CANADA
DEPARTMENT OF MINES
Hon. W. A. Gordon, Minister; Charles Camsell, Deputy Minister

MINES BRANCH
John McLeish, Director

INVESTIGATIONS IN ORE DRESSING AND METALLURGY

(Testing and Research Laboratories)

1929

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

Investigations of Mineral Resources and the Mining Industry.

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

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

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

Other reports on Special Investigations are issued as completed.

Part of the pyrometallurgical laboratory. On the left, Ajax high frequency induction furnaces with a capacity of 50 and 300 pounds respectively; on the extreme right, a rotary electric retort.

MINES BRANCH INVESTIGATIONS IN
ORE DRESSING AND METALLURGY, 1929

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm

Chief of Division

The investigative work of the Ore Dressing and Metallurgical Laboratories is steadily growing with the progress being made in the Canadian mining and metallurgical industries. Each year more applications and requests are received for experimental tests and research work on ore treatment. To meet the demands of the industry increased laboratory facilities and staff are required so that more investigative work can be undertaken at the same time. The assistance being given the industry by this service has been highly commended by those actively engaged in the development of mining properties and has helped materially the growth and progress of the industry.

In view of the growing importance of the iron and steel industry, a new pyrometallurgical laboratory was built for work on iron and steel. It is equipped with laboratory and semi-commercial scale, sintering, metallizing, melting, heat-treating, and standard laboratory testing and metallographic equipment for conducting extensive test and research on ferrous and non-ferrous ores, metals and their alloys.

It is proposed to carry out extensive investigations on the metallization of Canadian iron ores, by both the solid carbon and gaseous reduction methods. Some preliminary work has already been done in this laboratory on sponge iron or the direct reduction of iron ores and a co-operative arrangement has been entered into with the Ross-Playfair interests, a group of Canadian financiers, for the investigation and demonstration of the "Musso Semi-direct Steel Process." This is a solid carbon process. Later, the investigation of gaseous reduction processes will be undertaken to determine which is more adaptable to Canadian ores and conditions.

The co-operative arrangements entered into with the Base Metals Extraction Company, Limited, and the Cassel Cyanide Company were still existent. Progress was made by the metallurgists and chemists of the Base Metals Extraction Company in working out the details of their process for the treatment of bulk concentrates from refractory base metal sulphide ores. Special attention was paid to the treatment of residues. The company now control the rights for eastern Canada of the Waelz process for the treatment of residues. A large-scale test was conducted on the mixed concentrates from the Errington mine ore, the results of which are given under Section IV dealing with the hydrometallurgical and electrochemical investigations.

The Cassel Cyanide Company maintained a research fellow for work on the use of cyanide in the selective flotation of base metal sulphide ores. The investigative work was chiefly confined to the copper-zinc-iron sulphide admixtures. During the year a co-operative arrangement was made with the American Cyanamid Company, Niagara Falls, Ontario, for the placing of a research fellow in the laboratories beginning with the new year for investigative work on flotation agents and reagents in general. During the year, several large shipments containing the radioactive mineral uranite were received from the Ontario Radium Corporation to determine whether the occurrence was a commercial source of radium. The results of the concentration tests are given in Section III on non-metallic minerals. The investigation of the physical properties of calcined gypsum was continued. Calcining and physical tests of the calcined gypsum from a number of Canadian occurrences were made. A summary of the results is given in Section III dealing with non-metallic minerals, and the detailed results are given in "Investigations of Mineral Resources and the Mining Industry, 1929."

The larger part of the investigative work was on the selective flotation of base metal sulphide ores and the treatment of refractory gold ores. The laboratories of the Division are well equipped for this class of work, with small laboratory grinding and flotation machines, as well as continuous intermediate and semi-commercial scale units for tests on a few pounds up to carload lots. The equipment for amalgamation and cyanide tests is the latest. This is evident from the co-operative arrangements made with the Cassel Cyanide and American Cyanamid Companies.

G. B. O'Malley, lecturer in Metallurgy, in the University of Melbourne, New South Wales, Australia, was attached to the staff of the Division during the summer months, for the purpose of studying the laboratory methods of conducting the investigations.

The services of T. W. Hardy were placed at the disposal of the Chairman of the Advisory Board on Tariff and Taxation in connexion with the classification of materials in the proposed new schedule of iron and steel products, and a considerable amount of time was given by him to this important work.

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

Section II contains the results of the investigations on metallic ores under the supervision of C. S. Parsons, engineer in charge. He was assisted by A. K. Anderson, J. S. Godard, D. S. Halford, and G. B. O'Malley.

Section III contains the results of investigations on non-metallic minerals, under the supervision of R. K. Carnochan. R. A. Rogers was associated with him and performed the investigative work on gypsum.

Section IV contains the results of investigations in hydrometallurgy and electrochemistry under the supervision of R. J. Traill, engineer in charge. He was assisted by W. R. McClelland and J. D. Johnston. The co-operative work with the Base Metals Extraction Company is also included.

Section V contains the results of investigations on iron and steel, under the supervision of T. W. Hardy, engineer in charge.

Section VI contains the work and investigations in the chemical laboratories conducted by B. P. Coyne, R. A. Rogers, H. L. Beer, L. Lutes, and laboratory assistants, under the direction and supervision of H. C. Mabee, chief chemist.

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

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

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

Report No.	Page	Ore or product	Source of shipment	Shipper and address	Weight
					lb.
317	5	Copper	Boston Creek, Ont.	Patterson Copper Mines, Ltd., Boston Creek, Ont.	63
318	6	Copper-nickel	Michipicoten, Ont.	Engineers Holding Co., Toronto, Ont.	50
319	9	Gold	Jackson-Manion mine, Woman Lake, Ontario.	Jackson-Manion Mines, Ltd., Toronto, Ont.	1,457
320	13	Gold	Mine Centre, Ont.	British Canadian Mines, Ltd., Toronto.	200
321	17	Gold	Howey gold mine, Red Lake, Ont.	Howey Gold Mines, Ltd., Toronto, Ont.	8,000
322	29	Iron	Kaministikwia, Ont.	N. M. Paterson, Fort William, Ont.	45,000
323	42	Lead-zinc	Geneva, Ont.	Towagmac Exploration Co., Montreal, Que.	3,000
324	48	Gold	Mongowin township, Sudbury district, Ontario.	McMillan Gold Mines, Ltd., Sudbury, Ont.	50
325	53	Gold	Perrier mine, Nelson, B.C.	Perrier Syndicate, Nelson, B.C.	135
326	58	Gold	Gem Lake mine, east central Manitoba.	Gem Lake Mines, Ltd., Winnipeg, Man.	5,700
327	65	Copper	Windsor mine, La Sarre, Que.	Windsor Mines, Ltd., Toronto, Ont.	28
328	66	Gold	Sylvanite mine, Kirkland Lake, Ont.	Sylvanite Gold Mines, Ltd., Kirkland Lake, Ont.	1,000
329	69	Gold	Evangeline mine, Liscomb Mills, N.S.	Evangeline Gold & Copper Mines, Ltd., Liscomb Mills, N.S.	100
330	71	Gold	Belledat-Goudreau mine, Goudreau, Ont.	Towagmac Exploration Co., Montreal, Que.	1,070
331	82	Copper	Coxheath mine, Coxheath, N.S.	Arno Mines, Ltd., Coxheath, N.S.	380
332	84	Gold	St. Anthony mine, Savant Lake, Ont.	St. Anthony Gold Mines, Ltd., Savant Lake, Ont.	2,280
333	86	Gold	Malartic mine, Amos, Que.	Malartic Mines, Ltd., Amos, Que.	14,000

