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

MINES BRANCH
JOHN MCLEISH, DIRECTOR

INVESTIGATIONS IN ORE DRESSING AND METALLURGY

(Testing and Research Laboratories)

1924

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Annual reports on Mines Branch investigations are now issued in four parts, as follows:—

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Investigations in Ore Dressing and Metallurgy (Testing and Research Laboratories).

Investigations of Fuels and Fuel Testing (Testing and Research Laboratories).

Investigations in Ceramics and Road Materials (Testing and Research Laboratories).

Other reports on Special Investigations are issued as completed.

MINES BRANCH INVESTIGATIONS IN

ORE DRESSING AND METALLURGY, 1924

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm
Chief of Division

During the year 1924, the laboratories of the Division were devoted entirely to experimental and research work in connexion with the treatment of Canadian ores and metallurgical products. The results of the various investigations are given in the reports by the officials directly engaged on the work. Section I is a brief review of the investigations given in detail in Sections II, III, IV, and V, together with a list of the ores submitted for test purposes.

Section II (pages 8-91, inclusive)

The results of the investigations conducted in the Ore Dressing and Metallurgical Laboratory by Messrs. W. B. Timm, C. S. Parsons, R. K. Carnochan, and J. S. Godard, are included in this section. A brief review and summary of the investigations is as follows:—

Report No. 204 (pages 8-19, inclusive)

THE CONCENTRATION OF THE ZINC-LEAD ORES OF NOTRE DAME DES ANGES, QUE., BY C. S. PARSONS

The experimental work was conducted on two small shipments of zinc-iron middlings, one small shipment of ore, and a carload shipment. The results obtained on the zinc-iron middlings indicated that these middlings could be concentrated by flotation, with a separation of the zinc from the iron, and the production of a marketable zinc product, with high recoveries of the zinc content. The preliminary results on the small ore shipment showed that the ore could be concentrated by selective flotation or by a combination of gravity concentration and selective flotation. The results obtained by tonnage check tests on the carload shipment verified the results obtained from the preliminary tests and showed that the ore could be readily concentrated, with the production of marketable lead and zinc products with high recoveries of the values in these metals, and furthermore that the silver and gold values, together with the small amount of copper mineral present in the ore, reported almost entirely with the lead product. A concentrating mill for the treatment of the ore has been built at the mine following a flow-sheet based on the results of these experimental tests.

Report No. 205 (pages 20-21, inclusive)

THE MANUFACTURE OF A WATERPROOFING COMPOUND FOR USE IN PAPER BOARD, BY W. B. TIMM

Waterproofing compounds for use in paper board have been manufactured from hot colloidal mixtures of various mineral substances. This report covers the preparation in the cold of a bentonite-asphalt mixture suitable for incorporating into felt, paper, or other materials.

Report No. 206 (pages 22-24, inclusive)

THE CONCENTRATION OF A LEAD-ZINC ORE FROM RIONDEL, B.C., BY R. K. CARNOCHAN

The ore was a mixture of lead and zinc sulphides with pyrite and arsenopyrite in a gangue of manganese carbonate, calcite, and quartz, carrying values in silver and gold. The results of the experimental tests indicated that the ore could be satisfactorily concentrated by selective flotation with a separation of the lead and zinc sulphides, producing marketable products of these minerals with good recoveries of the values.

Report No. 207 (pages 24-46, inclusive)

EXPERIMENTAL TESTS ON WINDPASS ORE, BY J. S. GODARD

The test work was conducted on two lots representative of the two types of ore from the Windpass mine, Chu Chua, B.C. The results indicate that the ore can be successfully treated by amalgamation, concentration by flotation, and cyanidation, with good recoveries of the values. The report covers in detail the various tests conducted on both types of ore, and on a mixture of the two types.

Report No. 208 (pages 46-49, inclusive)

THE CONCENTRATION OF GARNETIFEROUS GNEISS FROM BARRY'S BAY, ONT., BY R. K. CARNOCHAN

The experimental work conducted showed that wet concentration was preferable to dry concentration on this class of garnetiferous gneiss. It showed that a process of jigging and tabling would work very satisfactorily on the material, giving a product suitable to the requirements of the trade.

Report No. 209 (pages 49-51, inclusive)

GOLD ORE FROM MINE CENTRE, ONT., BY C. S. PARSONS

The test work conducted showed the ore very amenable to treatment by amalgamation followed by cyanidation, or by straight cyanidation. The ore submitted contained considerable free gold, and on this grade of ore amalgamation before cyanidation is recommended. If, however, the average grade of the milling ore was below one ounce to the ton, and very little coarse gold was present, amalgamation could be dispensed with and straight cyanidation used.

Report No. 210 (pages 51-53, inclusive)

THE CLEANING OF MICA PREPARATORY TO TRIMMING AND SPLITTING, BY R. K. CARNOCHAN

The experimental tests show that the removal of the thin and small flakes by screening and suction is beneficial, giving a product more satisfactory for trimming and splitting, with a saving in hand labour. Much of the fine particles adhering to the surfaces of the larger flakes is removed, permitting of a thinner portion being discarded in splitting. Washing tests were also made and the results obtained are given in the report.

Report No. 211 (pages 53-54, inclusive)

EXAMINATION OF MINERALIZED DIKE MATERIAL FROM LARDER LAKE, ONT., BY J. S. GODARD

Experimental tests were made to determine whether the gold values were in the arsenopyrite or in the pyrite. It was found that the values were associated with both minerals, but to a greater extent with the arsenopyrite.

Report No. 212 (pages 55-58, inclusive)

THE CONCENTRATION OF A GOLD-COPPER ORE FROM SPROAT LAKE, B.C., BY C. S. PARSONS

The experimental work showed the ore very amenable to concentration by flotation and table concentration. The gold values were found to be associated with the sulphides of copper and iron. A simple separation of the sulphides from the gangue was required. This could either be accomplished by classification, table concentration, and flotation, or by flotation and table concentration. The latter method was recommended.

Report No. 213 (pages 58-61, inclusive)

METALLURGICAL TESTS ON GOLD ORE FROM THE REX MINE, HERB LAKE, MAN., BY C. S. PARSONS

The report on this ore covers the results obtained by amalgamation as against blanket concentration, for the recovery of the free gold. Table concentration and cyanide tests were also made. It was found that the ore was very amenable to treatment by amalgamation and cyaniding, and that amalgamation showed a higher recovery than blanket concentration for the recovery of the free gold.

Report No. 214-214a (pages 62-67, inclusive)

EXAMINATION AND EXPERIMENTAL TESTS ON GOLD ORE FROM DOMINION CLAIMS, COPPER LAKE, MAN., BY W. B. TIMM AND J. S. GODARD

The ore was submitted to determine in what form the gold and silver values were present, and also to determine its amenability to treatment. It was found that the gold values were present both as free gold and tellurides; that the silver was present as ruby silver, and possibly as the sulphide, although this mineral was not detected by microscopic examination, and a small portion may be associated with the small amount of galena in the ore. The ore is amenable to treatment by cyanidation with high recoveries of the gold and silver values, and with low consumption of cyanide and lime.

Report No. 215 (pages 68-71, inclusive)

THE CONCENTRATION OF MOLYBDENITE ORE FROM THE MOSS MINE, QUYON, QUE., BY C. S. PARSONS

The report covers the concentration of two carload shipments of ore, and the results obtained prove conclusively the amenability of this type of molybdenite ore to concentration by flotation. A high-grade concentrate of 93 per cent MoS_2 was obtained, with recoveries of over 95 per cent of the molybdenite content in the ore. The correct conditions of operation to obtain these results are described in detail.

Report No. 216 (pages 71-74, inclusive)

EXAMINATION OF KIRKLAND LAKE MILL TAILINGS, BY C. S. PARSONS

The examination included screen analysis, classification tests, and analysis of the classified products, flotation tests, etc. The results of the screen tests and classification tests indicated that too coarse grinding was responsible to a large extent for the high tailing losses. No satisfactory results were obtained from flotation of the cyanide tailings.

Report No. 217 (pages 74-82, inclusive)

CONCENTRATION OF A SILVER-LEAD ORE FROM PORTLAND CANAL DISTRICT' NEAR STEWART, B.C., BY C. S. PARSONS

The experimental work was of a preliminary nature indicating possible methods of treatment. These methods include selective flotation, with the production of both lead and zinc products, the silver reporting in both products, and straight flotation of the sulphides from the gangue. Cyanidation of the flotation tailings may be required.

Report No. 218 (pages 83-84, inclusive)

CONCENTRATION OF TITANITE (SPHENE) FROM PEGMATITE, BY C. S. PARSONS

A simple method of gravity concentration was followed for this material. The percentage of sphene in the rock was too low to make its separation of economic importance at the present time.

Report No. 219 (pages 85-87, inclusive)

EXPERIMENTAL TESTS ON CADWALLADER CREEK MILL TAILINGS, BY J. S. GODARD

Examination of these tailings showed them to contain fairly high gold values. The test work included amalgamation, concentration, and cyanidation. Very good recoveries were made by cyanidation. As cyanidation was used for the recovery of the values in the Cadwallader Creek ores, it would seem that increased recoveries could be expected by more careful attention to the details of this practice.

Report No. 220 (pages 87-91, inclusive)

CONCENTRATION OF THE LEAD-ZINC ORE OF THE READER MINE, CALUMET ISLAND, QUE., BY W. B. TIMM AND C. S. PARSONS

The experimental work showed that the ore was very amenable to treatment by selective flotation, with the production of high-grade lead and zinc products, with good recoveries. The tests also showed a good recovery of the silver values in the lead product.

Section III (pages 92-102, inclusive)

This section covers the progress of the work in the hydrometallurgical laboratory on the investigation being conducted by R. J. Traill and W. R. McClelland on the treatment of iron sulphide ore for the production of electrolytic iron, and the recovery of sulphur and other metals as by-products.

Section IV (pages 103-108, inclusive)

A brief summary of the work of the chemical laboratory is given in this section, together with a special investigation which was started during the year by H. C. Mabee, on the recovery of the iron and sulphur content, in addition to the nickel, copper, and precious metal values, from the nickeliferous pyrrhotite ores, and ores of a similar character.

Section V (pages 109-115, inclusive)

C. S. Parsons has prepared a short report which deals with the special problems connected with the concentration of lead-zinc ores of eastern Canada. Included in this section is a report, also by C. S. Parsons, on the concentration of the Lake George antimony ores, which is a summary of his experimental work in the laboratories.

List of Ores and Metallurgical Products on which Experimental Test and Research Work was Conducted

In the following table is given a list of the ores and metallurgical products received at the Ore Testing Laboratories for experimental test and research. The tabulated statement includes the report numbers, class of ore or product, source of shipment, name of shipper, and weight of shipment.

Report No.	Ore or product	Source of shipment	Shipper and address	Weight lbs.
204	Lead-zinc.	Notre Dame des Anges, Que.	Tetreault Bros., Notre Dame des Anges, Que.	54,242
205	Asphalt-bentonite.	Mansons, Ltd., Hawkesbury, Ont.	Mansons, Ltd., Hawkesbury, Ont.	500
206	Lead-zinc.	Riondel, B.C.	Shepherd Mining Co., Riondel, B.C.	177
207	Gold-copper.	Windpass mine, Chu Chua, B.C.	Windpass Mining Co., Chu Chua, B.C.	381
208	Garnet.	Barry's Bay, Ont.	Bancroft Mines Syndicate Ltd., Barry's Bay, Ont.	3,631
209	Gold.	Seine river, Mine Centre, Ont.	J. S. Hillyer, Esq., Mine Centre, Ont.	200
210	Mica.	Blackburn Bros., Ottawa, Ont.	Blackburn Bros., Ottawa, Ont.	564
211	Gold-arsenic.	Associated Goldfields, Larder Lake, Ont.	Dr. H. C. Cooke, Geological Survey, Ottawa	11
212	Gold-copper.	Sproat lake, near Alberni, B.C.	Messrs. Smith and Beavan, Alberni, B.C.	25
213	Gold.	Rex mine, Herb Lake, Man.	Manitoba Metals Co., Herb Lake, Man.	142
214	Gold-silver.	Dominion claims, Copper lake, Man.	A. L. Stewart, Esq., Winnipeg, Man.	205
215	Molybdenite.	Moss mine, Quyon, Que.	Canadian Wood Molybdenite Co., Quyon, Que.	135,200
216	Gold.	Kirkland Lake, Ont.	Wright Hargreaves Mines Ltd., Kirkland Lake, Ont.	50
217	Lead-zinc-silver.	Stewart, B.C.	S. G. O. Chalmers, Esq., Stewart, B.C.	54
218	Titanium.	Alleyn township, Pontiac district, Que.	Messrs. Spence and Cole, Mines Branch, Ottawa.	42
219	Gold.	Cadwallader creek, B.C.	A. W. Davis, Esq., Kamloops, B.C.	96
220	Lead-zinc.	Reader mine, Calumet island, Que.	W. B. Timm, Esq., Mines Branch, Ottawa.	50

II

REPORTS OF INVESTIGATIONS: ORE DRESSING AND METALLURGICAL LABORATORY

Report No. 204

CONCENTRATION OF THE ZINC-LEAD ORES OF NOTRE DAME DES ANGES, QUEBEC

C. S. Parsons

Shipments:—Four separate shipments were received from the property of Dr. J. L. A. Tetreault and associates of 730 Delorimier Ave., Montreal. The shipments were made up as follows:

Shipment No. 1 consisted of 100 pounds of zinc-iron middlings from a dump, produced from the operation of a gravity concentrator. This shipment was received in July, 1923.

Shipment No. 2 was a sample of the crude ore taken from the mill feed to the gravity concentrator then in operation. The shipment weighed 42 pounds and was received in July, 1923.

Shipment No. 3 consisted of 100 pounds of zinc-iron middling from tables in a gravity concentrator then in operation. This was received in July, 1923.

Shipment No. 4 was a carload of 27 tons of crude ore representing the mill feed to the gravity concentrator then in operation. This shipment was received in November, 1923.

Characteristics and Analyses of Shipments Nos. 1 and 3.—The zinc-iron middling product received consisted of a mixture of zinc and iron sulphides which had been freed by crushing to 40 mesh. Both pyrrhotite and pyrite were present, the pyrrhotite predominating. Analyses:

	Shipment No. 1	Shipment No. 3
Zinc	17·23 per cent	15·6 per cent
Lead	2·16 "	2·2 "
Iron	37·57 "	38·0 "
Gold	0·12 oz./ton	0·12 oz./ton
Silver	4·68 "	4·70 "

Characteristics and Analyses of Shipments Nos. 2 and 4.—These shipments represent the crude ore. The minerals are probably entirely freed by crushing to 65-mesh (Tyler standard screen). Mineralogically the ore is composed of galena, a dark zinc blende, pyrite, pyrrhotite, and a small amount of chalcopyrite in a siliceous gangue containing chloritic minerals and calcite. The galena is argentiferous and there is gold present, some of which is apparently in the free state. Analyses:

	Shipment No. 2	Shipment No. 4
Zinc	6·43 per cent	5·59 per cent
Lead	1·86 "	2·53 "
Iron		6·78 "
Gold	0·08 oz./ton	0·06 oz./ton
Silver	2·98 "	3·75 "

Object of Experimental Work.—The property from which these samples were taken has been operated from time to time for a number of years. Numerous attempts have been made to separate and recover both a lead and zinc concentrate which could be marketed. The attempts to produce a zinc concentrate were failures, but the mine was operated, and is being operated at present, to produce a lead concentrate. The lead is concentrated by graded crushing and tabling. Both flotation and magnetic separation have been tried on the ore, and mills were built to use these processes. During the war period a magnetic plant operated for some time, but it is reported that it was not a success. When flotation was used, the iron and zinc were both floated, the bulk of the lead having been previously eliminated by tables. The flotation concentrate of zinc and iron was given a magnetic roast and the iron eliminated. This process was not a commercial success.

The purpose of the experimental work in the case of the shipment of zinc-iron table middlings taken from the dump produced by previous milling operations, was to determine whether a successful method could be worked out whereby a high-grade zinc concentrate could be produced. Shipment No. 3 was the same material as No. 1 except that it was freshly produced middling, whereas the material of shipment No. 1 had lain exposed to the weather for some time. The purpose of submitting shipment No. 3 was to determine whether there was any material difference between the two products with respect to their response to various methods of separation.

Shipments Nos. 2 and 4 which consisted of the crude ore were submitted to determine whether a possible method could be worked out so that both a lead and zinc concentrate could be recovered as marketable products.

EXPERIMENTAL TEST ON SHIPMENTS NOS. 1 AND 3

The work done on the shipments of zinc-iron middlings was confined to small-scale magnetic and flotation tests.

Magnetic Separation Tests.—Two methods were tried: separation of the raw middlings, and of middlings which had been given a flash roast to change the pyrite to the magnetic sulphide.

Results.—No results were obtained by the first method. By the second method the results were very promising. The following is typical of a number of tests made:—

Product	Weight per cent	Analysis					Per cent of zinc values
		Zn %	Pb %	Fe %	Au. oz./ton	Ag. oz./ton	
Non-magnetic zinc	41·0	39·91	4·36	8·3	0·04	4·76	91·2
Magnetic iron	59·0	2·69	0·87	54·5	0·05	4·10	8·8

Conclusions.—Several adjustments were made to the magnetic separator used, but very little difference in either grade or recovery was obtained. The zinc product averaged about 39 per cent with a recovery of 93 per cent. An exceptionally good elimination of the iron was obtained. The reason why a higher grade zinc product was not obtained was because the gangue minerals in the feed reported in the non-magnetic product. These could probably be eliminated by air jigs or tables, and the grade of the zinc product increased.

Flotation Tests.—Only small-scale batch flotation tests were made on the middlings. The flotation of the middling product was given careful consideration, particularly the dump material. To float a zinc sulphide from such a product involves a difficult problem in selective flotation. A zinc concentrate was required which would contain 42 to 45 per cent zinc, the tailing consisting almost entirely of iron sulphide.

General Procedure.—The middlings were ground in a small ball mill to 65 mesh. The density of the pulp in the mill was 1 : 1. A charge was prepared for each test. Some of the reagents were added to the ball mill and others directly to the flotation cell.

Results of Flotation Tests.—Very promising results were obtained. The following is typical of a number of tests made:—

Shipment No. 1—Old Dump Middling

Product	Weight per cent	Analysis					Per cent of zinc values
		Zn %	Pb %	Fe %	Au. oz./ton	Ag. oz./ton	
Concentrate	36·6	42·08	4·69	13·23	0·16	9·90	90·4
Middling	7·5	11·60	1·75	45·85	0·02	4·20	5·1
Tailing	55·9	1·25	trace	52·92	0·03	1·65	4·5

Shipment No. 3—Freshly Produced Middling

Product	Weight per cent	Analysis					Per cent of zinc values
		Zn %	Pb %	Fe %	Au oz./ton	Ag. oz./ton	
Concentrate	34·1	42·15	....	....	0·10	12·74	92·1
Middling	6·0	7·99	....	....	0·12	11·52	3·1
Tailing	59·9	1·25	....	....	0·02	4·48	4·8

Conclusions.—Although the results of these tests could very probably be duplicated in actual practice on the freshly produced middling, difficulty might be encountered in the case of the old dump material. Small batch tests have been found to give in some cases unreliable results on such material. It would be necessary to run large-scale tonnage check tests on a truly representative shipment of the dump material before a definite opinion could be given on this method of treatment.

EXPERIMENTAL TESTS ON SHIPMENT NO. 2

Flotation Test.—Only one test was made on the ore of this shipment which represents the crude ore. The results are given in the following table:—

Product	Weight per cent	Analysis			Per cent of zinc values
		Zn %	Au. oz./ton	Ag. oz./ton	
Concentrate	14·8	41·3	0·12	16·46	81·6
Middling	9·2	12·5	0·12	10·60	15·4
Tailing	76·0	0·3	0·04	0·66	3·0

Conclusions.—A high-grade zinc concentrate can be obtained from the crude ore by flotation and a good recovery made without any apparent difficulty.

SMALL-SCALE EXPERIMENTAL TESTS ON SHIPMENT NO. 4

This shipment consisted of a carload of crude ore from the mine. Small-scale selective flotation tests were run to determine whether it was possible to make a separation between the lead and zinc minerals and to produce a marketable concentrate of each.

General Procedure.—The ore was ground in a ball mill to 65 mesh. The density of the pulp in the mill was 1 : 1. For the flotation of the lead some of the reagents were added to the ball mill; others directly to the flotation cell. The lead tailing was dewatered, the pulp density raised to 1 : 3 by the addition of fresh water, and the reagents for the flotation of the zinc added to the flotation cell.

Results of Selective Flotation Tests.—The results obtained are given in the following table:—

Product	Weight per cent	Analysis				Per cent of values	
		Zn %	Pb %	Au oz/ton	Ag oz/ton	Zn	Pb
Lead concentrate	8·6	7·52	24·42	0·32	32·48	12·9	93·5
Zinc concentrate	9·7	40·45	0·51	trace	1·38	78·4	2·2
Middling	5·5	4·43	0·41	0·02	1·44	4·9	1·0
Lead concentrate	4·8	4·12	25·44	0·28	34·86	4·1	55·5
Zinc concentrate	10·5	40·87	7·50	0·04	9·08	86·0	35·8
Zinc middling	6·5	4·92	1·18	0·22	5·12	6·4	3·5
Tailing	78·2	0·21	0·15	trace	0·24	3·4	5·3
Lead concentrate	7·7	8·49	34·47	....	....	10·9	90·5
Zinc concentrate	15·1	34·94	1·13	....	....	83·5	5·8
Tailing	77·2	0·47	0·14	....	....	5·7	3·7

Conclusions.—The results of these tests were very satisfactory, showing the possibility of selective flotation of the ore with the production of marketable lead and zinc concentrates. In conducting the tests, the lead concentrate was not recleaned, and in the last test given above, the zinc concentrate was not recleaned.

Tonnage Check Tests on Shipment No. 4

Nine tonnage check tests were run on the ore of shipment No. 4 in order to obtain more reliable data on the treatment of the ore. The procedure and results of the tests are summarized and given below:

Test No. 1

General Description of Test.—The ore was crushed to ½ inch in a jaw crusher and set of rolls. The ½-inch material was fed to a 4½-foot Hardinge ball mill, first passing through a Vezin sampler where $\frac{1}{20}$ was cut out for a head sample. The ball mill was operated in closed circuit with a 15-inch standard Dorr classifier. The classifier overflow which would pass 48 mesh went direct to two 6-foot by 12-inch Callow flat-bottom flotation cells where a rougher concentrate and final tailing were made. The rougher concentrate was recleaned twice in two 3-foot by 12-inch cleaners. The cleaner tailing was returned to the head of the roughers by an air lift. The heavy tar oil was fed to the ball mill, but the soluble oils and pine oil were fed at the classifier overflow. The soda ash and lime were added to the ball mill. The final concentrate was passed over a Wilfley table, using a standard deck.

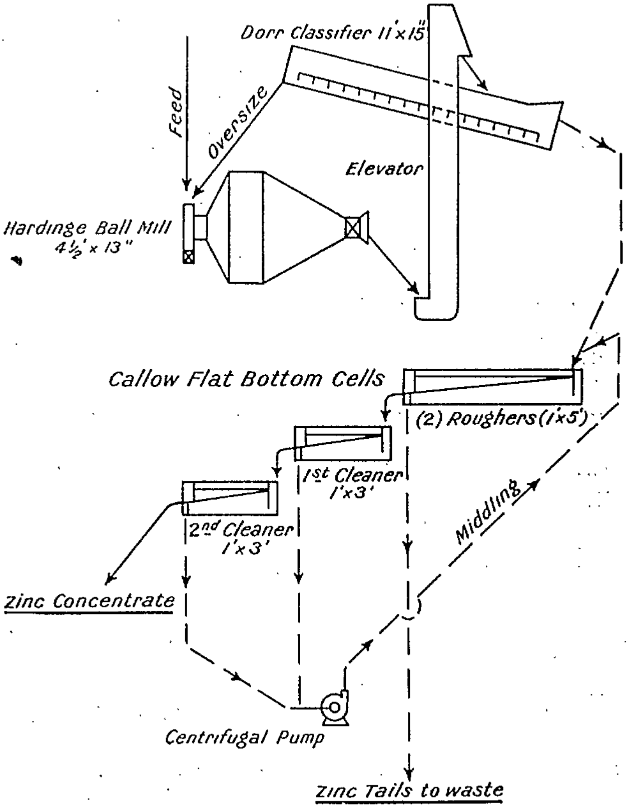

Figure 1. Flow-sheet, Shipment No. 4, Test No. 1.

Rate of feed per hour, 1,050 pounds.

Oils used were Barrett Co's. water-gas tar, General Engineering Co.'s TT mixture, and steam-distilled pine oil from Hercules Powder Co.

Analyses of Samples

Product	Zinc %	Lead %	Iron %	Gold oz./ton	Silver oz./ton
Head sample (Vezin sampler)	5·28	2·13	6·93	0·08	3·90
Classifier overflow (feed to cells)	5·78	1·86	4·68	0·03	2·60
Concentrate sample No. 1	30·88	7·61	19·31	0·08	13·68
Tailing, sample No. 1	0·78	0·28	3·05	0·02	0·38
Concentrate, sample No. 2	32·29	11·32	15·26	0·03	13·17
Tailing, sample No. 2	0·58	0·28	4·06	0·02	0·80
Table concentrate (lead)	2·25	68·78	....	0·36	62·04
Table tailing (zinc)	32·48	5·88	....	0·03	11·31

Recapitulation of Results Using Classifier Overflow as Head Sample

Zinc—	Average grade of concentrate	32·48	per cent
	Average grade of tailing	0·68	"
	Total recovery of zinc in feed	90·30	"
Lead—	Grade of table concentrate	68·78	"
	Total recovery of lead in feed	44·9	"
	Recovery of lead by tabling flotation concentrate	51·0	"

Summary of Distribution of Lead in Products

Lead lost in zinc product	43·1	per cent
Lead lost in flotation tailing	12·0	"
Lead recovered in table concentrate	44·9	"

Test No. 2

General Description of Test.—The ore was crushed to ½-inch and a head sample cut out by a Vezin sampler. The ore after passing through the sampler was fed to a 4½-foot Hardinge ball mill in closed circuit with a Dorr classifier. The classifier overflow which would pass 48 mesh was passed over a Wilfley table where a lead concentrate was made. The table tailing carrying the zinc was dewatered in an 8-foot Callow cone, and the thickened feed went to two 6-foot by 12-inch Callow flat-bottom cells, each cell receiving half the feed. As in the previous test the final tailing was produced from these two rougher cells and the rougher concentrate was recleaned twice in two 3-foot by 12-inch cleaner cells, the cleaner tailing being returned to the rougher cells. The final flotation zinc concentrate was tabled on a Wilfley table to remove a second lead concentrate.

Rate of feed per hour, 1,200 pounds.

Oils used were Barrett's No. 634, TT mixture and YZ mixture.

Steam-distilled pine oil.

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %	Gold oz. per ton	Silver oz. per ton
Whole........	Head sample (Vezin sampler).........	5·78	2·68	....	0·05	3·97
	Classifier overflow (table feed)........	5·88	2·15	....	0·02	3·78
	Table concentrate, lead................	1·93	71·41	6·5	0·70	75·60
10.30– 1.00	Thickened feed to cells, sample No. 1.	5·48	0·72	....	0·01	1·99
10.30–12.00	Flotation concentrate, sample No. 1...	25·68	2·58	....	trace	6·40
10.30–12.00	Flotation tailing, sample No. 1........	0·46	0·21	....	trace	0·54
12.00– 1.05	Flotation concentrate, sample No. 2...	45·55	4·78	....	0·10	10·90
12.00– 1.05	Flotation tailing, sample No. 2........	0·81	0·21	....	trace	0·60
1.00– 3.00	Thickened feed to cells, sample No. 2.	6·08	1·23	....	0·01	2·59
1·05– 1.30	Flotation concentrate, sample No. 3...	38·95	4·97	....	0·10	11·60
1.05– 1.30	Flotation tailing, sample No. 3........	0·25	0·31	....	trace	0·40
1.30– 3.00	Flotation concentrate, sample No. 4...	36·18	4·92	12·4	0·10	10·70
1.30– 3.00	Flotation tailing, sample No. 4........	0·35	0·31	....	trace	0·35
Whole........	Flotation concentrate................	35·80	6·20	....	0·10	11·60
"	Table concentrate, lead, No. 2........	18·19	25·86	....	0·40	41·60
"	Table tailing, final zinc concentrate....	37·89	3·49	....	0·04	6·56
	Average of lead, table concentrate....	7·78	55·0	....	....	....

Recapitulation of Results using Classifier Overflow as Head Sample

Average grade of lead concentrate..................................	55·0	per cent
Total recovery of lead values..	72·73	"
Recovery of lead values from flotation concentrate..................	50·4	"
Average grade of zinc concentrate..................................	37·89	"
Average recovery of zinc values......................................	93·75	"

Summary of Distribution of Lead Values in Products

Lead recovered by first tabling......................................	55·6	per cent
Lead recovered by tabling flotation concentrate....................	17·13	"
Lead values lost in flotation tailing..............................	10·3	"
Lead values lost in zinc concentrate................................	16·97	"

Test No. 3

General Description of Test.—The same procedure was followed as in test No. 2.

Rate of feed per hour, 1,500 pounds.

Reagents Used.—A fuel oil manufactured by the G.N.W. Oil Co., Cleveland, Ohio, was tried, but found to be of no use, as both a dirty concentrate carrying iron and a high tailing were made. During the operations it was noticed that considerable gangue was floating with the zinc. An analysis was made on concentrate sample No. 4 to determine whether it was the iron or the gangue that was reducing the grade of the concentrate.

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %	Gold oz.	Silver ton
Whole........	Head sample (Vezin sampler).........	5·86	2·57	....	0·05	3·61
"	Classifier overflow..................	5·56	2·46	....	0·02	3·68
"	Table concentrate (lead).............	3·29	63·62	....	0·60	66·9
"	Table tailing.........................	6·28	1·23	....	trace	1·78
10.35–11.40	Flotation concentrate (zinc), sample No. 1..............................	40·58	7·50	....	0·10	12·9
10.35–11.40	Flotation tailing, sample No. 1........	1·30	0·41	....	trace	0·74
11.40–12.40	Flotation concentrate (zinc), sample No. 2..............................	42·78	7·39	....	0·10	14·20
11.40–12.40	Flotation tailing, sample No. 2........	0·82	0·36	....	trace	0·62

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %	Gold oz. per ton	Silver oz. per ton
12.40– 1.40	Flotation concentrate (zinc), sample No. 3	27·95	7·71	....	0·10	13·8
12.40– 1.40	Flotation tailing, sample No. 3	2·63	0·31	....	trace	0·76
1.40– 2.10	Flotation concentrate (zinc), sample No. 4	41·97	5·85	....	0·10	11·9
1.40– 2.10	Flotation tailing, sample No. 4	2·11	0·51	....	trace	0·92
2.30–3.45	Flotation concentrate (zinc), sample No. 5	42·80	7·70	....	0·10	15·8
2.30–3.45	Flotation tailing, sample No. 5	2·26	0·62	....	trace	1·42
Whole	Combined flotation concentrate	38·75	8·72	9·6	0·10	15·48
	Table concentrate No. 2 lead	8·02	62·79	....	0·55	68·55
	Table tailing, final zinc concentrate	41·36	4·92	....	0·08	10·46

Recapitulation of Results using Classifier Overflow as Head Sample

Average grade of lead concentrate	63·4 per cent
Total recovery of lead values	66·7 "

The recoveries of the zinc values varied according to the changes made in the use and regulation of the flotation reagents. Recovery based on sample No. 2 obtained with 0·20 pound per ton No. 634 oil and 0·15 pound YZ mixture:—

Zinc concentrate	42·78 per cent
Recovery of zinc	88·6 "

Summary of Distribution of Lead Values in Products

Recovery of lead in first table concentrate	51·0 per cent
Recovery of lead in second table concentrate	15·7 "
Loss of lead in zinc concentrate	17·5 "
Loss of lead in tailing	15·8 "

Test No. 4

General Description of Test.—The ore was crushed to 1 inch and a head sample cut by a Vezin sampler. The ore after passing through the sampler was fed to a 4½-foot Hardinge ball mill in closed circuit with a Dorr classifier. The classifier overflow which would pass 48 mesh was passed over a Wilfley table where a lead concentrate was made. The table tailing was thickened in a Callow cone and the thickened feed went to two 6-foot by 12-inch Callow flat-bottom cells. A final tailing and a rougher concentrate were made. The rougher concentrate was recleaned in two 3-foot by 12-inch cleaner cells, the cleaner tailing being returned with the feed to the rougher cells. This zinc flotation concentrate was tabled to remove as much lead as possible. Rate of feed per hour, 1,000 pounds. Reagents used, same as in Test No. 2.

