour, Mapier...........	1	0·02 0·02	0·004 0·004	.... 0·09	 1,200	
“ Plough Lead.....	1	0·02	0·004	0·28	1,500	
“ Provintial........	1	0·02	0·008	trace	....	
Cochran Hill, McDonald........	1	0·04	0·01	0·72	none	
Goldenville, Bluenose dump.....	6	0·02	....	0·19	12,600	Sand
“ “	2	0·02	trace	0·35	....	Slime
“ Royal Oak.........	11	0·02	0·01	0·12	13,500	
“ McFarlane lake.....	8	0·025	....	0·19	180,000	
“ Sherbrook Mining Co...............	3	....	....	....	....	
“ “ “	1	....	....	....	....	
Harrington Cove...............	1	0·03	0·01	0·12	1,000	
Dufferin Mine..................	3	0·01	0·003	0·15	3,500	
“ Old Archibauld No. 1 dump.....	2	0·03	0·01	0·81	2,000	
Moose River, G. & K. Mill No. 2 dump.........	3	0·015	0·004	0·28	2,000	
“ Montreal Mill......	2	0·04	0·01	0·31	1,200	
Caribou, Elk Mill..............	3	0·015	0·003	trace	3,000	Cyanided
“ Lake Mine............	6	0·021	0·004	0·206	20,000	
“ Caffrey Mine..........	6	0·02	0·007	0·50	6,000	
Tangiers, Brunswick crusher.....	1	0·02	0·004	trace	....	
“ Essex crusher..........	4	0·015	trace	0·22	3,000	
Mt. Uniacke, No. 3 dump........	1	0·01	0·002	0·19	1,000	
“ Great belt No. 2 dump............	6	0·02	trace	0·25	8,000	
“ Montreal dump....	6	0·02	0·008	0·34	8,000	
Lake Catcha, Oxford dump—						
Holes 1-4.....................	4	0·06	0·03	0·10	40,000	Wash from rest of dump, slime
“ 6......................	1	0·02	trace	0·60		
“ 5 and 7................	7	0·04	0·02	0·12		
Lake Catcha, Anderson dump—						
Holes 1-5.....................	5	0·12	0·02	0·52	10,000	
“ 6......................	1	0·01	0·01	0·06		
“ 7......................	1	0·04	0·01	0·30		Slimes
Lake Catcha, Hanwright mine..	1	trace	trace	trace	....	
Rawdon, McNaughton dump....	2	0·021	0·004	0·25	5,500	
Renfrew, Pictou Development Co.—						
Dump No. 1..................	1	0·015	trace	trace	1,800	
“ No. 2.................	2	0·015	trace	trace	1,800	
Oldham, upper dump, holes 3, 4, 5	3	0·05	0·01	0·57	12,000	
“ lower dump, holes 1 and 2..................	2	0·08	0·01	0·02	3,000	
Waverley, Taylor-Hardman—						
Holes 1 and 2................	2	0·01	trace	trace	50,000	
“ 3 and 4................	2	0·025	0·005	0·89		
“ 5......................	1	0·025	0·005	0·93		
Waverley, Gold Mining Co.......	1	0·01	0·003	trace	....	All in lake
“ American dump.......	2	0·028	0·004	0·54	10,000	
“ Wilsons Hall dump....	2	0·02	0·001	0·35	1,000	

TABLE No. II—*Concluded*

Samples and Assays of Old Tailing Dumps—Concluded

Locality	No. of samples	Gold ozs.	Silver ozs.	Arsenic per cent	Approx. tonnage	Remarks
Gold River, London mine No. 1.	2	0·01	trace	0·22	2,500	
" " No. 2.	1	0·02	0·003	trace	....	
Malaga, Ballow dump, hole No. 1	1	0·017	0·003	0·31	16,000	Cemented material
" " " No. 2	1	0·02	0·005	0·35		
" " " No. 3	1	0·03	0·004	0·60		
" " " No. 4	1	0·022	0·003	0·38		
" " " No. 5	1	0·065	0·012	2·45		
" Caladonia dump........	3	0·025	0·005	0·206	700	
" Parker-Douglas—						
Hole No. 1...........	1	0·033	0·006	0·412	10,000	
" No. 3 and 4......	2	0·01	trace	0·38		
" No. 2...........	1	0·025	trace	0·33		
" Malaga Mining Co.—						
Dump A.............	4	0·035	0·003	0·28	3,500	
" B.............	3	0·03	0·006	0·190	700	
" C.............	3	0·025	0·009	0·100	4,000	Sand
" C.............	2	0·060	0·010	0·380		Slime
Brookfield, King mine—						
Hole No. 1........	1	0·024	0·006	0·285	very little	
" No. 2........	1	0·032	0·003	0·760		
Whiteburn—						
Dump A, hole 1..............	1	0·005	trace	trace	2,000	
" A, " 2..............	1	trace	"	"		
" A, " 3..............	1	0·006	"	"		
" A, " 4..............	1	0·005	"	"		
" A, " 5..............	1	0·03	0·009	"		
" B, " 1..............	1	0·05	0·02	"	....	No tonnage
" C, " 1..............	1	0·04	0·02	"	1,400	Sand
" C, " 2..............	1	0·025	0·01	"		Sand
" C, " 3..............	1	0·015	0·003	"		Slime
" C, " 22.............	1	0·015	trace	"		Slime

TABLE No. III

Some Rock Dumps

Locality	No. of samples	Gold ozs.	Silver ozs.	Arsenic per cent	Approx. tonnage	Remarks
Lake Catcha, Oxford...........	1	0·02	0·004	trace	....	
" Anderson..........	1	0·028	0·007	"	....	
Dufferin Mine..................	1	0·012	0·003	0·38	....	
Caribou, Lake mine............	1	0·01	trace	trace	....	
" Caffey mine...........	1	0·025	0·003	0·254	....	
Rawdon, McNaughton...........	1	0·02	0·003	0·19	....	
" "	1	0·02	0·003	0·174	....	
Moose River, Torquay..........	1	0·01	trace	1·07	....	
" G. & K...........	1	0·013	0·002	trace	....	
Goldenville, Sherbrooke mines..	1	0·028	0·004	0·285	....	
Mt. Uniacke.....................	1	0·02	0·004	0·174	....	

TABLE No. IV

Samples Taken of Tailings Running to Waste From Operating Mills Using Amalgamation Only

	No. of samples	Gold ozs.	Silver ozs.	Arsenic per cent	Approx-tonnage	Remarks
Small 10 stamp mill, working on approximately $12 ore..........	1	0·22	0·04	nil	....	
Small 20 stamp mill, working on $3-$4 ore—samples from five shifts—						
Sample No. 1................	1	0·05	0·08	....	....	
" No. 2................	1	0·04	0·06	....	....	
" No. 3................	1	0·02	0·05	....	....	
" No. 4................	1	0·04	0·06	....	....	
" No. 5................	1	0·02	0·06	....	....	....

TABLE No. V

West Gore Antimony Tailing and Waste Dumps

Sample	Approx. tonnage	Per cent Sb	Per cent As	Ozs. Au	Ozs. Ag
Sample No. 1, coarse sand tails............	1,500	1·30	0·285	0·06	0·01
" No. 2, fine sand tails...............	14,000	2·30	0·15	0·12	trace
" No. 3, rock dumps................ (Rock dumps from old shaft.)	27,000	0·38	0·25	0·02	"
" No. 4, rock dumps................ (From below 800 ft. level.)	27,000	trace	0·396	0·02	"

NOTE.—These sands contain no slimes. The slimes were run into the river and lost. They probably contained very high values in antimony and gold.