Report No.	Page	Ore or product	Source of shipment	Shipper and address	Weight
					lb.
334	94	Gold	Goudreau-Localsh area, Ont.	R. R. Hedley, Sault Ste. Marie, Ont.	22
335	96	Silver	Castle-Trethewey mine, Bestel, Ont.	Castle-Trethewey Mines, Ltd., Bestel, Ont.	4,000
336	99	Copper	McGinn claims, Baldwin tp., Sudbury dist., Ont.	W. P. Murdock, Espanola, Ont.	90
337	101	Tin	Snowflake mine, Albert Canyon, B.C.	Snowflake Mining Co., Ltd., Albert Canyon, B.C.	300
338	116	Gold-copper	Greene-Stabell mine, Dubuisson, Que.	Greene-Stabell Mines, Ltd., Toronto, Ont.	3,000
339	127	Gold-copper	Central Manitoba mine, Wadhope, Man.	Central Manitoba Mines, Ltd., Wadhope, Man.	2,000
340	139	Lead-zinc	Haslat-Duck Lake mine, Schreiber, Ont.	Haslat-Duck Lake Mines, Ltd., Toronto.	120
341	143	Gold	Granada-Rouyn mine, Rouyn, Que.	Granada-Rouyn Mines, Ltd., Rouyn, Que.	440
342	155	Barite-lead-copper.	Giant mine, Spillamacheen, B.C.	A. B. Trites, Vancouver, B.C.	363
343	159	Asbestos	Ville Marie, Que.	L. J. Jodouin, Sudbury, Ont.	55
344	161	Silica sands	Guigues township, Temiscamingue dist., Que.	F. W. Arnot, Fabre, Ont.	22,320
345	166	Radium	Cardiff township, Haliburton, Ont.	Ontario Radium Corporation, Toronto, Ont.	82,000
346	170	Mica	W. Ahearn, Ottawa, Ont.	W. Ahearn, Ottawa, Ont.	3,160
347	178	Iron-vanadium	Bad Vermillion lake, Mine Centre, Ont.	W. M. Goodwin, Gardenvale, Que.	100
348	192	Copper-lead-zinc.	Errington mine, Bradley, Ont.	Treadwell-Yukon Co., Ltd., Bradley, Ont.	20,000
349	175	Gypsum	Magdalen islands, Que.	Mines Branch, Ottawa, Ont.	3 samples

II

REPORT OF INVESTIGATIONS: METALLIC ORES SECTION

Report No. 317

CONCENTRATION OF COPPER ORE FROM PATTERSON COPPER MINES, LTD., BOSTON CREEK, ONT.

J. S. Godard and D. S. Halford

Shipment. One bag of ore weighing 63 pounds was received January 28, 1928, submitted by Mr. W. E. Simpson, Patterson Copper Mines, Ltd., Boston Creek, Ontario.

Characteristics and Analysis of the Ore. The ore consists of sulphides banded in a dark grey siliceous gangue. The sulphides were chalcopyrite and pyrite with the former predominating. Some calcite is present but quartz is the chief constituent of the gangue. A representative sample was cut out, which showed on analysis:

Copper	8·51 per cent.	Lime	4·75 per cent
Iron	12·10 "	Gold	Trace
Insoluble	45·5 "	Silver	0·22 oz./ton

Purpose of Tests. The following tests were run to determine whether the ore was amenable to concentration by flotation.

Experimental Tests

Test No. 1

Ore	1,000 grammes crushed to −18 mesh
Water	1,000 grammes
Soda ash	1·0 lb./ton
Thiocarbanilide	0·1 "
Cyanide	0·1 "

The above charge was ground 35 minutes in Abbé ball mill; floated in Ruth machine using cresylic acid to froth.

Test No.	Product	Weight, %	Assay Cu %	Per cent of values Cu
H-1	Concentrate	29·7	28·10	98·3
	Tailing	70·3	0·20	1·7

Test No. 2

Ore	1,000 grammes crushed to 18 mesh
Water	1,000 grammes
Soda ash	1·0 lb./ton
Cyanide	0·1 "

The above charge was ground 35 minutes in Abbé ball mill; floated in Ruth machine using 0·08 lb. per ton of potassium xanthate for promoter, and pine oil to froth.

Test No.	Product	Weight,	Assay	Per cent of values
		%	Cu %	Cu
H-2	Concentrate	31·1	26·60	97·3
	Tailing	68·9	0·33	2·7

CONCLUSION

This ore is a very simple one to concentrate by flotation as can be seen from the results of Test No. 1, where there was a recovery of 98·3 per cent of the copper in a concentrate that ran 28·1 per cent Cu without cleaning. The ore has a remarkably clean gangue, and a relatively small amount of iron sulphide, which is an ideal condition for concentrating purposes.

Report No. 318

CONCENTRATION OF COPPER-NICKEL-BEARING GABBRO FROM ENGINEERS HOLDING COMPANY, MICHIPICOTEN, ONTARIO

A. K. Anderson

Shipments. Two bags of material, gross weight 50 pounds, were received on January 8, 1929, consigned from Michipicoten, Ont., by Mr. R. N. Bond of the Engineers Holding Company, Limited, Toronto, Ont.

Characteristics of the Ore. The material consisted of a schisted, dark-coloured gabbro, containing finely disseminated sulphides of copper and iron.

Purpose of Tests. The shipment was forwarded with the request that concentration tests be made to determine the most suitable method of treatment.

Sampling and Analysis. The lot as received was crushed to pass 14 mesh, passed through a Jones riffle sampler and a representative head sample obtained. This gave the following metal contents:

Copper	1·23 per cent	Gold	0·01 oz./ton.
Nickel	0·51 "	Silver	0·25 "
Zinc	0·14 "	Platinoids (chiefly palladium)	0·06 "
Lead	None		
Bismuth	None		

EXPERIMENTAL TESTS

The ore after crushing to pass 14 mesh was thoroughly mixed, and representative portions of 1,000 grammes were taken and ground in porcelain mills containing iron balls and water equal in weight to the ore. Flotation tests were conducted in a laboratory-size Ruth flotation machine. The concentrates obtained were not cleaned. The tests in detail follow:—

Test No. 1

Bulk Concentration by Flotation

Reagents—

To ball mill—		To flotation cells—	
Soda ash.............	3·0 lb./ton	Potassium xanthate.....	0·10 lb./ton
		Pine oil................	0·08 "

Grinding, 99 per cent, 150 mesh.

Product	Weight, %	Assay					Per cent of values				
		Cu %	Ni %	Au, oz./ton	Ag, oz./ton	Plat-inoids, oz./ton	Cu	Ni	Au	Ag	Plat-inoids
Concentrate..	12·76	8·97	3·11	0·04	1·76	0·14	97·98	76·49	59·30	86·55	33·90
Tailing.......	87·24	0·03	0·14	0·004	0·04	0·04	2·22	23·51	40·70	13·45	66·10

This test shows that a high recovery of the copper in a concentrate containing the major amount of the nickel can be secured. The grade of the concentrate could be raised by cleaning. The precious metals accompany the copper and nickel into the concentrate.

Test No. 2

Bulk Concentration by Flotation in Acid Circuit

Reagents—

To ball mill—	To flotation cells—	
None	Sulphuric acid..........	0·5 lb./ton
	Potassium xanthate.....	0·10 "
	Pine oil	0·08 "

Product	Weight, %	Assay		Per cent of values	
		Cu %	Ni %	Cu	Ni
Concentrate..............................	14·8	8·30	2·58	96·9	71·4
Tailing....................................	85·2	0·05	0·18	3·1	28·6

This test shows no advantage is to be gained by floating in an acid circuit.

Test No. 3

Selective Flotation of the Sulphides

This test was made to see if a separation could be obtained between the copper and nickel minerals.