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %	Insol. %	Gold oz. per ton	Silver oz. per ton
Whole........	Head sample (Vezin sampler).	5·98	2·36	6·46	....	0·07	3·53
“	Head sample (Classifier overflow)........................	5·54	2·77	6·77	....	0·05	3·41
“	Table concentrate No. 1, lead	2·27	58·5	13·74	....	0·90	68·3
“	Table tailing after thickening..	5·68	1·85	7·37	....	0·03	2·55
11.50–1.20	Flotation concentrate, sample No. 1.....................	44·03	7·28	9·09	1·51	0·10	10·8
11.50–1.20	Flotation tailing, sample No. 1	1·19	0·36	....	....	....	....
1.20–3.30	Flotation concentrate, sample No. 2.....................	43·9	7·39	8·89	2·76	0·10	13·98
1.20–3.30	Flotation tailing, sample No. 2	1·76	0·41	....	....	....	....
3.30–4.00	Flotation concentrate, sample No. 3.....................	42·25	10·18	8·80	4·90	0·10	12·46
3.30–4.00	Flotation tailing, sample No. 3	0·95	0·40	6·4	....	trace	0·58
4.00–4.45	Flotation concentrate, sample No. 4.....................	43·0	17·25	8·30	4·7	0·12	14·12
4.00–4.45	Flotation tailing, sample No. 4	1·08	0·30	6·4	....	trace	0·08
Whole........	Flotation concentrate.........	44·0	9·23	....	....	....	....
“	Table concentrate No. 2, lead.	9·26	63·8	4·85	....	0·90	76·6
“	Table tailing, final zinc concentrate..................	44·13	7·28	9·4	1·51	0·12	12·36

Recapitulation of Results using Classifier Overflow as Head Sample

Lead—Average grade of concentrate.............................. 59.1 per cent
Zinc—The recovery varied during the test:—
Sample No. 1 Recovery of zinc.......................... 80·7 per cent
“ “ 2 “ “ .... 71·4 “ “
“ “ 3 “ “ .... 85·2 “ “
“ “ 4 “ “ .... 82·6 “ “

The lead in the flotation concentrate from this test was so fine that a poor recovery was made on the table.

Test No. 5

General Description of Test.—The procedure followed was similar to that of the preceding tests. The zinc concentrate was not tabled for the removal of the lead as sufficient data had already been collected on this operation. In figuring the recovery of the lead, a recovery of 50 per cent of the lead in the flotation concentrate was assumed as this figure corresponds with the results obtained in the preceding tests. Rate of feed per hour, 1,200 pounds. Oils used same as in former tests.

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %
Whole..........	Head sample (Vezin sampler)..............................	5·68	3·18	6·72
“	Head sample (classifier overflow)..............................	5·17	2·05	6·00
“	Table concentrate No. 1, lead..............................	2·69	59·84	11·00
“	Table tailing (feed to cells)..............................	4·81	0·92	5·30
11.30–12.45	Flotation concentrate, sample No. 1..............................	43·53	6·67	9·39
11.30–12.45	Flotation tailing, sample No. 1..............................	0·42		
	Mill shut down 12.45–3.20.			
3.20– 3.45	Flotation concentrate, sample No. 2..............................	42·39	6·06	10·86
3.20– 3.45	Flotation tailing, sample No. 2..............................	0·23		
3.45– 4.10	Flotation concentrate, sample No. 3..............................	45·51	5·54	9·29
3.45– 4.10	Flotation tailing, sample No. 3..............................	0·36		
4.10– 4.40	Flotation concentrate, sample No. 4..............................	42.81	6·98	9·24
4.10– 4.40	Flotation tailing, sample No. 4..............................	0·47		
Whole..........	Average flotation tailing (calculated)..............................	0·37		
“	Final flotation concentrate..............................	42·39	6·78	10·00

Recapitulation of Results using Classifier Overflow as Head Sample

Lead:	Average grade of concentrate	59·84 per cent
	Average recovery assuming 50 per cent lead in flotation concentrate recovered by tabling	78·0 "
Zinc:	Average grade of zinc concentrate (not tabled to remove lead)	42·4 "
	Average recovery for whole test	93·6 "

Test No. 6

General Description of Test.—This test was a selective flotation test. The ore was crushed as in the previous tests and the classifier overflow which would pass 35 mesh was fed to a four-cell Ruth flotation machine where the lead was floated. The tailing from the Ruth went direct to the Callow cells where the zinc was floated. Rate of feed per hour, 1,200 pounds.

Reagents for flotation of lead: sodium cyanide, oil mixture, cresylic acid.

Reagents for flotation of zinc: water-gas tar, copper sulphate, No. 634 oil, and pine oil No. 5.

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %	Insoluble %
Whole	Head sample (Vezin sampler)	5·65	2·87	6·87	
"	Classifier overflow, head sample	6·10	2·36	6·46	
"	Lead concentrate (Ruth machine)	4·65	35·51	6·72	28·3
"	Lead tailing—feed to zinc flotation	5·26	0·21	6·00	
1.15–2.15	Zinc flotation concentrate, sample No. 1	50·66	0·61	10·91	
1.15–2.15	Zinc flotation tailing, sample No. 1	1·96	0·10		
2.15–3.00	Zinc flotation concentrate, sample No. 2	51·18	0·72		
2.15–3.00	Zinc flotation tailing, sample No. 2	7·64	0·11		
3.00–3.45	Zinc flotation concentrate, sample No. 3	50·05	0·92		
3.00–3.45	Zinc flotation tailing, sample No. 3	2·53	0·13		
3.45–4.20	Zinc flotation concentrate, sample No. 4	48·02	0·92		
3.45–4.20	Zinc flotation tailing, sample No. 4	1·24	0·11		
Whole	Final zinc concentrate	50·04	0·72		
"	Final zinc tailing, average	1·76			

Recapitulation of Results using Classifier Overflow as Head Sample

Lead:	Average grade of concentrate	35·51 per cent
	Average recovery	90·0 "
Zinc:	Average grade of concentrate	50·09 "
	Average recovery	73·8 "

Note.—At times during test, the recovery was over 85 per cent.

Test No. 8

General Description of Test.—This test, together with test No. 9, was run to demonstrate the two methods of concentration for Messrs. Tetreault and their engineer, who were present by appointment. The procedure followed in this test is similar to that used in tests Nos. 2 to 5. Rate of feed per hour, 1,000 pounds.

Analyses of Samples

Time	Product	Zinc %	Lead %	Iron %
Whole	Head sample (Vezin sampler)	4·92	1·96	7·02
"	Classifier overflow (feed to table)	4·87	1·62	
"	Table concentrate (lead)	10·85	68·26	
"	Table tailing (feed to cells)	4·42	0·45	
12.20-2.30	Flotation concentrate, sample No. 1	38·65	5·12	
	Flotation tailing, sample No. 1	0·48	0·22	
Whole	Flotation concentrate	37·65	4·76	

Recapitulation of Results using Classifier Overflow as Head Sample

(Assuming that by re-tabling the zinc concentrate 50 per cent of the lead in it would be recovered)

Lead:	Average grade of concentrate	68·26	per cent
	Average recovery of lead	78·80	"
Zinc:	Average grade of concentrate	37·65	"
	Average recovery of zinc	91·4	"

Test No. 9

General Description of Test.—This was a selective flotation test. The object was to produce a high-grade lead and zinc concentrate. The classifier overflow which would pass 48 mesh was fed directly to a four-cell Ruth flotation machine where a lead concentrate was made in one operation, no cleaner cell being used. The concentrates from the last two cells were returned to the feed end of the machine, a finished concentrate being taken from the first two cells. The tailing was pumped to Callow cells consisting of two rougher cells operated in parallel, and two cleaners in parallel. The middling product from the two cleaners was returned to the feed end of the rougher cells. Rate of feed per hour, 1,400 pounds.

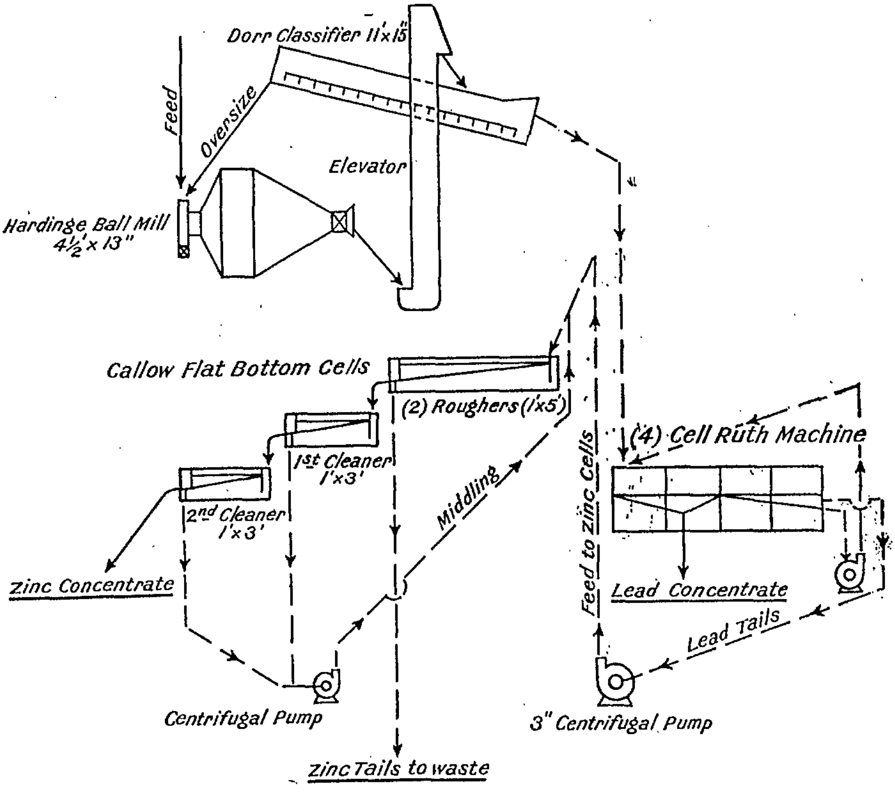

Figure 2. Flow-sheet, shipment No. 4, Test No. 9.

Lead flotation reagents: soda ash, sodium cyanide, cresylic acid and oil mixture.

Zinc flotation reagents: copper sulphate, No. 634 oil, TT mixture, and pine oil No. 5.

Analyses of Products

Product	Zinc %	Lead %	Iron %	Insol. %	Gold oz/ton	Silver oz/ton
Classifier overflow (head sample)	5·52	1·96	7·42	....	0·02	3·28
Lead concentrate, sample No. 1	4·17	51·09	2·60	16·13	0·30	
Lead tailing, sample No. 1	5·12	0·22	6·16	49·04	tr.	0· 7
Zinc concentrate, sample No. 1	41·46	2·72	....	....	....	....
Zinc tailing, sample No. 1	0·15	0·22	9·39	....	....	....
Zinc concentrate, sample No. 2	47·18	1·15	....	....	....	....
Zinc tailing, sample No. 2	0·53	0·17	....	....	....	....
Zinc concentrate, total sample	46·19	1·09	....	....	....	....
Zinc tailing, total sample	0·30	0·15	....	....	....	....
Special sample first cleaner, zinc concentrate	20·86	1·29	11·92	11·30	....	....
Special sample of table concentrate by tabling final flotation tailing	....	3·64	....	....	0·06	6·44

Recapitulation of Results using Classifier Overflow as Head Sample

Average grade of lead concentrate	51·09 per cent
Average recovery of lead	89·4 "
Average grade of zinc concentrate	46·17 "
Average recovery of zinc	94·8 "

GENERAL SUMMARY AND CONCLUSIONS

Two methods of concentration are suggested for the treatment of this ore, namely tabling to recover the lead followed by flotation of the table tailing to recover the zinc and re-tabling the flotation zinc concentrate to recover additional lead. The alternative method is selective flotation, whereby a slightly lower grade lead concentrate is obtained, but with a much higher recovery than by the first method, and a higher grade zinc concentrate, lower in lead, with approximately the same recovery as by the first method.

By the first method, provided that a system of graded crushing, screening, and classification is used, and a classified product fed to the tables, and the flotation concentrate is re-tabled, 80 per cent of the lead values should be recovered in a high-grade lead concentrate containing 65 to 70 per cent lead. The gold and silver values in the ore are practically all recovered and chiefly report in the lead concentrate. The zinc concentrate consistently contains only 0·10 ounce of gold, and approximately 12·0 ounces of silver per ton. The zinc recovery is high, and 90 per cent can be expected with a grade of concentrate assaying from 43 to 45 per cent zinc. The chief difficulty in obtaining a high-grade zinc concentrate is due to the tendency of the chloritic gangue mineral to float. This may not be so marked in a larger flotation unit.

By the second method of selective flotation, a 50 per cent lead concentrate can be obtained with a 90 per cent recovery of the lead values, and containing, from figures obtained in small-scale flotation test No. 1, 95 per cent of the gold and 87 per cent of the silver. A 50 per cent zinc concentrate can be obtained, having a lead content of less than 1·5 per cent and with a recovery of better than 90 per cent of the zinc values.

Report No. 205

EXPERIMENTAL TESTS ON THE USE OF BENTONITE IN THE MANUFACTURE OF WATERPROOFING COMPOUND FOR PAPER BOARD

W. B. Timm

Shipment.—Several small lots of asphalt, china clay, bentonite, and resin, were received March 5, 1924 from Mansons Limited, Hawkesbury, Ont.

Purpose of Experimental Tests.—The experimental tests on the above material were conducted to determine whether an asphaltic waterproofing compound could be obtained by grinding a mixture of these substances in the cold, suitable for incorporating into felt, paper, or other materials.

Preliminary Preparation of Material.—The asphalt was first crushed to ½ inch in a small jaw crusher and the bentonite ground to 100 mesh in a Raymond pulverizer. Bentonite from two sources was used separately, for comparison, one lot being from Rosedale, Alberta, and the other from Medicine Bow, Wyoming; the latter was supplied by the Owyhee Chemical Products Co., Chicago, being the product marketed as "Wilkinite." No material difference was perceptible in the products made from the two bentonites. Extremely fine preliminary grinding of the bentonite is not necessary, nor would it be necessary to crush the asphalt to ½ inch for large-scale grinding operations.

EXPERIMENTAL TESTS

The test work consisted of dry and wet grinding of the asphalt-bentonite, asphalt-china clay, and resin-bentonite mixtures, using various grinding devices.

Test No. 1

Mixture: 80 per cent petroleum asphalt
20 per cent bentonite

Ground in a 12 by 12-inch ball mill for 2½ hours with twice the weight of water. Charge consisted of 4 pounds asphalt, 1 pound bentonite, and 10 pounds water.

Result: A fine homogeneous, gelatinous mass which when washed on a 200-mesh screen gave 20 per cent coarser than 200 mesh.

Test No. 2

Mixture: 85 per cent petroleum asphalt
15 per cent bentonite

Ground in a 12 by 12-inch ball mill for 2½ hours with 1½ times the weight of water. Charge consisted of 4¼ pounds asphalt, ¾ pound bentonite, and 7½ pounds water.

Result: Product not so fine or gelatinous as in test No. 1. When washed on a 200-mesh screen gave 37·5 per cent coarser than 200 mesh.

Test No. 3

Mixture: 90 per cent petroleum asphalt
10 per cent bentonite

Ground in a 12 by 12-inch ball mill for 2½ hours with an equal weight of water. Charge consisted of 4½ pounds asphalt, ½ pound bentonite, and 5·0 pounds water.

Result: Product coarser and less gelatinous than in test No. 2. When washed on a 200-mesh screen gave 52·5 per cent coarser than 200 mesh.

Test No. 4

Comparative wet-grinding tests were also conducted on asphalt-bentonite and asphalt-china clay mixtures.

Mixture A: 80 per cent asphalt
20 per cent bentonite
Twice amount of water, by weight
Mixture B: 80 per cent asphalt
20 per cent china clay
Equal amount of water, by weight

Ground in a 12 by 12-inch ball mill for 2½ hours. Samples were taken at the end of ½, 1, 1½, 2, and 2½ hours. Examination of the samples, after standing for some time, showed that the bentonite mixture was in each case much finer, more homogeneous, and a superior product in every way, to the china clay mixture. The bentonite mixture after 2½ hours grinding was an extremely fine product, whereas the china clay mixture was much coarser, not so homogeneous, and the china clay settled out to some extent on standing.

Test No. 5

Dry grinding tests on asphalt-bentonite mixtures were made in two different types of grinding machines. In the 12 by 12-inch ball mill the asphalt softened almost immediately and balled up, forming a sticky mass, and practically no grinding took place. In the laboratory type of Raymond pulverizer careful adjustment of feed and discharge resulted in a fairly fine product, but this was not nearly so fine or uniform as that made by wet grinding. The higher temperature generated in the ball mill probably accounted in a large measure for the "balling" of the asphalt.

Test No. 6

A wet-grinding test was also made on a mixture of resin and bentonite:
Mixture: 80 per cent resin
20 per cent bentonite
Twice amount of water by weight

Ground in a 12 by 12-inch ball mill for ½ hour. The resultant product was a fine homogeneous paste from which no water separated on standing.

SUMMARY AND CONCLUSIONS

The above tests show that the addition of 20 per cent bentonite in the wet grinding of asphalt assists materially in the securing of a fine, uniform product. The paste formed with the amount of water added in test No. 1 was exceedingly smooth and homogeneous, and no separation of water took place on standing. More prolonged grinding, owing to the effecting of more perfect emulsification, might permit of the addition of even a larger amount of water for a paste of the same consistency.

Report No. 206

CONCENTRATION OF A LEAD-ZINC ORE FROM RIONDEL, B.C.

R. K. Carnochan

Shipments.—Two shipments of a lead-zinc ore were received from the Shepherd Mining Co., Riondel, B.C. Shipment No. 1 of 5 pounds was received November 10, 1923. Shipment No. 2 of 172 pounds was received December 12, 1923.

Purpose of Experimental Tests.—Tests were desired to determine a suitable method of recovering the lead and zinc values in marketable products.

Characteristics of the Ore.—The chief mineral constituents of the ore are galena, zinc blende, arsenopyrite, pyrite, calcite rhodochrosite (manganese carbonate) and quartz.

Sampling and Analysis.—Lot No. 1 was crushed to 20 mesh, a sample for analysis was cut out and ground to 100 mesh. Lot No. 2 was crushed to ½ inch, a sample was cut out and crushed to 20 mesh. A sample for analysis was obtained from the 20-mesh sample and ground to 100 mesh. The analyses of the samples were as follows:—

	Lot No. 1	Lot No. 2
Lead	5·38 per cent	3·03 per cent
Zinc	8·58 "	4·31 "
Manganese	13·70 "	3·57 "
Arsenic	0·62 "	0·58 "
Gold	0·04 ounce per ton	0·01 ounce per ton
Silver	6·69 "	5·58 "

EXPERIMENTAL TESTS

A number of tests were conducted on shipment No. 2. These consisted of: first, tabling to remove as much lead as possible in a high-grade concentrate followed by flotation; second, selective flotation of the lead and zinc values. Although a high-grade lead product was obtained by tabling, using the first method, the recoveries were low and a satisfactory zinc product was not obtained. The second method gave the better results. A description of a few of the tests with the results obtained are given below:

Test No. 21

Procedure.—A 1,000-gramme sample was ground wet in a small ball mill for 30 minutes with 2·5 grammes of soda ash and 0·1 c.c. of a mixture of 40 per cent coal tar and 60 per cent coal-tar creosote. The ground charge was then transferred to a small Ruth flotation machine and the lead floated by the use of 0·05 gramme sodium cyanide and 0·1 c.c. cresylic acid. The lead concentrate was re-run with 0·1 c.c. cresylic acid to clean it. The lead tailings were put back into the Ruth machine after decanting off the excess water, and the zinc floated by using 0·5 gramme of copper sulphate, 1 gramme soda ash, and 0·15 c.c. Barrett's No. 634, 0·1 c.c. TT mixture and 0·05 c.c. Aldol. The zinc concentrate was re-run to clean it.

Results.—The results of the test are given in the following table:—

Product	Weight per cent	Analysis		Per cent of values	
		Lead per cent	Zinc per cent	Lead	Zinc
Lead concentrate	5·60	44·90	9·30	69·2	10·3
Lead middling	7·10	12·23	11·70	23·9	16·3
Zinc concentrate	7·96	1·00	43·20	2·2	67·7
Zinc middling	5·55	0·89	1·90	1·3	2·1
Tailing	72·85	0·17	0·25	3·4	3·6

A screen test on the tailing gave:—

−65 + 100	2·9 per cent
−100 + 150	5·5 "
−150 + 200	12·2 "
−200	79·4 "

Test No. 22

Procedure.—This test is similar to test No. 21, except that different flotation reagents were used.

Reagents for flotation of lead—		
Added to ball mill	Soda ash	1 gramme
	Water-gas tar	0·1 c.c.
	Coal-tar creosote	0·1 c.c.
Added to flotation cell	Sodium cyanide	0·05 gramme
	Cresylic acid	0·1 c.c.
To clean lead concentrate	Cresylic acid	0·05 c.c.
Reagents used for flotation of zinc:		
Added to flotation cell	Copper sulphate	0·5 gramme
	Water-gas tar	0·1 c.c.
	TT mixture	0·15 c.c.
To clean zinc concentrate	Cresylic acid	0·05 c.c.
	Aldol	0·05 c.c.

Results.—The results of the test are given in the following table:—

Product	Weight per cent	Analysis		Per cent of values	
		Lead per cent	Zinc per cent	Lead	Zinc
Lead concentrate	8·15	33·36	8·55	84·6	13·9
Lead middling	9·82	2·45	10·60	7·5	20·8
Zinc concentrate	5·12	0·78	52·30	1·3	53·5
Zinc middling	4·04	1·28	9·80	1·6	7·9
Tailing	73·14	0·22	0·27	5·0	3·9

Remarks.—The results of the above tests indicate that the minerals were not entirely freed. The grade of the lead concentrate was low and the recoveries of both lead and zinc only fair. A number of tests were made in which the ore was ground wet to pass 200 mesh.

Test No. 27

SELECTIVE FLOTATION AT 200 MESH

Procedure.—1,000 grammes of the ore was ground for 30 minutes in a small ball mill with 0·05 c.c. of a mixture of 40 per cent coal tar and 60 per cent coal-tar creosote. The ball mill charge was then screened wet on 200 mesh and the oversize returned to the mill and ground for the same time with the same amount of oil. The charge was again screened and the oversize reground as before. After the third grinding only a small amount of 200-mesh material remained on the screen. The charge was then dewatered and returned to the mill for mixing with the flotation reagents: 0·05 c.c. mixture of coal tar and coal-tar creosote and 2·5 grammes soda ash. The charge was then floated in a similar manner to test No. 21, except that no Aldol was used in making the zinc rougher concentrate, but 0·05 c.c. Aldol was used to clean it.

Results.—The results of the test are given in the following table:—

Product	Weight per cent	Analysis				Per cent of values			
		Lead per cent	Zinc per cent	Gold oz/ton	Silver oz/ton	Lead	Zinc	Gold	Silver
Lead concentrate...	4·91	59·16	7·60	0·20	77·50	80·5	7·3	74·6	55·4
Lead middling.....	6·11	7·52	7·20	0·02	17·82	12·7	8·6	9·3	15·8
Zinc concentrate...	6·42	0·96	55·70	0·01	8·35	1·7	69·8	2·4	7·8
Zinc middling.....	3·00	1·19	12·80	0·06	6·14	1·0	7·5	13·7	2·7
Zinc tailing........	77·45	0·19	0·45	0·00	1·62	4·1	6·8	0·0	18·3

CONCLUSIONS

The results of the tests show that the ore is amenable to treatment by selective flotation. The above results are only indicative of what might be expected under operating conditions on a similar grade and class of ore. A 60 per cent lead concentrate should be obtained with a recovery of 85 per cent of the lead. A 55 per cent zinc concentrate should be obtained with a recovery of 75 per cent of the zinc.

In conducting the small-scale tests, sufficient concentrate was not obtained of either product to maintain the proper conditions as to pulp density for cleaning the rougher concentrates. This condition is believed to be the reason why the excessive fine grinding was necessary to obtain the results of test No. 27. To obtain reliable results, experimental tests should be conducted on a larger scale under continuous operating conditions.

The gold and silver values seem to be associated with the lead. In test No. 27, 75 per cent of the gold values and 55 per cent of the silver values reported in the lead concentrate. The low recovery of silver was no doubt due to excessive sliming.

Report No. 207

EXPERIMENTAL TESTS ON WINDPASS ORE

J. S. Godard

Shipments.—A shipment of 381 pounds of ore was received December 18, 1923 from the Windpass Mining Company, Chu Chua, B.C.

Lot No. 1 consisted of 191 pounds, and Lot No. 2 of 190 pounds.

The lots were representative of two types of ore taken from different sections of the Windpass vein.

The ore was from the Windpass properties situated in the North Thompson River valley, B.C. Access to the property is gained by Canadian National railway from Kamloops to Chu Chua station, a distance of 50 miles, thence five miles by motor-road, and the remaining four miles by pack trail.

Characteristics of the Ore.—It is reported that two distinct types of ore are found on the property. In the eastern mineralized zone there is a replacement of the brecciated country rock by massive magnetite, pyrrhotite, and chalcopyrite. Associated with these minerals are free gold, gold telluride, native bismuth, and cobaltite. Lot No. 1 is supposed to be representative of this type.

Ore from the westerly extension contains minute amounts of chalcopyrite, pyrrhotite, and pyrite scattered throughout a quartz gangue. Associated with the above minerals are grains and blades of native bismuth, free gold, and some tellurides. Lot No. 2 is supposed to be representative of this latter type.

Purpose of Experimental Tests.—An economical metallurgical treatment for the ores was desired.

Sampling and Analysis.—To obtain a sample for analysis the following procedure was carried out on both lots. The entire lot was crushed to 8 mesh and cut once using a Jones riffle sampler. One-half thus obtained was reduced to −14 mesh and cut twice. One of these portions was crushed to −48 mesh and cut to about one pound. The sample submitted for assay was cut from this latter quantity. The assays were as follows:—

	Lot No. 1	Lot No. 2
Gold, ounces per ton	1·65	1·35
Silver, ounce per ton	0·28	....
Copper, per cent	1·10	0·45
Bismuth, per cent	0·03	0·03
Nickel and cobalt, per cent	0·03	0·02

EXPERIMENTAL TESTS

In order to avoid unnecessary repetition, the following reference to procedure is given:

All samples of ore were cut by means of a Jones riffle sampler. Screen tests were made on Tyler standard screens. All grinding previous to amalgamation was done in a small porcelain laboratory ball mill using 15 pounds grey iron balls, and maintaining a pulp density of 66·7 per cent solids. Amalgamation tests of two hours' duration were carried out in a laboratory ball mill omitting the balls, with 100 grammes mercury, and a pulp density of 50 per cent solids. Separation of the mercury from the pulp was effected by means of a hydraulic classifier. Tabling was done on a laboratory size Wilfley table. Flotation tests were made in a small Ruth type flotation machine. Cyanide tests were of the usual bottle type.

Any deviation from the above methods is mentioned in the test.

A few preliminary tests were made to obtain some idea of the behaviour of the ore. Lot No. 1 was treated first, followed by lot No. 2, which in turn was followed by tests on ore consisting of a combination of both lots in equal proportions. Some tests are omitted because of very poor results, others are included, not because of the results obtained, but to illustrate the behaviour of the ore under certain methods of treatment.

LOT NO. 1

Test No. 1—Amalgamation and Tabling of Amalgamation Tailing

Procedure.—1,000 grammes ore at −14 mesh was ground wet to 48 mesh and amalgamated. The tailing was tabled making two products, a concentrate and tailing; 45 per cent of the table tailing was used for assay purposes, and the remainder for a screen test.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Table concentrate	16·0	2·52	3·40	34·3	49·3
Table tailing (1)	37·6	0·88	0·67	28·2	22·7
Table tailing (2)	46·4	0·95	0·67	37·5	28·0

Screen Test

Mesh	Weight per cent	Au oz/ton	Per cent of values
+48	15·2	0·88	14·2
+65	12·2	0·90	10·7
+100	18·3	0·92	17·8
+150	10·1	0·82	8·7
+200	10·4	0·80	8·8
−200	33·8	1·12	39·8

Summary.—The ore is amenable to amalgamation; 28·8 per cent of the total gold was recovered by this method. The ratio of concentration by tabling was 6·5: 1 and the loss during this operation was 4·63 per cent of original feed. The table concentrate was low grade and the tailing high in both gold and copper. A considerable amount of float copper passed into the tailing.

Test No. 3.—Amalgamation and Flotation of Amalgamation Tailing

Procedure.—1,000 grammes of ore at −14 mesh was ground wet to 48 mesh and amalgamated. The tailing was reground to 100 mesh and floated. About one-half of the flotation tailing was used for a screen test.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Flotation concentrate	8·1	5·50	12·16	38·3	93·2
Flotation tailing (1)	45·9	0·76	1·01	30·0	3·3
Flotation tailing (2)	46·0	0·80	0·10	31·7	3·5

Screen Test

Mesh	Weight per cent	Au oz/ton	Per cent of values
+100	7·4	0·68	6·2
+150	15·9	0·90	17·8
+200	24·0	0·92	27·5
−200	52·7	0·74	48·5

Summary.—The recovery by amalgamation was 29·5 per cent of total gold. The ratio of concentration was 12·3: 1. The flotation concentrate containing 12·26 per cent copper with a recovery of 93 per cent as well as 38·3 per cent of the gold remaining after amalgamation, might be considered satisfactory. The flotation tailing assayed 0·10 per cent copper and 0·90 ounce gold per ton, fairly uniformly distributed as shown by the screen test.

Test No. 4.—Amalgamation followed by Cyanidation

Procedure.—1,000 grammes of the ore at −14 mesh was ground to 48 mesh and amalgamated. The amalgamation tailing was dewatered and cyanided for 24 hours using 0·10 per cent solution of potassium cyanide and lime equal to 5 pounds per ton. Pulp density was 38 per cent solids. A screen test was made on the cyanide tailing.

Screen Test

Mesh	Weight per cent	Au oz/ton	Per cent of values
+ 48	3·2	1·12	8·8
+ 65	14·9	0·54	19·8
+100	20·2	0·46	22·6
+150	14·4	0·46	16·2
+200	16·6	0·36	14·6
−200	30·7	0·24	18·0

Consumption of cyanide amounted to 2·52 pounds per ton, and that of lime was 4·9 pounds per ton.

Summary.—The total recovery amounted to 75·2 per cent of the gold, of which 29·10 per cent was due to amalgamation. The results of the cyanidation were poor, due to the gold values remaining in the coarser sizes as indicated by the screen test. These results indicated that an all-slime process of cyanidation was necessary for a high extraction of the gold values.

Test No. 6.—Amalgamation, Tabling, Flotation of Table Tailing, and Cyanidation of Flotation Tailing

Procedure.—1,000 grammes ore at −14 mesh was ground to 48 mesh and amalgamated. The amalgamation tailing was tabled and the table tailing reground to 100 mesh and floated. The flotation tailing was sampled and the remainder reground to −200 mesh and cyanided for 48 hours using 0·10 per cent potassium cyanide solution and lime equivalent to 6·2 pounds per ton, with a pulp density of 33 per cent solids.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Table concentrate	19·3	3·02	3·32	45·8	60·2
Flotation concentrate	7·5	3·56	4·88	21·0	34·4
Flotation tailing	73·2	0·58	0·08	33·2	5·4

Flotation reagents: Coal tar 40 per cent } 0·5 pound per ton
Coal-tar creosote 60 per cent }
Pine oil No. 5 2 drops

Cyanide reagents: Potassium cyanide to the extent of 2·0 pounds per ton, and lime equivalent to 5·8 pounds per ton were consumed.

Summary.—The recovery by amalgamation was 22·6 per cent, which is roughly 8 per cent below the average test; 94·6 per cent of the copper was obtained in the concentrate which also contained 66·8 per cent of the gold remaining after amalgamation. The results of the cyanidation were again unsatisfactory. The cyanide heads assayed 0·58 ounces per ton, and the tailing 0·24 ounces per ton, making a recovery of 58·6 per cent. The total recovery amounted to 85·5 per cent.

Test No. 7.—Amalgamation, Flotation, and Cyanidation

Procedure.—1,000 grammes of the ore at −14 mesh was ground to 48 mesh and amalgamated. The amalgamation tailing was ground to 100 mesh and floated. The flotation tailing was sampled and the remainder divided into three parts, each part being cyanided separately. The cyanide tailing was screened on 200 mesh.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Flotation concentrate	13·9	3·24	6·92	37·8	92·7
Flotation tailing	86·1	0·86	0·09	62·2	7·3

Flotation reagents.—Same reagents used as in test No. 6.

Details of cyanidation.—

	Part A	Part B	Part C
Potassium cyanide solution, per cent	0·10	0·10	0·10
Lime, pounds per ton	10·0	7·0	7·5
Ratio ore to water	1 : 3·6	1 : 2·5	1 : 2·7
Time, hours	24	36	48
Heads, ounce per ton gold	0·86	0·86	0·86
Tails +200 " "	0·36 } 0·23	0·34 } 0·24	0·30 } 0·20
−200 " "	0·20 }	0·20 }	0·20 }
Recovery, per cent	73·2	72·2	74·4
Cyanide consumption, pounds per ton	2·9	2·4	3·0
Lime " " "	7·7	5·5	6·1

Summary.—Amalgamation was responsible for a recovery of 27·9 per cent of the gold. The flotation concentrate was low in copper but represented 92·7 per cent of the total; 37·8 per cent of the gold remaining after amalgamation was in the flotation concentrate which assayed 3·24 ounces per ton. The cyanide tailing was high in gold especially in the +200 size. Extraction of gold from −200 size showed no increase by extending the time of agitation. A slight increase in the extraction though not proportional to the increase in time was made in the +200 mesh.