Assays of Gold

(Mason)

District	Tailings from plates		Tailings from dumps		Concentrates	
	Maximum	Minimum	Maximum	Minimum	Maximum	Minimum
	oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.	oz. dwt. gr.
Cow Bay............	0 1 18	0 – 21	– – –	– – –	– – –	– – –
Sherbrooke..........	0 3 12	0 – 21	0 6 10	– 3 6	1 8 0	1 8 0
Caribou.............	0 2 22	0 – 14	0 13 2	– 2 23[1]	12 0 20	0 2 22
Uniacke.............	0 8 4	0 – 21	0 – –	– – –	13 3 3	1 17 21
Stormont............	0 15 8	0 1 18[2]	0 16 22	– 3 6	136 1 19[1]	1 7 0
Brookfield..........	– – –	– – –	0 13 1	– – 14	12 17 0	1 3 0
Waverley............	0 2 15	0 – 21	0 6 12	– – –	4 12 1	1 8 7
Central Rawdon.....	0 2 22	traces	0 14 17	– 4 16	5 15 0	1 2 4
Wine Harbour.......	0 3 12	0 1 0	0 6 17	– 3 12	– – –	– – –
Tangier.............	0 1 4	0 – 21	– – –	– – –	– – –	– – –
Fifteenmile Strm....	0 13 10[1]	0 – 7	0 19 20	– – 7	– – –	– – –
Oldham.............	– – –	– – –	– – –	– – –	3 8 0	1 8 4
Gold River..........	– – –	– – –	7 12 20[1]	– – –	– – –	– – –
Lunenburg Co.......	– – –	– – –	– – –	– – –	99 16 18[1]	21 1 8
Montague............	8 2 8[3]	0 1 4	0 7 0	– – –	7 11 16	5 9 16

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[1] Contained amalgam in appreciable quantities.

[2] The ore was giving only $2 in free milling at time the tailings were taken.

[3] Tailings from ore containing upwards of 40 per cent mispickel.

Test No. 176

CONCENTRATION TESTS ON TAILING SAMPLES FROM TWO MILLS OPERATING ON TYPICAL NOVA SCOTIA GOLD ORES

C. S. Parsons and R. K. Carnochan

A sample of tailings from an amalgamation mill at Mt. Uniacke, Nova Scotia, operated by Mr. Jas. Crease, was received November 9, 1922, at the Ore Dressing and Metallurgical Laboratories.

An assay for gold and silver was desired on the sample and also some experimental work to determine if the values present were unamalgamated free gold and silver, or gold and silver locked up in the sulphides.

The sample as received weighed 360 grams, and upon being dried the weight was 357 grams. This 357 grams was screened on 90 mesh and the oversize was crushed on a muller board by gently rocking the muller, the purpose of this being to roll out any free gold or silver into flakes and not to break up these flakes. The crushed material was screened and the +90 put back on the board and crushed again. This alternate crushing and screening was continued until only a small amount of oversize remained. An assay of this oversize gave only a trace of gold and silver.

Forty grams of the −90 mesh was cut out and crushed in the same manner as described above to pass 200 mesh. The small amount of +200 gave only a trace of gold and silver upon assay. The remainder of the −90 mesh, 310 grams, was floated in a small Janney flotation machine, the concentrate being re-run to clean it. This gave three products from flotation—a concentrate, a middling, and a tailing.

The flotation concentrate was tabled on a small Wilfley table making a concentrate and a tailing. The flotation middling and tailing were mixed and tabled making a concentrate and a tailing. The overflow from the boxes during the tabling of the flotation products was run to a large settling tank and in this way a slime product was obtained. The two table concentrates were so small that they were combined. All products were dried, weighed and assayed.

Product	Weight grams	Au oz./ton	Ag oz./ton	Au oz./ton x grams	Ag oz./ton x grams	Per cent of total values	
						Au	Ag
Table concentrate from flot. prod....	5·05	4·12	1·38	20·81	6·97	30·5	56·2
Table tailing from flot. conc.........	4·25	1·05	0·35	4·46	1·49	6·5	12·0
Table tailing from flot. mid. and tailing..........................	78·0	0·05	0·01	3·90	0·78	5·7	6·3
Slime from tank....................	133·0	0·05	0·01	6·65	1·33	9·8	10·7
Loss................................	89·70	0·36	0·02	32·38	1·83	47·5	14·8
Heads..............................	310·00	0·22	0·04	68·20	12·40	100·0	100·0

Owing to the small size of the sample submitted and the number of operations carried out on it, no great accuracy can be expected in the above table. Although far from conclusive, the results of the test work indicate that the gold and silver in the sample are locked up in the sulphides and are not present in the free state.

TABLE TEST ON TAILING FROM SAME MILL

Although a head sample could not be obtained this mill was treating ore of an average grade of about $12 per ton. A table test was made on the sample of amalgamation tailing which contained 0·22 oz. gold per ton. A concentrate was produced which contained 2·63 oz. per ton with a recovery of 74·5 per cent of the gold. The tailing assayed 0·06 oz. per ton. This mill depended wholly on amalgamation for the recovery of the gold values.

TABLE TEST ON TAILING FROM MILL OPERATED BY J. A. WARNER, SHERBROOKE MINING AND MILLING CO., GOLDENVILLE

This mill also depended wholly on amalgamation for the recovery of the gold. Five samples taken of the tailing from the battery plates on five different shifts, gave the following assays:

	Gold	Silver
Sample No. 1	0·05	0·08
" No. 2	0·04	0·06
" No. 3	0·02	0·06
" No. 4	0·04	0·06
" No. 5	0·02	0·06

This mill was operating on about $3 ore.

To obtain sufficient material for a table test, the five samples were mixed. A head sample cut from the lot assayed 0·03 oz. per ton gold.

Product	Weight grams	Per cent by weight	Gold			Arsenic per cent
			Assay ozs.	Content ozs. x grams	Recovery	
Concentrate	81	2·02	0·66	53·46	44·6	13·25
Middling	257	6·45	0·04	10·28	8·6	....
Tailing	3,653	91·53	trace	55·99	46·8	....
Totals	3,991	100·00	0·03	119·73	100·0	....

In practice the middling would be returned to the head of the table, and a 50 per cent recovery of the gold in it could be safely figured upon. In this case it would be equivalent to 30 cents per ton of ore. An average of about 40 pounds of concentrate would be obtained per ton of ore treated.

There was not enough concentrate for experiments on the recovery of the gold from the concentrate. Pan amalgamation is suggested as a possible method of recovery.

Test No. 177

THE SEPARATION OF DOLOMITE FROM MAGNESITE IN THE GRENVILLE MAGNESITES

R. K. Carnochan

Shipments of magnesite from the North American Magnesite Company, Calumet, Que., were received at the Ore Dressing and Metallurgical Laboratories as follows:

Feb. 2, 1920..................	Lot No. 1	84 lbs. gross	Calcined magnesite
Mar. 26, 1920..................	" No. 2	5,849 " net	Raw magnesite
Mar. 23, 1921..................	" No. 3	24,000 " "	"

The magnesite contains dolomite in sufficient quantity to make the lime content very high. Magnesites from Austria, Greece, and the state of Washington and California are very low in lime, and are preferred.

The shipments received gave the following analysis:—

Lot No. 1......	CaO 23·95 per cent	CO_2 1·22 per cent	Calcined magnesite
" No. 2......	CaO 12·85 "	MgO 24·94 "	Raw magnesite
" No. 3......	CaO 11·35 "	MgO 35·84 "	"

Lot 1 before calcination would have run about 12 per cent CaO.

Tests were desired on the magnesite to determine if it would be possble to reduce the lime content. The method used to remove the lime consists of three operations:—

1. Calcining the magnesite at a temperature ranging from 950°C. to 1,100° C.
2. Slaking the calcined magnesite with water. The lime slakes into a thin white slurry, while the magnesia remains coarse.
3. Washing out the lime.

Calcining.—Electric, gas, and oil fired furnaces were used to calcine the magnesite. Any one of these furnaces will give a fair product. Care must be taken not to overheat any part of the charge, as this causes clinkers to form. Difficulty was experienced in getting a thorough calcination. However products as low as 1·0 per cent Co_2 were obtained.

Slaking.—The following methods of slaking were tried:

1. Sprinkling the calcined magnesite both when hot and cold with sufficient water to slake, and covering with bagging to keep the heat generated by slaking from escaping.
2. With steam. The calcined magnesite was placed on a screen in a closed vessel, the steam introduced at the bottom passed through the layer of magnesite and escaped around the edges of a wooden cover.
3. With boiling water. The calcined magnesite when cold was thrown into boiling water.