Reagents—

To ball mill—		To flotation cells—	
Soda ash	5·0 lb./ton	Potassium xanthate	0·10 lb./ton
Sodium cyanide	0·10 "	Pine oil	0·08 "
		Copper sulphate	1·0 "
		Amyl xanthate	0·20 "

Grinding, 95 per cent, 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Cu %	Ni %	Cu	Ni
Concentrate No. 1	8·6	14·03	4·25	91·5	66·1
Concentrate No. 2	11·1	0·70	0·54	6·2	10·8
Tailing	80·3	0·40	0·16	2·3	23·1

No separation of the copper and nickel is obtained under these conditions.

Test No. 4

Table Concentration

A sample, 1,000 grammes, of the ore was ground in a ball mill and passed over a Wilfley table. The middling product was re-passed over the table and the middling product obtained added to the concentrate.

Product	Weight, %	Assay		Per cent of values	
		Cu %	Ni %	Cu	Ni
Table concentrate	6·5	3·29	1·36	17·1	16·3
Table tailing	93·5	1·09	0·44	82·9	83·7

This test shows that table concentration yields very poor recoveries.

CONCLUSIONS

From the above tests it appears that a bulk flotation of the finely ground material yields high recoveries of the copper, and a large percentage of the nickel in the ore. The assays of the products from Test No. 1 show that the major amounts of the precious metals will be contained in the flotation concentrate. Cleaning of the concentrate will doubtless raise the grade, and yield a product acceptable for smelting purposes.

It is evident that the material is readily amenable to concentration by flotation. A small pilot mill erected on the property will enable the operators to determine the most suitable reagent combinations, grinding conditions, and other factors of mechanical operation to yield the highest returns.

Report No. 319

THE RECOVERY OF GOLD FROM THE ORE OF THE JACKSON-MANION MINES, LTD., WOMAN LAKE, ONT.

A. K. Anderson

Shipments. Two shipments of ore were received. The first consisted of 8 bags, gross weight 481 pounds. The second was of 15 bags, 976 pounds weight.

Characteristics of the Ore. Both shipments were of clean white quartz, carrying a very small percentage of iron sulphides. Free gold was visible. Shipment No. 1 was of such a high-grade nature that apart from sampling and assaying, no test work was done on it. All tests were made on Shipment No. 2.

Purpose of Experimental Tests. The investigation was carried out to determine what recovery of the gold can be obtained by amalgamation and by cyanidation. Such pertinent factors as fineness of grinding required to secure maximum results, and cyanide consumption were also determined.

Sampling and Analysis. Due to free gold present, both shipments were sampled and assayed in duplicate. Each lot was crushed to pass 14 mesh, and cut through a Jones riffle sampler. Each half thus obtained was handled separately to secure duplicate samples. They were again cut in half, thus giving one-quarter of the original bulk. This quarter portion was crushed and screened through 35 mesh, and again cut in half, yielding one-eighth of the original lot. This portion was ground to pass 65 mesh, and cut twice giving one thirty-second of the total. This was ground and screened to pass 100 mesh, and the final sample prepared for assay. All metallics remaining on the different size screens were collected and the gold in them included in the calculation of the assays.

The assays of the two lots are as follows:

Shipment No. 1—	
A	14·16 oz. Au.
B	15·04 "
Average assay	14·70 "
Shipment No. 2—	
A	0·78 oz. Au.
B	0·77 "
Average assay	0·77 "

As the ore contains such a small percentage of sulphides, concentration either by flotation or tabling would not be practical. The investigation consisted wholly of amalgamation and cyanide tests for the extraction of the gold content.

The behaviour of the ore at different screen sizes was investigated. Three samples of 1,000 grammes each were crushed to pass, 1st 65 mesh, 2nd 100 mesh, and 3rd 150 mesh. Amalgamation was tried on the coarsely ground lot, −65 mesh. Cyanide tests were made on all three. The tailings from the cyanide tests were screened through 150 mesh, and the

+150 and −150 portions assayed separately. This was done to note any difference in the gold content of the coarser and finer material in the sample under investigation. The tests in detail follow.

Test No. 1

Amalgamation at 65 mesh

A sample, 1,000 grammes, of ore −65 mesh, was amalgamated with mercury. The amalgam was separated, and the tailing assayed.

Screen Sizes—	Per cent
+ 65	0·0
−65 +100	48·0
−100+150	10·5
−150+200	12·5
−200	29·0
	100·0

Head assay	0·77 Au, oz./ton
Tailing assay	0·44 "
Recovery	42·9 per cent

It appears that the gold is not freed at this fineness of crushing, as indicated by the low recovery by amalgamation. Finer grinding would not be practical as loss of fine gold would result.

Test No. 2

Cyanidation at −65 mesh

A sample, 200 grammes of ore, −65 mesh, was agitated for 48 hours with 400 grammes of sodium cyanide solution containing 2 pounds KCN per ton. Lime was added at the rate of 5 pounds per ton of ore.

Strength of solution after 48 hours, KCN	1·9 lb./ton
CaO	0·28 "
Consumption, KCN	0·20 lb./ton
CaO	4·4 "

—	Weight, %	Assay, Au, oz./ton
Heads		0·77
Tailing, +150 mesh	59·5	0·12
Tailing, −150 mesh	40·5	0·04
Average tailing	100·0	0·09

Recovery .... 88·3 per cent

As shown by the assays of the +150 and −150 portions of the tailing, it is necessary to grind the ore finer than −65 mesh to secure a high recovery of the gold.

Test No. 3

Cyanidation at −100 mesh

A sample, 200 grammes of ore, −100 mesh, was cyanided under the same conditions as those of Test No. 2.

Strength of solution after 48 hours, KCN.......................... 1·85 lb./ton
CaO.......................... 0·10 "

Consumption, KCN.. 0·30 lb./ton
CaO.. 4·8 "

—	Weight, %	Assay, Au, oz./ton
Heads..		0·77
Tailing, +150 mesh..	25·4	0·08
Tailing, −150 mesh..	74·6	0·03
Average tailing..	100·0	0·04

Recovery... 94·8 per cent

The assays of the +150 and −150-mesh portions of the tailing again show a marked difference. Finer grinding than −100 mesh will yield higher recoveries.

Test No. 4

Cyanidation at −150 mesh

A sample, 200 grammes of −150-mesh ore, was cyanided under the same conditions as Tests Nos. 2 and 3.

Strength of solution after 48 hours, KCN.......................... 1·80 lb./ton
CaO.......................... 0·10 "

Consumption, KCN.. 0·40 lb./ton
CaO.. 4·8 "

—	Weight, %	Assay, Au, oz./ton
Heads..		0·77
Tailing, +200 mesh..	21·6	0·05
Tailing, −200 mesh..	78·4	0·02
Average tailing..	100·0	0·026

Recovery... 96·6 per cent

This test shows that fine grinding is necessary to secure a high recovery of the gold.

Test No. 5

This test was made on the same class of material as Test No. 4, the ore crushed to pass 150 mesh. Conditions of cyanidation were also the same, with the exception that 6 pounds of lime per ton of ore was added instead of 5 pounds. Time of agitation was 48 hours.

Strength of solution after 48 hours, KCN	1·80 lb./ton
CaO	0·95 “

Consumption, KCN	0·40 lb./ton
CaO	4·1 “

—	Assay, Au, oz./ton
Heads	0·77
Tailing	0·02

Recovery.. 97·4 per cent

This test shows that a higher addition of lime is slightly beneficial, resulting in an increase in the recovery, and also in a small saving in lime consumed.

Test No. 6

This test was made under the same conditions as Test No. 5, and run for 72 hours, instead of 48 hours.

Strength of solution at start, KCN	2·0 lb./ton
Lime added	6·0 “
Strength of solution after 72 hours, KCN	1·80 “
CaO	0·80 “

Consumption, KCN	0·40 lb./ton
CaO	4·4 “

—	Assay, Au, oz./ton
Heads	0·77
Tailing	0·02

Recovery.. 97·4 per cent

This test shows that no advantage is obtained by increasing the time of agitation longer than 48 hours. The cyanide consumed remains the same, 0·40 pound per ton of ore, while the lime consumption shows a slight increase.