Test No. 8.—Amalgamation, Flotation, and Cyanidation

Procedure.—1,021 grammes of the ore at −14 mesh was ground to 48 mesh and amalgamated. The amalgamation tailing was reground to 65 mesh and floated. The flotation tailing was reground to 150 mesh and sampled. The remainder of the flotation tailing was divided into three parts and cyanided separately. Screen tests on 200 mesh were made.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Flotation concentrate	9·8	4·56	9·20	38·6	89·2
Flotation tailing +200	21·1	0·80	0·09	14·6	1·9
Flotation tailing −200	69·1	0·78	0·13	46·8	8·9

Flotation reagents.—Same as in test No. 6.

Details of cyanidation:—

	Part A	Part B	Part C
Weight, grammes	235·3	231·8	227·2
Ratio ore to water	1 : 3	1 : 3	1 : 3
Cyanide solution at start	0·05	0·10	0·2
Lime, pounds per ton	6·7	6·8	7·0
Sodium peroxide, pound per ton	0·85	0·86	0·88
Time, hours	24	24	24
Cyanide consumption, pounds per ton	1·5	2·3	3·7
Lime " " "	5·5	6·0	6·1

Screen Test on Cyanide Tailing

—	Weight grammes	Weight per cent	Au oz/ton	Per cent of values	Per cent recovery
Part A +200	65·5	27·8	0·30	42·0	62·5
−200	169·8	72·2	0·16	58·0	79·4
Part B +200	55·7	24·1	0·28	42·4	65·2
−200	176·1	75·9	0·12	57·6	84·6
Part C +200	55·0	24·2	0·24	43·3	70·0
−200	172·2	75·8	0·10	56·7	87·2

Summary.—30·1 per cent of the gold was recovered by amalgamation. The flotation concentrate contained 9·20 per cent copper with a recovery of 89·2 per cent and 38·6 per cent of the gold remaining after amalgamation. It assayed 4·56 ounces per ton gold. The highest recovery by cyanidation is 87·2 per cent on −200-mesh material using 0·20 per cent solution of cyanide. Test 8C showed a total gold recovery of 94·5 per cent.

Test No. 9.—Amalgamation, Flotation, and Cyanidation

Procedure.—1,025 grammes of the −14-mesh ore was ground to 65 mesh and amalgamated. The amalgamation tailing was ground to 100 mesh with suitable flotation reagents and then floated. The flotation tailing was ground to 200 mesh and a sample equal to about one-quarter of the tailing was taken for assay purposes. The remainder was divided into three parts and each part was cyanided for 48 hours. The only variable in this test was the dilution, which was 1 : 2, 1 : 3, and 1 : 4. The cyanide solution was 0·15 per cent in each case, and lime equivalent to 8 pounds per ton was used.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Flotation concentrate	9·5	4·60	9·46	40·7	88·4
Flotation tailing	90·5	0·70	0·13	59·3	11·6

Flotation reagents.—Same as in test No. 6.

Details of cyanidation:—

	Part A	Part B	Part C
Weight, grammes	229·2	232·4	232·4
Ratio, ore to water	1 : 2	1 : 3	1 : 4
Cyanide per cent solution at start	0·15	0·15	0·15
Lime, pounds per ton at start	8·7	8·7	8·7
Time, hours	48	48	48
Cyanide consumption, pounds per ton	2·8	4·4	4·5
Lime " " "	8·5	8·0	7·5

The cyanide tailing was screened on 200 mesh and the products assayed.

Screen Test

—	Weight grammes	Weight per cent	Au oz./ton	Per cent of values	Per cent recovery
Part A +200	9·4	4·1	0·25	6·3	64·2
−200	219·8	95·9	0·16	93·7	77·1
Part B +200	7·9	3·4	0·22	4·6	68·5
−200	224·5	96·6	0·16	95·4	77·1
Part C +200	8·7	3·7	0·20	5·2	71·4
−200	224·2	96·3	0·14	94·8	80·0

Summary.—Amalgamation yielded 35·1 per cent of the gold and showed an increase over that obtained at 48 mesh. Evidently finer grinding previous to amalgamation would increase the recovery by amalgamation. Little variation is shown in the flotation from that of previous tests. The cyanide tailing averaged 0·155 ounce gold per ton. Small advantage is gained in cyanidation in a more dilute pulp. The total recovery averaged 90·5 per cent.

Amalgamation Tests.—It was thought that insufficient data had been obtained on the amalgamation phase of this work, so the following tests were made.

Each test was divided into two parts, A and B. Part A consisted of grinding wet in a ball mill 1,000 grammes of ore at −14 mesh for a certain length of time, varying in each test, then screening the ground ore and assaying the products on each screen. Part B consisted of grinding the same amount of ore the same length of time as in part A and under, as far as possible, the same conditions. The ground ore in Part B was then amalgamated using 100 grammes mercury, and a pulp density of 50 per cent solids. The mercury was separated in a hydraulic classifier, and the amalgamation tailing screened and the products assayed.

The difference in assays of each screen product before and after amalgamation based in per cent will give the amalgamation at that mesh after grinding that length of time. The time in each test number was varied, and the difference between the head samples before and after amalgamation permit the calculation of the per cent of gold amalgamated at the size that grinding that length of time would produce.

Amalgamation Tests on Lot No. 1

		PART A			PART B.				
Test No.	Product	Weight per cent	Assay Au oz/ton	Per cent of values	Weight per cent	Assay Au oz/ton	Per cent of values	Per cent head sample amalgamated	Remarks
10	−28+35	1·4	1·19	1·0	1·0	1·14	0·9	4·2	Ground 15 minutes.
	−35+48	5·9	1·22	4·3	6·3	1·10	5·7	9·8	Head sample 1·68 oz./ton
	−48+65	14·8	1·32	11·6	17·7	1·18	17·3	10·6	After amalgamation 1·21 oz/ton.
	−65+100	17·2	1·40	14·3	15·6	1·18	15·3	15·6	Per cent amalgamated, 28·0.
	−100+150	12·4	1·72	12·7	11·6	1·26	12·2	26·7	
	−150+200	15·6	1·72	16·0	15·2	1·18	14·8	31·4	
	−200	32·7	2·06	40·1	32·6	1·25	33·8	39·3	
11	−48+65	3·4	1·66	3·4	7·1	0·98	5·8	40·9	Ground 25 minutes.
	−65+100	15·1	1·30	11·8	11·4	1·04	10·0	20·0	Head sample 1·65 oz/ton.
	−100+150	14·9	1·30	11·8	14·7	1·16	14·5	10·8	After amalgamation 1·18 oz/ton.
	−150+200	25·3	1·56	23·9	22·5	1·16	22·4	25·7	Per cent amalgamated, 28·5.
	−200	41·3	1·96	49·1	44·3	1·26	47·3	35·7	
12	−65+100	9·8	1·56	9·4	6·3	0·90	5·2	42·3	Ground 35 minutes.
	−100+150	11·6	1·24	8·8	10·9	0·98	9·8	20·9	Head sample 1·63 oz/ton
	−150+200	38·0	1·50	34·9	19·0	0·98	17·0	34·7	After amalgamation 1·09 oz/ton.
	−200	40·6	1·88	46·9	63·8	1·16	68·0	38·3	Per cent amalgamated, 33·1.
13	+150	2·0	4·55	5·3	2·3	0·90	2·0	80·2	Head sample 1·71 oz/ton
	−150+200	6·3	1·62	6·0	6·6	0·92	5·9	43·2	After amalgamation 1·03 oz/ton
	−200	91·7	1·66	88·7	91·1	1·04	92·1	37·3	Ground 60 minutes.
									Per cent amalgamated, 39·7.

Summary.—From the above tests, it would seem that amalgamation increases with finer grinding. At 65 mesh gold begins to accumulate on the coarse screens until it reaches 4·55 ounces per ton. At 150 mesh the best recovery, 39·7 per cent of the gold, is made.

Test No. 31.—Flotation and Cyanidation

Procedure.—A sample of 783 grammes of the ore at −14 mesh was ground to 65 mesh and floated. The flotation tailing was ground to −200 mesh in a ball mill using pebbles in place of grey iron balls. About one-half the flotation tailing was cyanided for 24 hours under the conditions as described below:—

Results of Flotation Test

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Flotation concentrate	8·0	11·84	12·80	56·8	95·7
Flotation tailing	92·0	0·78	0·05	43·2	4·3

Flotation reagents:—

Dupont flotation oil	0·50 lb/ton
Pine oil No. 5	2 drops

Details of cyanidation:—

Flotation tailing	332·2 grms.
Water, c.c	700
Cyanide, per cent solution	0·2
Sodium hydroxide, pounds per ton	5
Heads, gold, ounce per ton	0·78
Tailing, gold, ounce per ton	0·10
Recovery per cent	87·2
Cyanide consumption pounds per ton	2·8
Sodium hydroxide consumption, pounds per ton	4·0

Summary.—The flotation results from this test were very satisfactory; 56·8 per cent of the gold, and 95·7 per cent of the copper were obtained in the concentrate. Cyanidation of the reground flotation tailing was satisfactory and the cyanide consumption was moderate. The total recovery was 94 per cent.

Cyanidation Tests.—A few tests were undertaken to ascertain the possibilities of direct cyanidation, that is, omitting any other metallurgical treatment previous to cyanidation. It was thought that in such a complex ore as that of lot No. 1, soluble sulphates, water soluble cyanicides, and the presence of copper would interfere with this method, tending to increase the consumption of cyanide to such an extent as to make its application practically impossible. A test for soluble sulphates was first attempted and revealed in a soluble form 0·074 gramme sulphur per 1,000 grammes of ore, a quantity that is negligible. A water soluble cyanicide test indicated that no trouble need be expected from this source. The effect of copper in the ore on cyanide was next determined by cyaniding a sample of 1,000 grammes of ore at −200 mesh for 24 hours using 0·20 per cent solution of cyanide, and a pulp density of 33 per cent solids. The copper found in 200 c.c. of this solution amounted to 0·0582 gramme, equivalent to 5·29 per cent of that present in the ore. This would amount to roughly 1·2 pounds copper per ton of ore, and necessitates, on the basis of 2 units cyanide for each cyanide soluble unit of copper, 2·4 pounds of cyanide per ton of ore for this element alone. No remedy other than that of first removing the copper by some method such as flotation, can be suggested.

Cyanidation Tests on Ore Ground to –200 mesh

Test No.	Weight grammes	Water c.c.	KCN %	NaOH lb./ton	Assay, Au oz/ton		Re-covery %	Consumption lb/ton		Remarks
					Heads	Tailing		KCN	NaOH	
19	500	1,000	0·40	8	1·65	0·16	90·4	13·4	7·0	Ore crushed dry—lime used in place of caustic soda (NaOH). Time, 24 hours.
25	250	500	0·30	8	1·65	0·18	89·2	9·2	4·6	Ore ground wet, using pebbles. Time, 24 hours.
27	252	500	0·25	5	1·70	0·25	85·3	11·7	3·9	Ore crushed dry. After 21 hours cyanide solution was 0·012% cyanide and was raised to 0·20%. Time 24 hours.
28	247	500	0·25	5	1·70	0·20	88·3	12·8	....	Ore crushed dry. After 21 hours cyanide solution was 0·12% KCN and was raised to 0·2%. Bromo-cyanide equal to 0·1% added. Time, 24 hours
29	210	420	0·30	5	1·72	0·35	79·6	11·5	3·7	Ore crushed dry, 200 mesh. Time, 45 hours.
30	215	430	0·30	5	1·72	0·20	88·4	11·6	....	Ore crushed dry, 200 mesh. Cyanided for 45 hours. KCN then raised to 0·10%. Bromo-cyanide 0·04% added and agitation continued for 4 hours. Total time, 49 hours.

Test No. 24.—Amalgamation and Cyanidation

Procedure.—A sample of 1,000 grammes of ore at −14 mesh was ground to −48 mesh in a ball mill using pebbles in place of the usual grey iron balls, and then amalgamated with 100 grammes of mercury in a pulp density of 50 per cent solids. The mercury was separated using a hydraulic classifier, and the amalgamation tailing was ground to 200 mesh, again using pebbles. The reground amalgamation tailing was divided into four parts, one of which, No. 4, was assayed. The remaining three parts were cyanided separately, the details of which will be found tabulated below:—

	Part 1	Part 2	Part 3
Ore, grammes	227·3	229·0	227·0
Water, c.c	500	500	500
Cyanide solution, per cent	0·2	0·3	0·4
Caustic soda, pounds per ton	8	8	8
Time, hours	41	41	41

The results of the cyanidation tests are tabulated below:—

	Part 1	Part 2	Part 3
Heads, gold, ounces per ton	1·17	1·17	1·17
Tailings, gold, ounce per ton	0·33	0·28	0·2
Recovery, per cent	71·8	76·0	82·9
Cyanide consumption, pounds per ton	6·8	9·1	10·5
Caustic soda consumption, pounds per ton	4·7	3·8	4·0

Summary.—The recovery by amalgamation amounted to 29·1 per cent of the total gold. The total recoveries are:

Part 1—80 per cent. Part 2—83 per cent. Part 3—87·9 per cent.

The results of the cyanidation of the amalgamation tailings were disappointing, the tailings were high, and the recoveries were only fair. Considerable increase in extraction is found by using 0·40 per cent cyanide solution instead of 0·3 per cent. The difference in cyanide consumption in Parts 2 and 3 would be warranted by the increased extraction.

SUMMARY OF RESULTS OF EXPERIMENTAL TESTS ON LOT NO. 1

1. *Amalgamation:*—

30 per cent of the gold values were recovered by amalgamation at 48 mesh.

2. *Amalgamation and Flotation:*—

Amalgamation at 48 mesh......29·5 per cent of the gold values recovered.
Flotation at 48 mesh...........copper concentrate 8·1 per cent of weight of feed.
  Assay of concentrate........gold, 5·50 ounces per ton....copper, 12·26 per cent.
  Recoveries in concentrate...gold, 38·2 per cent.........copper, 93·2 per cent.
  Total recoveries...........gold, 56·5 per cent.........copper, 93·2 per cent.

3. *Amalgamation, Flotation, and Cyanidation:*—

Amalgamation at 48 mesh......30·1 per cent of the gold values recovered.
Flotation at 65 mesh...........copper concentrate 9·8 per cent of weight of feed.
  Assay of concentrate........gold, 4·56 ounces per ton...copper, 9·2 per cent.
  Recoveries in concentrate...gold, 38·6 per cent.........copper, 89·2 per cent.
Cyanidation at 200 mesh........Heads (flotation tailing)....gold, 0·78 ounce per ton.
  Cyanide tailing............gold, 0·10 ounce per ton.
  Recovery..................gold, 87·2 per cent.
Total recoveries................gold, 94·5 per cent..........copper, 89·2 per cent.
Reagent consumption..........cyanide, 3·7 pounds per ton..lime, 6·1 pounds per ton.

4. *Flotation and Cyanidation:—*

Flotation at 65 mesh	copper concentrate, 8 per cent of weight of feed.	
Assay of concentrate	gold, 11·84 ounces per ton	copper, 12·8 per cent.
Recoveries in concentrate	gold, 56·8 per cent	copper, 95·7 per cent.
Cyanidation at 200 mesh:—		
Heads (flotation tailing)	gold, 0·78 ounce per ton.	
Cyanide tailing	gold, 0·1 ounce per ton.	
Recovery	gold, 87·2 per cent.	
Total recoveries	gold, 94 per cent	copper, 95·7 per cent.
Reagent consumption	cyanide, 2·8 pounds per ton	caustic soda, 4 pounds per ton.

5. *Straight Cyanidation at 200 mesh:—*

Heads, gold	1·65 ounces per ton.	
Cyanide tailing, gold	0·18 ounce per ton.	
Recovery of gold values	89·2 per cent.	
Reagent consumption	cyanide, 9·2 pounds per ton	caustic soda, 4·6 pounds per ton.

CONCLUSIONS FROM RESULTS OF EXPERIMENTAL TESTS ON LOT NO. 1

1. Thirty per cent of the gold values are recovered by amalgamation. Transportation difficulties would give favour to this method as the first step in the treatment of the ore.

2. Very favourable results were obtained on the recovery of the copper values by flotation following amalgamation at 48 mesh. Only 56·5 per cent of the gold values are recovered, showing that this method must be supplemented for a further recovery of the gold in the ore. A high-grade copper concentrate is produced assaying 12·26 per cent copper and 5·50 ounces gold per ton.

3. Very favourable results were obtained by amalgamation followed by flotation and cyanidation of the flotation tailing. The flotation concentrate produced was of lower grade, but raising this is only a matter of manipulation. The final cyanide tailing assayed 0·1 ounce per ton in gold, showing a total recovery of 94·5 per cent of the gold values. The recovery of the copper values was 89·2 per cent.

4. Very favourable results were also obtained by flotation at 65 mesh, followed by cyanidation at 200 mesh. The flotation concentrate was very high grade, the gold recoverable by amalgamation reporting in the concentrate. The assay of the flotation concentrate showed 11·84 ounces gold per ton, and 12·8 per cent copper. The final cyanide tailing assayed 0·1 ounce gold per ton. The total recoveries were 94 per cent of the gold values and 95·7 per cent of the copper values. As far as the metallurgy of the ore is concerned, this method shows that amalgamation is not a necessary part of the flow-sheet. The use of this method would depend on transportation facilities to the smelters or some method of treatment for the copper concentrate at the mine. Should flotation be adopted good results can be obtained by using coal tar and coal-tar creosote mixed in the proportions of 40 and 60 per cent respectively. Pine oil No. 5 is a suitable frothing reagent as the pulp is neutral. The best flotation results are obtained on the ore ground to between 48 and 65 mesh.

5. Straight cyanidation shows a recovery of 90 per cent of the gold values. The chief objection is the high consumption of cyanide. The removal of the copper by flotation reduces the consumption from 9 pounds to 2·8 pounds per ton of ore treated. Fine grinding is essential for a good recovery of the gold values by cyanidation. Cyanidation of the amalga-

mation tailing shows no increased total recovery over straight cyanidation. The last 0·2 ounce per ton of remaining values is very difficult to extract.

6. The introduction of table concentration into the flow-sheet is not necessary. Besides complicating the flow-sheet no increased recoveries are shown.

LOT NO. 2

Amalgamation.—A series of amalgamation tests were conducted following a similar procedure to those on lot No. 1. Tabulated below are the results obtained:—

Test No. 3 is a screen analysis of the −14 mesh ore ground for 20 minutes in a small ball mill.

Test No. 4 is a screen analysis of the amalgamation tailing from a similar sample of the −14 mesh ore ground for 20 minutes and amalgamated.

Tests Nos. 5 and 6, 7 and 8, and 9 and 10, are similar tests, grinding for 30, 45, and 60 minutes respectively.

Summary of Results.—The percentage of gold values amalgamated increased with finer grinding until on ore ground to 100 mesh, 51 per cent was amalgamated. As in lot No. 1, gold began to accumulate when the ore was ground to 65 mesh until it reached 3·88 ounces per ton when ground to 100 mesh.

Amalgamation Tests on Lot No. 2

Test No.	Product	Weight %	Au oz/ton	Per cent of values	Test No.	Product	Weight %	Au oz./ton	Per cent of values	Per cent head sample amal-gamated	Remarks
3	+ 48	4·7	0·86	3·0	4	+ 48	1·9	0·80	1·9	7·0	Head sample.......... 1·35 oz./ton, gold
	− 48+ 65	14·6	1·08	11·7		− 48+ 65	11·5	0·78	11·2	27·7	After amalgamation... 0·80 oz./ton, gold
	− 65+100	19·3	1·10	15·7		− 65+100	22·9	0·80	22·8	27·3	Per cent amalgamated 40·4
	−100+150	13·3	1·22	12·0		−100+150	15·2	0·74	14·1	39·3	
	−150+200	16·5	1·40	17·2		−150+200	9·2	0·70	8·0	50·0	
	−200	31·6	1·72	40·4		−200	39·2	0·86	42·0	50·0	
5	+65	4·1	1·84	5·2	6	+65	3·2	0·70	2·7	62·0	Head sample.......... 1·45 oz./ton, gold
	− 65+100	14·5	1·34	13·4		−65+100	14·0	0·74	12·7	44·8	After amalgamation... 0·81 oz./ton, gold
	−100+150	17·6	1·26	15·3		−100+150	16·1	0·74	14·6	41·2	Per cent amalgamated 44·1
	−150+200	24·0	1·26	21·0		−150+200	29·0	0·74	26·3	41·2	
	−200	39·8	1·66	45·1		−200	37·7	0·94	43·5	43·4	
7	+100	5·3	2·94	11·1	8	+100	6·7	0·68	6·2	76·9	Head sample.......... 1·42 oz./ton, gold
	−100+150	11·1	1·20	9·4		−100+150	9·3	0·66	8·2	45·0	After amalgamation... 0·74 oz./ton, gold
	−150+200	16·8	1·08	12·7		−150+200	26·4	0·62	22·1	42·6	Per cent amalgamated 47·8
	−200	66·8	1·40	66·8		−200	57·6	0·82	63·5	41·4	
9	+100	2·1	3·88	6·0	10	+100	1·3	0·72	1·4	81·4	Head sample.......... 1·35 oz./ton, gold
	−100+150	9·5	1·54	10·8		−100+150	3·0	0·58	2·6	62·2	After amalgamation... 0·66 oz./ton, gold
	−150+200	8·4	1·02	6·3		−150+200	14·0	0·56	11·8	45·0	Per cent amalgamated 51·0
	−200	80·0	1·30	76·9		−200	81·7	0·68	84·3	47·6	

Amalgamation, Flotation, and Cyanidation

Procedure.—A sample of 1,000 grammes was ground to 65 mesh and amalgamated. The amalgamation tailing was floated. The flotation tailing was sampled, screened, and the remainder reground to −200 mesh, divided into two parts, and each part cyanided separately for 24 hours.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu %	Au	Cu
Flotation concentrate	5·1	2·76	6·40	19·6	85·2
Flotation tailing	94·9	0·61	0·06	80·4	14·8

Flotation reagents:	Water-gas tar 50 per cent Coal-tar creosote 50 per cent	0·50 pound per ton.
	Pine oil, No. 5	2 drops.

Result of screen test on 317 grammes flotation tailing:—

Mesh	Weight per cent	Assay		Per cent of values	
		Cu %	Au oz/ton	Cu	Au
−65+100	14·3	0·64	0·04	14·8	10·2
+150	22·1	0·64	0·05	23·3	18·6
+200	13·4	0·60	0·05	13·0	11·9
−200	50·2	0·60	0·07	48·9	59·3

Details of cyanidation:—	Part 1	Part 2
Ore—grms	310	304·2
Water, c.c	600	600
Potassium cyanide, per cent solution	0·10	0·20
Lime, pounds per ton	6·8	6·8
Results of cyanidation:—		
Heads, gold, ounces per ton	0·61	0·61
Tailing, gold, ounces per ton	0·06	0·06
Recovery, per cent	91·8	91·8
Cyanide consumption, pounds per ton	3·0	5·2
Lime consumption, pounds per ton	6·2	6·11

Summary.—The head sample assayed 1·35 ounces per ton and the tailing 0·06 ounce per ton, making a total recovery of 95·5 per cent of the gold values. Amalgamation yielded 46·6 per cent. The flotation concentrate is low both in grade and recovery of gold. A fair recovery of the copper values was made. Cyanidation results on this test are above the average.

Test No. 23.—Amalgamation followed by Cyanidation

Procedure.—1,000 grammes of the ore at −14 mesh was ground to 35 mesh and amalgamated. The amalgamation tailing was ground to 200 mesh, sampled, and divided into three parts, each of which was cyanided separately.

Details of cyanidation:—

	Part 1	Part 2	Part 3
Ore, grms	239·7	243·7	234·4
Water, c.c	500	500	500
Potassium cyanide, per cent solution	0·11	0·2	0·3
Caustic soda, pounds per ton	8·0	8·0	8·0
Time, hours	24	24	24
Results of cyanidation:—			
Head sample, gold, ounce per ton	0·86	0·86	0·86
Tailing, gold, ounce per ton	0·15	0·12	0·13
Recovery, per cent	82·5	86·1	84·4
Cyanide consumption, pounds per ton	3·1	2·5	3·1
Caustic soda consumption, pounds per ton	7·0	4·3	4·3

Summary.—Amalgamation yielded 36 per cent of the total gold. The total recovery averaged 90·5 per cent. Little advantage is gained by using stronger cyanide solution.

Straight Cyanidation.—Previous to attempting the direct cyanidation of ore of this lot, several tests similar to those conducted on lot No. 1 were made.

A test for soluble sulphates showed only small amounts present in the ore. A water soluble cyanicide test using 0·30 per cent solution of potassium cyanide showed that 1·2 pounds per ton cyanide would be consumed in this manner. Cyanidation in 0·20 per cent solution cyanide showed that 1·53 per cent of the copper present was soluble in a cyanide solution of this strength. This would account for the consumption of 0·28 pound per ton cyanide on the basis that one unit copper would require two units of cyanide.

Test No. 14—Cyanidation of Dry Ground Ore

Procedure.—A 500-gramme sample was crushed to 150 mesh and divided into two equal parts. One part was screened on 200 mesh, the other was cyanided for 24 hours using 0·1 per cent solution of cyanide, and lime equivalent to 8 pounds per ton.

—	Mesh	Weight %	Au oz/ton	Per cent of values	Recovery per cent
Head sample	+200	15·1	1·28	13·7	....
	−200	84·9	1·42	86·3	....
Tailing	+200	13·6	0·22	30·3	82·8
	−200	86·4	0·08	69·7	94·4

Cyanide consumption, 1·2 pounds per ton. Lime consumption, 7·3 pounds per ton.

Summary.—The average recovery was 92·8 per cent. Finer grinding is essential to the production of a low tailing.

Test No. 24.—Cyanidation of Wet Ground Ore

Procedure.—A 500-gramme sample at −14 mesh was ground in a ball mill, using pebbles, to −200 mesh, and cyanided for 24 hours; 0·1 per cent solution cyanide, and caustic soda equal to 6 pounds per ton used. Pulp density 50 per cent solids. The cyanide tailing was filtered and washed, and a sample equal to about one-half assayed. The remainder was recyanided for 24 hours using 0·1 per cent solution cyanide and caustic soda equal to 4 pounds per ton. Pulp density 25 per cent solids.

Results:—

First cyanidation:—	
Heads, gold, ounces per ton	1·35
Tailing, gold, ounce per ton	0·09
Recovery, per cent	93·3
Cyanide consumption, pounds per ton	1·3
Caustic soda consumption, pounds per ton	4·4
Second cyanidation:—	
Heads, gold, ounce per ton	0·09
Tailing, gold, ounce per ton	0·05
Recovery, per cent	44·5
Cyanide consumption, pounds per ton	0·13
Caustic soda consumption, pounds per ton	2·4
Totals:—	
Cyanide consumption, pounds per ton	1·43
Caustic soda consumption, pounds per ton	6·8

Summary.—The total recovery amounted to 96·3 per cent, of which 93·3 per cent was due to the first cyanidation. From this it hardly seems necessary to re-cyanide the tailing from the first operation.

SUMMARY OF RESULTS OF EXPERIMENTAL TESTS ON LOT NO. 2

1. *Amalgamation:*—

At 35 mesh, 36 per cent of the gold is recovered.
" 48 " 40 " " " "
" 65 " 45 " " " "
" 100 " 51 " " " "

2. *Amalgamation, Flotation, and Cyanidation:*—

Amalgamation at 65 mesh	46·6 per cent of the gold recovered.
Flotation at 65 mesh	Copper concentrate, 5·1 per cent weight of feed. Assay concentrate, gold, ounces per ton, 2·76; copper, 6·40 per cent. Recovery in concentrate, gold, 19·6 per cent; copper, 85·2 per cent.
Cyanidation at 200 mesh	Head (flotation tailing), gold, 0·61 ounce per ton. Tailing, gold, 0·06 ounce per ton. Recovery, gold, 91·8 per cent.

Total recoveries gold, 95·5 per cent; copper, 85·2 per cent.
Reagent consumption: cyanide, 3 pounds per ton; lime, 6 pounds per ton.

3. *Amalgamation and Cyanidation:*—

Amalgamation at 35 mesh	36 per cent of the gold recovered.
Cyanidation at 200 mesh	Head (amalg. tailing), gold, 0·86 ounce per ton. Tailing, gold, 0·13 ounce per ton. Recovery, gold, 85 per cent.

Total recoveries: gold, 90·5 per cent.
Reagent consumption: cyanide, 2·5 to 3 pounds per ton; caustic soda, 4·3 pounds per ton.

4. *Straight Cyanidation at 200 mesh:*—

Assay of heads	Gold, 1·35 ounces per ton.
Assay of tailing	Gold, 0·09 ounce per ton.
Recovery	Gold, 93·3 per cent.
Reagent consumption	Cyanide, 1·3 ounces per ton. Caustic soda, 4·4 pounds per ton.

CONCLUSIONS FROM RESULTS OF EXPERIMENTAL TESTS ON LOT NO. 2

1. Amalgamation showed a slightly increased recovery of the gold values as compared with lot No. 1.

2. The flotation concentrate was lower grade due to the smaller copper content as compared with lot No. 1. The grade could be increased by cleaning operations. By amalgamation, flotation, and cyanidation, a total recovery of 95 per cent of the gold values, and 85 per cent of the copper values was obtained.

3. Amalgamation and cyanidation gave a recovery of 90 per cent of the gold values. This should have showed as good results as straight cyanidation.

4. The ore of this lot is comparatively clean as compared with lot No. 1. Straight cyanidation gave a recovery of 93 per cent of the gold values. The cyanide and lime consumption was moderate. This class of ore could very probably be treated successfully by this method.

EXPERIMENTAL TESTS ON LOTS NOS. 1 AND 2 COMBINED

Several tests were made on ore consisting of equal parts of lots Nos. 1 and 2. These tests are lettered alphabetically and the results of those which showed any merit are given.

Test B—Amalgamation, Flotation, and Cyanidation

Procedure.—500 grammes of each lot of ore at −14 mesh were cut in a Jones riffle sampler. The two were combined and ground to 65 mesh and amalgamated. The amalgamation tailing was floated. The flotation tailing was sampled, screened, and assayed, and the remainder ground to −200 mesh, then divided into two parts, each part being cyanided separately.

Results.—Amalgamation yielded 37·3 per cent of the total gold contained in the head sample.

Flotation:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu %	Au	Cu
Flotation concentrate	5·8	5·60	11·0	34·4	90·8
Flotation tailing	94·2	0·71	0·07	65·6	9·2

Analysis of flotation concentrate:—

Gold	5·60 ounces per ton	Iron	24·30 per cent
Copper	11·00 per cent	Sulphur	21·46 "
Nickel and cobalt	0·17 "	Tellurium	trace
Bismuth	1·40 "	Insoluble	29·35 per cent

Reagents:—

Water-gas tar 50 per cent and coal-tar creosote 50 per cent	0·50 pound per ton.
Pine oil No. 5	2 drops.

Pulp density in flotation cell—33 per cent solids.

Screen test on 270 grammes of flotation tailing:—

Mesh	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu %	Au	Cu
+100	21·1	0·78	0·05	23·3	15·9
+150	18·3	0·78	0·05	20·2	13·9
+200	10·6	0·75	0·06	11·3	9·7
−200	50·0	0·64	0·08	45·2	60·5

Details of cyanidation:—

	Part 1	Part 2
Ore, grms	322	324
Water, c.c	600	600
Cyanide solution, per cent	0·10	0·15
Time, hours	24	24
Cyanide consumption, pounds per ton	3·2	4·9
Lime consumption, pounds per ton	6·0	6·1
Assay of tailing, gold, ounce per ton	0·1	0·1
Per cent of total gold cyanided	35·3	35·3
Recovery from flotation tailing	86%	86%

Summary.—The gold recoveries may be summarized as follows:—

By amalgamation	37·3 per cent
By flotation	21·6 "
By cyanidation	35·3 "

In cyaniding the flotation tailing no difference is found in extraction between the 0·1 per cent and 0·15 per cent cyanide solutions.

Test C.—Cyanidation of Slimed Ore

Procedure.—1,000 grammes of ore, 500 grammes from each lot, at −14 mesh was ground to −200 mesh in a ball mill. One-quarter of the pulp was used for head sample and the remainder cut into three parts, each part being cyanided separately. Head sample assayed 1·48 ounces per ton, gold.

	Part 1	Part 2	Part 3
Ore, grms	248	252	244
Water, c.c	500	500	500
Cyanide, per cent solution	0·1	0·2	0·3
Time, hours	24	24	24
Cyanide consumption, pounds per ton	4·0	5·2	5·5
Lime consumption, pounds per ton	7·2	6·9	6·9
Assay of tailings, ounce per ton	0·48	0·10	0·10
Recovery per cent	67·5	93·4	93·4

Summary.—All the cyanide in Part 1 was consumed. This would account for the low recovery. A recovery of 93·4 per cent of the gold values was obtained using 0·2 per cent cyanide solution. The stronger solution used in Part 3 produced no better results.