Of all these methods the slaking by means of boiling water gives the best results. The lowest lime content obtained before using this method was 9·0 per cent, but on using it, products running 7·1 per cent lime were obtained.

Washing.—The following methods of washing were used during the test work:

1. Agitating in excess water and decanting, then adding fresh water, agitating and decanting again, this being repeated until all the milk of lime was removed.

2. Stirring in a cylindrical tank with stirring arms close to the bottom, the arms and shaft upon which they operate being hollow, with holes provided in the arms, so that water forced down the shaft enters the tank through these holes, washes the magnesite and overflows at the top of the vessel.

3. Running the slaked magnesite over a Wilfley table, making a sand product and a slime product.

4. Classifying the slaked magnesite by means of a Dorr classifier into sand and slime.

All these methods give the same results. Any one seems to be as suitable as any other.

SUMMARY AND CONCLUSIONS

A washed magnesite product, 7·1 per cent lime, was produced from raw magnesite running 10·7 per cent lime, which had been calcined to 1·2 per cent Co_2. The material washed from the slaked calcined magnesite contains 21·6 per cent lime.

It is desirable to reduce the lime to 3 per cent or less. This has not been done. Starting with magnesite lower in lime than that used and getting a better calcination than that secured in the test work, it would be possible to reduce the lime to less than 7 per cent. It is doubtful if a magnesite product running 3 per cent or less in lime could be made from magnesite similar to that submitted.

Test No. 178

THE CONCENTRATION AND SEPARATION OF THE MINERALS IN THE ORE OF THE STIRLING MINE, RICHMOND CO., CAPE BRETON

C. S. Parsons and R. J. Traill

Location of Property.—The Stirling zinc mine is situated in the northeastern corner of Richmond county, Cape Breton island, half a mile to the south of the Framboise road which connects Loch Lomond with Framboise and Fourchu on the seaboard. The mine is, therefore, in close proximity to points from which the ore could be shipped and to which supplies could be brought by water.

Exploratory Work.—A shaft has been sunk on the ore for 30 to 40 feet. The deposit has been trenched in a number of places, in which can be seen bands of high grade and low grade ore. The property has been drilled by two different companies and some twelve bore holes have been put down.

NOTE.—See Report by D. D. Cairns, Summary Report of Geological Survey, 1916, for further details.

Shipments for Experimental Purposes.—Two shipments of 200 pounds each, one of high grade ore and the other of low grade ore were received at the Ore Testing Laboratories on January 26, 1922. These shipments were submitted by F. M. Connell, Esq., of the Eastern Mining and Milling Company, Toronto, Ont., who had obtained control of the property.

Purpose of Test Work.—It was desired that experimental work be conducted on the two classes of ore to determine methods of concentration and separation of the minerals, and gangue constituents. Saleable products were desired.

Low Grade Ore

The shipment consisted of low grade material, containing the sulphides of iron, zinc, lead, and copper, oxidized by weathering, taken from near the surface or from material which had been broken for some time. It showed on analysis to contain:—

Copper	0·95	per cent
Zinc	3·55	"
Lead	0·76	"
Iron	8·50	"

On examination under the microscope it was found that the minerals were present in a very fine state, and that fine grinding to at least 200 mesh was necessary to free them in order to obtain a separation.

Flotation tests were made, but owing to the oxidized condition of the ore no favourable results were obtained. Table concentration tests made showed that the gangue could be eliminated, with the production of a sulphide product the equivalent of the higher grade ore.

On account of the oxidized condition of the ore, the sample was not considered to be a true representative one and extensive experimental test work was not conducted.

Higher Grade Ore

The shipment consisted of the sulphides of zinc, lead, copper and iron representing 80 per cent and a siliceous gangue of about 20 per cent. It showed on analysis to contain:—

Copper	2·80	per cent	Magnesia	0·63	per cent
Zinc	27·60	"	Soluble iron	0·06	"
Lead	8·10	"	Manganese..trace, less than 0·1 per cent		
Iron	11·50	"	Gold	0·08	oz.
Lime	0·26	"	Silver	6·58	"

Examination of polished surfaces of this massive sulphide ore under the microscope showed the minerals to be in a very fine state, closely associated with each other. In comparison with the ore of the Sullivan mine, Kimberley, B.C., where grinding to 200 mesh is practiced to obtain a good separation, this ore showed that even finer grinding is necessary.

Flotation tests were conducted on the raw ore, and on the ore after it was given a slight roast. Leaching tests in a weak solution of sulphuric acid were also conducted on the roasted ore.

Flotation Tests on Raw Ore.—The ore was crushed to pass a 200 mesh screen. Some fourteen tests, namely tests Nos. 1 to 8, and 13 to 18 inclusive, were conducted using various reagents and combinations to effect a separation of the minerals. No results worthy of note were obtained. This may have been due to the oxidized condition of the sample. Further experimental work on the freshly broken ore would be necessary to determine if flotation could be applied to the raw ore.

FLOTATION TESTS ON ROASTED ORE

A number of tests were made on the roasted ore with encouraging results, as follows:—

Test No. 9

The ore was crushed to 20 mesh and given a slight roast below 450 C. The roasted ore was then ground in a ball mill to 200 mesh and floated with K.K. fuel oil and pine oil with about 15 lbs. of acid per ton or ore. 1,000 grams were used for the test. The results were as follows:

Product	Weight grams	Analysis per cent Zn	Content grams Zn	Percentage of Zn values
Concentrate	580	43·50	252·1	81·7
Middling	110	23·40	25·7	8·3
Tailing	305	10·10	30·8	10·0
Heads	995	31·00	308·6	100·0

Test No. 10

Test No. 9 was duplicated at a temperature of 340 to 370 C. Time in furnace was 6 minutes. The ore was crushed to 30 mesh before roasting. The same procedure for flotation was followed as in test No. 9, 905 grams of ore being used for this test, with the following results:

Product	Weight grams	Analysis per cent Zn	Content grams Zn.	Percentage of Zn values
Concentrate	592	41·95	247·2	92·1
Middling	105	13·75	14·4	5·3
Tailing	198	3·50	6·9	2·6
Heads	895	30·00	268·5	100·0

Test No. 11

This test was conducted on 1,000 grams of the roasted ore as in test No. 10. The oil used was No. 34 gravity fuel oil. The results were as follows:—

Product	Weight grams	Analysis per cent Zn	Content grams Zn	Percentage of Zn values
Concentrate	358	46·70	167·2	55·7
Middling	238	35·15	83·7	27·8
Tailing	377	13·15	49·6	16·5
Heads	973	30·88	300·5	100·0

Test No. 12

This test was conducted on 735 grams of the roasted ore as in tests Nos. 10 and 11. The oil used for the flotation of the zinc was K. and K. fuel oil, and pine oil with about 15 lbs. of acid per ton of ore. The results were as follows:—

Product	Weight grams	Analysis					Content grams Zn	Per cent of Zn values
		Per cent Zn	Per cent Pb	Per cent Cu	Au oz.	Ag oz.		
Concentrate	391	44·85	8·68	3·10	0·10	9·50	175·4	81·0
Middling	105	23·04	10·58	4·15	0·04	5·86	24·2	11·2
Table concentrate	24	16·65	23·35	6·57	0·14	13·10	4·0	1·8
Table tailing	186	7·00	5·70	2·57	0·02	5·58	13·0	6·0
Heads	706	30·68	....	....	0·08	6·58	216·6	100·0

Test No. 19

Product	Weight		Zinc		Per cent of Zn values	Remarks
	Grams	Per cent	Assay per cent	Assay per cent x weight per cent		
1st concentrate	587	59·2	36·3	214·8	75·1	H_2SO_4 10 lb./ton coal tar creosote 1 lb./ton.
2nd concentrate	165	16·6	27·4	45·5	16·2	
Tailing	240	24·2	10·2	24·7	8·7	

Test No. 20

1st concentrate	616	61·3	36·9	226·1	78·6	K.K. oil 10 dr.; P.T. and T. No. 350 3 dr., coal tar creosote 2 dr.
2nd concentrate	171	17·1	26·4	45·2	15·6	
Tailing	217	21·6	7·7	16·6	5·8	

Leaching Tests on the Roasted Ore

A small 100 gram sample was crushed to 20 mesh and roasted for 6 hours at a temperature below 700° C. The roasted ore was leached with 10 per cent hot H_2SO_4. The recovery of the zinc in solution was 90 per cent.