The following tabulation of the cyanide tests shows the results secured in the various tests.

Test No.	Mesh grinding	Time of agitation, hours	KCN consumed, lb./ton	Lime consumed, lb./ton	Tailing, Au, oz./ton	Recovery, per cent
2....................	− 65	48	0·20	4·4	0·09	88·3
3....................	−100	48	0·30	4·8	0·04	94·8
4....................	−150	48	0·40	4·8	0·026	96·6
5....................	−150	48	0·40	4·1	0·02	97·4
6....................	−150	72	0·40	4·4	0·02	97·4

CONCLUSIONS

The test work done shows quite clearly that cyanidation of the ore will yield a high recovery of the gold content. Fine grinding is indicated, as shown by the analysis of the tailing from the various tests. When the grinding is such that 59·5 per cent of the ore remains on 150 mesh, the tailing assay is 0·04 ounce, and when all passes 150 mesh, with 78 per cent through 200 mesh, the tailing is 0·02 ounce.

The process indicated is an all-slime cyanide mill, using from 6 to 7 pounds lime per ton of ore, and a cyanide solution of from 1 to 2 pounds cyanide per ton.

The ore as supplied contains practically no sulphides. Concentration, therefore, will find no place in the flow-sheet unless at depth the nature of the ore-body should change.

Due to the absence of refractory minerals or substances harmful to the cyanide process, the problem is a simple one, a high recovery being obtained on fine grinding.

Report No. 320

THE RECOVERY OF GOLD FROM THE ORE OF THE BRITISH CANADIAN MINES, LTD., MINE CENTRE, ONTARIO

A. K. Anderson

Shipments. Two hundred pounds of ore was received at the Ore Dressing Laboratories, consigned from Mine Centre, Ontario.

Characteristics of the Ore. The ore consisted of quartz and basic rock, containing small amounts of iron pyrite. These sulphides were segregated along fractures in the predominating quartz portion of the ore.

Purpose of Experimental Tests. The investigation was made to determine what recovery of the gold in the ore can be obtained by amalgamation and by cyanidation, and to determine the most suitable method of milling to employ.

Sampling and Analysis. The total lot was crushed to pass 14 mesh, passed through a Jones riffle sampler and after crushing through 35 mesh and 65 mesh, with intervening cuts being made through the sampler, a portion was secured for analysis. This showed an assay value of 0·60 ounce gold per ton. A trace of copper was also reported, no other economic minerals being present.

As the percentage of sulphides present was low, concentration was not applied, the investigation consisting of amalgamation and cyanide tests to determine the adaptability of the ore to these processes.

The behaviour of the material at different sizes was investigated. Three samples of 1,000 grammes each were crushed to pass, 1st 65 mesh, 2nd 100 mesh, and 3rd 150 mesh. Amalgamation was tried on the coarsely ground lot, −65 mesh. Cyanide tests were made on all three. The tailings from the cyanide tests were screened through 150 mesh, and the +150 and −150-mesh portions assayed separately. This was done to note any difference in the gold content of the coarser and finer material in the sample under investigation. The detailed tests follow:

Test No. 1

Amalgamation at 65 mesh

A sample, 1,000 grammes of the ore, −65 mesh, was amalgamated with mercury, the amalgam separated and the tailing assayed.

Screen Sizes—	Per cent
+ 65	0·0
− 65+100	41·7
−100+150	11·6
−150+200	13·8
−200	32·9
	100·00

—	Assay, Au, oz./ton
Heads	0·60
Tailing	0·11

Recovery........ 81·7 per cent

This test shows that a large amount of the gold is readily recovered by amalgamation. However, the residue still contains considerable gold, which to be recovered, would necessitate finer grinding of the amalgamation tailing.

Test No. 2

Cyanidation at 65 mesh

A sample, 200 grammes of ore, −65 mesh, was agitated for 48 hours with 400 grammes of sodium cyanide solution containing 2·0 pounds KCN per ton. Lime was added at the rate of 5 pounds per ton of ore.

Strength of solution after 48 hours, KCN	1·85 lb./ton
CaO	0·30 "
Consumption, KCN	0·30 lb./ton
CaO	4·4 "

—	Weight, %	Assay, Au, oz./ton
Heads		0·60
Tailing +150 mesh	60·0	0·01
Tailing −150 mesh	40·0	0·01
Average tailing		0·01

Recovery	98·3 per cent

This test shows that material crushed to pass 65 mesh, with 41·7 per cent remaining on 100 mesh, yields quite readily to cyanidation.

Test No. 3

Cyanidation at −100 mesh

A 200-gramme sample of ore, −100 mesh, was cyanided under the same conditions as those of Test No. 2.

Strength of solution after 48 hours, KCN	1·85 lb./ton
CaO	0·20 "
Consumption, KCN	0·30 lb./ton
CaO	4·6 "

—	Weight, %	Assay, Au, oz./ton
Heads		0·60
Tailing +150 mesh	22·2	0·01
Tailing −150 mesh	77·8	0·005
Average tailing		0·006

Recovery	99·0 per cent

This test confirms the results obtained in the preceding test, the extraction of the gold is high at coarse sizes. A slight increase is noted by grinding finer than 65 mesh.

Test No. 4

Cyanidation at −150 mesh

A 200-gramme sample of ore −150 mesh was cyanided under conditions similar to those of Tests Nos. 2 and 3.

Strength of solution after 48 hours, KCN	1·70 lb./ton
CaO	0·15 "
Consumption, KCN	0·60 lb./ton
CaO	4·7 "

—	Weight, %	Assay, Au, oz./ton
Heads		0·60
Tailing +200 mesh	16·1	0·015
Tailing −200 mesh	83·9	0·005
Average tailing		0·007

Recovery .... 98·8 per cent

This test shows that no advantage is to be gained by fine grinding. A higher cyanide consumption results from cyaniding at this mesh, −150.

Test No. 5

This test was made on the same class of material as Test No. 4, ore ground to pass 150 mesh. Conditions of cyanidation were also the same, with the exception that 6 pounds lime per ton of ore was added instead of 5 pounds. Time of agitation was 48 hours.

Strength of solution after 48 hours, KCN	1·75 lb./ton
CaO	1·00 "
Consumption, KCN	0·50 lb./ton
CaO	4·00 "

—	Assay Au, oz./ton
Heads	0·60
Tailing	0·005

Recovery .... 99 per cent

This checks the results of the previous tests, showing that the gold is quite readily soluble in cyanide solution, and that a higher cyanide consumption results from agitating the finely ground material.

The following summary of the various cyanide tests shows the results obtained.

Test No.	Mesh grinding	Time of agitation, hours	KCN consumed, lb./ton	Lime consumed, lb./ton	Tailing, Au, oz./ton	Recovery, %
2	− 65	48	0·30	4·4	0·01	98·3
3	−100	48	0·30	4·6	0·006	99·0
4	−150	48	0·60	4·7	0·007	98·8
5	−150	48	0·50	4·0	0·005	99·0

The ease with which the gold is extracted by cyanide makes the milling of this ore a simple problem; 81·7 per cent of the metal is recovered by amalgamation, but leaves $2.20 in the residue. Cyanidation of this amalgamation tailing would result in the saving of the contained values. However, 98·3 per cent of the gold is recovered by cyaniding at the same coarse grinding that yielded 81·7 per cent recovery by amalgamation. This shows that no metallurgical advantage will be gained by employing amalgamation.

By grinding to pass 100 mesh, with 22 per cent remaining on 150 mesh, a recovery of 99 per cent is secured. No advantage is obtained by grinding finer, in fact a slightly higher consumption of cyanide results.