Test D.—Flotation and Cyanidation

Procedure.—1,000 grammes of ore, 500 grammes of each lot, at −14 mesh was ground to 48 mesh with suitable flotation reagents and floated in a Ruth machine. The flotation concentrate was cleaned and the tailing ground to −200 mesh. About two-thirds of the tailing was used for sampling and assaying and the remainder cyanided for 24 hours using 0·3 per cent cyanide solution and caustic soda equal to 14 pounds per ton.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu %	Au	Cu
Flotation concentrate	0·3	32·77	28·08	7·2	13·0
Flotation middling	49·2	9·04	5·83	31·3	42·6
Flotation tailing	95·0	0·88	0·30	61·5	44·4

Reagents:—

Coal tar	0·50 pounds per ton
Fumol, No. 6	2 drops

No further reagents added during clean-up.

Details of cyanidation:—

Ore, grms	320
Water, c.c	600
Cyanide, per cent solution	0·3
Caustic soda, pounds per ton	14·0
Cyanide consumption, pounds per ton	6·5
Caustic soda consumption, pounds per ton	5·7
Heads, gold, ounce per ton	0·88
Tailing, gold, ounce per ton	0·10
Recovery, per cent	88·7

An addition of cyanide was made at 19 hours.

Summary.—Although the assay values of the flotation concentrate were very high, the quantity was so small that the recovery was only 7·2 per cent of the gold and 13·0 per cent of the copper. The middlings carried fair values and the tailing was high both in gold and copper. The distribution of the gold is:—

In flotation concentrate	7·2	per cent
In flotation middling	31·3	"
In cyanide solution	54·5	"
In cyanide tailing	7·0	"

In addition to the above the flotation concentrate carried 6·59 ounces per ton silver.

Test E.—Flotation and Cyanidation

Procedure.—1,000 grammes ore, 500 grammes of each lot, at −14 mesh was ground to −48 mesh in a ball mill with flotation reagents and floated in a Ruth machine. The flotation tailing was ground to −200 mesh and about two-thirds used for assaying, the remainder being cyanided for 24 hours.

Results:—

Product	Weight per cent	Assay		Per cent of values	
		Au oz/ton	Cu %	Au	Cu
Flotation concentrate	2·1	21·94	22·08	31·6	73·4
Flotation middling	12·0	2·56	0·90	21·1	17·1
Flotation tailing	85·9	0·80	0·07	47·3	9·5

Reagents:—

P.T. and T. Co., No. 400	0·50 pounds per ton
Pine oil, No. 350	2 drops

No further reagents used for clean-up

Details of cyanidation:—

Ore, grms	282
Water, c.c	600
Cyanide, per cent solution	0·1
Caustic soda, pounds per ton	8·6

Heads, gold, ounce per ton	0·8
Tailing, gold, ounce per ton	0·14
Recovery, per cent	82·5
Cyanide consumption, pounds per ton	5·5
Caustic soda consumption, pounds per ton	4·4

A sample of the cyanide solution was tested after 18 hours and found to contain no free cyanide. Cyanide was added to bring the strength up to 0·15 per cent.

Summary.—The flotation concentrate contained 31·6 per cent of the gold and 73·4 per cent of the copper and showed a marked improvement over test D. A more satisfactory copper tailing was also obtained. Cyanidation was not so effective as in test D, due to the absence of free cyanide when first tested.

The head sample assayed 1·45 ounces gold per ton, and the cyanide tailing 0·14 ounce per ton, showing a total recovery of 90·4 per cent of the gold values.

Test G.—Amalgamation and Cyanidation

Procedure.—1,000 grammes ore, 500 grammes of each lot, at −14 mesh was ground wet to 100 mesh and amalgamated in a 1:1 pulp for 2 hours using 100 grammes mercury. The amalgamation tailing was reground to −200 mesh and a sample equal to about one-quarter of the total weight removed. The remainder was divided into three parts, each part being cyanided separately.

Cyanidation Results:—

	Part 1	Part 2	Part 3
Ore, grms	244·2	240·2	244·0
Water, c.c	500	500	500
Caustic soda, pounds per ton	8	8	8
Cyanide, per cent solution	0·1	0·2	0·25
Heads, gold, ounce per ton	0·83	0·83	0·83
Tailing, gold, ounce per ton	0·14	0·14	0·15
Recovery, per cent	83·2	83·2	81·9
Cyanide consumption, pounds per ton	2·0	3·2	3·5
Caustic soda consumption, pounds per ton	3·3	3·2	3·1

Summary.—The total recoveries were as follows: Part 1—90·6 per cent; Part 2—90·6 per cent; Part 3—90·0 per cent. Amalgamation tailing assayed 0·83 ounces per ton making a recovery of 44·6 per cent by amalgamation.

SUMMARY OF EXPERIMENTAL TESTS ON LOTS 1 AND 2 COMBINED

1. *Amalgamation, Flotation, and Cyanidation:*—

Amalgamation at 65 mesh	37·3 per cent of gold values recovered.
Flotation at 65 mesh	Copper concentrate—5·8 per cent of feed. Assay concentrate, gold, 5·6 ounces per ton, copper 11·0 per cent. Recovery, gold 34·4 per cent, copper 90·8 per cent.
Cyanidation at 200 mesh	Heads (flotation tailing), gold, 0·71 ounce per ton Cyanide tailing, gold, 0·10 ounce per ton. Recovery, gold 86 per cent.
Total recoveries	Gold, 94·2 per cent; copper, 90·8 per cent.
Reagent consumption	Cyanide, 3·2 pounds per ton; lime, 6 pounds per ton.

2. *Flotation and Cyanidation:—*

Flotation at −48 mesh	Copper concentrate, 2·1 per cent of feed. Assay of concentrate, gold, 21·94 ounces per ton, copper 22·08 per cent. Recovery in concentrate, gold, 31·6 per cent, copper 73·4 per cent.
Cyanidation at 200 mesh	Heads (flotation tailing), gold, 0·80 ounce per ton. Cyanide tailing, gold, 0·14 ounce per ton. Recovery, gold, 82·5 per cent.
Total recoveries	Gold, 90·4 per cent; copper, 73·4 per cent.
Reagent consumption	Cyanide, 5·5 pounds per ton; caustic soda, 4·4 pounds per ton.

3. *Amalgamation and Cyanidation:—*

Amalgamation at 100 mesh	44·6 per cent of gold values recovered.
Cyanidation at 200 mesh	Heads (amalgamation tailing), gold, 0·83 ounce per ton. Cyanide tailing, gold, 0·14 ounce per ton. Recovery, gold, 83·2 per cent.
Total recovery of gold values	90·6 per cent.
Reagent consumption	Cyanide, 2 to 3·5 pounds per ton. Caustic soda, 8·3 pounds per ton.

4. *Straight Cyanidation at 200 mesh:—*

Heads, gold, ounces per ton		1·48
Cyanide tailing, gold, ounce per ton		0·10
Recovery per cent, gold ounces per ton		93·4
Reagent consumption	Cyanide, pounds per ton	5·2
	Lime, pounds per ton	6·9

CONCLUSIONS FROM RESULTS OF EXPERIMENTAL TESTS ON LOTS 1 AND 2 COMBINED

1. Similar results were obtained by amalgamation as were obtained on the separate lots.

2. Flotation at 48 mesh gave good results. By recleaning the rougher concentrate, a copper concentrate was obtained representing 2·1 per cent of the feed and assaying 22 ounces per ton gold and 22 per cent copper. In the test work on lot No. 1 the rougher concentrate was not recleaned. Flotation and cyanidation gave a recovery of 90 per cent of the gold values.

3. Amalgamation and cyanidation showed a recovery of 90 per cent of the gold values.

4. Straight cyanidation gave a good recovery of the gold values but with a high consumption of cyanide and lime.

GENERAL CONCLUSIONS FROM RESULTS OF EXPERIMENTAL TESTS

1. The ores represented by the sample lots submitted can be treated separately or mixed together. If mixed, the flow-sheet of the more refractory ore as represented by lot No. 1 will have to be followed.

2. Thirty per cent of the gold values can be recovered by amalgamation at 35 mesh. The adoption of amalgamation will depend on whether it is more economical to recover this percentage of gold or allow it to report in the flotation concentrate.

3. Flotation is essential to remove the refractory minerals in the ore of lot No. 1 before cyaniding, otherwise the consumption of cyanide will be excessive and prohibitive. A high-grade flotation concentrate can be obtained which will have to be shipped to the smelter or treated at the

mine for the recovery of the copper and gold values. This concentrate will amount to 2 to 4 per cent of the feed and assay 20 ounces gold and 20 per cent copper if amalgamation is not included in the flow-sheet, and 10 ounces gold and 20 per cent copper if amalgamation is included. Flotation between 48 and 65 mesh gives the best results.

4. Cyanidation is essential for a high recovery of the gold values. An all-slime process, grinding to 200 mesh is necessary for a high extraction.

5. The ore of lot No. 2 being comparatively clean and having a low copper content, good results were obtained by straight cyanidation with a moderate consumption of cyanide. It might be feasible to grind this ore separately and allow it to enter the cyanide circuit of the ore from lot No. 1.

6. The results of the experimental tests showed recoveries of both the gold and copper values above 90 per cent.

Report No. 208

THE CONCENTRATION OF GARNETIFEROUS GNEISS FROM BARRY'S BAY, ONTARIO

R. K. Carnochan

Shipment.—Three lots of garnetiferous gneiss were received at the Ore Dressing and Metallurgical Laboratories, as follows:—

Lot No. 1	received	December	5, 1923—gross weight	40	pounds.
" 2	"	"	13, 1923— "	591	"
" 3	"	January	4, 1924— "	3,000	"

The shipments were submitted by the Bancroft Mines Syndicate Ltd., 18 Toronto Street, Toronto, Ont., from the company's property in the vicinity of Barry's Bay, Ont.

Purpose of Experimental Tests.—The company had decided to erect a new concentrator to replace the one recently destroyed by fire. Markets had been built up for their product and in order to retain these it was necessary to lose no time in re-commencing operations. The old method of concentration, which consisted of crushing to 6 mesh, sizing, and tabling, did not give a product high enough in garnet content to meet the demands of the trade. The company requested the assistance of the Department in determining a method of concentration that would produce a concentrate suitable for the trade they had already built up.

Conditions Governing the Method of Concentration to be Applied.—The methods to be applied to the concentration of the garnet rock of these particular deposits were governed by the following conditions:

The trade required a product of 90 per cent garnet content consisting of 6-mesh material with very little fines. The manufacturers desired to finish the crude product to meet their own requirements for the manufactured article.

The absence of an available supply of sufficient water in close proximity to the mill site for wet concentration; the advantages of operating a dry mill during the winter months; and the cost of drying wet-concentration products were factors in favour of a dry-concentration process, provided that a high-grade product could be obtained by this means. The comparatively small tonnage to be treated, not more than 30 tons of rock per day; the high garnet content of the rock, approximately 40 per cent garnet; the abundant supply of rock of this grade, considering the tonnage to be treated, were important factors in favour of a simple process. The tonnage to be milled was not large enough to permit of the erection of an elaborate plant to obtain extremely high recoveries of the garnet content, and too large a percentage of fines was not acceptable to the trade. The high garnet content of the rock and the abundant supply of such rock permitted a waste of considerable garnet in the tailing, especially of fines that could not be sold. If it were found desirable and economical to improve recoveries, regrinding mills and further concentration units could be added as conditions demanded, and would depend to a large extent on the sale of a larger proportion of fines.

Characteristics of the Garnet Rock.—The rock containing the garnets is a gneiss, and consists of about 40 per cent garnet mineral with large amounts of black mica and hornblende. The garnets in the deposits being worked at present range in size from about $\frac{3}{8}$ inch to minute crystals. Garnets of an inch and larger are found in certain of the deposits. The garnets are of a deep red colour and have been found to be very suitable for abrasive purposes.

EXPERIMENTAL TESTS

Lot No. 1.—Wet Concentration by Jigging and Tabling

This lot was crushed to pass a 6-mesh Tyler standard screen, and sized on 8-, 10-, 14-, and 20-mesh standard Tyler screens, and on 40- and 100-mesh screens. The sizes coarser than 20 mesh were jigged on a laboratory Richards pulsating-jig and the finer sizes tabled on a laboratory Wilfley table.

A concentrate was produced representing 35 per cent of the rock treated. A clean tailing was obtained. The concentrates, both jig and table, contained some black hornblende, but would average 90 per cent garnet. The test indicated that this method of concentration could be applied successfully to the recovery of the garnet of this particular sample.

Lot No. 2.—Dry Concentration on a Sutton, Steele and Steele Table

This lot was crushed to 6 mesh and sized on 8-, 10-, 14-, 20-, 28-, 35-, 48-, 65-, and 100-mesh Tyler standard screens. Concentration tests were made on all the sizes with the exception of the -100 mesh, on a Sutton, Steele and Steele table. Various adjustments were made to the table, such as the cloth used for the table top, the distribution and volume of the wind-pressure, slope of the table, speed and stroke of the table, rate of feed, etc. These tests showed that a separation of the micaceous gangue could be made, but the garnet concentrate contained a large percentage of black hornblende. It was decided that dry concentration would not be successful on this class of garnet rock on account of the large amount of hornblende present, which is too nearly of the same specific gravity as the garnet to permit of a close separation by this means.

Lot No. 2.—Wet Concentration on a Wilfley Table

The sized material of the previous test run on the dry table was concentrated on a standard Wilfley table. No difficulty was experienced in obtaining a good separation. The garnet concentrate obtained represented 51·8 per cent of the feed to the table. It contained an appreciable amount of black hornblende, but would contain approximately 90 per cent garnet. The tailing was clean. The good separation, especially on the coarser sizes, depended on the close sizing of the feed and the manipulation of the table. The grade of the rock treated (50 per cent garnet) was a factor in obtaining such a good separation.

Lot No. 3.—Wet Concentration by Jigging Unsized Material

This lot was crushed to pass a 6-mesh Tyler standard screen. An attempt was made to concentrate the −6-mesh material, without sizing, in a two-compartment James jig. It was found that a good concentrate and hutch were produced by the first jig, but a very poor concentrate and hutch were obtained from the second jig if a clean tailing were made. The test indicated the possibility of treating unsized material by jigging with the subsequent regrinding and jigging of a middling product from some of the compartments, and the tabling of the hutch products of these compartments.

Lot No. 3.—Wet Concentration by Jigging and Tabling Sized Material

The products from the previous test were dried and mixed with the remainder of the lot. The lot was then sized on 8-, 10-, 14,- 20-, 28-, and 35-mesh Tyler standard screens. The sizes coarser than 20 mesh were concentrated in a two-compartment James jig and the finer sizes on a standard Wilfley table. This lot was not so high grade as the previous ones and contained a larger amount of hornblende. Although a good separation was made with a comparatively clean tailing, the concentrate was not so high grade as in the previous small-scale tests that had been made on the higher grade rock. The concentrate represented 32 per cent of the material treated.

SUMMARY OF EXPERIMENTAL TESTS

The three lots submitted varied in garnet content, in the amounts of hornblende present, and consequently in their amenability to concentration.

Dry concentration was not successful. On the higher grade lot on which it was tried, a satisfactory concentrate could not be obtained due to the hornblende being of almost the same specific gravity as the garnet.

Lots Nos. 1 and 2 concentrated very nicely by close sizing, jigging or tabling the various sizes. Lot No. 3 being of lower grade material, the concentrates produced by the same methods of treatment, were not so high grade nor the tailing so clean as those from the other lots.

CONCLUSIONS

The results of the test work showed three simple methods of wet concentration that should be given due consideration in producing a suitable product for the trade. All three gave good results and can be tried out at the mine in more detail and on a much larger scale to prove definitely which is the most economical.

1. Table Concentration of the Sized Material. The success of this method will depend on very close sizing of the rock crushed to 6 mesh, and on the careful manipulation of the tables in concentrating the coarser sizes.

2. *Jig and Table Concentration of the Sized Material.* Jigs, as a general rule, have been found to be more adaptable than tables to concentration of material coarser than 20 mesh. In most cases they will give a cleaner product, have a greater capacity, do not require as closely graded material, and are more easily manipulated. Tabling does better work on the finer sizes.

3. *Jig and Table Concentration of the Unsized Material.* It may be found more economical to jig the unsized material, thus eliminating the cost of sizing. By using jigs with several compartments, a clean concentrate and hutch could be obtained from the first two compartments, a middling product and hutch middling from the remaining compartments, and a clean tailing. The middling product could be reground in wet rolls and returned to the jig circuit, and the hutch middling concentrated on tables.

Report No. 209

GOLD ORE FROM MINE CENTRE, ONTARIO

C. S. Parsons

Shipment.—A shipment of 200 pounds of gold ore was received January 4, 1924, from J. S. Hillyer, Esq., 4 Chester Terrace, Duluth, Minn., U.S.A.

Location of Deposit.—The sample was taken from the property known as the Lucky Coon mine, situated on mining location P-655, near Mine Centre, Rainy River district, Ont.

Characteristics of Sample.—The sample is a free milling gold quartz ore, containing a small amount of iron pyrite and is supposed to be representative of what would be considered the milling ore from the mine.

Sampling and Analysis.—The 200-pound sample was crushed to 8 mesh and cut in a riffle sampler to 100 pounds. The 100 pounds was further reduced to 14 mesh. The sample for assay was obtained from the 14-mesh material by careful graded reduction.

Assay of sample..........Gold..........2·22 ounces per ton.

Purpose of Shipment.—Information was desired as to the best method of treatment of the ore for the design of a mill to recover the gold.

Scope of Experimental Work.—There were five possible methods of treatment, namely: amalgamation; amalgamation followed by concentration; amalgamation followed by concentration and cyaniding; amalgamation followed by cyaniding; and the simple method of straight cyaniding. In order to obtain necessary data for comparing the merits of each, tests were conducted using each method.

Experimental Tests

Test No. 1

Amalgamation.—A sample of the ore was crushed to 35 mesh and amalgamated.

Procedure.—1,000 grammes of ore was agitated with 100 grammes of mercury in a pulp of 1: 1 density for 2 hours.

Results—

Assay of heads	2·22 ounces per ton
Assay of amalgamation tailing	0·52 " "
Recovery by amalgamation	76·4 per cent

Test No. 2

Amalgamation followed by Table Concentration.—The amalgamation tailing from test No. 1 was tabled:—

Product	Weight grammes	Weight per cent	Assay oz./ton	Recoveries per cent
Table concentrate	30·4	3·06	6·76	39·20
Table tailing	963·4	96·94	0·33	60·80
Heads	993·8	....	0·52	....

Recapitulation—

Recovery by amalgamation	76·4 per cent.
Recovery in table concentrate	9·25 "
Total recovery	85·65 "
Loss in tailing	14·35 "

Test No. 3

Amalgamation, Table Concentration, and Cyaniding.—The table tailing from test No. 2 was cyanided, the tailing first being recrushed to pass 100 mesh. A cyanide solution of 0·25 per cent potassium cyanide and lime 10 pounds per ton was used. Time of agitation was 24 hours.

Assay of table tailing before cyaniding	0·33 ounce per ton.
Assay of cyanide tailing	0·02 "
Extraction by cyaniding	94·0 per cent.

Recapitulation—

Recovery by amalgamation	76·4 per cent.
Recovery in table concentrate	9·25 "
Extraction by cyaniding	13·5 "
Total recovery	99·15 "

Test No. 4

Amalgamation followed by Cyanidation of Amalgamation Tailing.—A new sample was crushed to 35 mesh and amalgamated as in test No. 1.

Assay of heads	2·22 ounces per ton.
Assay of amalgamation tailing	0·56 "
Recovery by amalgamation	74·7 per cent.

The amalgamation tailing was recrushed to pass 100 mesh and cyanided. The strength of solution used was 0·25 per cent potassium cyanide; 10 pounds lime per ton was used. The time of agitation was 24 hours.

Assay of amalgamation tailing before cyaniding	0·56 ounce per ton.
Assay of cyanide tailing	0·02 "
Extraction by cyaniding	96·5 per cent.

Recapitulation—

Recovery by amalgamation	74·7 per cent.
Extraction by cyaniding	24·4 "
Total recovery	99·1 "

Test No. 5

Cyaniding.—A sample of ore was crushed to 100 mesh and cyanided. A solution strength of 0·25 per cent cyanide was used; 10 pounds lime per ton added; the time of treatment was 24 hours.

Assay of heads before cyaniding	2·22 ounces per ton
Assay of cyanide tailing	0·02 "
Extraction by cyaniding	99·1 per cent.

SUMMARY AND CONCLUSIONS

1. *Amalgamation.*—76·4 per cent of the gold values were recovered by amalgamation. This is probably higher than what could be expected from actual milling operations, but consideration should be given to amalgamation as a preliminary operation for the recovery of a portion of the gold values.

2. *Amalgamation followed by Table Concentration.*—A further recovery of 9·25 per cent of the gold values is made in a table concentrate, making a total recovery of 85·65 per cent by amalgamation and tabling of the tailing. The concentrate would require further treatment for the recovery of the gold, and the tailing is too high to be discarded. Some subsequent treatment of the tailing is necessary to recover more of the gold.

3. *Amalgamation, Table Concentration, and Cyaniding.*—By this method a total recovery of 99·15 per cent of the gold values is obtained. There is produced, however, a table concentrate requiring further treatment of regrinding and cyaniding, or disposal to the smelters.

4. *Amalgamation and Cyaniding.*—By this method a total recovery of 99·1 per cent of the gold values is obtained, showing that it is not necessary to make a table concentrate.

5. *Straight Cyaniding.*—By this method a recovery of 99·1 per cent of the gold values is obtained, showing that on the sample submitted, a recovery is made equally as good as by any of the above methods.

The choice of methods is, therefore, between amalgamation followed by cyaniding the amalgamation tailing, and straight cyaniding. Both methods give the same recoveries of the gold values. The method adopted will depend on many factors, such as local conditions at the mine, financial conditions, and the general characteristics of the ore-bodies, etc. If coarse gold is present in the ore in any appreciable amounts, it would seem advisable to amalgamate preliminary to cyaniding. It provides for quick returns from a portion of gold values in the ore. After operating for some time, it can be eliminated from the flow-sheet if found unnecessary. Straight cyaniding would be the simpler and cheaper process both as regards cost of installation and operation. The cyanide consumption should not be high, as the sample shows the ore to be remarkably clean.

Report No. 210

THE CLEANING OF MICA PREPARATORY TO TRIMMING AND SPLITTING

R. K. Carnochan

Shipment.—Two barrels of mica, net weight 564 pounds, were received January 28, 1924, from Blackburn Bros., Ottawa.

Characteristics of the Mica.—The mica received was amber mica in pieces ranging in size from about 4 by 3 inches by ½ inch thick down to very small flakes. All of the larger pieces had small thin flakes of mica adhering to their faces. A small amount of rock was present in the shipment.

Purpose of Experimental Tests.—Tests were desired to see what could be done to remove the unattached fine flakes and to loosen and remove the adhering fine particles from the faces of the larger flakes. At present it is customary to split off a layer of mica from each face of the larger pieces so as to remove all adhering matter. It was thought that this method was wasteful, and might be improved on.

Arrangements for Experimental Tests.—Mr. H. L. Forbes of Blackburn Bros. Ltd., was present during most of the test work and gave valuable advice.

Test No. 1 (Suction)

A lot of 167½ pounds of mica was dried and then run over a 1-inch mesh Ferraris shaking screen. A suction pipe leading to a large collector was arranged at the discharge end of the screen to remove any thin flakes of mica that were too large to pass through the 1-inch mesh screen. A fan was used to create the suction. A small amount of rock was picked off the screen by hand.

Product	Weight pounds	Weight per cent
Rock	6·5	3·9
Oversize, +1 inch	134·0	80·0
Suction +1 inch	3·5	2·1
Fines, −1 inch	18·0	10·7
Loss	3·0	1·8
Moisture	2·5	1·5
Heads	167·5	100·0

Test No. 2 (Washing and Suction)

A lot of 283 pounds of mica was washed in a small revolving trommel fitted with a ½-inch screen. The mica was placed in the trommel, which was closed at both ends, through a small door. The door was closed and the trommel was revolved while half-submerged in a tank of water. The faces of the larger pieces were cleaned and the finer flakes passed through the screen and settled to the bottom of the tank. After washing, the mica was dried and run over the Ferraris screen fitted with a suction pipe as described under test No. 1.

Product	Weight, pounds	Weight, per cent
Rock	12·5	4·4
Washings, −½ inch	45·0	15·9
Oversize, +1 inch	152·5	53·9
Suction, +1 inch	12·5	4·4
Fines, −1 inch	46·5	16·5
Loss	10·0	3·5
Moisture in heads	4·0	1·4
Heads	283·0	100·0

Test No. 3 (Suction, Washing and Suction)

A lot of 113·5 pounds of mica was dried and put over the 1-inch Ferraris shaking screen fitted with a suction pipe. The oversize +1-inch product was washed in the revolving trommel. The washed product after drying, was run over the 1-inch Ferraris shaking screen fitted with a suction pipe. Similar products from the two runs on the Ferraris screen were combined.

Product	Weight, pounds	Weight, per cent
Rock	3·5	3·1
Washings, −½ inch	15·0	13·2
Oversize, +1 inch	63·5	55·9
Suction, +1 inch	5·0	4·4
Fines, −1 inch	22·0	19·4
Loss	3·0	2·7
Moisture in heads	1·5	1·3
Heads	113·5	100·0

Summary of Experimental Tests

Product	Test No. 1 Weight per cent	Test No. 2 Weight per cent	Test No. 3 Weight per cent
Rock	3·9	4·4	3·1
Washings, −½ inch		15·9	13·2
Oversize, +1 inch	80·0	53·9	55·9
Suction, +1 inch	2·1	4·4	4·4
Fines, −1 inch	10·7	16·5	19·4
Loss	1·8	3·5	2·7
Moisture	1·5	1·4	1·3

CONCLUSIONS

Running the mica over the 1-inch Ferraris shaking screen fitted with a suction pipe is very beneficial. All the free fine mica is removed and even some of that adhering to the faces of the larger pieces. This results in an oversize +1-inch product that would be ideal for trimming, except that there is still a small amount of fine mica adhering to the faces of the larger pieces. The fines from the screen would be suitable for making ground mica. The suction product consists of flake mica all +1 inch and ranging up to about 2 by 2 inches. Some of it is rather too thick to allow the whole product to be sold as split mica, but it seems quite probable that either by more careful adjustment of the size of screen and amount of suction used, or re-running, a suction product could be made that would sell as split mica.

Washing in the trommel followed by treatment in the shaking screen with suction results in a very nice oversize product quite free of any small pieces of mica adhering to the faces of the larger flakes, it has the disadvantage, though, of cutting down the percentage of oversize from 80 to 55. The trommel crushes the mica up as well as washes it, and on this account cannot be recommended. If, instead of washing in the trommel the washing were done by passing the mica under water sprays on a moving belt and some suitable arrangement were used to turn the mica over, so as to wash both sides, it is quite likely that a large percentage of oversize free from adhering particles would be obtained.

Report No. 211

MINERALIZED DIKE MATERIAL FROM ASSOCIATED GOLDFIELDS, LTD., LARDER LAKE, ONTARIO

J. S. Godard

Shipment.—Two small shipments were received during April, 1924, from Dr. H. C. Cooke of the Geological Survey, Ottawa. Shipment No. 1 consisted of 4 pounds, and shipment No. 2 of 7 pounds.

Purpose of Experimental Tests.—The object of the test work was to determine whether the gold values were associated with the arsenopyrite or the pyrite.

Characteristics of the Shipment.—The shipments submitted were designated "dike ore" and were supposed to be typical of a large body containing aresenopyrite and pyrite as the chief sulphide minerals.

Analyses:

	Shipment No. 1	Shipment No. 2
Gold	0·12 ounce per ton	0·10 ounce per ton
Arsenic	1·33 per cent	0·86 per cent
Iron	9·90 per cent	7·98 per cent

EXPERIMENTAL TESTS

A number of tests were made to obtain a separation of the arsenopyrite from the pyrite. These consisted of flotation tests; sizing and tabling of the sized material, hydraulic classification, air jigging, electrostatic separation, and roasting of the material at a temperature below the decomposition point of pyrite. None of these were successful with the exception of tabling of closely-sized material. A separation was noticed on the table, but there was not sufficient material in shipment No. 1 to obtain clean products for determination of the gold content. A further shipment was requested and shipment No. 2 consisting of 7 pounds was received.

Shipment No. 2 was crushed to 35 mesh and sized on 100- and 200-mesh screens. The sized products were tabled making a sulphide concentrate and a gangue tailing. The ratio of concentration was 1 : 13·5 showing the rock to contain about 7·7 per cent sulphides. The sulphide concentrates were closely sized on 48-, 65-, 80-, 100-, 115-, 150-, 170- and 200-mesh Tyler standard screens. Each size was tabled one after the other and the middling products re-run. By this close sizing and tabling four products were obtained, grading from practically pure arsenopyrite in No. 1 to pyrite in No. 4.

Results of Experimental Tests.—Each product was assayed with the following results:—

Product	Weight grms.	Arsenic per cent	Gold oz/ton
No. 1 concentrate	2·05	24·04	1·29
No. 2 "	8·50	22·70	0·80
No. 3 "	11·00	15·71	0·96
No. 4 "	157·20	7·19	0·70

CONCLUSIONS

The above results show that the gold values are associated with both the arsenopyrite and the pyrite, but to a greater extent with the arsenopyrite. The table tailings contained 0·03 ounce per ton gold.

Report No. 212

THE CONCENTRATION OF GOLD-COPPER ORE FROM SPROAT LAKE, NEAR ALBERNI, BRITISH COLUMBIA

C. S. Parsons

Shipment.—A shipment of 25 pounds of gold-copper ore was received January 18, 1924, from Messrs. Smith and Beavan, Alberni, B.C. The sample was from their property at Sproat lake, near Alberni.

Characteristics of the Ore.—The sample submitted was a sulphide ore consisting of chalcopyrite associated with iron pyrite in a siliceous gangue. The copper and iron sulphides carry the gold and silver values.

Purpose of Experimental Tests.—The object in conducting experimental tests was to determine whether the ore could be concentrated with the production of a marketable concentrate containing the copper and precious metal values.

Sampling and Analysis.—The shipment was sampled by stage-crushing and cutting in a Jones riffle sampler. Analysis of the sample showed it to contain:—

Copper	1·22 per cent	Gold	0·34 ounce per ton
Arsenic	trace	Silver	1·06 ounce per ton
Sulphur	9·57 per cent	Iron	8·69 per cent
Insoluble	76·52 per cent		

EXPERIMENTAL TESTS

Test No. 1—Selective Flotation

Object of Test.—The purpose of this test was to determine whether the gold was with the copper mineral only, and if such were the case, to produce a high-grade copper concentrate carrying the gold, by the elimination of the iron sulphide with gangue.

Procedure.—1,000 grammes of the ore was crushed in a ball mill to pass 65 mesh and floated in a laboratory Ruth flotation machine. The following reagents were fed to the ball mill with the ore:—

Lime	5·0 pounds per ton
Coal tar and coal-tar creosote mixture	0·5 pounds per ton

Lime was added to prevent the iron sulphide from floating with the copper.

Results.—The results of the test are given in the following table:—

Product	Weight per cent	Analysis		Per cent of values	
		Cu Per cent	Au oz/ton	Cu	Au
Concentrate	9·0	11·36	1·22	91·9	37·6
Tailing	91·0	0·10	0·20	8·1	62·4

Conclusions.—First: the results given in the above table show that only part of the gold is associated with the copper. Second: that the use of lime will prevent the iron sulphides from floating and will give a high-grade copper concentrate.

Test No. 2—Straight Flotation

Object of Test.—The purpose of this test was to float all the sulphides in the ore.

Procedure.—1,000 grammes of the ore was crushed in a ball mill to pass 65 mesh and floated in a laboratory Ruth flotation machine. A coal tar and coal-tar creosote mixture, 0·5 pound per ton of ore, was added at the ball mill.

Results.—The results of the test are given in the following table:—

Product	Weight per cent	Analysis		Per cent of values	
		Cu per cent	Au oz/ton	Cu	Au
Concentrate	23·6	4·76	1·12	98·7	89·7
Tailing	76·4	0·02	0·04	1·3	10·3

Conclusions.—First: the results of this test show that a high recovery of the gold can be obtained by the flotation of the sulphides. Second: that the gold is associated with both the copper and iron sulphides. Third: that the sulphide content of the ore can be concentrated by straight flotation in a neutral pulp.

Test No. 3.—Straight Flotation Followed by Tabling

Object of Test.—This test was made to determine whether the addition of acid to the flotation pulp would float more sulphide and increase the recovery of the gold, as the results of the previous tests indicated that the gold was associated with the sulphides.

Procedure.—1,000 grammes of the ore was crushed to 65 mesh in a ball mill and the following reagents used:

Coal tar and coal-tar creosote,	0·5 pound per ton added to ball mill
Sulphuric acid,	10·0 pounds per ton added to flotation cell.

Results.—The results of the test are given in the following table:

Product	Weight per cent	Analysis		Per cent of values	
		Cu per cent	Au oz/ton	Cu	Au
Flotation concentrate	25·2	4·86	1·24	98·2	79·3
Table concentrate	0·5	....	8·86	....	11·2
Table tailing	74·3	0·03	0·05	1·8	9·5

Combined concentrates—Concentration ratio 3·89 : 1
Analysis— Copper, 4·86 per cent; gold, 1·396 ounces per ton.
Recoveries—Copper, 98·2 per cent; gold, 90·5 per cent.