A sample of 50 pounds was crushed to 20 mesh and roasted in a large muffle furnace. The temperature range was 300°C. in the top muffle, to 700° C. in the lower muffle. 95 per cent of the zinc was soluble in acid.

A small sample was roasted in a laboratory muffle furnace for 1½ hours at temperature 550° to 700° C.

Water soluble....................	Zinc 0·9	per cent of total....	– 0·26	per cent
Acid soluble in 10 per cent H_2SO_4	Zinc 99·3	" "	–27·40	"
" 10 " "	Cu 95·4	" "	– 2·67	"

A small sample was roasted in a laboratory muffle furnace for 4 hours at temperature 300° to 700° C.

Water soluble....................	Zinc 2·5	per cent of total...	– 6·10	per cent
Acid soluble in 10 per cent H_2SO_4	Zinc 100	" " ...	–27·60	"
" 10 " "	Cu 94·3	" " ...	– 2·64	"

SUMMARY AND CONCLUSIONS

Due to the oxidized condition of the sample of low grade ore submitted, a separation of the minerals in this class of ore could not be obtained. Table tests show, however, that the gangue material can be eliminated and the minerals constituents collected in a concentrate approaching the grade of the higher grade ore.

Flotation tests on the higher grade ore gave no satisfactory results on the raw ore. This, however, is not conclusive proof that a separation of the minerals could not be obtained on freshly broken ore.

Flotation tests on the higher grade ore after it was given a slight roast gave very satisfactory results. A zinc concentrate containing 45 per cent zinc was obtained by this method.

Leaching tests on the roasted higher grade ore showed the zinc and copper content readily soluble in weak sulphuric acid.

Test No. 179

CONCENTRATION TESTS ON LE ROI No. 2 DUMP ORE

C. L. Dewar

A shipment of 100 pounds of Le Roi No. 2 dump ore was received February 21, 1922, from Douglas Lay, Esq., Rossland, B.C. The shipment was submitted in connection with our investigation on the use of Canadian manufactured reagents for flotation of Canadian ores.

On analysis the ore was found to contain:—

Copper..	0·52 per cent
Iron..	8·90 "
Gold...	0·16 oz. per ton
Silver..	0·18 "

A number of tests were conducted using various reagents, the results of which are given in the following tables:—

Concentration Tests on Le Roi No. 2 Dump Ore

LE ROI No. 2 MINE—FLOTATION OF RAW ORE

Test No.	Product	Weight per cent	Analysis		Recovery per cent		Remarks
			Cu per cent	Au oz.	Cu	Au	
1	Concentrate. Tailing.......	13·4 86·6	3·65 0·05	0·70 0·08	92·0 8·0	57·5 42·5	Coal tar creosote 1·0 lb./ton Pine oil (G.N.S. No. 5) 0·15 " Lime 4·0 "
2	Concentrate.. Middling...... Tailing.......	3·3 5·7 92·0	11·60 1·00 0·05	1·26 0·21 0·10	78·8 11·7 9·5	28·6 8·2 63·2	No. 34 Fuel oil 0·7 lb./ton F.P.L. No. 25 0·4 "
4	Concentrate.. Middling...... Tailing.......	2·9 9·4 87·7	14·20 0·75 0·05	1·80 0·24 0·12	78·3 13·4 8·3	29·0 12·5 58·5	Lime 5·0 lb./ton Mix. "A" 0·7 " G.N.S. No. 28 0·2 "
6	Concentrate.. Middling...... Tailing.......	6·3 11·7 82·0	7·28 0·15 0·02	1·38 0·16 0·03	93·1 3·6 3·3	66·8 14·4 18·8	Mix. "A" 1·2 lb./ton (Deep froth)
7	Concentrate.. Tailing.......	23·0 77·0	2·10 0·02	0·50 0·04	96·9 3·1	79·0 21·0	Wood tar, 50 per cent Heavy wood oil No. 30, 50 per cent } 0·8 lb./ton
9	Concentrate.. Tailing.......	7·7 92·3	5·60 0·12	....	79·6 20·4	....	LeRoi mix. {No. 34 fuel oil water gas tar H. W. creosote and P.T.T. No. 350
10	Concentrate.. Tailing....... Clean up......	3·3 95·2 1·5	10·05 0·15 1·45	....	67·8 27·9 4·3	....	Mix. "A" 1·2 lb./ton (Callow cells)
11	Concentrate.. Tailing.......	14·3 85·7	3·35 0·05	1·02 0·03	91·8 8·2	85·0 15·0	Mix. "A" 2·3 lb./ton H_2SO_4 10·0 "
12	Concentrate.. Tailing.......	19·3 80·7	2·45 0·06	0·56 0·07	90·7 9·3	65·7 34·3	H_2SO_4 10·0 lb./ton. Wood tar mix. 0·8 " P.T.T. No. 350 0·05 "
17	Concentrate.. Tailing.......	3·0 97·0	7·99 0·21	1·56 0·13	54·5 45·5	26·9 73·1	Ground with mix. "A" 2·0 lb./ton H_2SO_4 10·0 " Made very rough concentrate, cleaned with fizzol, midds put in with tailing.

Mix. "A"—Coal tar creosote 60 per cent. Both these from Dominion Tar and Chemical Co.
Coal tar......... 40 "

LE ROI No. 2 MINE—FLOTATION AND TABLING OF RAW ORE

Test No.	Product	Weight per cent	Analysis		Recovery per cent		Remarks
			Cu per cent	Au oz.	Cu	Au	
3	Table conc....	9·4	1·38	0·36	25·0	20·5	2,000 gm. −14 mesh tabled, Table tails and midds ground; floated with K.K. oil 0·5 lb./ton G.N.S. No. 28 0·6 " F.P.L. No. 31 0·1 " P.T.T. No. 350 0·2 "
	Flot. conc.....	2·3	11·45	2·04	50·5	28·5	
	" midd....	5·5	0·65	0·18	6·7	6·0	
	" tails.....	74·2	0·08	0·10	11·3	45·0	
	Table slime...	8·6	0·40	....	6·5	....	
5	Table conc....	17·5	1·13	0·40	38·8	36·6	1,500 gm. −14 mesh tabled. Midd. retabled. Conc. and midd. mixed. Tails reground, floated with mix. "A" 1·4 lb./ton G.N.S. No. 28 0·25 " P.T.T. No. 400 0·4 " (Very deep froth)
	Flot. conc.....	4·9	5·60	1·20	53·8	30·8	
	" midd....	9·5	0·02	0·23	0·4	11·4	
	" tails.....	67·5	0·05	0·06	6·6	21·2	
	Table slime..	0·6	0·37	....	0·4	....	