The process indicated is treatment by cyanide, with ore ground between 65 and 100 mesh. The absence of refractory material and minerals harmful to the cyanide process ensures clean mill solutions, with no decrease in efficiency.

Report No. 321

THE INVESTIGATION OF THE RECOVERY OF GOLD AND SILVER IN THE ORE OF THE HOWEY GOLD MINES, LTD.

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

Shipments. Two shipments of ore were received from the Howey Gold Mines, Limited, consigned from Hudson, Ont. The first consisted of 34 bags, gross weight 4,000 pounds and received on January 23, 1929; while the second, contained in 42 bags, arrived on February 8, 1929. All tests were made on the first shipment.

Characteristics of the Ore. The ore consists of hard quartz and porphyry containing about 5 to 6 per cent iron sulphides, and containing gold and silver values.

Purpose of Experimental Tests. Due to the location of the property, and high transportation charges on material and supplies, a careful study of metallurgical practice was requested in order to devise the most economical method to apply to the treatment of the ore. The manager of the company desired that representative samples of the shipment be sent to the Southwestern Engineering Company, Los Angeles, California, and to the General Engineering Company, Salt Lake City, Utah.

Sampling and Analysis. These two samples, 150 pounds to the Southwestern Engineering Company, and 250 pounds to the General Engineering Company, were secured as follows: The total shipment No. 1, 4,000 pounds, was crushed in a jaw crusher to ¾-inch, and then through rolls to 4 mesh. One-tenth of this material was cut out by an automatic sampler, giving a portion weighing 399 pounds. This was crushed through the rolls to pass 10 mesh, and cut in half through a Jones riffle sampler yielding two portions of 200 pounds each. One of these was cut in half, and one of the 100-pound portions added to the other 200-pound portion. This 300-pound lot was cut in half, thus giving 150 pounds, which was sent to the Southwestern Engineering Company. The other 150-pound half was added to the 100 pounds remaining from the preceding cut giving 250 pounds, which was forwarded to the General Engineering Company.

The remaining 3,600 pounds was again sampled similarly to yield a representative 10-mesh portion on which the test work was done in the Ore Dressing and Metallurgical Laboratories. This gave on assay:—

Gold	0·71 oz./ton	Zinc	0·04 per cent
Silver	0·32 "	Lead	0·03 "
Iron	2·68 per cent	Sulphur	2·27 "

Experimental Tests

All tests were carried out on material coarsely ground, −40 mesh. These were conducted with the object of comparing the results secured by straight all-cyaniding and by combined concentration and cyaniding. Concentration was made both by flotation and by tabling, followed by cyanidation of the concentrate.

TABLING AND FLOTATION

Test No. 1

A sample of 5,000 grammes of ore, −10 mesh, was ground for 10 minutes in a ball mill, in 1 : 1 pulp. After grinding, the sample was allowed to settle in a bucket, and the water decanted through a filter. The solids were then tabled. Considerable pyrite was observed to pass into the tailing during this operation. To recover these sulphides, the sand portion of the tailing was floated. The slimes were not floated, but were added to the slimes from the decantation made after grinding.

Product	Weight, %	Assay, Au, oz./ton	Per cent of values
Table concentrate	6·16	8·08	69·75
Table middling	7·04	1·49	14·70
Flotation concentrate	2·35	3·05	10·08
Flotation tailing + 35 mesh	5·79	0·10	0·84
− 35+ 48	12·67	0·05	0·84
− 48+ 65	12·78	0·05	0·84
− 65+100	15·12	0·05	0·98
−100+150	2·56	0·05	0·14
−150+200	5·72	0·06	0·42
−200	8·35	0·02	0·28
Slimes not floated	21·46	0·04	1·13
	100·00		100·00

From the above results, it appears that from every 100 tons of feed, there are recovered 6·16 tons of table concentrate assaying 8·08 ounces gold per ton, containing 69·75 per cent of the gold values, 7·04 tons of table middling, assaying 1·49 ounces gold, and containing 14·70 per cent of the values; 2·35 tons of flotation concentrate assaying 3·05 ounces gold per ton, containing 10·08 per cent of the gold; a slime of 21·46 tons that was not floated, assaying 0·04 ounce gold, containing 1·13 per cent of the values, and a sand tailing from flotation weighing 62·99 tons, containing 0·05 ounce per ton and carrying 4·34 per cent of the gold in the original tonnage.

The combined table middlings, and the table and flotation concentrates contain 94·53 per cent of the gold.

Test No. 2

This test is a duplicate of Test No. 1, with the exception that the grinding was considerably coarser, approximately 10 per cent remaining on 48 mesh in this test, than in the preceding one. Also, the table middling produced was cleaned, and added to the concentrate.

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Heads		0·71	0·32		
Heads calculated from products		0·706	0·39		
Table concentrate	7·73	8·10	3·74	88·75	74·45
Table tailing		0·09	0·10		
Flotation concentrate	1·40	2·34	2·30	4·65	8·29
Flotation tailing + 35 mesh	11·87	0·11	0·07	1·85	2·14
− 35+ 48	16·79	0·08	0·06	1·90	2·60
− 48+ 65	20·30	0·04	0·04	1·15	2·09
− 65+100	8·46	0·03	0·03	0·35	0·64
−100+150	2·69	0·03	0·02	0·12	0·13
−150+200	5·10	0·02	0·03	0·14	0·39
−200	12·79	0·02	0·06	0·37	1·98
Slimes not floated	12·87	0·04	0·22	0·72	7·29

This test shows that for every 100 tons of feed there are recovered 7·73 tons of table concentrate assaying 8·10 ounces gold per ton and containing 88·75 per cent of the gold and 74·45 per cent of the silver, and a flotation concentrate weighing 1·4 tons, assaying 2·34 ounces gold, 2·30 ounces silver per ton, and carrying 4·65 per cent of the gold and 8·29 per cent of the silver in the original feed, or a combined recovery of 93·4 per cent of the gold and 82·7 per cent of the silver.

The lower results secured in this test can be attributed to coarser grinding. There is 10·2 per cent more material remaining on 48 mesh than in the preceding test, accounting for 3·75 per cent of the gold in the original feed as against 1·68 per cent in Test No. 1.

FLOTATION AND TABLING

Test No. 3

In this test, the procedure employed in the two preceding ones was reversed. The coarsely ground ore was floated first and then tabled to recover coarse pyrite and gold.

A sample, 5,000 grammes of the ore, was ground in a ball mill for 10 minutes, in 1 : 1 pulp, and transferred to the flotation machine. Following the addition of 0·12 pound per ton amyl xanthate and a small amount of wood tar together with pine oil sufficient to froth, a concentrate was removed. After flotation, the pulp was withdrawn from the machine and the slimes decanted. The sands were then tabled. The table middling product was recleaned and added to the table concentrate. Some of the larger sulphide particles were observed to pass into the tailing in the tabling operation.

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Flotation concentrate	7·95	5·30	2·90	67·79	74·28
Flotation tailing		0·18	0·08		
Table concentrate	4·33	1·77	0·46	12·40	6·43
Table tailing + 35 mesh	14·53	0·34	0·13	7·89	6·10
− 35+ 48	15·09	0·26	0·11	6·28	5·14
− 48+ 65	15·85	0·13	0·06	3·38	2·89
− 65+100	6·93	0·06	0·02	0·64	0·32
−100+150	6·99	0·03	0·02	0·32	0·32
−150+200	6·04	0·02	0·02	0·17	0·33
−200	12·36	0·04	0·04	0·81	1·62
Slimes	9·93	0·02	0·08	0·32	2·57
Average table tailing		0·12	0·05		

This test does not show so good results as the preceding one, a recovery of only 80 per cent of the gold and 81 per cent of the silver being obtained as against 93·4 per cent and 82·7 per cent in Test No. 2. The material was ground to the same size, 29 per cent of the ore remaining on 48 mesh.