Conclusions.—First: a high recovery of the gold and copper is obtained by flotation and tabling but the grade of concentrate is low. Second: the use of acid increases the recovery of the sulphides by flotation but the flotation tailing is higher in gold than in test No. 2, where the flotation was made in a neutral pulp without the addition of lime or acid. Third: the high-grade gold concentrate obtained from the tabling indicates the presence of some free gold.

Test No. 4.—Selective Flotation followed by Tabling

Object of Test.—The test was made to determine whether a concentrate containing a higher content of copper could be obtained by first producing a high-grade copper concentrate by selective flotation and then tabling the flotation tailing for the recovery of the gold not reporting with the copper flotation concentrate.

Procedure.—1,000 grammes of the ore was crushed to 65 mesh in a ball mill. The following reagents were added in the ball mill:—

Lime	5·0 pounds per ton
Coal tar and coal-tar creosote	0·5 pound per ton
Steam-distilled pine oil	0·05 pound per ton

Results.—The results of the test are given in the following table:—

Product	Weight per cent	Analysis		Per cent of values	
		Cu per cent	Au oz/ton	Cu	Au
Flotation concentrate	8·9	12·66	1·12	92·6	30·0
Table concentrate	13·8	0·20	1·24	2·3	51·4
Table tailing	77·3	0·08	0·08	5·1	18·6

Combined concentrates—Concentration ratio 4·4 : 1.
Analysis—Copper, 5·08 per cent; gold, 1·19 ounces per ton.
Recoveries—Copper, 94·9 per cent; gold, 81·4 per cent.

Conclusions.—First: a high-grade copper concentrate can be made by selective flotation in an alkaline pulp, but the recovery of the gold is very low. Second: the tabling of the tailing from the selective flotation test did not give so high a total recovery as was obtained in the preceding test No. 3. The reason for this was due to the table not saving the fine iron sulphides which carry the gold and which were thrown into the flotation tailing by the selective action of the alkaline pulp.

Test No. 5—Amalgamation

Object of Test.—An amalgamation test was run to determine whether there was any free gold present in the ore.

Results—

Heads before amalgamation	0·34 ounce per ton
Tailing after amalgamation	0·32 ounce per ton
Recovery by amalgamation	0·02 ounce per ton = 5·9 per cent

Conclusions.—The above results show that there is very little free gold present in the ore and that the gold is associated with and contained in the copper and iron sulphides.

Summary.—A concentrate containing 4·86 per cent copper and 1·4 ounces per ton gold can be obtained with a concentration ratio of 3·9 : 1, and a recovery of 98·2 per cent of the copper and 90·5 per cent of the gold values. This was obtained by flotation followed by tabling the flotation tailing as in test No. 3.

A comparison of the flotation results from the four tests shows that selective flotation combined with tabling does not give so high a recovery of the gold values as straight flotation in a neutral or acid pulp. Flotation in a neutral pulp, as in test No. 2, gives a higher recovery of the gold values by flotation alone than flotation in an acid pulp, as in test No. 3. The amalgamation test shows that very little gold, not over 5·9 per cent of the total amount present, could be recovered by introducing amalgamation into the flow-sheet.

CONCLUSIONS AND RECOMMENDATIONS

The tests show conclusively that the gold is associated with the sulphides, and very little, if any, is free milling. This means that there will be an equivalent loss of gold for any loss of sulphide in the tailing, or in other words, a high recovery of the sulphides means a high recovery of the gold.

The procedure followed in the above tests was to grind the ore to the required fineness for flotation and to table the flotation tailing. As the results of these tests show that to obtain a high recovery of the gold values a high recovery of all the sulphides in the ore is necessary, it therefore would make no material difference to the recoveries of the copper and gold values whether tabling was done before or after flotation.

Tabling before flotation will require a more extensive plant, and the cost of operation will be higher. Classification of the pulp for table feed, regrinding of the coarse table tailing, and thickening of the slimes for flotation would be necessary, requiring more floor space with increased cost of installation, operation, and maintenance. The efficiency of flotation would not be so high as on freshly crushed ore feed.

By using flotation followed by tabling, the mill will be compact, and require considerably less floor space. The grinding can be done in one operation in ball mills in closed circuit with drag classifiers, the ball mill receiving a 1½-inch feed, and the discharge from the circuit going direct to the flotation cells. The flotation tailings will pass straight to the tables which will only have to handle a small amount of coarse sulphide that the flotation cells fail to pick up. These tables can be crowded and their capacity will be double the capacity of tables working on the crude ore feed containing all the sulphides. The number of tables required will be about half that necessary for tabling before flotation. The flotation operator would be able to look after the tables as well as the flotation cells. If Callow flat-bottom flotation cells are used, the cleaning and roughing can be done in the one cell, thus saving considerable head-room in the mill.

Report No. 213

METALLURGICAL TESTS ON GOLD ORE FROM THE REX MINE, HERB LAKE, NORTHERN MANITOBA

C. S. Parsons

Shipment.—A shipment of 142 pounds of gold ore consisting of four separate samples was received April 4, 1924, from the Manitoba Metals Co., Herb lake, Man. The samples were supposed to be representative of the ore of the Rex mine, which is being developed by the above company, a subsidiary of the Mining Corporation of Canada.

Characteristics of the Ore.—The samples submitted consisted of white quartz containing free gold visible to the naked eye, and small amounts of sulphides, chiefly iron pyrite.

Purpose of Experimental Tests.—The object in conducting the experimental tests was to determine a simple process for the recovery in a small plant of the values in the ore broken during development, and to determine the amenability of the ore to various methods of treatment.

Analyses of Samples.—The weights and assays of the samples received were as follows:—

Sample mark	Weight lbs.	Gold oz./ton	Silver oz./ton
Main sample	89·0	1·096	0·17
Bag No. 312	11·0	0·90	0·17
" " 362	21·5	0·945	0·17
" " 370	20·0	0·90	0·17

Note.—No particular difference was noticed in the characteristics of the ore of all four samples, and the assays show that they were all of practically the same grade.

Experimental Tests

Test No. 1.—Amalgamation

An amalgamation test was made on 38 pounds of ore taken from the main sample.

Description of Test.—The ore was crushed dry to 28-mesh Tyler standard screen and fed through a mercury-trap amalgamator on to amalgamation plates at the rate of one ton of ore to 2·5 square feet of plate area.

Results—

Heads before amalgamation	1·096 ounces per ton
Tailing after amalgamation	0·16 ounce per ton
Recovery of gold values	85·4 per cent

Test No. 2—Blanket Test

A blanket test was made on 49 pounds of ore from the main head sample.

Description of Test.—The ore was crushed dry to 28-mesh Tyler standard screen, and fed over corduroy blankets at the rate of 1·1 tons of ore to 2·5 square feet of blanket area.

Results—

Heads to blankets	1·096 ounces per ton
Tailing from blankets	0·52 ounce per ton
Blanket concentrate obtained	882·4 grammes
Assay of blanket concentrate	15·45 ounces per ton
Gold values in blanket concentrate	56·2 per cent
Gold values remaining in tailing	43·8 per cent

Test No. 3.—Table Test on Amalgamation Tailing

A 5,000-gramme sample of the amalgamation tailing from test No. 1, was tabled on a laboratory Wilfley table. The results were as follows:—

Product	Weight		Assay Au oz/ton	Per cent of gold values
	Grammes	Per cent		
Table concentrate	68·7	1·4	7·35	49·0
Table tailing	4768·0	98·6	0·11	51·0

A very clean table concentrate was made. It is quite probable that a much lower tailing could be obtained by cutting out more concentrate. The total recovery by amalgamation and tabling was approximately 93 per cent.

Test No. 7.—Amalgamation Test on Blanket Concentrate

Description of Test.—685 grammes of blanket concentrate from test No. 2 was amalgamated with 10 per cent by weight of mercury in a pulp density of 50 per cent solids.

Results—

Assay before amalgamation	15·45 ounces per ton
Assay after amalgamation	1·20 ounces per ton
Recovery of gold values	92·3 per cent.

Test No. 8.—Amalgamation Test on Table Concentrate

Description of Test.—40 grammes of table concentrate from test No. 3 was amalgamated in a mortar. The mercury was panned out and the tailing assayed.

Results—

Assay before amalgamation	7·35 ounces per ton
Assay after amalgamation	1·03 ounces per ton
Recovery of gold values	86·0 per cent.

Cyanide Tests on Amalgamation Tailing.—Tests Nos. 9 and 10 were run on amalgamation tailing containing 0·24 ounce gold per ton. The details of the tests and the results obtained are given below:—

Test No. 9—Part 1

	A	B	C
Degree of grinding, mesh	−65	−100	−150
Weight of tailing, grammes	200	200	200
Water, c.c	400	400	400
Potassium cyanide, per cent solution	0·25	0·25	0·25
Lime, pounds per ton	5	5	5
Time of agitation, hours	24	24	24
Results—			
Cyanide heads, ounce per ton	0·24	0·24	0·24
Cyanide tailing, ounce per ton	0·008	0·005	0·005
Recovery, per cent	96·8	98·0	98·0
Cyanide consumption, pounds per ton	0·866	0·91	0·56
Lime consumption, pounds per ton	3·9	4·2	3·72

Test No. 9—Part 2

	A	B	C
Degree of grinding, mesh	−65	−100	−150
Weight of tailing, grammes	200	200	200
Water, c.c	400	400	400
Potassium cyanide, per cent solution	0·25	0·25	0·25
Lime, pounds per ton	5	5	5
Time, hours	48	48	48
Results—			
Cyanide heads, ounce per ton	0·24	0·24	0·24
Cyanide tailing, ounce per ton	0·01	0·01	0·01
Recovery, per cent	95·8	95·8	95·8
Cyanide consumption, pounds per ton	0·62	0·98	0·786
Lime consumption, pounds per ton	3·8	4·18	4·02

Test No. 10—Part 1

Degree of grinding, mesh	−65	−100	−150
Weight of tailing, grammes	200	200	200
Water, c.c	600	600	600
Potassium cyanide, per cent solution	0·05	0·05	0·05
Lime, pounds per ton	5	5	5
Time, hours	24	24	24
Results—			
Cyanide heads, ounce per ton	0·24	0·24	0·24
Cyanide tailing, ounce per ton	0·05	0·02	0·02
Recovery, per cent	79·2	91·7	91·7
Cyanide consumption, pounds per ton	0·632	1·23	1·11
Lime consumption, pounds per ton	3·54	3·81	3·56

Test No. 10—Part 2

Degree of grinding, mesh	−65	−100	−150
Weight of tailing, grammes	200	200	200
Water, c.c	400	400	400
Potassium cyanide, per cent solution	0·05	0·05	0·05
Lime, pounds per ton	5	5	5
Time, hours	48	48	48
Results—			
Cyanide heads, ounce per ton	0·24	0·24	0·24
Cyanide tailing, ounce per ton	0·04	0·01	0·01
Recovery, per cent	84·4	95·8	95·8
Cyanide consumption, pounds per ton		0·4	0·91
Lime consumption, pounds per ton		3·67	4·19

SUMMARY AND CONCLUSIONS

The tests show a recovery of the gold values of 85 per cent by amalgamation alone. Amalgamation shows a much higher recovery than concentration on corduroy blankets and amalgamation of the blanket concentrate.

Tabling after amalgamation and amalgamation of the table concentrate increases the recovery of the gold values.

Grinding to 30 mesh for amalgamation gives a high recovery of the gold values. The condition of the mercury was good, both inside and on the plates, showing that the ore is very amenable to amalgamation.

Cyanide tests on the amalgamation tailing show very high recoveries of the remaining gold values after amalgamation.

Report No. 214

EXAMINATION OF AND EXPERIMENTAL TESTS ON GOLD ORE FROM DOMINION CLAIMS, COPPER LAKE, NORTHWESTERN MANITOBA

W. B. Timm

Shipments.—Two small shipments of gold ore were received from A. L. Stewart, Esq., 280 Smith St., Winnipeg, Man. Shipment No. 1 consisting of two small pieces of ore weighing approximately ½ pound was received on February 27, 1924. Shipment No. 2 consisting of two samples of the ore, weighing 6½ pounds and ½ pound respectively, was received April 29, 1924.

Location of Claims.—The samples submitted were taken from a 14-inch stringer in an ore zone of 32 feet in width on the Dominion claims on Copper lake. The claims are situated approximately 80 miles north of The Pas, Man., and 12 miles northeast of the proposed railway to the Flin-Flon camp.

Characteristics of the Ore.—The samples consisted of mineralized quartz, the chief minerals being iron pyrites and small amounts of galena. No free gold was visible in the samples of No. 1 shipment, but in the samples of No. 2 shipment free gold which showed up as metallics in grinding the samples for analysis and experimental tests was detected.

Purpose of Examination and Experimental Tests.—An examination of the samples was requested to determine in what form the gold and silver values occurred in the ore. Panning tests made on the ore by the owners revealed no gold, although similar samples gave high values in gold on assay.

Analysis of Ore Samples.—The samples received gave the following values on assay:—

	Gold oz/ton	Silver oz/ton	Tellurium %
Shipment No. 1	13·92	45·56	0·09
Shipment No. 2, large sample	4·91	16·37	
Shipment No. 2, small sample	7·50	23·47	

Experimental Tests on Shipment No. 1

The two small pieces of ore were crushed to 40 mesh and screened on 80 mesh. This gave: −40+80 83 grammes
−80 126 "

Each of the above sizes was tabled separately on a laboratory Wilfley table and the combined table tailings were floated in a laboratory Ruth flotation machine. The results of this test were as follows:—

Product	Weight grms.	Gold oz/ton	Silver oz/ton	Tellurium per cent
Ore	....	13·92	45·56	0·09
+80 table concentrate	10·3	9·80	27·00	....
−80 table concentrate	6·6	84·40	150·80	....
Flotation concentrate	4·7	140·00	380·30	....
Flotation middling	47·0	10·32	45·48	....
Flotation tailing	144·2	3·28	16·40	....

The table concentrates were practically clean sulphides, mostly pyrite, whereas the flotation concentrate contained very little mineral. The flotation middling and tailing consisted of quartz gangue, very little mineral being noticeable on examination under the microscope.

Conclusions from Results of the above Test.—No free gold was visible and no metallics were obtained in preparing the sample for analysis and for the above test, which indicated that if the gold were present as free gold it was in a very finely divided state. The presence of tellurium in the ore as indicated by the analysis suggests that the gold may occur as the telluride.

The coarse table concentrate, +80 mesh, which is practically clean sulphides, gave a lower assay than the ore, and the lowest of any of the concentration products with the exception of the tailings. This would indicate that the iron pyrite is not carrying the gold values. If it were, there would be a concentration of the gold in this product.

The −80 mesh table concentrate and the flotation concentrate, which would be 200 mesh and finer, show the highest values in both gold and silver. This would indicate that in whatever form the values occur, they are in a finely divided state.

Microscopic Examination of Products from Shipment No. 1.—The −80-mesh table concentrate and the flotation concentrate carrying the high gold values were examined under a high power microscope. No free gold could be detected. There was not sufficient of the products available to make a thorough examination. It is, however, very probable that the gold is present as tellurides. Larger samples are required for a more extended examination in order to reach a definite conclusion on this point.

Experimental Tests on Shipment No. 2

A concentration test by tabling and flotation, an amalgamation test and two cyanide tests were made on the larger sample of ore submitted with shipment No. 2.

Concentration Test.—This test was conducted to determine in what products the gold and silver values were reporting and to obtain products for microscopic and chemical examination. 1,000 grammes of the ore was crushed to 80 mesh. In crushing the sample for assay some metallics were obtained and free gold was observed in the ore. The −80 mesh material was sized on 100-, 150-, and 200-mesh Tyler standard screens. Each size was tabled on a laboratory Wilfley table and the combined tailings floated in a laboratory Ruth flotation machine. The results of this test were as follows:—

Product	Weight grms.	Gold oz/ton	Silver oz/ton
Table concentrate +100	13·1	10·21	22·45
Table concentrate +150	8·9	14·29	20·90
Table concentrate +200	7·2	28·58	60·66
Table concentrate −200	9·0	144·90	226·00
Flotation concentrate	11·8	105·80	322·00
Flotation middling	83·0	6·76	34·74
Flotation tailing	784·0	0·60	2·76
Loss in test	83·0	....	....
Ore	1000·0	4·91	16·37

Amalgamation Test.—This test was conducted to determine the percentage of gold present as free gold that would amalgamate; 40 per cent of the gold values and 7 per cent of the silver values were recovered from the mercury used in the test. This checked with the assay of the amalgamation tailing. The test was made on the ore ground to 200 mesh, in order to obtain the maximum recovery by amalgamation.

Cyanide Tests.—Two cyanide tests were made on the ore ground to 200 mesh. One was given 48 hours' agitation, the other 96 hours. The results were as follows:—

Tailing assay test No. 1—48 hours' agitation	Gold, 0.08;	silver, 0.78
“ “ 2—96 “ “	“ 0.06;	“ 0.80
Recoveries by cyanidation	“ 98%;	“ 95%

Conclusions from Results of above Tests on Shipment No. 2.—The ore of shipment No. 2 contains free gold. Small particles were detected in the ore visible to the naked eye, and more so under the microscope. Metallic gold was obtained in preparing the sample for analysis. The amalgamation test showed that 40 per cent of the gold values and 7 per cent of the silver values were recovered by amalgamation. This test also showed that the silver was not alloyed with the gold to any extent, and that it was not present as native silver, but that it was either associated with the galena in the ore, or as a separate mineral, possibly argentite or ruby silver. The low recovery of the gold values by amalgamation could only be accounted for by the remaining gold being either locked up with the sulphides in a very finely divided state or present in the form of tellurides.

The results of the concentration test on shipment No. 2 were very similar to those on shipment No. 1. The high values in both gold and silver reported in the fine table concentrate and flotation concentrate, showing that the coarse pyrite carried relatively small values as compared with the finer products, and that the gold and silver minerals were present in a very finely divided state.

The high recoveries of both gold and silver obtained in the cyanide tests, and the low consumption of cyanide and lime, indicated that the ore contained no objectionable impurities; that the gold and silver minerals were in a finely divided state, and readily soluble in cyanide solution. The small amounts of tellurium, arsenic, and antimony present had but little effect on the solubility of the gold and silver.

Chemical Tests.—To determine the presence of tellurium in the ore, a qualitative analysis was made on 50 grammes. Tellurium was determined in small amounts, indicating that a portion of the gold values was present in the form of gold tellurides.

To determine the presence of ruby silver in the ore, a qualitative analysis was made for arsenic and antimony in the ore, in the flotation concentrate which contained the high silver values, and in the +200 mesh table concentrate. Very small amounts of arsenic were present in all three, but much more antimony was present in the flotation concentrates than in the ore or in the +200 table concentrate. The +200 table concentrate contained a much larger proportion of iron pyrite than the flotation concentrate. The higher antimony content in the flotation concentrate would indicate that the antimony was not associated with the pyrite but with the silver mineral in the flotation concentrate, indicating the presence of ruby silver.

Microscopic Examination.—Free gold was detected in the ore and in the concentration products. Ruby silver was detected in the ore and in the flotation concentrate. Although no tellurides could be detected in any of the products, the chemical analysis shows the presence of tellurium beyond a doubt.

SUMMARY OF EXAMINATION AND EXPERIMENTAL TESTS

The ore submitted for examination and test contains free gold. It also contains tellurium, indicating the presence of gold tellurides. The silver is present as ruby silver, a small portion may be present associated with the small amount of galena in the ore, and a portion may be present as silver sulphide, argentite, although this mineral was not detected by microscopic examination.

The ore is amenable to treatment by cyanidation, with high recoveries of the gold and silver values and low consumption of cyanide and lime.

Report No. 214a

EXPERIMENTAL TESTS ON GOLD ORE FROM DOMINION CLAIMS ON COPPER LAKE, NORTHWESTERN MANITOBA

J. S. Godard

Shipment.—A further shipment of 198 pounds of ore was received September 3, 1924, from A. L. Stewart, Esq., 280 Smith street, Winnipeg.

Purpose of Shipment.—The shipment was made to check the accuracy of the results obtained from experimental tests on former small shipments received for examination as to the form in which the gold and silver values occurred in the ore.

Sampling and Analysis.—The ore was crushed to half inch and cut once in a riffle sampler. One half was ground to 14-mesh and cut twice in the sampler. One of these portions was ground to pass 48 mesh. The sample for analysis was cut from this portion and gave the following:—

Gold	2·88 ounces per ton
Silver	11·14 ounces per ton

Experimental Tests

A series of amalgamation tests was made to determine the amount of gold and silver values that could be recovered by amalgamation at various stages of grinding. For each test 1,000 grammes of ore was used with 100 grammes of mercury, and a pulp density of 50 per cent solids. The contact period in each case was two hours. The amalgamation tailings were screened and the products assayed. The results are given in the following table.

A series of cyanide tests was made to determine the amenability of the ore to treatment by the cyanide process. The accompanying table shows the details of each test and the results obtained.

Amalgamation Tests

Test No.	Screen analysis of tailings				Per cent of values		Per cent amalgamated	
	Mesh	Weight per cent	Assay oz. per ton					
			Gold	Silver	Gold	Silver	Gold	Silver
2	+100	3·9	1·26	5·55	2·5	2·1	32·6	0·64
	−100+150	13·4	1·24	5·39	8·5	6·9	....	....
	−150+200	17·2	2·61	6·06	23·2	10·0	....	....
	−200	65·5	1·95	12·92	65·8	81·0	....	....
4	+100	1·0	2·57	9·20	1·7	0·9	....	....
	−100+150	6·3	1·39	5·77	5·6	3·7	45·4	11·4
	−150+200	12·6	1·11	5·19	8·8	6·6	....	....
	−200	80·1	1·65	10·95	83·9	88·8	....	....
6	+150	2·3	1·81	6·81	3·0	1·7	....	....
	−150+200	11·5	1·11	4·71	9·0	5·8	51·4	15·6
	−200	86·2	1·43	10·10	88·0	92·5	....	....
23	+48	6·0	2·37	....	7·1	....	....	....
	−48+65	7·1	2·33	....	8·2	....	....	....
	−65+100	15·0	1·88	....	14·1	....	30·2	....
	−100+150	12·4	1·56	....	9·6	....	....	....
	−150+200	11·3	1·32	....	7·4	....	....	....
	−200	48·2	2·23	....	53·6	....	....	....

SUMMARY

1. Amalgamation.—At 48 mesh 30·2 per cent of the gold was recovered by amalgamation. This recovery was increased to 51·4 per cent with ore ground to 150 mesh. Less than 1 per cent of the silver was amalgamated at 100 mesh. This amount increased with fine grinding until a maximum of 15·6 per cent was obtained on 150-mesh material.

2. Cyanidation.—The recovery by cyanidation at 48 mesh was 84·1 per cent of the gold and 61·9 per cent of the silver. On material ground to 150 mesh and cyanided with 0·30 per cent potassium cyanide for twenty-four hours, the recovery was 96·5 per cent of the gold and 92·4 per cent of the silver. A small increase in recovery is obtained by increasing the time. Both the cyanide and lime consumption was moderate. Better results were obtained using a dilute pulp.

CONCLUSIONS

The results of the experimental tests on the larger shipment show that the ore is amenable to treatment by cyanidation with high recoveries of the gold and silver values, and that the results check closely with those reported on the former smaller shipments.

Cyanidation Tests

Test No.	Weight grms.	esl Mesh.	Water grms.	Per cent cyanide	Lime lb/ton	Time hours	Heads		Tailing		Per cent recovery		Reagents consumed	
							Gold oz./ton	Silver oz./ton	Gold oz./ton	Silver oz./ton	Gold	Silver	Cyanide lbs/ton	Lime lbs/ton
7	295	—48	590	0·20	5	24	2·88	11·14	0·46	4·24	84·1	61·9	2·3	4·2
8	297	—150	594	0·10	5	24	2·88	11·14	0·36	3·57	87·6	68·0	1·9	4·5
9	299	—200	600	0·20	5	48	2·88	11·14	0·23	0·79	92·1	93·0	3·3	4·5
10	300	—150	600	0·20	5	24	2·88	11·14	0·15	1·26	94·8	88·6	2·5	4·4
10b	300	—150	600	0·20	5	48	2·88	11·14	0·16	0·70	94·4	93·8	2·9	4·3
17	300	—150	900	0·10	4	24	2·88	11·14	0·38	2·24	86·8	80·0	2·1	2·0
18	305	—150	915	0·20	4	24	2·88	11·14	0·16	1·04	94·4	90·6	2·9	2·0
19	303	—150	909	0·30	4	24	2·88	11·14	0·10	0·80	96·6	92·8	3·0	1·9
20	300	—150	900	0·30	4	24	2·88	11·14	0·10	0·84	96·5	92·4	3·0	2·3
21	300	—150	900	0·30	4	48	2·88	11·14	0·07	0·69	97·6	93·8	3·1	2·8
22	300	—150	900	0·30	4	72	2·88	11·14	0·06	0·68	97·9	93·8	3·2	2·5
24	300	—150	300	0·20	5	24	2·88	11·14	0·88	3·60	69·4	67·6	2·4	3·9
25	300	—150	600	0·20	5	24	2·88	11·14	0·36	1·88	87·5	83·1	2·7	3·1
26	300	—150	900	0·20	5	24	2·88	11·14	0·21	1·31	92·7	88·2	2·7	2·3

Report No. 215

CONCENTRATION OF MOLYBDENITE ORE FROM THE MOSS MINE, QUYON, QUE.

C. S. Parsons

Shipments.—Two shipments of molybdenite ore, each a carload lot, weighing 25·3 and 42·3 tons, respectively, were received May 16, and June 11, 1924, from the Moss mine, the property of the Canadian-Wood Molybdenite Co., Quyon, Que.

Purpose of Experimental Tests.—The company were re-opening the mine, and starting milling operations. They had secured a contract from a chemical firm for high-grade molybdenite concentrates and desired a large-scale test run to determine the best practice to follow in the concentration of their ore to produce the highest grade of concentrate with good recoveries.

Characteristics of the Ore.—The Moss mine ore consists of segregations of pyrite, pyrrhotite, fluorite, quartz, and orthoclase in quartz syenite. The iron sulphide content is low in comparison to the silica content, and this feature makes this ore more amenable to concentration than those of high iron content. The molybdenite flake is not large, rarely exceeding one-quarter inch in diameter, and is fairly well disseminated throughout the gangue rock. The rock is quite hard to crush, but grinding need not be carried to extreme fineness as the molybdenite is freed at 35 to 48 mesh, the desirable point for flotation. The cost of preparation of the ore for concentration is, therefore, not excessive.

CONCENTRATION RESULTS

Shipment No. 1

One car of ore, net weight	50,600 lb.
Analysis of sample cut from dry ore by Vezin sampler	2·21% MoS_2
Average analysis of daily sample of wet feed to cells	2·45% MoS_2
Average of dry and wet samples	2·33% MoS_2
Content of molybdenite, using average analysis	1,178·98 lb.
Amount of concentrate obtained	1,213·0 lb.
Analysis of concentrate	92·93% MoS_2
Content of molybdenite in 1,213 pounds of concentrate	1,127·24 lb.
Calculated analysis of tailing by difference	0·10% MoS_2
Recovery of molybdenite from average assay and calcul. tailing	95·6%

Shipment No. 2

One car of ore, net weight	84,600 lb.
Analysis of sample cut from dry ore by Vezin sampler	2·33% MoS_2
Average analysis, daily sample of wet feed to cells	2·60% MoS_2
Average of dry and wet samples	2·46% MoS_2
Content of molybdenite, using average analysis	2,081·16 lb.
Amount of concentrate obtained	2,135·0 lb.
Analysis of concentrate	93·44% MoS_2
Content, 2,135 pounds, 93·44 per cent	1,994·94 lb.
Analysis of tailing, average of daily samples	0·12% MoS_2
Calculated analysis of tailing by difference	0·12% MoS_2
Recovery of molybdenite	95·8%

Procedure Followed.—The ore was crushed in jaw crusher to one inch and elevated to a storage bin from which it was fed through a Vezin sampler taking a one-twentieth cut. The ore after passing the sampler went direct to a $4\frac{1}{2}$-foot Hardinge mill operating in closed circuit with a simplex Dorr classifier. The classifier overflow went direct to two 1 by 6-foot, flat-bottom Callow roughing cells. The rougher concentrate was cleaned in two 1 by 3-foot cleaner cells operated in series. The cleaner tailing was returned to the head of the rougher cells by an air lift. The cleaner concentrate from the last cleaner cell was pumped to a Callow belt-screen equipped with an 80-mesh screen. The product remaining on the screen was the finished concentrate and was collected in a storage tank. The material passing through the screen was returned to the head of the rougher cells.

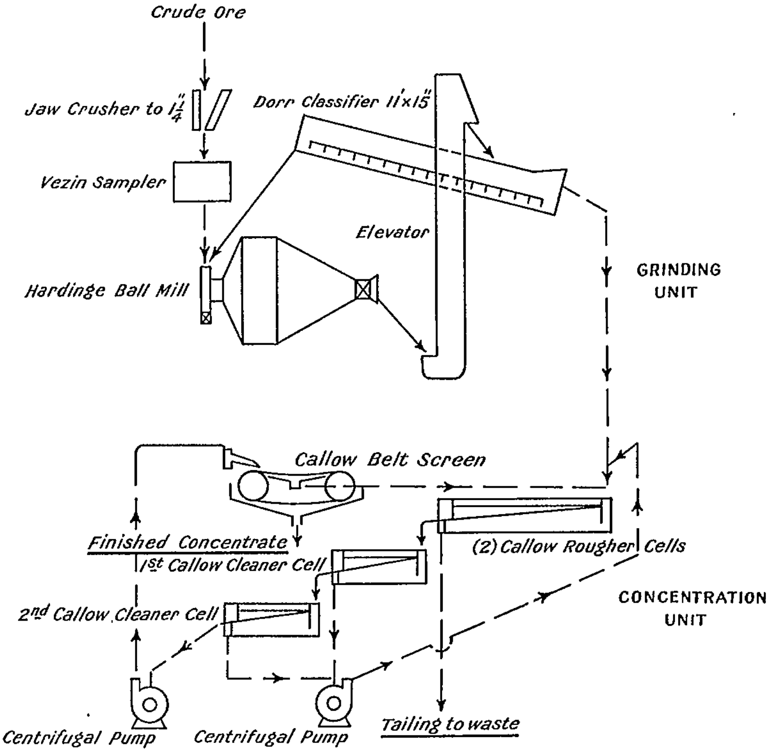

Figure 3. General flow-sheet for concentration of molybdenite ore.

NOTES ON FLOTATION

Considerable experimental work was done during the treatment of the first carload. The work was confined chiefly to the operation of the classifier and Callow belt-screen, and to the use of addition reagents such as lime and soda ash.

In order to produce a high-grade concentrate an endeavour was made to keep the molybdenite in the classifier overflow as coarse as possible, but at the same time free from any attached particles of gangue or other sulphide. In this connexion it was found necessary to crush until the classifier overflow did not contain more than 1 per cent on a 35-mesh Tyler screen. The classifier was equipped with air and water sprays and the feed was allowed to plunge into the classifier so as to produce a boiling action in the settling-area of the classifier which tended to bring the flake to the surface. Water sprays were used on the rakes and caused considerable flake to be washed out of the oversize. An air spray at the water edge was used to blow the flake which had been washed out of the oversize over to the overflow as fast as it was washed down. This arrangement was found very satisfactory and reduced the quantity of flake returning to the mill by approximately 50 per cent.

The concentrate from the last cleaner cell was fed to a small 1-inch centrifugal pump together with a large volume of fresh water. The control of the quantity of water fed with the concentrate to the pump was found to be very important. Too much water broke up the flocculation of the molybdenite allowing too much molybdenite to pass through the Callow belt screen and be returned to the circuit. Too little water did not allow the molybdenite to flocculate free from the entangled pyrite and gangue, thus lowering the grade of the concentrate obtained on the screen. A series of analysis of the concentrate before and after passing over the screen showed that the use of the screen raised the grade 7 per cent to 8 per cent in MoS_2 content. The object of using the centrifugal pump was to re-flocculate the molybdenite in a pulp of clean water so that the pyrite and gangue would be delivered on the screen free from any entanglement in the flocculated molybdenite.

The oils used were kerosene and steam-distilled pine oil. The kerosene found to be most satisfactory was British American Oil Co's. "Lampolene" brand. The amount of kerosene used varied slightly from time to time, but at no time when a neutral pulp was used did it exceed 0·80 pound per ton. The quantity of pine oil used averaged only 0·4 pound per ton. The quantity of pine oil was found to affect the grade of the concentrate, too much oil lowering the grade. This effect was probably due to the well-known action of pine oil in a closed water circuit. Most of the selective power of pine oil is used up in its first contact with the ore. In this case practically all the pine oil in the flotation concentrate is returned back with the undersize from the screen. This building up of dead pine oil produced a voluminous froth which had very little carrying property for molybdenite. The result of this could be observed by watching the molybdenite gradually start to spread down the cells and instead of coming off right at the feed-end it comes off all over the cell. This building up of dead pine oil not only lowers the grade of the concentrate but also increases the loss in the tailing.