FLOTATION AND TABLING OF ROASTED ORE

Test No.	Product	Weight per cent	Analysis		Recovery per cent		Remarks
			Cu per cent	Au oz.	Cu	Au	
16	Table conc....	2·8	0·38	0·60	2·1	26·9	1,200 gm. −18 mesh tabled. Tails reground with W.G. tar 1·8 lb./ton and floated with P.T.T. No. 350 0·2 lb./ton.
	Flot. conc.....	1·8	1·12	2·32	3·9	30·0	
	" midd....	10·7	0·55	0·12	11·4	9·2	
	" tails.....	84·7	0·50	0·08	82·6	48·7	

FLOTATION OF ROASTED ORE

Test No.	Product	Weight per cent	Analysis		Recovery per cent		Remarks
			Cu per cent	Au oz.	Cu	Au	
13	Concentrate..	4·9	....	0·95	....	47·3	Roasted 6 minutes at 800° F.
	Midds........	8·3	....	0·31	....	26·2	Ground with mix. "A" 1·8 lb./ton
	Tailing.......	86·8	....	0·03	....	26·5	P.T.T. No. 350, 0·1 lb./ton
14	Concentrate..	8·4	....	0·49	....	39·1	Ground with mix. "A" 1·2 lb./ton
	Tails.........	91·6	....	0·07	....	60·9	
15	Concentrate..	6·4	....	1·00	....	?	Ground with mix. "A" 2·5 lb./ton
	Tailing.......	93·6	....	trace	....	?	Floated with P.T.T. No. 350 0·1 lb./ton

CONCLUSIONS

From the results of the above tests, reagents of Mixture "A" in an acid circuit seem to give the better results. The results, using these reagents, are given under Test No. 11 on raw ore. Satisfactory results were not obtained on the roasted ore.

Test No. 180

THE USE OF FLOTATION REAGENTS MANUFACTURED IN CANADA

C. S. Parsons and C. L. Dewar

This investigation was undertaken at the request of Canadian manufacturers of flotation oils and other flotation reagents. The object was to encourage the use of the Canadian products in preference to those imported from foreign countries.

There are only five sources of flotation reagents in Canada, namely, the petroleum industry, the distillation of coals, the distillation of hardwoods, the Canadian Electro Products Co., and the pulp industry where the sulphite process is used and from which a reagent is obtained possessing high frothing qualities.

There is no commercial source of resinous wood oils in Canada. If it is necessary to use oils of this nature, such as pine oils, they must be imported.

In order to obtain information on the consumption of the various reagents used for flotation purposes and their source of supply, the mining companies using flotation in the treatment of their ores were circularized and asked to fill out a questionnaire. On the whole these were filled out and returned promptly, but the accuracy of some of the figures supplied seems to be open to doubt, and it has been necessary to put our own interpretation on them in certain cases. The tables following are compiled from the answers received. A table is also given, compiled from the records of the Mineral Resources Division showing the consumption of oils for the year 1920. The quantity of oils used for flotation probably reached a maximum in that year.

Pine oil makes up 26 per cent of the total oils imported into Canada for flotation purposes. It has been found that with few exceptions, the use of pine oils is not necessary, and other less expensive oils and reagents will give equally good results. The chief instances of where pine oil is essential are in the flotation of molybdenite and graphite ores. Steam distilled pine oils are the only known oils to give satisfactory results in frothing the pulp in the presence of kerosene oil which is used as the collector in the case of these minerals.

There is at present on the market a Canadian flotation reagent which has a frothing power far greater than pine oil, and which can be used even in the presence of kerosene. This reagent is not an oil, has no collecting power for sulphides, and from the results of experimental work with it, can be used in place of steam distilled pine oil in the flotation of practically every ore where pine oil is used. The reagent is known as Fumol No. 6, manufactured by the Canadian Electro Products Co., Shawinigan Falls, Que. It was tried on the flotation of the porphyry copper ores in the United States in one of the largest mills, and found to be equally as satisfactory as pine oil.

It is one thing to recommend the use of certain reagents for those already in use, but it is a very difficult matter to persuade mill men to use them. The reasons are obvious. The operators become familiar with the characteristics displayed by certain oils in the flotation cells, and can tell at a glance how the cells are working. In 1916 this Department, with the co-operation of the Forest Products Laboratories undertook to find a substitute for pine oil for the flotation of the Cobalt silver ores. This was done at the urgent request of the Cobalt mining companies. The reasons given were that they were unable to obtain an adequate supply of imported pine oil. A substitute was found, namely, hardwood creosote, manufactured in Canada by the Standard Chemical Company of Montreal. To demonstrate finally and conclusively that this oil could be used with equal success, the use of the Buffalo Mill at Cobalt was obtained through the co-operation of Mr. T. R. Jones. This mill was run, using hardwood creosote oil under standard operating conditions, for two weeks, during which time no difficulties were encountered, and the recovery and grade of concentrate was as good as obtained with steam distilled pine oil. In the meantime, the General Naval Stores of New York had obtained long term contracts with most of the southern producers of pine oil and came forward with the guarantee that they would be able to meet the requirements of the mining companies at Cobalt. Not one of the mills ever used the hardwood oils, and even to-day, when pine oil is costing double what it did at that time, and hardwood oil can be obtained considerably cheaper, they have not made the change.

It will be noted that in 1921, 2,800 gallons of hardwood creosote was imported into Canada from Cleveland Cliffs, United States. The Standard Chemical Company, the only producer of hardwood oils in Canada, produces hardwood oils which, in our opinion, have superior flotation qualities to the Cleveland Cliffs product. In our flotation laboratories we have been able to obtain as good results with the Canadian hardwood oils as with the Cleveland Cliffs imported oils, and we recommend their use in preference to the imported hardwood oils.

Samples of 100 pound lots of ore were received from the following companies using flotation, and the test work conducted during the year on these samples is given elsewhere in the Summary Report under their respective test numbers. The work on the remainder will be included in the Summary Report for 1923.

Le Roi No. 2 Limited, Rossland, B.C.	Test No. 179
Consolidated Mining & Smelting Co. of Canada, Trail, B.C.	" " 187
Canada Copper Corporation, Allenby, B.C.	" " 188
Galena Mining and Milling Co., Silverton, B.C.	" " 189
Belmont Surf Inlet Mines Ltd., Surf Inlet, B.C.	" " 190

A list of Canadian mills which our records show use flotation is as follows:—

CONCENTRATION PLANTS IN CANADA USING OIL FLOTATION

Name of Company	*Location*
Copper Ores—	
Eastern Mining and Milling Co.	Eastman, Que.
Eustis Mining Co. (under erection)	Eustis, Que.
Belmont Surf Inlet Mines, Ltd.	Surf Inlet, B.C.
Britannia M. & S. Co., Ltd.	Britannia Beach, B.C.
Canada Copper Corporation	Allenby, B.C.
Consolidated Mining and Smelting Co. of Canada,	Trail, B.C.
Granby Consolidated M. S. & P. Co. (under erection)	Anyox, B.C.
Highland Valley Mining and Development Co.	Highland Valley, B.C.
Kamloops Copper Co.	Kamloops, B.C.
Tidewater Copper Co.	Sydney Inlet, B.C.

Concentration Plants in Canada Using Oil Flotation—*Concluded*

Name of Company	*Location*
Copper-Nickel Ores—	
Mond Nickel Co.	Conniston, Ont.
Lead-Zinc Ores—	
Zinc Company, Ltd.	Notre-Dame des Anges, Que.
Consolidated M. & S. Co. of Canada (under erection)	Kimberley, B.C.
Consolidated M. & S. Co. of Canada	Trail, B.C.
Cunningham, Clarence (Slocan ores)	Alamo, B.C.
Galena Mining and Milling Co.	Silverton, B.C.
Noble Five	Cody, B.C.
Ottawa Mine	Slocan City, B.C.
Roseberry-Surprise Mines Co., Ltd.	Roseberry, B.C.
Silversmith Mines, Ltd.	Sandon, B.C.
Standard Silver-Lead Mining Co., Ltd.	Silverton, B.C.
Van Roi Mining Co.	Silverton, B.C.
Gold Ores—	
Premier Gold Mining Co.	Stewart, B.C.
Argonaut Gold Mines, Ltd.	Dane, Ont.
Silver Cobalt Ores—	
Bailey Silver Mines, Ltd.	Cobalt, Ont.
Cobalt Provincial Mining Co., Ltd.	" "
Coniagas Mines, Ltd.	" "
Dominion Reduction Co.	" "
McKinley-Savage-Darragh Mines, Ltd.	" "
Mining Corporation of Canada " " " Buffalo Mill	" "
M. J. O'Brien, Ltd. (Bailey Mill)	" "
National Mines, Ltd.	" "
Peterson Lake Silver-Cobalt Mining Co., Ltd.	" "
Molybdenum Ores—	
Dominion Molybdenite Co.	Quyon, Que.
St. Maurice Mines, Ltd.	Indian Pen., Que. Amos, Que.
Molybdenum Products Co.	Wilberforce, Ont.
Renfrew Molybdenum Mines, Ltd.	Mt. St. Patrick, Ont.
Steel Alloy Corporation	Dacre, Ont.
Molybdenum Mining and Reduction Co.	Alice Arm, B.C.
Graphite Ores—	
Black Donald Graphite Co.	Calabogie, Ont.
Globe Graphite Mining and Refining Co., Ltd. (The)	Port Elmsley, Ont.
National Graphite, Ltd.	Harcourt, Ont.
Timmins Graphite Mines (The)	Westport, Ont.
Consolidated Graphite M. & M. Co.	Buckingham, Que.
Laurentian Graphite Co.	St. Rémi, Que.
North American Graphite Co., Ltd. (under erection)	Buckingham Que.
Quebec Graphite Co., Ltd. (The)	" "
Standard Graphite Co.	Guenette, Que.