FLOTATION

Test No. 4

In this test the effects of concentration by flotation were investigated.

A sample of 5,000 grammes of ore, −10 mesh, was ground to secure the same coarse sizes as in Test No. 3, and floated with amyl xanthate, 0·12 pound per ton, and pine oil.

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Flotation concentrate	8·16	6·15	3·47	79·76	80·79
Flotation tailing	91·84	0·13	0·07	20·24	19·21

The results secured are identical with those in Test No. 3, a recovery of 80 per cent of the gold, and 81 per cent of the silver being secured. Tabling followed by flotation appears to be superior to flotation followed by tabling and to straight flotation. In Test No. 2 there is a recovery of 93·4 per cent of the gold and 83 per cent of the silver, as against 80 and 81 per cent in Tests Nos. 3 and 4. The fact that sulphide particles passed into the tailing in the tabling operation in Test No. 3, is responsible for the lower recovery. On full-size tables properly operated, this condition would not present itself, consequently the discrepancy between these two tests may be disregarded.

TABLING, FLOTATION AND CYANIDING

Test No. 5

The concentration procedure in this test is similar to that in Tests Nos. 1 and 2. A sample, 25,000 grammes, of ore ground in a ball mill, 1 : 1 pulp. The slimes were decanted and the sands tabled. The table tailing was then floated and the concentrate produced combined with the table concentrate. These were then agitated in a 5-pound per ton KCN solution for 48 and 72 hours, without regrinding.

Due to the coarseness of grinding, free gold remained undissolved in the tailing. In practice, this gold would remain in the classifier and ball mill until finely ground. To get a fair comparison from this test, the tailing after cyaniding was amalgamated and the coarse gold removed before the sample was assayed.

Results of Concentration

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Heads		0·71	0·32		
Table concentrate	4·95	12·22	4·04	84·89	60·62
Flotation concentrate	1·47	4·03	2·75	8·31	12·25
Flotation tailing	75·88	0·05	0·09	5·32	20·70
Slimes not floated	17·70	0·06	0·12	1·48	6·43
	100·00			100·00	100·00

This shows that for every 100 tons of mill feed there is secured 6·42 tons of combined table and flotation concentrates with an assay value of 10·34 ounces gold and 3·74 ounces silver per ton, representing a recovery of 93·2 per cent of the gold and 72·9 per cent of the silver. The coarse crushing of the sample, 29 per cent on 48 mesh is responsible for the lower recoveries as compared with Test No. 1.

The combined flotation and table concentrates were divided into two portions and agitated for 48 and 72 hours, 1 : 3 dilution with sodium cyanide solution equivalent in strength to 5·0 pounds KCN per ton. Lime was added at the rate of 5 pounds per ton of ore.

—	48-hour agitation				72-hour agitation			
	Assay, oz./ton		Per cent of values		Assay, oz./ton		Per cent of values	
	Au	Ag	Au	Ag	Au	Ag	Au	Ag
Heads	10·34	3·74			10·34	3·74		
Cyanide tailing	0·81	1·16	7·84	31·02	0·55	0·88	5·32	23·53
Recovery from cyaniding			92·16	68·98			94·68	76·47

Details of Tests

	48 hours	72 hours
Strength of solution at start, KCN lb./ton	5·0	5·0
Strength of solution at finish, KCN lb./ton	3·3	3·1
Lime added, lb./ton ore	5·0	5·0
Alkalinity at finish, lb./ton of solution	0·2	0·10
Cyanide consumed, lb./ton ore	5·1	5·7
Lime consumed, lb./ton ore	4·4	4·7
Ratio of solution to ore	3:1	3:1

Calculation of Results based on Mill Feed

—	Weight, %	48-hour agitation				72-hour agitation			
		Assay, oz./ton		Per cent of values		Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag	Au	Ag	Au	Ag
Heads		0·71	0·32			0·71	0·32		
Flotation tailing	75·88	0·05	0·09	5·34	21·34	0·05	0·09	5·34	21·34
Slimes not floated	17·70	0·06	0·12	1·49	6·63	0·06	0·12	1·49	6·63
Cyanide tailing	6·42	0·81	1·16	7·32	23·28	0·55	0·88	4·97	17·66
Extraction by cyaniding				85·85	48·75			88·20	54·38
	100·00			100·00	100·00			100·00	100·00

The following screen analysis of the tailing from the 48-hour cyanidation of the concentrates is of interest.

Product, mesh	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
+ 35	7·60	1·52	0·66	14·23	4·31
− 35+ 48	10·72	1·42	0·88	18·49	8·10
− 48+ 65	25·01	1·18	0·84	36·36	18·03
− 65+100	20·43	0·52	0·78	13·09	13·68
−100+150	10·46	0·50	1·01	6·44	9·07
−150+200	9·96	0·34	1·33	4·18	11·37
−200	15·82	0·37	2·61	7·21	35·44
	100·00			100·00	100·00

This shows that the material remaining on 100 mesh, contains 82 per cent of the gold and 44 per cent of the silver not extracted by cyanide from the table and flotation concentrates. This indicates quite clearly that the concentrates require to be reground before cyanidation.

Test No. 6

This test parallels the procedure in Test No. 5 with the exception that finer grinding was attempted; 5,000 grammes of ore was ground for 22 minutes in a ball mill, 1 : 1 dilution with water, passed over the Wilfley table, and the sand portion of the tailing floated. The table and flotation concentrates were combined, and cyanided for 48 and 72 hours.

Results of Concentration

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Heads		0·71	0·32		
Table concentrate	4·37	12·92	3·82	89·55	65·42
Flotation concentrate	2·66	1·38	1·38	5·82	14·39
Table slimes, not floated	23·86	0·05	0·10	1·89	9·37
Flotation tailing	69·11	0·025	0·04	2·75	10·82
	100·00			100·00	100·00

This shows that from 100 tons of feed there is obtained a combined table and flotation concentrate of 7·03 tons, assaying 8·56 ounces gold and 2·90 ounces silver per ton. This represents a recovery of 95·4 per cent of the gold and 79·8 per cent of the silver.

The concentrate was divided into two parts and cyanided for 48 and 72 hours. The tailings after cyaniding were screened on 150 mesh, each part being assayed separately.

—	48-hour agitation					72-hour agitation				
	Weight, %	Assay, oz./ton		Per cent of values		Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag		Au	Ag	Au	Ag
Heads		8·56	2·90				8·56	2·90		
Cyanide tailing, +150	42·06	0·94	0·49	4·6	7·2	40·0	0·65	0·49	3·1	6·8
Cyanide tailing, −150	57·94	0·53	0·76	3·6	15·2	60·0	0·42	0·74	2·9	15·3
Average tailing		0·70	0·65				0·51	0·64		
Recovery by cyaniding				91·8	77·6				94·0	77·9

Details of Tests

	48 hours	72 hours
Strength of solution at start, KCN lb./ton	5·0	5·0
Strength of solution at finish, KCN lb./ton	2·6	2·1
Lime added, lb./ton ore	5·0	5·0
Alkalinity at finish, lb./ton solution	0·5	0·5
Cyanide consumed, lb./ton ore	4·8	5·8
Lime consumed, lb./ton ore	4·0	4·0
Ratio of solution to ore	2:1	2:1

Calculation of Results based on Mill Feed

—	Weight, %	48-hour agitation				72-hour agitation			
		Assay, oz./ton		Per cent of values		Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag	Au	Ag	Au	Ag
Heads		0·71	0·32			0·71	0·32		
Flotation tailing	69·11	0·025	0·04	2·39	8·75	0·025	0·04	2·43	8·64
Slimes not floated	23·86	0·05	0·10	1·55	7·50	0·05	0·10	1·68	7·46
Cyanide tailing	7·03	0·70	0·65	6·90	14·38	0·51	0·64	5·05	14·06
Extraction by cyanide				89·16	69·37			90·84	69·84
	100·00			100·00	100 00			100·00	100·00

A screen test made on the original feed in this test showed the following:

Mesh—	Per cent
+ 48	4·4
− 48+ 65	10·1
− 65+100	16·3
−100+150	7·1
−150+200	7·4
−200	54·7
	100·0

From an examination of the results of this test, it is seen that the material remaining on 150 mesh in the residues after cyanidation of the concentrates is of higher grade than that passing through 150 mesh. This confirms the conclusion drawn from Test No. 5, that to secure maximum results, the concentrates must be reground before cyaniding.