The addition of lime or soda ash to the pulp proved to be of no particular advantage. Lime was tried and the amount was varied from 1 to 5 pounds per ton of ore. A slight improvement was observed in the grade of the concentrate, but this was offset by a decided increased loss in the tailing. The increase in grade of the concentrate was about 1 per cent of MoS_2 and the increased loss in the tailing approximately 0·1 per cent. The same increase in the grade of the concentrate can be obtained in a neutral pulp with a corresponding tailing. The addition of lime gave a very different acting pulp from that obtained under neutral conditions; the quantity of kerosene could be doubled without any over-oiling effect being observed in cells. It was necessary to use over ¼ pound per ton more kerosene than in a neutral pulp to keep the tailing clean. The lime apparently tended to emulsify the kerosene oil forming an emulsion which did not readily break down when coming in contact with the flakes of molybdenite. The increased loss in the tailing was from very fine flake only.

Soda ash had much the same effect on the oils, it being necessary to use a much larger amount of kerosene. The grade of concentrate was not improved as there was a tendency to float gangue material. The tailing was also higher.

An important and significant fact was observed in connexion with the operation of the ball mill. At one time during the run the characteristic froth in the rougher cells started to disappear and be replaced by a more voluminous heavy and slow-breaking froth. The molybdenite instead of appearing on the convexed face of the bubbles near their dome and all coming off in the first foot of the cells, appeared in dead-looking clots carried in the troughs between the contact of bubble surfaces and spreading down the whole length of the cell before coming off. The reagents were checked and found to be unchanged and the ore looked as clean as usual. The density of the pulp discharging from the mill was observed to be much thicker than usual. With the addition of more water to the ball mill the cells immediately cleaned up and operation came back to normal. During the time the pulp in the mill was thick there was considerably more loss of molybdenite in the tailing, and what is more important, the grade of the concentrate was considerably reduced due to the flotation of more iron. The action of the oils on this thick pulp in the ball mill was affected in some manner, entirely changing the characteristics displayed by a thinner pulp.

The density of the pulp in the ball mill during the run was between 45 per cent and 50 per cent solids, but when increased above 60 per cent poor results were obtained in the flotation cells. Any thickening of the pulp in the ball mill was indicated in the cells immediately by the characteristic froth referred to above.

SUMMARY AND CONCLUSIONS

The ore is very amenable to concentration by flotation. With proper manipulation and under the right conditions, as outlined above, a high-grade concentrate, 93 per cent MoS_2, is obtained with recoveries in excess of 95 per cent of the molybdenite content in the ore. Concentrates assaying over 96 per cent MoS_2 were made during the runs under the best conditions. The concentrates are exceptionally free from deleterious substances; are suitable for the manufacture of molybdic acid, molybdic salts, molybdenum metal, ferro-molybdenum, and should command a high market price.

Report No. 216

EXAMINATION OF KIRKLAND LAKE MILL TAILINGS

C. S. Parsons

Shipment.—A shipment of 50 pounds of mill tailings from the Wright-Hargreaves Mines, Ltd., Kirkland Lake, Ont., was received October 16, 1924.

Purpose of Experimental Tests.—The test work was conducted to determine whether the values in the tailings could be recovered by flotation, and also to get some idea of the reasons for the high tailing losses.

Assay of Sample.—The assay of the tailings submitted for test purposes showed them to contain 0·06 ounce gold per ton. It is reported that at times the tailings exceed this value, depending on the refractoriness of the ore being treated.

Experimental Tests.—The experimental tests consisted of screen analysis of the tailings, classification tests, and flotation tests.

Screen Analysis of Tailings from Wet Sizing Test on Tyler Standard Screens.

Head sample: 0·06 ounce gold per ton.

Mesh	Weight per cent	Assay Au oz./ton	Per cent of values
−100+150	19	0·10	35
−150+200	12	0·07	15
−200	69	0·04	50
Totals	100	0·055	100

The above screen test shows that 31 per cent of the tailing stays on a 200-mesh screen; 69 per cent passes a 200-mesh screen; that this 31 per cent oversize contains 50 per cent of the gold values in the tailings.

Classification Tests on Tailing.—Several classification tests were made to determine the distribution of the values in the classified products, sands and slimes. The results of a number of these tests are given below:—

Product	Weight per cent	Assay Au oz./ton	Per cent of values
Sands	76·5	0·08	89·7
Slimes	23·5	0·03	10·3
Totals	100·0	0·068	100·0
Sands	61·8	0·10	80·2
Slimes	38·2	0·04	19·8
Totals	100·0	0·077	100·0
Sands	80·7	0·07	93·5
Slimes	19·3	0·02	6·5
Totals	100·0	0·06	100·0
Sands	73·3	0·08	95·6
Slimes	26·7	0·01	4·4
Totals	100·0	0·061	100·0

The above classification tests show that the high tailing values are in the coarser material and that cyanidation has extracted the values to a much greater extent from the fine slimes.

Flotation Tests on Classified Sands.—The following flotation tests were made on the sand products from the classification tests given above. The values in the flotation products do not check up with the values in the classified sands, probably due to the small amount of material being used in the tests, the low assay value of the material and products, and the difficulty in cleaning up the testing apparatus.

Heads: Classified sand—0·10 ounce per ton

Product	Weight %	Assay Au oz./ton	Per cent of values
Concentrate	7·2	0·40	60·6
Tailing	92·8	0·02	39·4
Totals	100·0	0·047	100·0
Concentrate	3·0	0·70	44·5 } 62·5
Middling	8·5	0·10	18·0 }
Tailing	88·5	0·02	37·5
Totals	100·0	0·047	100·0

Heads: Classified sand—0·08 ounce per ton

Product	Weight %	Assay Au oz./ton	Per cent of values
Concentrate	4·1	0·60	44·1 } 55·6
Middling	13·0	0·05	11·5 }
Tailing	82·9	0·03	44·4
Totals	100·0	0·056	100·0

The flotation results on the classified sand tailings are, so far, not very promising. They show a possible recovery of 40 per cent of the values in the cyanide tailings, in a concentrate assaying 0·6 ounce per ton or $12 per ton. This concentrate would represent about 4 per cent by weight of the ore fed to the mill.

SUMMARY

From the results of the screen test and classification tests given above it would seem that coarse grinding is responsible, to a large extent, for the high tailing losses. Cyanidation seems to have given good extractions on the −200-mesh material and on the classified slimes. Fifty per cent of the tailing values is in the 31 per cent of the tailings remaining on 200-mesh. Over 80 per cent of the tailing values is in the sands from classification, and less than 20 per cent in the slimes. An all-slime process should, therefore, increase the recovery materially and give a low tailing. It would seem that more efficient classification is required in the tube mill circuit. This may mean the installation of a bowl to receive the classifier overflow, and of an additional tube mill in circuit with the bowl, in order to make an all-slime process. The problem is whether the cost of this additional installation would be offset by the increased recoveries.

It is the opinion of many engineers that it is the presence of tellurides which accounts for the high tailing losses. If it is the case that tellurides grind readily, one would expect them to slime easily, and that the slimes would carry the higher values. This is not the case, and would lead one to believe that the losses are in the coarse pyrite. A microscopic examination is being made of the coarse pyritic concentrates in the tailings to determine if the tellurides are present with the pyrite. Flotation of cyanide tailings does not seem to work very satisfactorily. This was found to be the case when tried in the Cobalt camp.

Report No. 217

CONCENTRATION OF A SILVER-LEAD ORE FROM PORTLAND CANAL DISTRICT, STEWART, BRITISH COLUMBIA

C. S. Parsons

Shipment.—A shipment of 54 pounds of silver-lead ore was received August 26, 1924, from S. G. O. Chalmers, Stewart, B.C.

Location.—The shipment was made from the E. G. group of mineral claims situated on the Canadian side of Portland canal, five miles south of Stewart, and was supposed to be representative of the ore from these claims.

Characteristics of the Ore.—The ore is a complex mixture of lead, zinc, copper, and iron sulphides, carrying silver and gold values. The silver values predominate. The gangue is of a siliceous nature.

Sampling and Analysis.—The shipment was sampled and found to contain the following on analysis:—

Lead	2·48 per cent	Iron	4·00 per cent
Zinc	1·60 "	Gold	0·05 oz. per ton
Copper	0·30 "	Silver	37·00 "

Purpose of Experimental Tests.—Experimental work was desired to determine methods of treatment to recover the values in the ore.

Scope of Experimental Work.—Methods of treatment outlined in the following description of the experimental work include: First, fine grinding followed by flotation of all the sulphides to obtain a concentrate carrying the precious metal values which could be economically marketed to the smelters. Second, fine grinding followed by selective flotation to obtain a lead concentrate carrying part of the precious metals, and a zinc concentrate carrying the remainder of the precious metal values. Third, the cyanidation of the flotation tailings to obtain further recoveries of the precious metal values.

Experimental Tests.—The results of the experimental tests are given in tabulated form following:—

TESTS Nos. 1, 2, AND 3

FLOTATION OF ALL THE SULPHIDE MINERALS

This series of flotation tests was made to compare the results obtained in a lime, soda ash, and neutral pulp.

Test No. 1

Flotation was made in an alkaline pulp produced by the addition of lime. The ore was crushed in a ball mill to pass a 65-mesh screen. A screen test showed that 60 per cent passed through 200 mesh. The following reagents were added during the grinding:—

Lime	5·0 pounds per ton
Mixture of 50 per cent gas tar 50 per cent coal-tar creosote	0·5 pound per ton

A small amount of steam-distilled pine oil for frothing was added to flotation cell.

Test No. 2

The flotation was made in an alkaline pulp produced by the addition of soda ash. The same procedure and oils were used as in test No. 1. The flotation tailing was tabled.

Test No. 3

The flotation was made in a neutral pulp. The same procedure and oils were used as in tests Nos. 1 and 2. The flotation tailing was tabled.

Conclusions from Tests Nos. 1, 2, and 3.—As expected, the lime pulp caused the iron sulphides to report in the tailing and also interfered with the flotation. Using the soda ash pulp a high recovery of the zinc and lead was obtained, and the greater part of the iron sulphides reported with the lead and zinc concentrates. It will be noted that a much higher recovery of the silver and gold values was obtained than in test No. 1. The recoveries of lead and zinc were not so high in test No. 3 in which a neutral pulp was used, and a larger proportion of the iron sulphides reported in the tailing than in test No. 2. The results of the table tests on the flotation tailing of these three tests give some idea of the association of the values in the flotation tailing. One point of importance to note is that there is more silver in the tailing than there is combined zinc and copper minerals to account for its association being entirely with those minerals.

TESTS Nos. 4, 7, 9, AND 11, SELECTIVE FLOTATION TESTS

This is a series of selective flotation tests made to produce a lead concentrate and a zinc concentrate which could be marketed separately. No attempt was made to reclean the lead and zinc concentrates.

Test No. 4

The ore was ground in a ball mill to pass a 65-mesh screen. The following reagents were used.

Lead reagents—			
5·0	pounds per ton	soda ash	added to ball mill.
0·15	"	Z cake	added to ball mill.
0·30	"	sodium cyanide	added to cells.
0·05	"	steam-distilled pine oil	added to cells.
Zinc reagents—			
1·0	"	copper sulphate.	
0·25	"	XY mixture.	
0·25	"	No. 2 neutral creosote oil from Dominion Tar and Chemical Co.	

Test No. 7

The same procedure was followed as in test No. 4.

Lead reagents—		
5·0	pounds per ton	soda ash.
0·15	“	Z cake.
0·30	“	sodium cyanide.
0·05	“	pine oil.
Zinc reagents—		
1·0	“	copper sulphate.
0·25	“	YZ mixture.
0·25	“	No. 2 neutral creosote.

Test No. 9

The ore was ground as in the above tests.

Lead reagents—		
5·0	pounds per ton	soda ash.
0·01	“	QED
0·3	“	sodium cyanide.
0·05	“	pine oil.
Zinc reagents—		
1·0	“	copper sulphate.
0·25	“	YZ mixture.
0·25	“	No. 2 neutral creosote.

Test No. 11

The same procedure was followed as in the previous tests.

Lead reagents—		
4·0	pounds per ton	soda ash.
0·25	“	mixture of water-gas tar 50 per cent, coal-tar creosote 50 per cent.
0·2	“	sodium cyanide.
0·05	“	cresylic acid.
Zinc reagents—		
1·0	“	copper sulphate.
0·3	“	Barrett's No. 634.
0·05	“	pine oil.

Conclusions from Tests 4, 7, 9, and 11.—Tests Nos. 7 and 11 gave the best results. Test No. 4 was also good, the same reagents being used as in test No. 7. In test No. 9 for some unaccountable reason the lead and silver values were much lower than the head sample so that no definite conclusions can be drawn. However, there seems to be no advantage in using reagent QED in place of Z cake for the flotation of the lead. In test No. 11 oils were used to float the lead in place of Z cake, and a better selective action was obtained between the lead and zinc. Special note should be taken when comparing the grade of the lead and zinc concentrates in these tests that they were not recleaned. In practice these concentrates would be recleaned and the grade considerably improved. There is little doubt that a 50 per cent lead concentrate and a 40 per cent zinc concentrate could be obtained.

TESTS Nos. 5, 6, AND 8, FLOTATION TESTS WITH ADDITION OF SODIUM SULPHIDE

The following is a series of tests made to determine whether a high recovery of the silver values could be obtained by the addition of sodium sulphide.

Test No. 5

The ore was ground in a ball mill to pass 60 mesh with 5 pounds per ton sodium sulphide, and an attempt was made to obtain a product containing the silver values.

Reagents—
5·0 pounds per ton sodium sulphide (Canadian Electro-Products).
0·2 " Z cake.

It was noted that very little zinc was floating, so 0·2 pound per ton YZ mixture was added to bring it up.

Test No. 6

Same procedure was followed as in test No. 5, but in this test selective flotation of the lead and zinc was attempted.

Lead reagents—
5·0 pounds per ton sodium sulphide to ball mill.
0·2 " Z cake.........to ball mill.
0·3 " sodium cyanide to cells.
0·05 " pine oil.........to cells.
Zinc reagents—
1·0 " copper sulphate.
0·5 " No. 2 neutral creosote oil.

Concentrates 1 and 2 were taken off with the reagents used for the flotation of the lead and in the presence of cyanide. Concentrate 3 was taken off by the reagents used for the flotation of zinc after addition of copper sulphate.

Test No. 8

The procedure was that followed in tests Nos. 5 and 6. Selective flotation of the lead and zinc was not attempted.

Reagents—
0·75 pound per ton mixture of 60 per cent coal-tar creosote, 40 per cent coal tar.
0·05 " " steam-distilled pine oil.

Conclusions from Tests 5, 6 and 8.—The sodium sulphide instead of increasing the recovery of the silver had the effect of increasing the loss in the tailing. Comparing tests Nos. 5 and 8 it will be seen that although the grade of the concentrates differ, the recoveries are practically the same. The difference between the grade of concentrate is due to the use of oils in test No. 8 instead of Z cake as in test No. 5. Test No. 6 in which an attempt was made to selectively float a lead and zinc concentrate is worthy of careful consideration, as it throws considerable light on the association of the silver values. According to the results obtained in this test, at least a considerable portion of the silver must be present as a separate mineral and not associated with the other sulphides as it will be observed that the iron sulphides collected in No. 1 concentrate, but that only 5·5 per cent of the silver values reported there. The second concentrate consisted chiefly of lead values but they contained 18·3 per cent of the total silver, whereas the third concentrate contained 55 per cent of the silver values and assayed 344·8 ounces per ton.

Test No. 10—Table Concentration

This test was made on a small-size Wilfley table. A fair grade of concentrate was obtained, but the recovery of both the lead and silver values was poor.

TESTS Nos. 12 AND 13, FLOTATION TESTS WITH FINE GRINDING

These two tests were run to determine if the recovery of the silver could be increased by finer grinding.

Test No. 12

This is a selective flotation test for the production of a lead and zinc concentrate. The ore was ground to pass 100 mesh. The method of conducting the test was the same as in the previous selective tests:—

Lead reagents—
5·0 pounds per ton soda ash.
0·01 " QED
0·3 " sodium cyanide.

A lead concentrate was made and was recleaned, producing a lead middling product.

Zinc reagents—
1·0 pounds per ton copper sulphate.
0·2 " YZ mixture.
0·4 " No. 2 neutral creosote.

The concentrate was not recleaned as the amount was too small to work with.

Test No. 13

This was a straight flotation test to produce a concentrate containing the lead and zinc together with the silver and gold values:—

Reagents: 0·4 pound per ton coal-tar creosote (Dominion Tar & Chemical Co.)
0·4 " No. 2 neutral creosote " "

The acid creosote oil was used to float the lead, and the neutral creosote oil to float the zinc.

Conclusions from Tests Nos. 12 and 13.—Finer grinding did not improve the recovery of the silver and gold values in these tests, but the grinding may not have been carried far enough, because a screen test made on the tailing, given later, shows that the silver values are higher in the coarser products.

CYANIDE TESTS

J. S. Godard

Cyanidation of the Ore.—This series of tests was made on the crude ore ground to pass 100 mesh. The time of treatment was the only variable in this series.

Time: No. 1, 24 hours. No. 2, 30 hours. No. 3, 48 hours.
Details: 200 grammes ore, –150 mesh.
600 " water.
0·68 " sodium cyanide, 0·15 per cent solution.
0·5 " lime, 5 pounds per ton.

The above charge was used in each test.

Results:—

Test No.	Assay Silver oz/ton		Extraction per cent	Consumption lbs. /ton	
	Head	Tailing		Cyanide	Lime
1	37·09	22·19	40·2	5·60	2·90
2	37·09	22·09	40·4	4·92	3·05
3	37·09	21·80	41·5	4·74	3·25

Cyanide Tests Nos. 4, 5, and 6

Cyanidation of Flotation Tailing.—This series of tests was made on flotation tailing from test No. 11. The strength of solution was varied, but time kept constant at 48 hours. Strength of solution: No. 4, 0·05 per cent; No. 5, 0·10 per cent; No. 6, 0·20 per cent.

Time, 48 hours.
Details: 200 grammes ore, -150 mesh.
600 " water.

Results:—

Test No.	Ag. oz/ton		Au. oz/ton		Extraction Ag. per cent	Consumption lbs./ton	
	Heads	Tailing	Heads	Tailing		Cyanide	Lime
4	9·56	4·54	0·02	trace	52·5	1·7	1·8
5	9·56	4·08	0·02	trace	57·2	1·8	1·7
6	9·56	3·88	0·02	trace	59·4	2·8	1·5

Conclusions from Cyanide Tests Nos. 1-6.—The cyanidation of the crude ore was not successful, less than 50 per cent of the silver being extracted. The cyanidation of the flotation tailing was more successful from an extraction point of view. Over 50 per cent of the silver in the tailing was extracted, which in test No. 6 is 5·68 ounces per ton, or $3.86, with silver at 68 cents per ounce.

General Summary and Conclusions.—The following screen test was made on the flotation tailing from test No. 4 for the purpose of obtaining information on the condition of the remaining silver.

Screen Analysis on Tailing Test No. 4

Heads: Lead, 0·17%. Zinc, 0·18%. Silver, 7·68 oz/ton. Gold, 0·02 oz/ton.

Mesh	Weight		Analysis				Per cent of silver values
	Grms.	%	Silver	Gold	Lead	Zinc	
			oz/ton	oz/ton	%	%	
+65	3·5	0·5	17·12	....	....	....	0·1
+100	54·2	6·9	13·2	0·04	0·13	0·33	11·4
+150	132·5	16·9	11·03	0·03	0·18	0·32	23·3
+200	119·5	15·3	9·96	0·02	0·15	0·31	19·1
−200	471·5	60·4	6·11	0·01	0·16	0·10	46·1

The screen test indicated that the ore should be ground to at least 150-mesh.

The table tests made on the flotation tailing from tests Nos. 1, 2, and 3 showed that about 40 per cent of the silver remaining in the flotation tailing could be recovered, but the concentrate produced was rather low grade. It was necessary to cut considerable gangue with concentrate because the amount of concentrate in the feed was so small. In actual practice on a full-sized table a higher grade concentrate could be produced. The suggestion is made that this table concentrate, if not sufficiently high grade to ship, might be returned and reground with the feed, and pass again through the mill circuit.

There was not sufficient ore to continue the flotation tests to show which of the two methods of flotation would be the more satisfactory. The comparison of the two systems depends on the possibility of producing both a lead and zinc concentrate of marketable grade by selective flotation, and comparing the combined net value of the concentrate against the net value of a concentrate produced by straight flotation containing zinc in amounts which would subject it to heavy penalties by the smelter.

Cyanidation of the crude ore was not successful, only 40 per cent of the silver being extracted. Cyanidation of the flotation tailing gave a little higher recovery, approximating about 55 per cent of the silver in the flotation tailing. There was not sufficient ore to run a cumulative cyanide test to determine the tendency of the solutions to become fouled. It is possible that considerable fouling of the solution would take place which would result in increased cyanide consumption and precipitation difficulties. This test should be run before drawing any definite conclusions as to the advisability of cyaniding.

The series of tests conducted on this sample are purely of a preliminary nature, and it will be necessary to do more detailed work in order to determine the best flow-sheet for the ore. A sample of at least 500 pounds would be required for this work. It should represent accurately the milling grade of ore in the mine, and should be a fresh sample taken from all working-faces in proportion to the amount of material which will be mined from each face.

Results of Experimental Tests

Test No.	Product	Weight		Analysis						Per cent of values			
		Grms.	%	Lead %	Zinc %	Copper %	Iron %	Silver oz./ton	Gold oz./ton	Lead	Zinc	Silver	Gold
	Flotation of all the sulphides—												
1	Concentrate	97·2	9·7	23·7	9·33	....	....	259·65	0·35	91·1	54·7	65·8	48·6
	Tailing	900·0	90·2	0·25	0·83	....	....	14·50	0·04	8·9	45·3	34·2	51·4
2	Concentrate	124·7	12·5	18·7	12·37	....	....	275·45	0·35	94·7	89·8	86·6	71·0
	Tailing	873·0	87·5	0·15	0·20	....	....	6·04	0·02	5·3	10·2	13·4	29·0
3	Concentrate	104·0	10·4	21·30	11·46	....	....	264·35	0·35	89·2	71·5	74·1	57·2
	Tailing	897·5	89·6	0·30	0·53	....	....	10·73	0·03	10·8	28·5	25·9	42·8
	Table tests on flotation tailings—												
1	Table concentrate	42·0	8·1	2·56	9·43	....	....	94·68	0·20	58·5	82·2	43·2	30·2
	Table tailing	478·0	91·9	0·16	0·18	....	....	11·10	0·04	41·5	17·8	56·8	69·8
2	Table concentrate	67·0	13·8	0·42	0·8	....	....	18·08	0·04	40·3	29·7	30·2	22·7
	Table tailing	419·0	86·2	0·10	0·03	....	....	6·70	0·02	59·7	70·3	69·8	77·3
3	Table concentrate	54·0	12·9	1·13	4·05	....	....	40·8	0·08	56·3	77·9	39·3	27·8
	Table tailing	363·0	87·1	0·13	0·17	....	....	9·33	0·03	43·6	22·1	60·7	72·2
	Selective flotation tests—												
4	Lead concentrate	86·0	8·6	25·30	10·90	2·93	14·18	347·4	0·40	91·0	61·4	78·1	62·5
	Zinc concentrate	28·5	2·8	2·29	15·10	0·62	13·79	53·42	0·10	2·7	28·1	4·0	5·2
	Tailing	890·0	88·6	0·17	0·18	0·05	2·35	7·68	0·02	6·3	10·5	17·9	32·3
7	Lead concentrate	93·0	9·2	25·24	10·42	2·09	13·99	316·0	0·40	91·4	62·1	76·5	75·2
	Zinc concentrate	26·0	2·6	1·33	21·49	0·43	10·84	66·35	0·15	1·4	36·2	4·5	8·0
	Tailing	884·0	87·2	0·21	0·03	0·03	2·18	8·29	0·01	7·2	1·7	19·0	17·8
	Lead concentrate	41·4	4·1	24·35	10·75	2·70	12·51	304·4	0·40	76·2	26·5	43·2	38·5
	Zinc concentrate	87·5	8·7	1·97	13·97	0·83	18·36	109·0	0·20	13·1	73·0	32·7	41·0
	Tailing	878·0	87·2	0·16	0·01	0·06	2·19	7·99	0·01	10·7	0·5	24·1	20·5
	Lead concentrate	58·5	5·3	37·62	7·23	....	....	439·35	0·60	85·3	26·0	63·1	53·1
	Zinc concentrate	56·0	5·1	3·07	20·43	....	....	99·00	0·20	6·7	70·9	13·7	17·1
	Tailing	987·0	89·6	0·21	0·05	....	....	9·56	0·02	8·0	3·1	23·2	29·8

Results of Experimental Tests—Concluded

Test No.	Product	Weight		Analysis						Per cent of values			
		Grms.	%	Lead %	Zinc %	Copper %	Iron %	Silver oz/ton	Gold oz/ton	Lead	Zinc	Silver	Gold
	Flotation tests with addition of sodium sulphide—												
5	Concentrate	146·0	14·6	13·55	3·74	1·14	10·87	162·0	0·20	84·0	38·3	62·8	46·0
	Tailing	856·5	85·4	0·44	1·03	0·12	2·30	16·44	0·04	16·0	61·7	37·2	54·0
6	Concentrate No. 1	39·7	4·0	47·3	1·54	0·24	15·86	53·30	0·10	7·7	4·1	5·5	6·5
	Concentrate No. 2	40·7	4·1	38·55	1·74	0·78	8·70	171·50	0·20	64·1	4·7	18·3	12·9
	Concentrate No. 3	60·5	6·1	7·89	8·11	1·35	9·65	344·80	0·40	19·5	32·7	55·0	38·7
	Tailing	851·5	85·8	0·25	1·03	0·06	2·16	9·40	0·03	8·7	58·5	21·2	41·9
8	Concentrate	49·0	5·2	35·0	5·31	2·65	17·63	316·36	0·44	84·6	81·0	54·5	62·3
	Tailing	899·0	94·8	0·35	1·26	0·08	2·21	14·46	0·04	15·4	19·0	45·5	37·7
	Table concentration—												
10	Concentrate	96·7	3·3	40·72	8·98	1·49	15·28	228·74	0·30	57·2	22·2	19·8	13·9
	Middling	141·0	4·8	2·51	12·52	0·83	9·97	64·87	0·15	5·1	45·2	8·2	9·7
	Tailing	2,161·0	74·4	0·56	0·36	0·15	2·41	24·51	0·05	17·7	20·1	47·7	51·4
	Slimes	510·2	17·5	2·68	0·95	0·40	3·31	53·10	0·10	20·0	12·5	24·3	25·0
	Flotation tests—finer grinding—												
12	Lead concentrate	57·5	5·8	38·18	11·19	....	....	460·7	0·60	87·9	38·9	69·2	59·1
	Lead middling	56·7	5·7	2·00	4·51	....	....	50·0	0·10	4·4	15·1	7·5	9·8
	Zinc concentrate	34·7	3·5	2·40	20·38	....	....	77·85	0·15	3·2	43·4	7·2	9·0
	Tailing	851·1	85·1	0·13	0·04	....	....	7·16	0·015	4·4	2·1	16·1	22·1
13	Concentrate	142·0	14·2	18·14	10·99	....	....	230·30	0·30	94·5	98·1	83·7	71·2
	Tailing	858·0	85·8	0·17	0·04	....	....	7·44	0·02	5·5	1·9	16·3	28·8

Report No. 218

CONCENTRATION OF TITANITE (SPHENE) FROM PEGMATITE

C. S. Parsons

Shipment.—A shipment of 42 pounds of pegmatite rock containing titanite, or sphene, was submitted by Messrs. Spence and Cole of the Mines Branch, Ottawa.

Source of Shipment.—The shipment was obtained from a deposit located on lot 3, concession III, Alleyn township, Pontiac district, Quebec.

Analysis.—The sample assayed 5·3 per cent titanium oxide (TiO_2) contained in the mineral sphene.

Character of the Rock.—The sphene was associated with a considerable quantity of dark green hornblende and some calcite in a quartz pegmatite.

Experimental Tests

General Considerations.—The rock was tough and difficult to crush. Crushing costs would be the chief factor in the treatment. The reduction of the rock was therefore not carried any further than necessary, only to the point of separation of the sphene from the quartz-calcite-hornblende gangue. The sphene is more brittle than the gangue and consequently crushes finer, but in parting from the gangue especially the quartz, a thin shell of sphene is left attached. Fine grinding would be required to free this adhering shell, but this operation would not be economical. In view of the above considerations, the procedure followed was to rough out a clean tailing at as coarse a size as possible, so that any recrushing required to obtain a cleaner product would be done on a smaller amount of material.

General Description of Tests.—By crushing a sample to different degrees of fineness and carefully examining the sizes, it was determined that −3 mesh was the coarsest size from which there was any likelihood of obtaining a sufficiently clean tailing. The size −3+10 was jigged, and a final tailing and rougher concentrate made. The concentrate contained the greater part of the hornblende which has practically the same specific gravity as sphene. Considerable other gangue was also present. A concentrate and tailing were made from this −3+10 size. The concentrate was recrushed to 20 mesh and re-treated as described later. The −10+20-mesh material was jigged, and only a concentrate and final tailing were made. This concentrate was also recrushed. The −20-mesh material was tabled without further sizing. A final table tailing was made and a table concentrate which was added to the recrushed concentrate from the jig. All the recrushed concentrate was sized and recleaned on tables producing a sphene-hornblende product and a secondary tailing which was not further treated. The sphene-hornblende product was fed to an Ullrich magnetic separator and the hornblende was removed, leaving a sphene concentrate containing some quartz.

Results of Tests.—The weights of the sizes obtained by crushing to 3 mesh were as follows:—

Total weight of sample		42 lbs.
Assay		5·03 % TiO_2
Content		2·1126 lbs. TiO_2
−3+10 mesh	weight	30 lbs.
	Per cent total weight	71·44
−10+20 mesh	weight	6 lbs.
	Per cent total weight	14·28
−20 mesh	weight	6 lbs.
	Per cent total weight	14·28

Results of Jigging −3+10-mesh Material.—

Tailing	Weight	19 lbs.
	Assay	0·90% TiO_2
	Content	0·152 lbs. TiO_2

Summary.—45·2 per cent of the original sample of 42 pounds was eliminated as a final tailing containing 7·2 per cent of the total titanium oxide (TiO_2)

Results of Jigging −10+20-mesh Material.—

Tailing	Weight	3·48 lbs.
	Assay	0·50% TiO_2
	Content	0·0174 lbs. TiO_2

Summary.—8·28 per cent of the original sample of 42 pounds was eliminated as a final tailing, containing 0·82 per cent of total titanium oxide (TiO_2).

Results of Table Concentration −20-mesh Material.—

Tailing	Weight	2·88 lbs.
	Assay	1·0% TiO_2
	Content	0·0288 lbs. TiO_2

Summary.—6·85 per cent of the original sample was eliminated as a final tailing containing 1·365 per cent of the total TiO_2.

Recapitulation of Results

Product	Weight		Analysis TiO_2 %	Content lbs. TiO_2	Per cent of values
	lbs.	%			
Final concentrate	4·00	10·0	29·84	1·20	65·3
Final middling	1·75	4·4	7·61	0·133	7·3
Final tailing	25·50	63·7	0·60	0·196	10·7
Final secondary tailing	2·50	6·3	2·0	0·05	2·7
Final hornblende product	4·25	10·6	1·1	0·046	2·5
−100 mesh product	2·00	5·0	10·58	0·212	11·5
Total	40·00	100·0	4·6	1·837	100·0

Summary.—The results of the above concentration tests show a concentration ratio of 10 of rock to 1 of concentrate. The concentrates averaged 30 per cent TiO_2 with a recovery of 65 per cent of the titanium content of the rock. The recovery could probably be increased to 70 or 75 per cent by further treatment of the middling product, and concentration of the fines.

Report No. 219

EXPERIMENTAL TESTS ON CADWALLADER CREEK MILL TAILINGS

J. S. Godard

Shipment.—A shipment of two bags weighing 96 pounds, was received on October 13, 1924, from A. W. Davis, Resident Engineer, Kamloops, B.C. One bag was marked "Coronation mill tailings" and weighed 31 pounds; the other "Lorne mill tailings" and weighed 65 pounds.

Purpose of Experimental Tests.—Experimental tests were conducted to determine a method for the recovery of the values remaining in the tailings.

Sampling and Analysis.—A sample was taken from each lot of the tailings as received, using a riffle sampler. Assays of the samples showed them to contain:—

Coronation Tailing—

Gold	0·30 ounce per ton
Silver	0·08 " "
Arsenic	0·38 per cent
Iron	3·71 "

Lorne Tailing—

Gold	0·08 ounce per ton
Silver	
Arsenic	0·43 per cent
Iron	4·19 "

Experimental Tests.—Table concentration, amalgamation, and cyanide tests were made to determine the recovery of the values by these methods.