TOTAL OILS USED IN FLOTATION

1921-22

Oil	Quantity used	Pounds of oil used per ton of ore
	Pounds	
Coal tar creosote	223,120	0·44
Coal tar	123,750	0·25
Pine oil, G.N.S., No. 5	8,400	0·035
Pine oil, P. T. & T. Co., No. 350	56,270	0·28
Water gas tar	153,750	0·50
Solvent naphtha and pitch	5,760	4·0
Hardwood creosote oils	28,250	0·25
Total	599,320	

OILS IMPORTED FOR FLOTATION

Oil	Quantity
	Pounds
Pine oil, steam distilled, G.N.S. No. 5	8,400
Pine oil, Pensacola Tar and Turp. Co., No. 350	56,270
Cleveland Cliffs Hardwood Oils	28,250
Water gas tar	153,750
Total	246,670

Quantity of ore reported treated by flotation, 1921–1922	617,844 tons
Quantity of oil used	599,320 lbs.
Quantity of oil used per ton of ore	0·97 lbs.

ADDITIONAL REAGENTS

Reagent	Pounds	Average cost per pound	Total cost	Ore
			$	
Copper sulphate	303,750	3 to 10c.	24,375	Lead-zinc
Soda ash	1,800,000	3·3c.	59,400	Lead-zinc
Sodium silicate	11,000	....	330	Silver and copper
Sulphur	3,920	3·0c.	117	Cobalt-silver
Sod. sulphide	2,600	4·6c.	120	Copper, gold

FLOTATION OILS CONSUMED IN 1920

—	Quantity gals.	Average cost per gal.	Total cost
			$ cts.
Oil imported	116,173	49·0c.	57,219 40
Oil domestic	101,270	24·6c.	24,665 81
Total oil consumed	217,443	37·7c.	81,885 21

Of the above total 5 per cent of oils used were imported.

Of the above total 132,410 gals. were used in the Cobalt district.

Tests Nos. 181 and 182

THE METALLURGICAL TREATMENT OF CANADIAN PYRRHOTITE ORES BY LEACHING AND ELECTROLYTIC DEPOSITION, AND OF CANADIAN PYRITE ORES BY ROASTING, LEACHING, AND ELECTROLYTIC DEPOSITION, WITH THE PRODUCTION OF ELECTROLYTIC IRON, AND THE RECOVERY OF BY-PRODUCTS SUCH AS SULPHUR, COPPER, ETC.

R. J. Traill

INTRODUCTORY

An investigation to determine the suitability of a leaching and electrolytic method of treatment for pyrrhotite and pyrite ores in connection with their possible utilization in the manufacture of electrolytic iron, was started in November 1922.

The preparation of electrolytic iron has been a subject of scientific study and research for about 80 years, but it is only during the last 15 or 20 years, with the realization of its unusual physical and chemical properties, that metallurgists and chemists have turned their attention toward establishing for it a practical industrial process. Processes are in operation at the present time in France, Germany and America, producing an iron of 99·9 per cent to 99·97 per cent purity, from pig iron or scrap used as anodes in an electrolyte of ferrous salts, with and without additional reagents, on steel cathodes of either sheet or rotating mandrel form.

Electrolytic iron has unusual mechanical and electrical properties which open up many special fields of usefulness, and it is in demand in the making of special steel or alloys, boiler tubes, electrical machinery, wire, and articles to be nickel plated or galvanized, etc.

THE EUSTIS PROCESS

F. A. Eustis of Boston and C. P. Perin of New York have patented what is known as the Eustis process, with the idea of utilizing iron ores directly. It differs from preceding processes in that an insoluble anode is used, and a diaphragm separates the anolyte and catholyte. In the cell iron is deposited on the cathode, while oxidation of the ferrous chloride takes place at the anode. The ferric chloride thus formed in the anode compartment is led over the raw pyrrhotite ore (pyrite requires special treatment) effecting the solution of the iron and simultaneously reducing the ferric chloride to the ferrous state. Sulphur is deposited in the gangue and is recovered as brimstone. The chemistry of the cell reaction may be shown as

$$\underset{\text{Ferrous chloride}}{3FeCl_2} = \underset{\text{Iron}}{Fe} + \underset{\text{Ferric chloride}}{2FeCl_3}$$

The production of ferric chloride is directly proportional to the amount of iron deposited.

The reaction of the ferric chloride and ore may be shown by the equation:

$$\underset{\text{Ferric chloride}}{2FeCl_3} + \underset{\substack{\text{Mono-sulphide}\\ \text{or iron}}}{FeS} = \underset{\text{Ferrous chloride}}{3FeCl_2} + \underset{\text{Sulphur}}{S}$$

From these equations it will be seen that exactly the amount of iron removed by electrolysis is restored to the solution, and the process becomes cyclical.

In an ore containing copper the copper dissolved must be removed, probably by precipitation on scrap iron, and can be readily recovered as a by-product. It is claimed for the process that 90 per cent of the iron may be recovered as pure electrolytic iron, and 90 per cent or more of the sulphur as brimstone. The current density may be varied over a wide range, 20-100 amp. per sq. ft., and the voltage from 1-4·5 volts.

The Estelle Process

The Estelle process, which in some respects is similar to the above, has been patented by Axel Estelle.

In this process the suitably prepared ore is heated with a weak non-oxidizing acid, preferably hydrochloric, at a temperature of 40-80° C. The result of the reaction is the formation of ferrous chloride and sulphuretted hydrogen. The ferrous chloride goes to the electrolytic cell where a portion of the iron is deposited, and twice this amount of iron is converted to ferric chloride. The sulphuretted hydrogen set free in the acid-ore reaction is collected in absorption towers and used for the reduction of the ferric chloride from the cell, hydrochloric acid being set free with separation of sulphur. The hydrochloric acid previously combined with the non-electrolytically deposited is in this manner again liberated and made available for leaching fresh ore.

It is claimed for this process that other "sulphur containing metals" present remain with the gangue, and can be readily recovered in this concentrated condition, also that the sulphur is eliminated in a separate apparatus in a fairly pure condition.

Laboratory Investigations

The laboratories of the Ore Dressing and Metallurgical Division plan to investigate the possibilities of each of these processes, and is at the present time conducting tests on the Eustis process. The work done so far has been of a very preliminary nature and only on a small laboratory scale. The following brief description of tests are submitted to demonstrate the possibilities of the leaching process as applied to pyrrhotite and pyrite ores.

GENERAL PROCEDURE FOLLOWED

An enamel pail of 5-6 litres capacity, fitted with a wooden cover, was used as a leaching tank. Agitation was accomplished by a rotating wooden agitator. Steam was used for maintaining the temperature necessary for leaching. The ferric chloride used was ordinary grade, containing about 20 per cent iron.