ALL CYANIDATION

Test No. 7

A portion of the ore was ground dry to pass 48 mesh, giving the following screen test.

Mesh—	Per cent
+ 65	12·5
− 65+100	20·2
−100+150	16·0
−150+200	11·8
−200	39·5
	100·0

Three lots of this −48-mesh material was agitated in cyanide solution for 24, 48, and 72 hours, with the following results.

Time of agitation	Head assay, oz./ton		Tailing assay, oz./ton		Recovery, %	
	Au	Ag	Au	Ag	Au	Ag
24 hours	0·71	0·32	0·03	0·05	95·8	84·4
48 "	0·71	0·32	0·02	0·05	97·2	84·4
72 "	0·71	0·32	0·03	0·04	95·8	87·5

Details of Test

Time of agitation	Solution at start, KCN, lb./ton	Solution at finish, KCN, lb./ton	Lime added, lb./ton	Alkalinity of solution at finish, CaO lb./ton	KCN consumed, lb./ton	Lime consumed, lb./ton	Ratio of solution to ore
24 hours	2·45	2·40	6·0	1·50	0·10	3·0	2:1
48 "	2·45	2·30	6·0	1·30	0·30	3·4	2:1
72 "	2·45	2·40	6·0	1·20	0·10	3·6	2:1

These tests show that 97·2 per cent of the gold and 84·4 per cent of the silver in the ore is extracted in 48 hours. Finer grinding than −48 mesh, with 40 per cent through 200 mesh, is unnecessary. The consumption of cyanide is low, from 0·10 to 0·30 pound per ton of ore milled. The addition of 6 pounds lime per ton of feed leaves the alkalinity of the pulp after agitation sufficiently high to cause prompt settling in the thickeners.

GRINDING—CLASSIFICATION AND CYANIDING

Test No. 8

Ore crushed to pass 14 mesh was fed to a small rod mill in closed circuit with a drag classifier, at the rate of 22 pounds per hour. Lime was added at the rate of 4 pounds per ton, and cyanide solution containing 1·5 pounds KCN per ton. After adjusting the rod load to the fineness of grinding necessary, and the solution fed to give a classifier overflow of 47·5 per cent solids, the pulp was passed to a Pachuca agitator. This agitator was fitted with a bottom discharge through which the pulp was drawn by a centrifugal pump, and returned to the top of the tank. This ensured positive agitation of all coarse particles. After 6½ hours the agitator was full, containing approximately 160 pounds of ore. Agitation was continued for 42 hours, with additions of cyanide to maintain a solution strength of 1·5 pounds KCN per ton. Lime was added from time to time as indicated by titrations to maintain a protective alkalinity of about 0·4 pound CaO per ton of solution.

During the grinding periods, samples of the classifier overflow were taken to determine the percentage of dissolution in the grinding circuit. After agitation, the pulp was discharged through a filter press, washed, and sampled.

To secure information on the condition of the sulphides under the conditions of grinding adopted, a screen test was made on sulphides concentrated from a representative portion of the cyanide tailing.

Classifier Overflow

Sample No.	Grinding, %				Strength of solution, lb./ton		Tailing	Recovery, %
	+48	−48 +65	− 65 +200	−200	KCN	CaO	Assay, Au, oz./ton	
1		2·1	37·0	60·9	0·90	0·20	0·07	89·5
2	1·8	9·7	43·3	45·2	0·60	0·70	0·15	77·6

These samples show what is usually the case in cyanide practice, a high percentage of the gold is dissolved in the circuit.

Results of Agitation

Time of agitation	Tailing, assay, Au, oz./ton	Recovery, %
½ hour	0·10	85·1
24 hours	0·03	95·5
42 "	0·017	97·4

The above results indicate with what ease the gold is extracted by cyanide. Before the pulp reaches the agitation system, approximately 85 per cent of the gold is in solution. The remainder is dissolved without any interference by deleterious substances in the ore, so that within 42 hours a total extraction of 97·4 per cent is obtained.

Screen Analysis of Final Tailing

Product, mesh	Weight, %	Assay, Au, oz./ton	Per cent of values
+48	1·9	0·04	4·4
− 48+ 65	7·6	0·03	13·2
− 65+100	18·1	0·02	21·0
−100+150	10·5	0·03	18·3
−150+200	12·4	0·02	14·4
−200	49·5	0·01	28·7
	100·0		100·0
Average assay		0·017	

This analysis shows a consistent drop in the gold content as the sizes of the particles decrease.

A comparison between the above screen test made on the ore as agitated and one on the sulphides removed by concentration from the same material is of interest.

Mesh	Ore, %	Sulphides, %
+ 65	9·5	2·2
− 65+100	18·1	22·1
−100+150	10·5	16·3
−150+200	12·4	11·1
−200	49·5	48·3
	100·0	100·0

Although there is a marked decrease in the amount of material remaining on 65 mesh in the sulphides, as compared with that in the original grind, there is an increase of 10 per cent in the −65+150-mesh sizes, with approximately the same amount through 200 mesh. It would appear that the work done in the grinding circuit is expended on the coarser sizes with shielding of the sulphide particles. Manipulation of the grinding circuit is indicated in order to secure concentration of the sulphides in the classifier. In this way they will be returned to the grinding mills for further reduction in size.

SUMMARY

All Cyanidation. Tests on the ore ground to pass 48 mesh indicate that a recovery of 97·2 per cent of the gold and 84 per cent of the silver can be obtained with 48 hours' agitation. In 24 hours, 96 per cent of the gold and 84 per cent of the silver is secured. A solution containing 2·45 pounds KCN per ton was used, and a cyanide consumption of from 0·10 to 0·30 pound per ton of ore is indicated.

Grinding in cyanide solution 1·5 pounds KCN per ton, in closed circuit with a classifier to overflow at −40 mesh, gives an extraction of 97·4 per cent of the gold after 42 hours' agitation. An extraction of 85 per cent is obtained in the grinding circuit.

Tabling and Flotation. Tabling at coarse sizes, 18 per cent + 48 mesh, followed by flotation of the sands gives a combined concentrate and middling assaying 4·34 ounces gold per ton, and containing 94·5 per cent of the values. The discarded slimes, which were not floated, contained 1·13 per cent of the gold, and the sand tailing from flotation, the remaining 4·34 per cent. As the slime passes 200 mesh and the −200 portion of the tailing from flotation assays one-half that of the unfloated slime, it can be inferred that by floating this, one-half of the contained gold will be recovered, thus raising the total recovery to 95 per cent. A ratio of concentration of 6·4 : 1 is obtained.

When coarser crushing is employed, 28 per cent +48 mesh, there is a decrease in recovery from 95 per cent to 93·7 per cent with a ratio of concentration of 10·9 : 1.

As shown in Test No. 6, when the crushing is such that 4·4 per cent remains on 48 mesh, there is a recovery by concentration of 96·3 per cent of the gold and 80 per cent of the silver, with a ratio of concentration of 14·2 : 1.

Flotation. Concentration by flotation alone, at coarse sizes, 28 per cent +48 mesh, gives a recovery of 80 per cent of the gold and 81 per cent of the silver in a product assaying 6·15 ounces gold and 3·47 ounces silver, ratio of concentration 12·2 : 1.