CORONATION TAILING

Table Concentration Test.—A 3,000-gramme sample of the tailing as received, which showed by screen tests to be 48-mesh material was concentrated on a small Wilfley table with the following results:—

Product	Weight per cent	Gold oz/ton	Per cent of values
Concentrate	0·90	3·56	16·7
Middling	1·10	0·20	1·1
Tailing	98·00	0·16	82·2

Amalgamation Test.—A 1,000-gramme sample of the tailing as received was amalgamated for two hours in a pebble jar using 10 per cent by weight of mercury and a pulp density of 1:1. A screen analysis was made of the amalgamation tailing which showed the following results:—

Product	Weight per cent	Gold oz/ton	Per cent of values
+48 −48+65	8·8	0·10	3·6
−65+100	19·3	0·10	8·0
−100+150	25·4	0·14	14·9
−150+200	11·4	0·20	9·5
−200	35·1	0·44	64·0

Head sample	0·30 ounce per ton
Amalgamation tailing	0·24 " "
Recovery by amalgamation	20 per cent

Cyanidation Tests.—A series of bottle-type agitation cyanide tests was made, the results of which are given in the following table:—

Test No.	4	5	6	7
Weight, grms.	300	300	300	300
Mesh	48	48	48	−150
Water, c.c.	300	600	900	900
Potassium cyanide, per cent	0·2	0·2	0·2	0·2
Lime, pounds per ton	4	4	4	4
Heads, gold, oz/ton	0·30	0·30	0·30	0·30
Tailing, " "	0·06	0·04	0·04	0·03
Recovery per cent	80·0	86·7	86·7	90·0
Cyanide consumption, pounds per ton	0·62	0·73	0·42	2·4
Lime consumption, pounds per ton	3·4	2·9	2·7	3·4
Time, hours	24	24	24	24

Summary of Tests on Coronation Tailing.—16·7 per cent of the gold values was recovered in a table concentrate representing by weight 1 per cent of the feed and assaying 3·56 ounces per ton gold, 15·52 per cent arsenic, 24·96 per cent iron. Twenty per cent of the gold values was recovered by amalgamation. The test shows that the greater proportion of the values is in the −200-mesh material and is not recoverable by amalgamation. Ninety per cent of the gold values was recovered by cyanidation from the tailing ground to 150 mesh, using a pulp dilution of 3:1 and 0·20 per cent potassium cyanide solution. Eighty-seven per cent of the values was recovered without further grinding and with a low cyanide consumption.

LORNE TAILING

Table Concentration Test.—A 2,655-gramme sample of the tailing as received, which showed by screen test to be 28-mesh material, was concentrated on a small Wilfley table with the following results:—

Product	Weight per cent	Gold oz/ton	Per cent of values
Concentrate	2·8	1·12	51·6
Middling	15·8	0·04	9·7
Tailing	81·4	0·03	38·7

Amalgamation Test.—A 1042-gramme sample of the tailing as received was amalgamated in a pebble jar for two hours, using 100 grammes of mercury and a pulp density of 1:1. A screen analysis of the amalgamation tailing gave the following:—

Product	Weight per cent	Gold oz/ton	Per cent of values
+48 mesh	22·7	0·04	16·2
−48+65 mesh	10·1	0·04	7·1
−65+100 mesh	25·9	0·02	9·3
−100+150 mesh	10·9	0·06	11·6
−150+200 mesh	8·4	0·06	8·9
−200 mesh	22·0	0·12	46·9

Head sample..........0·08 ounce per ton
Amalgamation tailing..........0·056 " "
Recovery by amalgamation..........30·0 per cent

Cyanidation Tests.—A series of bottle-type agitation cyanide tests was made, the results of which are given in the following table:—

Test No.	4	5	6
Weight, grms	290	309	305
Mesh	−150	28	28
Water, c.c	870	927	915
Potassium cyanide, per cent	0·20	0·20	0·30
Lime, pounds per ton	4	4	4
Heads, gold, oz/ton	0·08	0·08	0·08
Tailing, " "	0·04	0·02	0·03
Recovery, per cent	50	75	62·5
Cyanide consumption, pounds per ton	2·3	0·71	1·10
Lime consumption, pounds per ton	2·7	2·5	1·9

Summary of Tests on Lorne Tailings.—51·6 per cent of the gold values was recovered in a table concentrate representing by weight 28 per cent of the feed and assaying 1·12 ounces per ton gold, 8·12 per cent arsenic, 19·79 per cent iron. Thirty per cent of the gold values was recovered by amalgamation. The test shows that about 50 per cent of the values is in the −200-mesh material on which a poor recovery is made by amalgamation. Seventy-five per cent of the gold values was recovered by cyanidation of the tailings as received, using 0·20 per cent potassium cyanide solution. The cyanide consumption was less than one pound per ton.

CONCLUSIONS

The tests conducted on the mill tailings show that cyanidation gives the highest recovery of the values. As cyanidation was used for the recovery of the values from the Cadwallader Creek ore, it would seem that increased recoveries could be expected by more careful attention to the details of this practice, especially in the case of the Coronation ore.

Report No. 220

CONCENTRATION OF THE LEAD-ZINC ORE OF THE READER MINE, CALUMET ISLAND, QUEBEC

W. B. Timm[1] and C. S. Parsons

The favourable position of lead in the metal markets of the world has resulted in an active search for new lead properties, and the re-opening of idle properties. Hence a visit was made by the writer on November 13, 1924, to the Reader mine on Calumet island.

Situation.—The property comprises lots 9, 10, 11, and 12, range IV, township of Calumet, district of Pontiac in the province of Quebec. It is situated in the southern part of the island, one-half mile from the Roche Fendue channel of the Ottawa river, three miles from the village of Bryson and the recent power development at Calumet falls, and six miles from Campbells Bay on the Ottawa-Waltham branch of the Canadian Pacific railway.

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[1] Introductory by W. B. Timm.

Operation.—The property was operated intermittently from 1896 to 1913 by the Grand Calumet Mining Co., Ltd., the Calumet Metals Co., and the Calumet Zinc and Lead Co. In 1898, 1,100 tons of high-grade ore was shipped to Belgium. In 1910 a small concentrating plant was built and trial shipments of lead and zinc concentrates were made to the Balbach smelter, Newark, N.J. In 1912 a 150-ton concentrator was built, equipped with jaw crusher, coarse rolls, screens, fine rolls, jigs, Huntingdon mill, Wilfley and Overstrom tables. This mill after being operated for only a few months, was closed down and was later destroyed by fire.

Occurrence.—There are two main zones, striking northwest, 1,400 feet apart at the southern exposures and 630 feet apart at the northern exposures. The western zone was opened up by prospect pits and shafts for 2,100 feet along the strike, and the eastern zone for 1,800 feet along the strike.

Development.—Development consists of four shafts varying in depth from 74 feet to 146 feet; several smaller shafts varying in depth from 17 feet to 35 feet, and numerous prospect pits. The total amount of drifting is approximately 500 feet.

Concentration Products from Milling Operations.—A marketable lead product was made containing on the average 65 per cent lead, 8 per cent zinc, 0·5 per cent copper, and 87 ounces of silver per ton. A low-grade zinc product was obtained containing on the average 3·5 per cent lead, 28 per cent zinc, 18 ounces silver per ton, and of high iron content. The tailing losses were high. Before the application of oil flotation, lead-zinc-silver ores possessing the characteristics of this ore presented a difficult problem in the matter of their concentration and separation of the minerals to obtain marketable products. The selective flotation of lead-zinc ores has been the solution to the problem in a great many cases and the results of the experimental work conducted by C. S. Parsons on this ore, show that marketable products with high recoveries can be obtained, so that one of the chief difficulties of the earlier operations has been overcome. Comparatively cheap power is also now available from the recent developments at Calumet falls, 3 miles distant. C. S. Parsons' report on the experimental work follows:—

Shipment for Experimental Purposes.—A shipment of four samples submitted by W. B. Timm, Mines Branch, Ottawa, was received at the Ore Testing Laboratories, November 15. The shipment comprised: a sample taken from the stock piles, weight 25 pounds, and representative of the ore mined; a sample of selected high-grade ore, for analysis, to determine if a marketable product could be sorted from the ore for direct shipment to the smelter; a sample of supposed ground mill feed for analysis, to obtain an idea of the assay value of the feed to the concentrator destroyed by fire; and a sample of the old concentrator tailings. From the analyses it will be seen that the latter two samples are not representative ones, especially that of the mill tailings, but they show that there were high tailing losses with the former methods of concentration.

Analyses of Samples.—The samples were designated Nos. 1, 2, 3 and 4, and gave on analyses the following values:—

Description of sample	Lead %	Zinc %	Iron %	Silver oz/ton	Gold oz/ton
No. 1—Stock pile ore	3·35	7·08	4·82	9·50	....
No. 2—Selected high-grade ore	16·52	28·26	8·35	39·78	0·02
No. 3—Supposed ground mill feed	2·12	10·04	8·93	3·15	....
No. 4—Mill tailings	1·77	12·15	8·16	8·20	....

Purpose of Experimental Tests.—The object in conducting experimental tests was to determine whether the ore could be concentrated by the improved methods of concentration applicable to this type of ore, with the production of marketable lead and zinc products and with high recoveries of the values. When the mine was last operated the only method available was gravity concentration on jigs and tables. The recovery of the lead and silver values was low, due to the tendency of the galena to slime during crushing operations. The iron pyrite and zinc blende being of practically the same specific gravity could not be separated by gravity means, and reported as a zinc-iron middling too low in zinc to be of market value. In order to obtain a marketable zinc product from this zinc-iron middling roasting was required to change the pyrite to the magnetic form, followed by magnetic separation. Such a method was not only costly but was hard to control, and with the exception of a few instances was not successful from an economic standpoint.

EXPERIMENTAL TESTS

The tests were conducted on sample No. 1, the stock pile ore, as oxidization had not taken place to any appreciable extent. A table test was made on the ore crushed to 20 mesh to determine the results from gravity concentration. Three flotation tests were conducted on the ore ground to varying degrees of fineness, as given in the screen tests following, to determine the results from selective flotation. The reagents used in making the flotation tests are given in a separate table, and the results of the table and flotation tests are also given in tabulated form.

Screen Tests of Flotation Tailing Showing Fineness of Grinding

Mesh	Test No. 1		Test No. 2		Test No. 3	
	Grms.	%	Grms.	%	Grms.	%
+48	2·5	0·5	0·1	0·0	0·3	0·1
−48+65	18·2	3·6	4·0	0·8	6·5	1·3
−65+100	77·4	15·5	19·6	3·9	39·5	7·9
−100+150	86·0	17·2	53·0	10·6	45·5	9·1
−150+200	62·5	12·7	62·0	12·4	71·5	14·3
−200	252·4	50·5	361·3	72·3	336·7	67·3
Totals	500·0	100·0	500·0	100·0	500·0	100·0

Reagents used in Flotation Tests.—

Test No. 1—1,000 grammes ore

Lead reagents:	5·0	pounds per ton	sodium carbonate
	0·2	"	Z cake
	0·4	"	sodium cyanide
	0·75	"	cresylic acid
Zinc reagents:	2·0	"	copper sulphate
	0·2	"	YZ mixture
	0·1	"	TT mixture
	0·01	"	pine oil

Test No. 2—1,000 grammes ore

Lead reagents:	5·0	pounds per ton	sodium carbonate
	0·1	"	QED
	0·4	"	sodium cyanide
	0·75	"	cresylic acid
Zinc reagents:	2·0	"	copper sulphate
	0·2	"	YZ mixture
	0·15	"	TT mixture
	0·01	"	pine oil

Test No. 3—1,000 grammes ore

Lead reagents:	5·0	pounds per ton	sodium carbonate
	0·2	"	Z cake
	0·25	"	sodium cyanide
	0·75	"	cresylic acid
Zinc reagents:	2·0	"	copper sulphate
	0·5	"	Neutral creosote oil No. 2. Dominion Tar & Chemical Co.
	0·01	"	potassium xanthate

SUMMARY OF RESULTS

The results from the table test show that a high-grade lead concentrate can be obtained, but with a low recovery of the lead and silver values in the concentrate. A low-grade zinc-iron middling was obtained. This test showed the difficulty of concentrating an ore of this type by gravity methods.

The flotation tests conducted were only of a preliminary nature but the separations were so readily obtained with little attention to manipulation that equally as good results, if not better, should be obtained in actual milling operations. Both the grade of the lead and zinc concentrates and the recoveries of these metals were good as well as the recovery of the silver values. One very important point has been determined, namely that fine grinding increases the proportion of silver values in the lead concentrate as can be seen by comparing the distribution of silver values in test No. 1 with that of tests Nos. 2 and 3. The screen tests show that much finer grinding was done in the latter tests. The results of fine grinding were that the silver content of the zinc concentrate was reduced from 33 ounces to 4·5 ounces per ton, thereby increasing the recovery of the silver values in the lead products from 60 to 87 per cent. This increased recovery of silver values with the lead concentrate is a very important item from an economic standpoint in the marketing of the concentrates.

CONCLUSIONS

It is reasonably safe to assume from the results of the small-scale experimental tests conducted that the ore of the Reader mine can be successfully concentrated by selective flotation with the production of high-grade lead and zinc products, and with good recoveries of the lead, zinc, and silver values.

Results of Table and Flotation Tests

Test No.	Product	Weight		Assays			Per cent of values		
		Grms.	%	Pb. %	Zn. %	Ag. oz/ton	Pb.	Zn.	Ag.
	Table test—								
1	Concentrate	50·5	2·5	63·68	6·58	139·00	48·4	2·5	33·3
	Middling	610·5	30·5	2·71	14·31	9·38	25·1	65·4	27·5
	Tailing	1,339·0	67·0	1·29	3·34	6·10	26·5	32·1	39·2
	Flotation tests—								
1	Lead concentrate	43·7	4·37	55·27	2·10	152·5	69·4	1·4	56·6
	Lead middling	77·2	7·72	11·20	7·63	5·5	24·8	8·3	3·6
	Zinc concentrate	110·7	11·07	0·50	52·21	33·1	1·6	84·3	31·1
	Zinc middling	53·0	5·30	0·62	4·92	6·56	0·9	3·8	3·0
	Tailing	715·9	71·54	0·16	0·21	0·94	3·3	2·2	5·7
2	Lead concentrate	95·0	9·5	34·66	4·10	91·7	89·2	5·5	79·4
	Zinc concentrate	162·0	16·2	1·48	38·70	8·8	6·8	88·8	13·1
	Tailing	748·0	74·8	0·20	0·54	1·1	4·0	5·7	7·5
3	Lead concentrate	59·7	6·0	46·54	2·74	134·0	80·9	2·2	72·2
	Lead middling	59·0	5·9	9·02	6·88	28·5	15·5	5·6	15·2
	Zinc concentrate	108·7	10·9	0·32	55·32	4·5	1·0	82·7	4·4
	Zinc middling	35·5	3·5	0·80	15·16	7·44	0·8	7·4	2·4
	Tailing	737·1	73·7	0·08	0·21	0·88	1·7	2·1	5·8

III

REPORTS OF INVESTIGATIONS: HYDROMETALLURGICAL LABORATORY

THE HYDROMETALLURGICAL TREATMENT OF IRON SULPHIDE ORES FOR THE PRODUCTION OF ELECTROLYTIC IRON AND THE RECOVERY OF SULPHUR AND OTHER METALS AS BY-PRODUCTS

R. J. Traill and W. R. McClelland

Introductory.—The preliminary experiments on this investigation, described in the report of 1923, showed results that were sufficiently promising to warrant a continuation of the investigation on a somewhat larger scale. Additional laboratory space was provided by the addition of a temporary laboratory, 40 by 18 feet, to the present chemical laboratories. This structure was ready for occupancy in March and the equipping of the laboratory with the necessary apparatus which was mostly of a special nature, occupied several months.

To assist in the carrying out of the larger tests additional technical help was required and Mr. W. R. McClelland was appointed in the capacity of junior chemist (temporary).

Many interesting problems have arisen during this investigation which cannot be covered nor incorporated in a report of this kind, and the information submitted in this report has mainly to do with the results of leaching on a larger scale under various conditions, and a general description of the apparatus used in this investigation.

Materials Used.—Because of the decided corrosive action of ferric chloride, it became necessary to make a study of various materials to find what would be the most satisfactory material in withstanding this action. Wood is quite readily decomposed by the action of hot ferric chloride. Enamelled ware withstands the action for a time, but has a tendency to crack at the temperature used and in addition the enamel is gradually worn off by the friction of the ore particles. Bitumen and asphalt coverings softened at the high temperatures. The common metals and alloys are readily dissolved by the ferric chloride. Rubber, hard or soft, is fairly satisfactory at temperatures below 50° C. but does not stand up at higher temperatures. Sulphur-coke, sulphur-asbestos and sulphur-sand mixtures withstand the action quite well but have a tendency to crack on sudden heating or cooling. Concrete impregnated with sulphur or iron oxide, seems to be the most suitable material found so far.

Two materials for construction of tanks or cells, which have proved entirely satisfactory, are acid-proof stoneware, and "ebony asbestos wood." The latter is an asbestos fibre with a binding cement, densely impregnated with some asphaltic compound. The high initial cost of the two materials would prohibit their use in a commercial-sized plant, but for the experimental work of the laboratory they were considered to be the most suitable, and have, therefore, been adopted for tanks, filters, and cells. For piping, it would appear that stoneware, bakelite, and cement-sulphur, are most satisfactory where high temperatures are used, but in the laboratory installation hard rubber has been used, as the temperature of the solutions to be transferred will be kept below 50° C. A hard rubber pump has also been used, but in commercial practice a stoneware pump would be better suited for handling the hot liquors.

Description and Plan of Layout.—The following is a brief description of the apparatus at present installed, with flow-sheet.

The leaching equipment consists of acid-proof stoneware tanks of the well-known kettle type, having a capacity of 15 gallons, and fitted with stoneware agitating-paddles which are driven by pulleys at about 80 r.p.m. The solution is heated by a special design of wire resistance heater, made in the laboratory, the wire being enclosed in pyrex glass tubing. A rheostat is used to vary the current consumption from 500 to 3,000 watts. The leach is discharged from the heating tank into stoneware suction filters fitted with alundum filter-discs, one inch thick, the hot solution filtering readily with the aid of suction from a small Crowell pump. The filtered liquor is passed to the cementation tanks. These tanks are 18 inches square and made of ebony asbestos wood. In this tank the copper is precipitated, with agitation, by sponge iron (made from hematite ore and charcoal). Any lead present may also be separated in this tank. The liquor

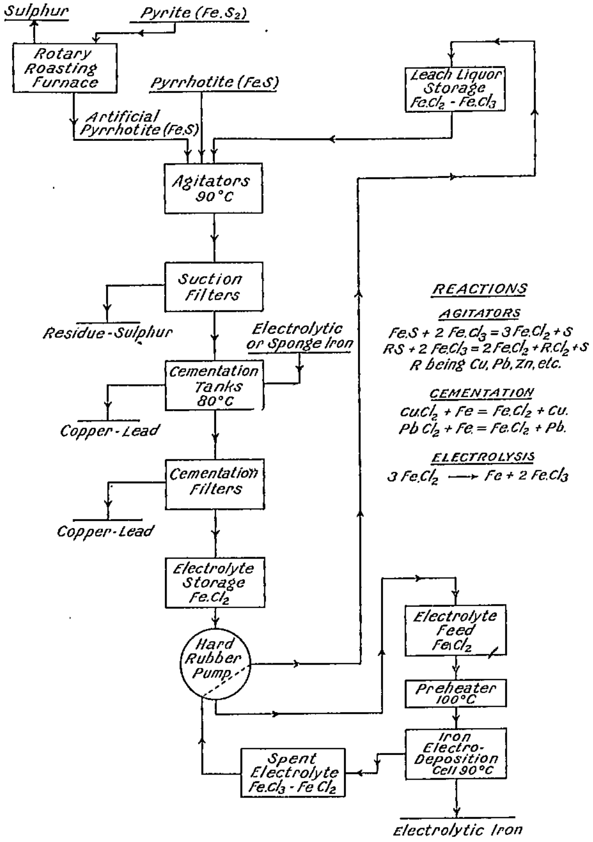

Figure 4. Flow-sheet, leaching of sulphide ores of iron and subsequent production of electrolytic iron.

thus purified is run through an ebony asbestos wood filter-box, 12 by 18 inches, having an alundum plate as the filtering medium. The filtered liquor is passed to the electrolyte storage tanks which have a capacity of 75 gallons, and are made of acid-proof stoneware.

When sufficient electrolyte has accumulated, it is pumped to the electrolyte feed tank, similar in capacity and design to the storage tanks. The electrolyte is now fed through a preheater to the cell. The preheater is simply a steel tube insulated and wound with a wire resistance sufficient to take about 1,500 watts. The hot electrolyte passes into the cell. The mandrel cell is built of ebony asbestos wood, with an asbestos diaphragm separating the anolyte and catholyte. The cathode is a steel mandrel 12 inches long by 3⅘ inches diameter, and the anode consists of graphite plates, the side plates being about 9 by 12 inches and the bottom plate 7 by 12 inches. The speed of the mandrel can be varied from 125 to 450 r.p.m. In the catholyte compartment a hydrostatic head of about ½ inch is maintained and the anolyte compartment is tapped at two levels for drawing off the ferric chloride formed during electrolysis. The ferric chloride thus obtained is returned by means of the hard rubber pump and piping to the leach-liquor storage tanks ready for use in leaching fresh ore.

Leaching.—The following is the general procedure carried out in the leaching process:—

The leach liquor is prepared from commercial grade ferric chloride and ferrous chloride obtained in previous leaching tests, or from the electrolytic cell, and which has been purified of copper, but not purified of zinc or calcium salts which are present in amounts too small to be separated readily. Fifty litres of leach liquor having a concentration of 64 to 74 grammes per litre ferric chloride, and a similar concentration of ferrous chloride, is prepared and run into the agitation tank, heated to about 60 to 70° C., and the ore charge, about 10 pounds, added in accordance with the particular conditions pertaining to each test. Samples of the liquor are withdrawn every 20 minutes for determination of ferrous iron, the leaching temperature being kept around 95° C. until the liquor is reduced to the ferrous condition. The reduced liquor is then discharged into the filter. The gangue and sulphur are separated, washed, dried, weighed, and analysed. Ferrous and ferric (if present) iron are determined in the filtered liquor.

Table I shows results of several leaching tests on a 50-litre scale, under various conditions; such as ore excess, concentration of ferric in leach liquor, and rate of ore feed to leach liquor. It may be observed here that the shipment of Smith pyrrhotite used in these tests contained about 10 per cent of its iron in the form of pyrite which is only slightly attacked by the ferric chloride. Analysis of the ore showed 41·6 per cent iron present as pyrrhotite, and 9·8 per cent present as pyrite, copper 0·83 per cent, and zinc 1·03 per cent, with some lime also present. The figures given in the column "per cent extraction of iron calculated from analysis of residue" are not strictly correct, as a certain amount of hydrolysis occurs during the leaching operation, and there is also a certain amount of iron precipitated by the lime in the ore, the amounts of iron so occurring in the residue being calculated as unleached iron. The actual extraction of iron, therefore, should be somewhat higher, probably by 2 to 5 per cent.

TABLE I

Leaching Tests

Ore—Smith Pyrrhotite (—200 mesh). Volume—50 litres. Temp.—93°–96° C.

Test No.	Weight of ore charge grammes	Excess of ore based on total iron present in the ore per cent	Time of agitation hours	Concentration of leach liquor in grammes of iron per litre		Rate of ore feed	Weight of residue grammes	Analysis of leached liquor in grammes of iron per litre		Extraction of copper per cent	Extraction of zinc per cent	Extraction of soluble iron calculated from analysis of residue per cent	Extraction of total iron calculated from analysis of residue per cent	Reduction of ferric iron in leach liquor per cent
				Fe″	Fe‴			Fe″	Fe‴					
X-2	4,042	25·0	14	77·3	66·4	All ore added at start.	2,048	162·5	8·4	91·0	99·5	75·2	69·2	87·4
X-3	4,400	38·3	9	58·0	65·9	All ore added at start, 13·3% excess added end of first day.	2,948	148·7	1·9	83·8	87·7	63·1	59·4	97·2
X-4	4,421	41·0	8	66·4	64·4	Ore added 10-min. intervals, 16% excess at fifth hour.	2,408	161·0	trace	86·1	94·7	75·5	67·7	100·0
X-5	4,000	25·0	7	67·9	65·9	Ore added 20-min. intervals.	2,129	160·1	1·4	88·0	93·8	79·4	69·5	97·8
X-6	4,550	25·0	7⅓	65·8	74·7	20-min. intervals last addition at fifth hour.	2,636	172·0	3·5	87·6	91·5	75·2	68·0	95·3
X-7	4,550	25·0	7⅙	68·4	74·4	20-min. intervals......	2,636	176·6	2·5	88·0	....	77·7	67·8	96·6

Test X2, it will be observed, was very slow in leaching, requiring two days, and even then the reduction in solution was only 87·4 per cent. The ferric chloride used in this test was of different manufacture than that used in the tests that follow. This particular ferric chloride shipment had given similar trouble in smaller tests and the only reason that can be given in explanation is the fact that it contained ferric chloride in some colloidal form, the presence of which was discovered in making a dialysis test. The total ore charge was added at the one time. Filtration was extremely slow.

Text X3

This test was run using a new shipment of ferric chloride. The initial ore charge was 4,000 grammes, and after 5 hours' agitation the reaction slowed up to about 2 grammes of iron per hour dissolving, so an additional 400 grammes of ore was added at the eighth hour to complete the reduction of the liquor. The low iron extraction in this test is therefore accountable to this 400-gramme addition, less than 100 grammes of this addition reacting to complete reduction. The filtration in this test was faster than the previous test, but occupied several hours, in which time the reduced solution became slightly oxidized, again showing a small amount of ferric in the filtered leach. The reduction time in this test is an improvement over test X2, due, no doubt, to the new ferric chloride which was free from colloidal material. There is a pronounced slowing up of the reaction, however, after 4 or 5 hours' agitation, which may be due to the ore particles becoming coated with sulphur set free in the reaction and thereby protected from the action of the ferric chloride. To overcome this effect it was decided to add the ore at short intervals. The following tests show the results of this procedure.

Test X4

The ore was added in about 500-gramme lots every 10 minutes until 3,921 grammes (25 per cent excess) had been added. The reaction showed an increase in reduction rate, but slowed up from the fourth to the fifth hour. An additional 500 grammes was added after 6 hours' agitation, resulting in complete reduction of solution in the next two hours. In comparison with test X3, this test shows a gain in iron extraction, and the filtration rate was also improved, the time required being 1½ hours.

Test X5

In this test the ore was added at 20-minute intervals, approximately 500 grammes at a time. The rate of reduction showed a slight increase over test X4, and although the reaction slowed up a little after 5 hours' agitation it was more progressive on the whole. The test was filtered after 7 hours' agitation, filtration being completed in 40 minutes. Figure 5 shows a curve representing the solution of iron on a time basis.

Test X6

The ore was added in 500-gramme amounts every 20 minutes until 4,000 grammes had been added, the last addition being withheld until the slowing up period of the reaction had occurred—after about 5 hours' agitation. The fresh ore added at this stage was readily reacted on as observed by an increase in the rate of reduction of the liquor. The higher ferric concentration of leach liquor as used in this test, apparently does not materially offset the time factor in leaching. The solution of iron curve is shown in Figure 5 and compares closely with X5. The effect of the addition of fresh ore is noticeable at the 360-minute period.

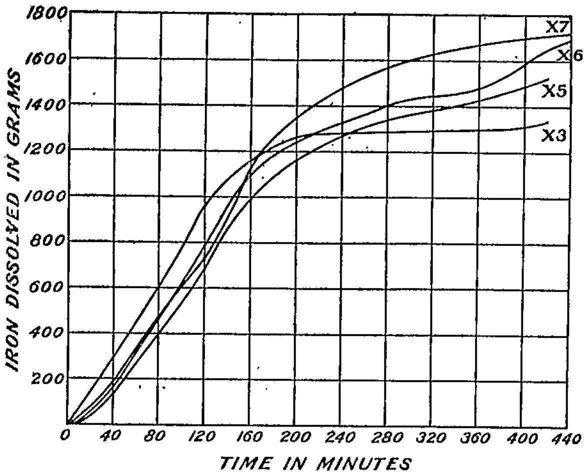

Figure 5.

Test X7

This test was run as a comparative test with test X6, using the same concentration of leach liquor (ferric) and same ore charge, but adding the charge at 20-minute intervals until all had been added. The reaction was progressive until the sixth hour; from the third hour to the sixth the test showed a higher rate of extraction than test X6. The solution of iron curve is shown in Figure 5.

CONCLUSIONS

The solution rate of iron compares favourably in all of these tests for the first three hours. The reaction tends to slow up at about this period, either because the leach liquor becomes depleted in ferric, or because the liberated sulphur adheres to the ore particles, thereby protecting them from the action of the ferric chloride. From the tests made, it would appear that the latter cause has the greater effect. The iron extractions are not so high as had been expected, but this is explained somewhat by the fact that the ore contains 10 per cent iron in the form of pyrite which scarcely reacts at all with ferric chloride.

The copper and zinc extractions are fairly satisfactory.

ROASTING OF PYRITE

In the Report of Investigations, 1923, under test No. 182, experiment showed that pyrite could be converted into an artificial pyrrhotite by roasting in a non-oxidizing atmosphere at about 750° C, one atom of recoverable sulphur being set free. A furnace was designed in an attempt to roast the ore and recover the sulphur. The furnace was of the rotary type, having a tube 4 feet long by 4 inches inside diameter, as roasting-chamber. Special bearing castings were fitted to each end of the tube with openings for feed, sulphur-gas exit, and roasted-product discharge. Provision was also made for circulation of the non-oxidizing gas through the furnace, this gas consisting mostly of sulphur dioxide (SO_2) formed in the initial stage of roasting. The furnace was heated by resistance wire elements of the muffle-furnace type, the middle section of the tube to the length of 2 feet being heated, the temperature being controlled by a rheostat. The furnace as designed was air- and gas-tight; the sulphur condenser and the roasted product receptacle being closed, the feed hopper closed by the ore, and the bearing castings in the tube furnished with gaskets. It was found that ordinary iron or steel tubes were quite unsuitable, the sulphur gas set free from the ore combining with the iron of the tube formed an iron sulphide which built up and destroyed the tube. Alundum refractory linings were tried, but, while withstanding the action of the sulphur gas, they had the tendency to crack badly. Various alloy tubes were tried, but they were all readily attacked by the sulphur gas. At present a calorized steel tube is in use and so far has proven quite satisfactory.

The drawing off and condensation of the sulphur gas is a problem that has not been satisfactorily solved, but results obtained are at least encouraging. To convert the pyrite completely to the monosulphide soluble form, it has been necessary to pass the charge through the furnace twice and occasionally three times. With a longer section of the tube heated it is quite probable that roasting would be complete in the one pass.

Test No. 182

The ore used in the roasting and subsequent leaching tests contains about 85 per cent pyrite (41·5 per cent Fe), 4·5 per cent chalcopyrite (1·6 per cent Cu), with a small amount of lead sulphide. The charge usually 30 pounds, is fed to the furnace at the rate of 15 to 20 pounds per hour, the temperature being from 750 to 800° C. After the first pass through the furnace a product analysing from 20 to 28 per cent soluble ferrous iron is obtained, which on the second pass is increased to 40 to 48 per cent soluble iron, and the third pass gives a product 48 to 52 per cent soluble iron. The sulphur recovery usually ranges from 50 to 65 per cent. Some sulphur is of course oxidized at the start of the roasting, and the ore contains ferric oxide which combines with some sulphur becoming ferrous sulphide, and sulphur also condenses in cooler parts of the furnace. The roasted product is quite grey in colour, porous in appearance, and is easily powdered by rubbing between the fingers. This last feature suggested the dispensing with further grinding before leaching, and experiment has shown that the material as obtained from the furnace is readily soluble. Lack of further grinding has, moreover, reduced to a minimum the formation of slime which had formerly been somewhat troublesome in filtration.

Leaching Roasted Pyrite.—The leach liquor was prepared from commercial ferric chloride and ferrous chloride made by treating ferric chloride with roasted pyrite and cementing out the copper and most of the lead. Leaching was carried out in 50-litre volumes, the necessary ore charge being added as outlined in the following tests. The roasted pyrite averaged 49 per cent iron, 1·8 per cent copper, and 0·7 per cent lead, with 1 to 2 per cent iron in an insoluble form, probably unaltered pyrite. Tests were made every 20 minutes to observe rate of reduction and solution of ore. Table II shows results obtained on roasted pyrite.

REMARKS

Test Y1

This test was run to obtain a supply of ferrous liquor for subsequent tests.

Test Y2

This test was run in a manner similar to the pyrrhotite tests, namely adding the ore at intervals of 20 minutes. The theoretical amount of ore required was thus added. The reduction proceeded at an even rate, but somewhat slower than with pyrrhotite. The additional excess ore was added after 5 hours' agitation, the reduction rate increasing a little thereby. It will be noticed from the curve, Figure 6, that the reaction was somewhat slow in starting, as up to the 40-minute period very little increase in ferrous iron was noted. The leach filtered in 15 minutes, being practically free from slime and therefore much easier to handle.

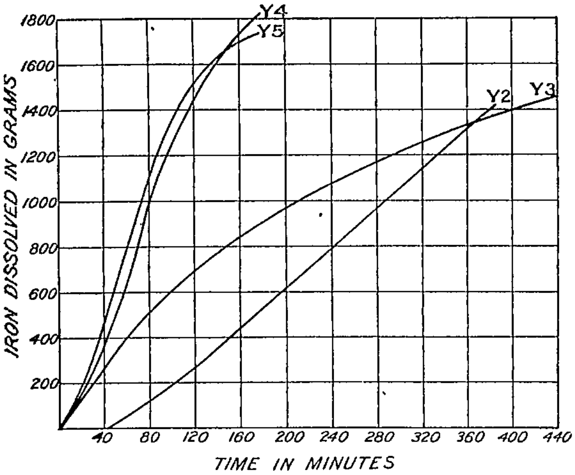

Figure 6.