It was found that steam passing into a cold or slightly warm solution of ferric chloride hydrolyzed the ferric chloride to an appreciable extent, and the iron so hydrolyzed would be lost as oxychloride of iron in the gangue. By passing the steam into a hot solution, or a slightly acid solution, the hydrolytic dissociation was greatly reduced. Heating directly with steam causes a dilution of solution due to condensation of the steam, and the increase in volume has been as much as 18 per cent in two hours running on a 3 litre test. This problem of heating the solution will probably be more readily overcome in a continuous circuit process, as the electrolyte will be hot, due to the action of the current. It may also be observed here that the reaction between the ore and the ferric chloride is exothermic to the extent of 20,400 calories or 700 B.Th.U. per pound of iron dissolved.

For laboratory tests a steam jacket method of heating was found most satisfactory; there was a loss of course in volume due to evaporation amounting to 5-15 per cent, but the hydrolytically dissociated iron was greatly reduced.

According to theory, in the leaching process 2 parts of iron as ferric chloride should dissolve 1 part iron from the ore, resulting in the formation of 3 parts iron as ferrous chloride.

Test No. 181

A shipment of pyrrhotite ore received in November from the Smith property, west side of lake Memphremagog, Que., was used in the following tests. This ore on analysis was found to be about 80 per cent pyrrhotite, 25 per cent chalcopyrite, 2·5 per cent lime (CaO). The iron content averaged 49·5 per cent, and the copper 0·85 per cent.

Leaching Test No. 1.—Quantities used:

3·0 litres ferric chloride at 70 grams per litre = 210 grms. Fe.
235 grams ore (−200 mesh at 49·5 per cent Fe = 116·5 grms Fe.

This amount of ore would be about 11 per cent in excess of theoretical requirement.

The ore and solution were put in a pail and heated by passing steam into the solution, agitating at the same time, until the temperature had reached 95° C. which temperature was maintained for 2 hours. The solution was allowed to cool and was then filtered, increase of volume being 18·3 per cent. Theoretically the 210 grams of Fe in the ferric chloride should react with 105 grams of Fe in the ore to form 315 grams of ferrous iron.

Analysis of solution showed that 252 grams of ferrous iron had been formed, which is 80 per cent of theoretical amount, or in other words 84 grams of iron were leached out of the ore. Had the time of leaching been extended, it is probable that a much higher recovery would have been made.

Leaching Test No. 3.—This test was run to obtain data concerning the rate of reaction when using a high excess of ore, and to obtain some information on counter-current extraction, and also on the behaviour of copper in the ore. Quantities used:—

3·5 litres ferric chloride at 66 grams per litre = 231 grams Fe.
335 grams ore (−200 mesh) at 49·6 per cent Fe = 165·8 grams Fe.
and at 0·85 per cent Cu = 2·85 grams Cu.

This amount of ore would be about 45 per cent in excess of theoretical.

The solution was heated to 75°C. by steam jacket method, ore added, and the temperature increased to and held at 95°C.

Solution sampled after 30 minutes leaching contained 85 g/l ferrous Fe.
Solution sampled after 60 minutes leaching contained 92·5 g/l ferrous Fe.
Solution sampled after 90 minutes leaching contained 105 g/l ferrous Fe.
Solution sampled after 120 minutes leaching contained 106 g/l ferrous Fe.
Reaction considered complete, no ferric iron present. Time 2 hours.
Volume of solution after cooling and filtering = 3·10 litres
Analysis showed total ferrous iron = 328·6 grams

Theoretically 231 grams of Fe in the ferric chloride would react with 115·5 grams Fe in the ore to give 346·5 grams of ferrous iron, so that 328·6 grams ferrous iron found is equal to 95 per cent of the theoretical, or 109·5 grams of iron were leached from the ore.

The residue weighed 206 grams and contained 30·6 per cent Fe or 63 grams. Theoretically Fe in residue should be the amount of iron in quantity of ore used, minus amount of iron dissolved in leaching, which would be 165·8 — 109·5 = 56·3 grams. It was found, however, that ferric chloride hydrolyzed and oxychloride of iron precipitated, hence this excess of 6·7 grams in the residue is probably due to (1) iron hydrolytically dissociated, and (2) iron precipitated by lime present in ore.

The copper in residue amounted to 1·17 per cent or 2·41 grams, therefore the amount of copper leached was 15·3 per cent.

Leaching of residue:—180 grams of this residue containing 55 grams Fe and 2·106 grams Cu were treated with 3·4 litres solution containing 45 grams/litre ferrous Fe = 153 gr. Fe. containing 32 grams/litre ferric Fe = 108·8 gr. Fe.

Conditions of leaching similar to first part of test. After 2½ hours leaching analysis of solution showed a content of 73 grams/litre ferrous Fe, and 17 grams/litre ferric Fe.

Volume of solution after cooling and filtering = 3·125 litres. Total ferrous iron is 228·3 grams and ferric iron 53 grams.

The result shows that 25 grams of Fe in the residue has been leached out.

The copper content of the residue was 0·65 per cent, weight of residue 150 grms.

Combining the results of these two tests it is found that:

Total extraction of iron = 83·4 per cent
Total extraction of copper = 61·6 per cent

Summing up it may be said that when a large excess of ore is used the reaction is very rapid and a completely reduced ferrous chloride solution is produced. In practice this reaction would take place in the primary agitators. In the treatment of the residue from this first action, which would take place in the secondary agitators, the reaction would be much slower, due possibly to the liberated sulphur forming a protective coating over the ore particles.

In the above test a higher extraction would probably have resulted had the period of extraction in the secondary action been extended.

As a result of the several tests made, it has been found that fine grinding of the pyrrhotite is essential and that the leaching process should be conducted at a temperature around 90-95°C. Best results have been obtained by grinding to minus 200 mesh (·0029 inch opening) and leaching at 95°C.

PYRITE TREATMENT

Pyrite (FeS_2) is not readily attacked by ferric chloride, and requires preliminary treatment. This treatment consists of converting the pyrite into a variety of pyrrhotite, soluble in ferric chloride, by roasting in a neutral or reducing atmosphere at a temperature of 700-750°C. Under these conditions one atom of sulphur is driven off and with suitable apparatus can be recovered as a by-product.

Test No. 182

A shipment of pyrite received November, 1922, from the Eustis mine, Eustis, Quebec, was used in the following test:—

Chemical analysis of ore: iron 41·2 per cent; copper 1·55 per cent; lead 0·75 per cent; sulphur 46·7 per cent. This would be about 85 per cent pyrite and 4·5 per cent chalcopyrite. Screen analysis showed:—

—	Sample weight	Per cent	Per cent accumulative
Held on 35 mesh (·0164″ opening)	14	7	7
" 48 " (·0116″ ")	28	14	21
" 65 " (·0082″ ")	40	20	41
" 100 " (·0059″ ")	48	24	65
" 150 " (·0041″ ")	28	14	79
" 200 " (·0029″ ")	28	14	93
Passing 200 " (·0029″ ")	14	7	100
	200	100	....

Pyrite roasting.—Three samples of 150 grams each were placed in a piece of wrought iron pipe 8″ x 1½″, closed at one end with a cap. These were placed in a small electric muffle furnace and roasted at a temperature ranging from 700 to 750 C. An examination after 45 minutes roasting showed incomplete roasting, particles of pyrite being observed. Analysis gave 39 per cent iron in ferrous condition. A further 45 minutes roast was made, and resulting product showed 48·2 per cent ferrous iron and a magnetic test showed 91 per cent of the roasted material to be magnetic. The samples all checked very closely, the third sample being roasted for two hours gave results similar to the 1½ hour roast. Analysis of roasted material was:—

	Per cent
Iron (total)	50·00
Copper	1·90
Lead	0·92
Total sulphur	33·30

Of the iron, 97·2 per cent was present in the ferrous condition. A small amount of SO_3 was present as originally in the raw ore. From further analytical data obtained the composition of the roast would be somewhat as follows:—

Fe_8S_9, 79·35 per cent Cu_2S 2·38 per cent FeS_2 2·67 per cent.

Leaching.—A quantity of the roasted material ground to pass a 200 mesh screen (·0029″ opening) was leached with a solution of ferrous and ferric chlorides under conditions of temperatures and agitation similar to those pertaining in the leaching of pyrrhotite.