Test Y3

In this test the theoretical quantity of ore required to react with the ferric chloride was used and the ore was added in total amount at the start. Reduction set in immediately and was much better than Y2 until

TABLE II

Leaching Tests

Ore—Roasted Eustis pyrite (—40 mesh). Volume—50 litres. Temp.—94°–96° C.

Test No.	Weight of ore charge	Excess of ore based on Fe″ (soluble) present in the ore	Time of agitation	Concentration of leach liquor in grammes of iron per litre		Rate of ore feed	Weight of residue	Analysis of leached liquor in grammes of iron per litre		Extraction of copper	Extraction of lead	Extraction of soluble iron (Fe″) calculated from analysis of residue	Extraction of total Fe″ calculated from analysis of residue	Reduction of ferric in leach liquor
				Fe″	Fe‴			Fe″	Fe‴					
	grammes	per cent	hours				grammes			per cent	per cent	per cent	per cent	per cent
Y-1	7,000	20	6⅓	0·0	116·7	20 minute intervals....	4,222	171·0	1·0	....	....	83·0	81·0	99·2
Y-2	4,846	26	11½	71·9	74·4	Theoretical at 20 min. intervals. Excess at 5th hour.	2,958	180·2	trace	71·3	100	70·2	68·3	100·0
Y-3	3,360	0	10⅓	69·4	64·9	All ore added at start.	1,729	160·5	2·5	95·5	100	85·2	83·7	96·2
Y-4	4,609	20	3	68·9	75·5	Two equal lots ½ hr. interval.	2,494	179·1	trace	73·9	100	74·8	....	100·0
Y-5	3,997	5·5	3	64·2	74·5	Two equal lots ½ hr. interval.	2,098	172·7	trace	78·2	100	77·2	....	100·0

the sixth hour when the rate slowed down. Had the leach liquor been of the same ferric concentration as in the former test the difference in rate of reduction would have been still more pronounced. The extraction of copper and iron is very much better. The total time is somewhat high and should be reduced by using excess ore. Filtration again proved rapid.

Test Y4

A 20 per cent excess of ore was added in this test in two equal lots at half-hour intervals. The rate of reduction was exceedingly rapid and was complete within 3½ hours. Filtration required 25 minutes. The time of reduction is surprising and the total extraction may be considered fairly good.

Test Y5

This test was a duplicate of the above except in quantity of excess ore added, and it checks test Y4 very closely. The total extraction is slightly higher on account of less excess ore charge. Filtration required 40 minutes.

CONCLUSIONS

Tests Y4 and Y5 would indicate that properly roasted pyrite leaches more rapidly than natural pyrrhotite and does not require fine grinding. From results obtained it would appear that the addition of the total ore charge within the first half hour gives the best results. This is the opposite of the procedure found most suitable for pyrrhotite and it is suggested that this may be due to the coarser condition of the ore charge, the iron in the porous coarse particles of ore being leached out and the sulphur being left in larger masses. Microscopic examination of the dried leach residue shows a minimum of fines present, in fact the residue appears even coarser than the original ore charge. Its fineness would be difficult of determination on account of its softness and consequent disintegration on a moving screen.

PURIFICATION OF ELECTROLYTE

The filtered leach liquor contains copper and zinc in the case of the pyrrhotite, and copper and lead from the roasted pyrite. Copper is readily separated out by cementation. Sponge iron containing 88 per cent iron made in a crucible from hematite ore and charcoal was used for precipitation of the copper. Difficulty has been experienced in filtering the copper-free liquor. At present it is not known definitely whether this is due to the presence of silica in the sponge iron becoming gelatinous in the solution and thereby clogging the filter plates, or whether a too neutral stage is reached whereby some of the iron takes a colloidal form or a readily hydrolised form. An examination of the washed residue on the plate shows the presence of iron and silica. The addition of a small amount of acid generally improves filtration. A study of the hydrogen-ion concentration of the liquor at this stage of the process will, it is believed, be of some value. Lead is partly precipitated out by iron but not completely. The elimination of zinc from the electrolyte has proved somewhat difficult, and has not so far been satisfactorily solved. The leach from the pyrrhotite ore contains approximately 0·8 gramme zinc per litre. Sodium or calcium sulphide precipitates about 25 to 40 per cent of the zinc and the limit would appear to be reached at 0·4 gramme zinc per litre. It has been suggested that the zinc be permitted to co-deposit with the iron and then allowed to volatilize in the melting process or possibly in the annealing process. The problem requires further and more definite investigation, and this will be undertaken.

ELECTRO-DEPOSITION

Several depositions of iron were made principally for the purpose of testing out the cell preliminary to the proper investigation of this phase of the process. The electrolyte used was that obtained from the leaching of the pyrrhotite and pyrite ores and contained zinc and lead as impurities, with a small amount of copper. The deposits obtained could only be considered fair, being dark grey in colour and somewhat pitted. The deposits were easily removed from the mandrel and showed silver-white on the cathode surface. The deposits contained about 2·5 to 5 per cent zinc and about 2 per cent lead, with a little copper, and were from about $\frac{1}{32}$ to $\frac{1}{10}$ of an inch in thickness.

The current density ranged from 101 to 105 amperes per square foot with a cell voltage of 4 to 4·2 volts, showing a current efficiency (corrected for impurities) of 88 to 90 per cent. The maximum temperature in the cell was 80° C., and the anolyte outflow ranged from 8 to 30 grammes ferric iron per litre. Under ideal conditions and proper regulation the anolyte outflow should have run about 64 per cent ferric iron.

IV

REPORTS ON THE WORK AND INVESTIGATIONS OF THE CHEMICAL LABORATORY

H. C. Mabee

Chemist in Charge

The work performed in the chemical laboratory of the Division during the year was similar in classification to that outlined in reports of previous years. It consisted of regular routine analytical work performed on samples connected with the experimental treatment of ores under investigation; special chemical problems connected with the treatment of complex ores, and research work. The latter involved a continuation of investigations in the electrolytic deposition of iron begun in previous years; and the study and development of hydrometallurgical methods for sulphide ores, the details and progress in connexion with these investigations being fully dealt with elsewhere in this report by members of the staff under whose supervision the work is being conducted.

Owing to the increased number of shipments of Canadian ores received and the large amount of experimental work involved in their treatment, the necessary chemical routine work was considerably increased during the year. From January 1 to December 31, over 5,000 chemical determinations were performed on 1,511 samples of ores and concentration products. This number represents an increase of over 15 per cent of that reported in 1923.

The following list of ores and minerals indicates the variety of determinations made:—

Arsenical gold ores	78	Magnesite	5
Calcite	8	Molybdenite	135
Copper	183	Nickel-copper	8
Dolomite	1	Platinum-gold	10
Gold-silver	521	Silver-lead-zinc	17
Gold-copper	74	Tungsten	2
Graphite shale	6	Titanium	18
Lead-zinc	255	Zinc-lead	164
Magnetite (iron ore)	11	Identification and valuation	19

Many of the ores, notably those from the Chu Chua district, B.C.; gold ores from The Pas district, northern Manitoba; and the gold-copper ores of northern Ontario and Quebec, presented unusual metallurgical problems, and, therefore, required an unusual amount of investigation with respect to their physical and chemical properties in order to establish suitable methods of treatment. B. P. Coyne, in addition to the regular routine analytical work, gave very valuable assistance to the engineers in this work. In July, R. A. Rogers, chemist in the Reclamation Service, Department of Interior, was transferred to the vacancy of Junior Chemist created by the resignation of D. T. Fotheringham. The addition to the present laboratory buildings, which was started late in the fall of the previous year was completed early in 1924, and installation and equipping were accomplished without any material interruption of routine work. The new addition will provide extra room for an assay laboratory and an electro-chemical research laboratory.

A COMBINED PYRO- AND HYDROMETALLURGICAL PROCESS FOR THE TREATMENT OF NICKELIFEROUS-PYRRHOTITE ORES WITH THE SUBSEQUENT RECOVERY OF IRON AND SULPHUR IN ADDITION TO NICKEL AND COPPER.

H. C. Mabee and A. E. Smaill

Introductory.—During the years that have elapsed since the process of smelting and bessemerizing as a method of treating the nickel-copper sulphide ores of Ontario was adopted, it has been fairly well demonstrated that with modern equipment and efficient management, good profits are to be obtained from the recovery of the nickel-copper values by this process. On the other hand, however, the metallurgical problems involved in the recovery and utilization of the sulphur and iron content, now entirely wasted, have never been satisfactorily solved, and anything that can be done to encourage or develop on a profitable basis a more complete recovery of the values in these ores would be an exceedingly valuable contribution to national economy and to the further development of our mineral industry in the recovery of the iron and sulphur.

Although the ores of the Sudbury district vary somewhat in their nickel-copper-iron-sulphur content, depending on the locality, the following table may be assumed to represent a fair average of their composition and to illustrate the losses by the present method of treatment.

TABLE I

Percentages and Values in One Ton of Ore

Metal	Per cent	Pounds	Price per lb.*	Value
			cents	$ cts.
Nickel	3·00	60	29·35	17 61
Copper	1·50	30	14·42	4 33
Iron	40·00	800	1·10	8 80
Sulphur	30·00	600	1·00	6 00
Precious metals	....	....	....	1 50

*The prices assigned to nickel and copper are those quoted by Dominion Bureau of Statistics report of Mineral Production of Canada for 1923. That for iron is based on recent quotations for furnace pig, and that for sulphur is the present price of sulphur f.o.b. Gulf States.

It will be seen from the above table that, omitting the precious metal values, $14.80, represented in the iron and sulphur content, is wasted from each ton of ore treated by present methods, in an endeavour to recover $21.94, or the total values in nickel and copper, or, in other words, 40 per cent of the total value of the ore.

The following report contains a brief outline of the results of a series of experiments which were conducted during the year in the research laboratories of the Ore Dressing and Metallurgical Division, consisting of a combined smelting and leaching process having as the ultimate object the recovery of the sulphur, the solution and separation of practically all of the copper and a portion of the nickel, leaving a sufficient quantity of nickel in the iron residues for the production of a 2·5 to 3·5 per cent nickel-steel.

General Procedure—Smelting.—The ore was first smelted to a low-grade matte in a covered, four-burner, round gas-furnace using a No. 60 Morgan graphite crucible, and in an atmosphere as strongly reducing as possible. The flux consisted of lime or limestone, in sufficient quantity to form an easily fusible silicate slag with the gangue, and yielding a matte containing all the nickel, copper, iron, and sulphur originally present in the ore. This operation is quite the reverse of that employed at the nickel plants at Sudbury, where the smelting operations are conducted in blast furnaces with the main object of eliminating sulphur and the oxidation and ultimate slagging of the iron as iron silicate.

The tests performed at the Ore Dressing Laboratories were made on lots of 20-pound melts, and were conducted on low-grade ore from the Frood mine, No. 3, of the International Nickel Company, which had been received for the purpose of carrying on concentration tests by flotation.

The analysis of the ore, together with that of the products from the smelting operations will be found in the following table:—

TABLE II

Composition of Ore, Matte, and Slag

—	Analysis per cent				
	Nickel	Copper	Iron	Sulphur	Total sulphides
Ore	1·35	1·45	18·79*	11·17	32·76
Matte	3·86	4·00	56·56	31·22	34·40
Slag	0·03	0·11	5·72	0·29	....

*This analysis of the ore does not include iron contained in the gangue material.

TABLE III

Weights and Recovery per Ton of Ore

—	Nickel lbs.	Copper lbs.	Iron lbs.	Sulphur lbs.	Total sulphides lbs.
In 1 ton of ore	27·0	29·0	375·8	233·4	655·2
Matte recovered	26·6	27·5	389·0	214·6	657·7
Losses in slag	0·4	1·5	76·7	8·8	....

The slag losses, particularly in nickel and copper, may be attributed partly to very small particles of matte, and also to the low iron content of the ore. In the blast furnace practice at Sudbury, the losses amount to about 8·5 per cent of the total nickel and 9·5 per cent of the total copper. It is interesting to note in Table II that the analysis of the ore indicates a possible 32·76 per cent total sulphides and the matte shows an actual yield of 34·40 per cent. This increase as noted in Table III appears entirely in the iron, and is probably due to a part of the iron in the norite gangue combining with the sulphur and passing into the matte. The matte, particularly that with a high iron content, exhibits a very unstable character. It is readily acted upon by the oxygen of the air, especially in a moist atmosphere, the whole mass disintegrating to a fine powder, over 80 per cent of which will pass a 100-mesh screen.

The finely divided matte was then desulphurized at a temperature slightly above 450° C. until 85 to 90 per cent of the total sulphur was eliminated. Owing to the lack of proper mechanical roasting-equipment, no attempt was made to secure a complete recovery of the sulphur content. Some experimental work has been carried on in this connexion and it is hoped that when the necessary apparatus is available a satisfactory method may be worked out.

The calcined product was next subjected to a chloridizing roast at a temperature of from 450 to 500° C. Although the method of chloridization of ores and the extraction of the metal chlorides produced has been developed extensively within the last few years, the chemical reactions involved are not yet definitely known. However, the experimental data obtained from chloridizing practice has revealed valuable information regarding the influence that some solid and gaseous substances exert during the operation and it is from this information that the most favourable conditions for the successful treatment of the matte has been studied. The best results so far have been obtained by the following procedure:—

Chloridizing Roast.—The roasted product was thoroughly mixed with about 10 per cent of sodium chloride and then moistened with 5 to 10 per cent water.

The mixture was charged into a fireclay cylinder, about 5 inches in diameter and 24 inches long, with a perforated bottom through which air was forced at a low pressure. The whole was then introduced by degrees into a vertical electric furnace maintained at a temperature of from 450 to 500° C. As the temperature of the mass in the lower end of the tube rises, the moisture is gradually driven to the upper part of the mixture forming a compact cake above the roasting-zone, which acts as a blanket. As the roasting continues, however, the roasting zone progresses upwards until the salt is all decomposed when the mass becomes more spongy and porous. This operation requires about 4 to 5½ hours to complete. The mass was allowed to cool slightly, but while still quite hot was discharged into a suitable leaching receptable and the chlorides of the metals leached out in hot water with constant stirring. Seventy-five per cent of the soluble metals passes into solution in a very short time. Leaching was continued for 2 to 3 hours and the liquid after settling was decanted, the residue washed twice with hot 5 to 10 per cent sulphuric acid solution, finally with hot water, and then sucked dry.

Results from Leaching Tests.—Table IV gives the results of leaching test from a chloridized roasted matte made from the Frood mine ore referred to in Table II. The roast was conducted on a thin bed of the material, and at a maximum temperature of 500° C.

TABLE IV

Leaching Test—Matte, Frood Mine Ore

—	Pounds per ton of ore		
	Nickel	Copper	Iron
Metals in 1 ton of ore	27·0	29·0	378·8
Recovered in matte	26·6	27·9	389·0
Extracted in leaching	16·1	24·1	5·8
Retained in iron residue	9·8	2·3	379·5
Total losses in process	1·1	2·6	3·7

Recovery	Nickel	Copper	Iron
In leach solutions	59·40 per cent	83·17 per cent	1·5 per cent
In iron residue	37·00 "	8·21 "	100·0 "

It will be noticed in the results that although the extraction of copper is fairly high, considerable losses occur. It was found on a repeated test that the results obtained from chloridizing in thin beds were not so satisfactory as those from deep fairly hot masses kept under chloridizing conditions for a longer period.

In the following tables are given the analyses and results of a smelting and leaching test on a nickel-copper-pyrrhotite ore from the Shebandowan Lake nickel area, the chloridization of which was conducted in a deep mass as described above.

TABLE V

Analysis of Shebandowan Ore and Products

—	Nickel %	Copper %	Iron %	Sulphur %
Ore	3·82	4·02	31·7	21·5
Matte	6·75	7·00	50·2	35·05
Slag	0·06	0·10	7·57	0·51

TABLE VI

Recovery per Ton of Matte

—	Nickel lbs.	Copper lbs.	Iron lbs.	Sulphur lbs.
Contained in matte	135·0	140·0	1004·0	701·0
Recovered in leaching	101·9	138·2	11·0	....
Remaining in residue	32·4	2·8	977·9	2·78

TABLE VII

Recovery and Losses of Matte Content

—	Per cent of total			
	Nickel	Copper	Iron	Sulphur
In the leach solution	75·5	98·74	1·1	....
In the residues	24·0	0·28	97·4	0·39
Losses	0·5	0·98	1·5	....

These results show an almost complete extraction of the copper with comparatively small total losses, and an iron oxide product still containing approximately 3·3 per cent nickel, and a sulphur content of less than 0·40 per cent.

Recovery of the Metals in the Leach Solutions.—The leach solutions obtained from each of the small-scale tests were naturally small in quantity, and were therefore treated by regular laboratory electro-deposition methods to recover the copper and nickel content for the purpose of ascertaining and checking the leaching results, rather than the investigation of a working method of recovery. However, some attention was given to this phase of the process, and it is expected that, as larger scale tests are undertaken and the necessary additional apparatus provided, a suitable electrolytic process will be developed under conditions approaching as nearly as possible those that would obtain in actual practice. It need scarcely be pointed out that the recovery of the platinum group metals and the additional values which they represent is also of great importance in establishing a successful process.

SUMMARY AND CONCLUSIONS

It is necessary in the preliminary smelting of the ore that the operation be conducted in an atmosphere as non-oxidizing as possible, in order to recover the sulphur and iron, and for this purpose a reverberatory or electric furnace could be operated quite successfully. By this preliminary treatment all substances such as lime, magnesia, and other gangue materials, which are not only unnecessary, but interfere ordinarily with successful chloridization, are eliminated, and the matte produced is more amenable to changes and decomposition than the original ore, or a concentration product of the ore.

It was found that the essential points governing a successful chloridizing roast were the presence of moisture, and oxygen, for the purpose of oxidizing the sulphur still present in the charge, furnishing as a result the necessary acid radicals for the liberation of chlorine from the chloridizing agent. By closely observing these conditions, an extraction of upwards of 98 per cent of the total copper and 75 per cent of the nickel was obtained without difficulty and a residue containing almost 100 per cent of the total iron content of the matte in the form of a finely divided oxide, and which still contained a sufficient amount of nickel to produce a steel, the quality of which would doubtless be much superior to that obtained from adding nickel direct to the molten steel bath.

V

CONCENTRATION OF THE LEAD-ZINC ORES OF EASTERN CANADA

C. S. Parsons

INTRODUCTORY

The steady advance in the price of lead and zinc in the metal markets of the world during the past few years has resulted in an active search for new deposits of these metals, and to the re-examination of known properties that are now idle.

There are a number of deposits of these metals in eastern Canada, the best known of which are the Tetreault mine at Notre Dame des Anges, Que.; the Reader mine on Calumet island, Que.; the Stirling property in Cape Breton, and the Federal mine in Gaspe district, Que. The author has run concentration tests on the ores from the first three mentioned properties. The ores from Notre Dame des Anges and Calumet island presented problems in concentration which were not successfully solved until the past year. Experimental work done on the ore from the Stirling mine has indicated the possibility of obtaining a high-grade zinc concentrate, but so far no satisfactory method of treatment has been developed.

CHARACTER AND TYPES OF ORES

Metallurgically the lead-zinc ores of eastern Canada may be classified into three distinct types, as follows:—

First.—Ores in which the sulphide minerals are fairly coarsely crystalline and can be freed by crushing to 40 or 60 mesh. The zinc blende has a high iron content (not marmatite) and gives a dark brown colour on grinding. The ores of Notre Dame des Anges and Calumet island, Que., may be cited as examples.

Second.—Ores in which the sulphide minerals are very finely disseminated and require extremely fine grinding. The zinc blende contains such high percentages of iron that it may be classed as marmatite. The ore of the Stirling mine, Cape Breton, is an example.

Third.—Ores in which the sulphide minerals are fairly coarsely crystalline and the zinc blende is of the light resin-coloured variety containing very little iron. The ore of the Federal Lead and Zinc Co.'s mine in Gaspe, Que., is an example.

EARLY METHODS OF CONCENTRATION

The ore at Notre Dame des Anges was discovered in 1910, and in 1912 Mr. Tetreault erected a concentrating mill in the hope of treating this complex ore by methods used in the Joplin district. The mill failed to produce a marketable grade of concentrate. In 1914 the Weedon Mining Company obtained a lease on the property and remodelled the mill. The galena was concentrated by graded crushing and tabling. A zinc-iron middling product was obtained on tables, and oil flotation was used to save the slimed blende, but as the iron sulphide floated with the zinc, the product was too low grade to market. To eliminate the iron sulphide a magnetic separation plant was built and the non-magnetic pyrite was given a roast to convert it to the magnetic form. A considerable tonnage of zinc concentrates produced in this manner was marketed. It is said that they averaged about 42 per cent zinc. The process was costly and could not be looked upon as a commercial success. A large dump of zinc-iron middlings containing about 24 per cent zinc was left on the property and these are now being successfully concentrated by the British Metals Corporation (Canada), Ltd.

The Reader property on Calumet island was operated intermittently from 1896 to 1913. In 1910 a small concentrator was built and trial shipments were made to Newark, N.J. In 1912 a 150-ton concentrator was built, equipped with jaw crusher, coarse rolls, screens, fine rolls, jigs, Huntingdon mill, Wilfley and Overstrom tables. The mill was only operated for a few months and then closed down. The records show that a marketable lead product was made containing on the average, 65 per cent lead, 8 per cent zinc, 0·5 per cent copper, and 87 ounces silver per ton. A low-grade zinc concentrate was obtained averaging about 3·5 per cent lead, 28 per cent zinc and 18 ounces silver per ton, the remainder being iron sulphides. A sample of the mill tailings recently taken from the dump contained 1·77 per cent lead, 12·15 per cent iron, and 8·3 ounces silver, showing that the recovery of lead and silver obtained was very poor and probably did not exceed 50 per cent of either mineral.

A detailed description of the concentration tests on the zinc-lead ores of Notre Dame des Anges is given in Report No. 204, pages 8-19.

The detailed results of tests on the ores of the Reader mine, Calumet island, are given in Report No. 220, pages 87-91.

A summary report of the above tests has been published in mimeograph form, "Concentration of Lead Zinc Ores of Eastern Canada," Memorandum Series No. 21.

THE CONCENTRATION OF THE LAKE GEORGE ANTIMONY ORES

C. S. Parsons

History.—Occurrences of antimony ores in Canada are rare. The deposit at West Gore, Hants county, Nova Scotia, and the one at Lake George, York county, New Brunswick, are the only ones that have produced any tonnage of ore.

The Lake George deposits were discovered in 1863 and the property has been worked intermittently for the past 60 years by various companies. Many years ago, the property was operated by the Lake George Mining and Smelting Company and a small smelting plant was erected which produced about one ton of metal per week when in full operation. Attempts were made to concentrate the ore by hand-cobbing, and ship in lump form to Swansea, but this met with failure owing to the disseminated nature of the mineral in the vein material. Gravity concentration in jigs and vanners was tried but the slime losses were so high that this method was also abandoned. The Canadian Antimony Company installed a dry method of concentration similar to a method used extensively in France. This process consisted of heating the ore, which was then supposed to decrepitate, the flying pieces to be picked up in a current of air. No previous test work was done on the ore, and after the process had been installed it was found that the ore would not decrepitate, so the process was abandoned.

In mining the ore considerable wall rock was broken which contained arsenical pyrites. In refining operations the arsenic condensed with the antimony and produced a metal with properties objectionable to the trade. Some Montreal people took up the matter of the elimination of the arsenic. They undertook to remove the arsenic by leaching the antimony oxide produced by roasting the ore. Most of the arsenic could be removed in this way, but the method was not successful from an economic standpoint.

The mineral occurs as lenses of stibnite (antimony sulphide) in fissure veins in the slates and quartzites. The veins have been opened for a distance of a mile in length and a large number of shafts have been sunk. The deepest shaft is about 375 feet.

CONCENTRATION

The many failures recorded in the history of the operation of the antimony deposits in Canada have been partly due to want of suitable methods for the concentration of the ores. Stibnite is an extremely friable mineral and slimes very badly, and it is impossible to save these slimes by any known method of gravity concentration.

Two samples of ore from the Lake George deposits were submitted to the Ore Dressing and Metallurgical Laboratories of the Mines Branch. The first was received in 1916 from A. P. Slipp, K.C., of Fredericton, who was at that time interested in the deposits. This sample was low-grade ore from the waste dumps, and on analysis gave:—

Antimony, 3·15 per cent; arsenic, 0·28 per cent; gold and silver, trace.

The second sample was received in 1922 and was submitted by the North American Antimony Smelting Company, Ltd. This sample was claimed to be a true representative sample of the milling ore from the company's mine. The sample on analysis gave:—

Antimony, 11·65 per cent; arsenic, 0·37 per cent.

Synopsis of Concentration Test on Shipment No. 1.—A brief synopsis will be given of the work done on the concentration of the low-grade ore. Gravity concentration on jigs and tables followed by flotation of the slimes was tried, and straight flotation of the ore.

GRAVITY CONCENTRATION

There was not enough mineral freed at the coarser sizes, and the jig tailing would have had to be recrushed and re-treated. Tabling of the sized material gave the following results:—

−20+80 mesh	Concentrate	Antimony	31·10	per cent
		Arsenic	0·65	"
	Tailing	Antimony	0·67	"
		Arsenic	0·34	"
−80 mesh	Concentrate	Antimony	31·02	"
		Arsenic	2·71	"
	Tailing	Antimony	2·10	"
		Arsenic	0·28	"

These results show conclusively the difficulty of concentrating stibnite by gravity concentration. The slime loss is very high and there is also a corresponding loss in the coarser sizes from flat flakes of mineral that are carried into the tailing. The concentrate is also low grade and another point of importance is that the concentrate still contains a relatively high percentage of arsenic. The above tests were made on standard size jigs and tables.

FLOTATION

A number of small-scale laboratory flotation tests were made on the original sample, using a standard Callow laboratory testing unit. Various combinations of reagents were tried with the object of producing a high-grade concentrate with the elimination of the arsenic.

Test No. 1

0·6 pounds per ton hardwood creosote
0·2 " " steam distilled pine oil
10·0 " " sulphuric acid

Product	Analysis		Per cent of values	
	Antimony%	Arsenic %	Antimony	Arsenic
Concentrate	47·52	0·92	91·8	23·0
Tailing	0·24	0·24	8·2	77·0
Heads	2·75	0·29	....	....

Test No. 2

0·6 pounds per ton hardwood creosote
0·2 " " pine oil
3·0 " " caustic soda

Product	Analysis		Per cent of values	
	Antimony%	Arsenic %	Antimony	Arsenic
Concentrate	49·00	1·0	75·5	13·5
Tailing	0·66	0·28	24·5	86·5
Heads	2·58	0·31	....	....

Test No. 3

0·5 pounds per ton fuel oil from asphalt base
0·3 " " pine oil
neutral pulp

Product	Analysis		Per cent of values	
	Antimony%	Arsenic %	Antimony	Arsenic
Concentrate	58·52	0·24	87·6	3·5
Tailing	0·40	0·28	12·4	96·5
Heads	3·06	0·29	....	....

These tests show that even with a low-grade ore a high-grade concentrate can be obtained with a good recovery, and that it is possible to eliminate 90 per cent or more of the arsenic contained in the feed. Flotation in a neutral pulp with a fuel oil from a petroleum oil with an asphalt base gave the best results.

SYNOPSIS OF CONCENTRATION TESTS ON SHIPMENT NO. 2

These tests were run to design a flow-sheet suitable for the treatment of the ore on a basis of 100 tons daily capacity, the erection of a mill being contemplated on the results of this work.

Hand Sorting.—The ore was examined for the possibility of sorting sufficient lumps of pure stibnite to warrant hand-picking, but this was found to be impracticable.

Jigging.—Considerable stibnite was freed at $\frac{1}{4}$-inch size and it was found possible to produce a jig concentrate containing 50 per cent antimony. A large proportion of true middling would have to be handled and the tailing would have to be recrushed. Considering the use of jigs from the standpoint of operating costs in a small mill, it was decided that their use would not be advisable owing to the necessity of recrushing machinery for the tailing and middling products, which would mean a more complicated flow-sheet.

Table Concentration.—Table concentration tests were tried on −14 to −48-mesh material. It was found possible to obtain a table concentrate by recleaning the primary table concentrate on a special cleaner deck, containing 58 per cent antimony and 0·25 per cent arsenic, but with a recovery of only 57 per cent of the antimony in the table feed. It was found after a careful examination of the table products under the microscope that tailing and middling products from sizes coarser than 24 mesh would have to be crushed again and that the middling from the finer sizes down to 48 mesh would have to be recrushed. The losses in the sizes from −35 to −48 mesh, due to flotation on the table, were very high and practically all the stibnite passing to the tailing was found to be entirely freed from the gangue.

Summary of Jigging and Tabling Tests.—It was found that by jigging or tabling carefully sized feed that had been crushed to −14 mesh, a recovery of 56 per cent of the antimony could be obtained in a concentrate averaging approximately 53 per cent antimony and 0·30 per cent arsenic. This grade of concentrate was considered too low to meet the company's requirements. By crushing to −24 mesh and tabling classified or sized feed a 58 per cent antimony concentrate could be obtained but the recovery dropped to 55 per cent and less.

Flotation Tests.—These tests were made on the original ore and consisted of a series of small-scale tests and one large-scale or tonnage test under conditions similar to actual mill operations. The ore for these tests was crushed so that 95 per cent passed 65-mesh Tyler standard screen.

SMALL-SCALE TESTS

Test No. 1.—Reagents used in this test were a mixture of coal tar and coal-tar creosote (40 and 60 per cent) and pine oil for frothing. A neutral pulp was used.

Test No. 2.—The Southwestern Engineering Company's oil KK 1, and steam-distilled pine oil for frothing. Neutral pulp used.

Test No. 3.—The Southwestern Engineering Company's oil KK 1, pine oil, and 4 pounds per ton of soda ash (pulp alkaline).

Test No. 4.—The Southwestern Engineering Company's oil KK 1, pine oil, and 4 pounds per ton of lime (pulp alkaline).

Test No. 5.—The Southwestern Engineering Company's oil KK 1, pine oil, and 10 pounds per ton of sulphuric acid (pulp acid).

Test No.	Product	Weight per cent	Analysis		Per cent of values	
			Sb %	As %	Sb	As
1	Concentrate	16·0	64·24	0·28	86·7	....
	Middling	2·8	6·38	1·62	1·4	....
	Tailing	81·2	1·75	0·35	11·9	....
2	Concentrate	17·7	61·80	0·62	81·4	35·6
	Middling	4·0	4·50	1·04	13·4	13·5
	Tailing	78·3	0·88	0·27	5·2	50·9
3	Concentrate	15·3	62·70	1·00	83·6	40·0
	Middling	4·1	13·10	1·44	4·7	15·4
	Tailing	80·6	1·65	0·21	11·7	44·5
4	Stibnite would not float in lime pulp.					
5	Concentrate	17·4	63·36	0·31	95·6	16·4
	Middling	4·3	5·94	1·29	2·2	16·7
	Tailing	78·3	0·35	0·28	2·2	66·9

Large-scale Pilot Tests

A large-scale tonnage check test was made in a Callow flotation unit consisting of two rougher cells and two cleaner cells of the flat-bottom type. The ore was reduced to ½ inch in a jaw crusher and rolls and fed to a 4½-foot Hardinge mill in closed circuit with a standard Dorr simplex classifier. The overflow of the classifier was −50 mesh and went direct to the flotation cells. The reagents used were a mixture of coal tar 40 per cent and coal-tar creosote 60 per cent from the Dominion Tar and Chemical Company, and sufficient pine oil to maintain a good froth. Sulphuric acid was used amounting to 5 pounds per ton of ore.

Concentrate—Antimony, 58·21 per cent; Arsenic, 0·28 per cent.
Tailing— Antimony, 0·94 per cent of total values.
Recovery— 93·5 per cent of total values.

The results from the large-scale test confirmed the results obtained from the small laboratory tests. The grade of concentrate is slightly lower, but this does not mean that a concentrate of higher grade cannot be obtained. It was difficult to gauge the grade of concentrate being produced, but as the operator becomes more familiar with the operation of the flotation cells, better work and even a higher grade concentrate than 63 per cent antimony, and containing less than 0·3 per cent arsenic, may be expected.

CONCLUSIONS

The following flow-sheet was recommended as the simplest and most feasible method of concentrating the ore: "Crushing the run-of-mine ore to 2 inches in a breaker, ball milling in closed circuit with drag classifier to 65 mesh, the classifier overflow to go to a flotation unit of the pneumatic type."

In the construction of a small concentrator of 50 to 100 tons capacity, simplicity of design is important. Jigging and tabling followed by flotation would have required regrinding, classifying, sizing, and thickening units, making a complicated flow-sheet and adding greatly to the initial cost of the mill as well as to the labour, power, and other operating costs. On the other hand a mill designed for straight flotation would have a very simple flow-sheet and would occupy about one-third of the floor space. The ore from the breaker, crushed to 2 inches, could be fed directly to a ball mill operated in closed circuit with a drag classifier to produce a product approximately −65 mesh, which would go directly to the flotation units.

The grade of the final concentrate from the flow-sheet using straight flotation would be 5 per cent higher than that from the flow-sheet using jigs and tables.