The reduction of the ferric chloride and leaching did not occur quite as rapidly as with natural pyrrhotite, 210 grams ore requiring 7 hours leaching, with a solution containing 308 grams Fe as ferrous chloride and 209 grams Fe as ferric chloride, excess of ore considering copper and lead, being about 6·5 per cent. However, results showed extraction of 85 per cent of the iron, 42 per cent of the copper, and practically 100 per cent of the lead.

GENERAL CONCLUSIONS

The results obtained in these tests, though being of a very preliminary nature, show that this method of treatment for pyrrhotite and pyrite is very promising. So far as the iron is concerned it would appear that recoveries of over 80 per cent could readily be made.

Concerning copper, the results are not so encouraging, the copper going partly into solution and partly with the gangue. This problem, as well as the separation and recovery of other metallic sulphides occuring in these ores, will be gone into in the investigation contemplated.

The laboratory at present is not equipped with electrolytic apparatus, and so it has not been possible to study the electro-deposition phase of the process. However, plans are being prepared with this end in view, and it is hoped that a continuous circuit suited to the proper study of this process will be installed in the near future, when the process may be investigated under conditions approaching more closely those that would obtain in actual practice.

Test No. 183

THE LOW GRADE ORE FROM THE MURRAY MINE

H. C. Mabee

This test was made on a low grade sample of ore from the Murray Mine, British America Nickel Corporation, Nickelton, Ont., and was conducted with the object of determining if the precious metal values were concentrating with the nickel and copper in the process of flotation, or in which of the products they were reporting. Attention was not paid so much to obtaining the best flotation results, as this had been done in former tests on the ore (See Summary Report of the Mines Branch 1921, Test No. 160), but to the getting of sufficient of the different flotation products for assaying.

To obtain this a number of flotation tests were run, using in all 13 kilos of the ore, and the different products combined.

The assay results of these products, as well as the percentages in the products, appears below.

Product	Weight grams	Assays						Value of precious metal content
		Copper per cent	Nickel per cent	Gold oz./ton	Platin. oz./ton	Palladm. oz./ton	Rho. Irid., etc. oz./ton	
							*	[1]
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
Feed............	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 fuzing and scorifying.

[1] Figured from December, 1922, quotations on these metals.

Product	Weight grams	Percentages in Products						Value of precious metal
		Copper	Nickel	Gold	Platin.	Palladm.	Rho., Irid., etc.	
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
Feed............	100·0	100·0	100·0	100·0	100·0	100·0	....	100·0

On this grade of ore the test shows that 60 per cent of the precious metal values are recovered in the concentrate, 25 per cent in the middling, and 15 per cent remains in the tailing. The middling product representing 10·9 per cent of the original feed would be reground and refloated, producing a total recovery of 75 per cent of the precious metal values from this grade of ore.

The result of this test demonstrates fairly conclusively that the gold and the platinum group metals are following the nickel-copper concentrate in the flotation process.

Test No. 184

THE PRECIOUS METAL VALUES IN THE NICKEL-COPPER PYRRHOTITE ORES OF THE SUDBURY DISTRICT. THE LOW GRADE ORE FROM No. 3 (FROOD) MINE

H. C. Mabee

This test was conducted on a low grade sample of ore from Mine No. 3 (Frood) of the International Nickel Co., Copper Cliff, Ont., with the object of determining if the precious metal values were reporting with the nickel and copper in the flotation concentrates, or in what flotation product they were reporting. Attention was not paid so much to obtaining the best flotation results, as this was demonstrated by former tests on this ore (See Summary Report of the Mines Branch, 1921, Test No. 155), but to obtaining sufficient quantities of the flotation products for accurate determination of the precious metal contents.

A sample consisting of 13 kilos of ore was crushed and flotation tests run on the whole in 1 kilo lots, and the products combined. The results of this test are contained in the table following:—

Product	Weight grams	Assay						Precious metal values content
		Nickel per cent	Copper per cent	Gold oz./ton	Platin. oz./ton	Palladm. oz./ton	Rhod., Irid., etc. oz./ton	
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
Feed.............	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.

Product	Weight grams	Percentages in Products						Precious metal values content
		Nickel	Copper	Gold oz./ton	Platin. oz./ton	Palladm. oz./ton	Rhod., Irid., etc. oz./ton	
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
Feed.............	100·0	100·0	100·0	100·0	100·0	100·0	....	100·0

The above results on this particular grade of ore show that 68 per cent of the total precious metal values are recovered in the concentrate, 17 per cent appears in the middling, and 15 per cent is retained in the tailing. In this case again, 11·4 per cent of the original feed represented in the middling would be reground and floated, producing a total recovery of 80 per cent of the precious metal values in the process of concentration by flotation.

The assay results of these products, as well as the percentages in the products, appears below.

Product	Weight grams	Assays						Value of precious metal content
		Copper per cent	Nickel per cent	Gold oz./ton	Platin. oz./ton	Palladm. oz./ton	Rho. Irid., etc. oz./ton *	[1]
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
Feed.............	13,000	0·43	1·14	0·0055	0·0067	0·0009	0·0009	$1 55

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[1] Figured from December, 1922, quotations on these metals.

Product	Weight grams	Percentages in Products						Value of precious metal
		Copper	Nickel	Gold	Platin.	Palladm.	Rho., Irid., etc.	
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Tailing...........	76·5	8·8	14·0	12·5	12·7	22·0	....	14·8
Feed.............	100·0	100·0	100·0	100·0	100·0	100·0	....	100·0

On this grade of ore the test shows that 60 per cent of the precious metal values are recovered in the concentrate, 25 per cent in the middling, and 15 per cent remains in the tailing. The middling product representing 10·9 per cent of the original feed would be reground and refloated, producing a total recovery of 75 per cent of the precious metal values from this grade of ore.

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The above results on this particular grade of ore show that 68 per cent of the total precious metal values are recovered in the concentrate, 17 per cent appears in the middling, and 15 per cent is retained in the tailing. In this case again, 11·4 per cent of the original feed represented in the middling would be reground and floated, producing a total recovery of 80 per cent of the precious metal values in the process of concentration by flotation.

IV

OTHER TEST WORK OF THE DIVISION

W. B. Timm

Crushing tests were made by R. K. Carnochan on 533 pounds of Nepean sandstone.

Several shipments of clay, totalling 2,540 pounds, were ground and prepared for the use of the Public Schools of Ottawa.

A shipment of supposed gold ore was submitted by Mr. Geo. Cheney. This was found to contain no values in gold and silver, so no further test work was performed.

A sample of rock from Mr. H. Stuart, Dauphin, Man., was identified as greenstone of no commercial value.

A sample, identified as hematite, was submitted by Mr. Dalmore Constantineau of Ottawa.

A supposed gold ore from Mr. Wm. Allen, N.B., was found to contain no gold or silver.

Some test work was performed on a shipment of feldspar by Mr. Norman Davis in the laboratories.

V

ADDITIONAL EQUIPMENT FOR THE LABORATORIES

W. B. Timm

A Callow pneumatic flotation unit, consisting of two rougher cells, and two cleaner cells, capacity 5-10 tons in 24 hours, was installed, to conduct the large scale or tonnage check tests on samples of over two tons. With this unit a mill flow sheet can be followed, closely approaching actual practice, and the results of the small scale laboratory tests verified. The capacity of this unit conforms to that of the grinding capacity of the 4′ 6″ Hardinge ball mill when grinding to 100 mesh or finer.

The unit is so arranged that the feed can be split going to both roughers, each followed by a cleaner cell, or followed by two cleaners, one after the other, or the entire feed can go to one rougher and the tailings to the other, the concentrate from both passing on to separate cleaners, or first one cleaner followed by another cleaner. This arrangement gives almost any desired combination.

A Heavi-Duty type of electric muffle furnace was installed for assay work. This furnace with its special cast alloy tray was found to be most satisfactory for assay work. It is much easier to manipulate and gives a more accurate control of temperatures than the oil and gas furnaces formerly used for this work.

A large electric drying oven was built and installed in the sample room for the drying of products from the small scale test work. The oven is giving the utmost satisfaction for this work.