Flotation at this size must be supplemented by table concentration to secure a high recovery.

Tabling, Flotation, Cyaniding of Concentrates. A combination of table and flotation concentration at −40 mesh yields a product containing 95·4 per cent of the gold and 80 per cent of the silver in the table feed. This concentrate assays 8·56 ounces gold and 2·90 ounces silver per ton. Ratio of concentration 14·2 : 1.

Cyanidation of this concentrate for 72 hours, without regrinding, yields 94 per cent of the gold and 78 per cent of the silver, or 90·8 per cent and 69·8 per cent respectively, calculated on the original table feed.

Cyanide consumption was 5·8 pounds KCN per ton of concentrate or 0·32 pound sodium cyanide per ton of table feed.

The above calculation does not take into consideration the additional recovery of 0·85 per cent of the total gold that can be expected from the flotation of the table slimes.

Comparison of All-cyanidation vs. Concentration and Cyanidation of Concentrates

—	Cyanidation		Concentration	
	Au	Ag	Au	Ag
Head assay, oz./ton	0·71	0·32	0·71	0·32
Tailing assay, oz./ton	0·02	0·05	0·06	0·09
Recovery, per cent	97·2	84·4	90·8	69·8
Cyanide consumption, lb. sodium cyanide per ton ore	0·24		0·32	

Allowing an additional recovery of 0·85 per cent of the gold from the unfloated slimes, thereby increasing the recovery by the combined concentration-cyanidation method to 91·6 per cent of the gold, there is a saving in favour of all-cyanidation of 80 cents per ton of ore milled, also a saving of 0·08 pound sodium cyanide.

Cyanide losses in filter cakes and from the formation of cyanide compounds throughout the system will increase slightly the total cyanide consumed approximately 0·1 pound over that indicated in the tests.

It is to be noted that 85 per cent of the gold is dissolved in the grinding and classification circuit. This will be of material advantage in reducing the time of pulp agitation below 48 hours, 36 hours should prove ample.

The recommendation made is that an all-cyanide plant be erected to treat this ore. Two-stage grinding is advised, a primary rod-mill, low discharge, in closed circuit with a Dorr simplex classifier, the overflow of which passes to a Dorr bowl classifier. The slimes from this second classifier to overflow through a thickener, the underflow of which passes to the agitators. The sands from the bowl classifier to pass to a trunnion-discharge Hardinge ball mill in closed circuit with a standard Dorr duplex classifier, the overflow passing through a Dorr tray thickener to the agitation system.

Investigation of the power situation in the district shows that the cost of slightly increasing the fineness of grinding will not materially affect operating expenditures. It is advised that the grinding be such that not more than 2 per cent remains on 65 mesh. This will ensure the sulphide particles being ground sufficiently fine to yield readily their contained gold, and will act as a safeguard against coarse gold finding its way to the agitators.

The ore yields to cyanidation with remarkable ease, and as it is free from any large amounts of mineral liable to cause a decrease in the efficiency of the cyanide solutions, trouble from fouling should not appear to any marked extent.

The simplicity of operation of a cyanide plant as compared with the rather close control necessary in a flotation mill is another reason for the adoption of cyanidation.

Everything taken into consideration, the all-cyanide system has many things to recommend its adoption, higher recovery, less complicated operation by workmen not so highly skilled as expert flotation operators, and less possibilities of mechanical loss.

Report No. 322

CONCENTRATION OF THE LOW-GRADE BANDED IRON ORE FROM KAMINISTIKWIA, ONTARIO

C. S. Parsons

Shipment. A carload containing 22·5 tons of iron ore was received on January 28, 1929, from N. M. Paterson, Fort William, Ontario.

Location of Deposit. The deposit from which the shipment was taken is near Kaministikwia on the Canadian Pacific railway, about 20 miles west of Port Arthur, Ontario.

Characteristics of Shipment. The ore was taken from the surface and consisted of bands of iron oxide (magnetite and hematite mixed) and red quartz (jasper). The banded structure of this ore has been badly folded and broken and pieces of the ore showed fractures across the bedding which had later been filled with hematite.

Very few bands of iron oxide were less than $\frac{1}{8}$ inch in thickness and very few bands were over $\frac{1}{2}$ inch.

It, therefore, should be possible to obtain a separation by crushing to $\frac{3}{4}$ inch and not finer than $\frac{1}{8}$ inch. The crumpled structure of the ore, however, increased the difficulty of separation at these sizes and this coarse crushing did not free some of the thin bands of iron which seem to be attached firmly to the walls of the jasper. It also did not free the iron in the fracture veinlets which cross the bedding.

The iron occurs as a mixture of magnetite and hematite. In this particular shipment the proportions of magnetite to hematite apparently vary from place to place even in the same band, and no distinct bands of hematite were observed.

The bands of iron themselves are not high grade; they contain finely disseminated silica which requires approximately 100-mesh grinding to free. The highest grade bands probably do not run over 57 per cent Fe and some not over 50 per cent Fe. All these factors complicate coarse concentration methods.

Analysis of Shipment: Sample dried at 212° F.

	Per cent
Fe	31·0
Fe_2O_3	35·92
FeO	7·56
Mn	0·01
Al_2O_3	0·50
CaO	0·80
MgO	0·58
P	0·075
*S	0·07
SiO_2	52·42

*(Probably high from salting)

H_2O less than 1 per cent.

The proportions of magnetite and hematite were calculated from the above analysis and are as follows:—

Hematite (Fe_2O_3)..	19·12 per cent
Magnetite (Fe_3O_4)..	24·36 "

Therefore of the total iron present—

56·9 per cent is present as Fe_3O_4
41·1 per cent is present as Fe_2O_3

Purpose of Experimental Work: An ore of the above grade is not commercial ore. No blast furnace will use such low-grade material particularly when the silica content is so high. An iron ore in order to be readily marketed must contain over 50 per cent iron and not over 12 per cent silica, and in order to be of Bessemer grade should not contain more than 0·1 per cent phosphorus and no additional bonus is added to the base rate for Bessemer ores until the phosphorus is below 0·045 per cent.

There is a market for a limited quantity of siliceous ore. The consumption of siliceous ores amounts to about 2 per cent of the total Lake shipments.

The purpose, therefore, of the experimental work was to raise the grade of the ore by some means of beneficiation. The owners in submitting this sample requested that beneficiation be confined to methods employing coarse crushing and two types of tests were suggested, namely concentration by jigs and tables and concentration by magnetic cobbing.

EXPERIMENTAL TESTS

Gravity Concentration in Jigs and Tables

Two gravity tests were made, Nos. 1 and 2 (*see* Figures 1 and 2). These two flow-sheets show each step taken in making the tests.

Test No. 1

The ore was crushed in a jaw crusher to $1\frac{1}{4}$-inch size and in rolls set at $\frac{3}{4}$ inch and then sized on $\frac{3}{4}$-inch screen, the oversize being returned to the rolls until all passed through the $\frac{3}{4}$-inch round hole screen. The following sizes were made on a shaking screen: $-\frac{3}{4}+\frac{1}{2}$ round hole, $-\frac{1}{2}+(\frac{1}{4} \times \frac{1}{2})$ tongue cap hole, $-(\frac{1}{4} \times \frac{1}{2}) + (\frac{1}{8} \times \frac{1}{2})$ tongue cap hole. Further screening of the $-(\frac{1}{8} \times \frac{1}{2})$ was done wet on a Hummer screen making $-(\frac{1}{8} \times \frac{1}{2})$ + 7-mesh Tyler and $-7+10$-mesh Tyler and -10 mesh. With the exception of the -10-mesh product each of the above sizes was jigged. The -10 mesh was classified in a Richards launder type hydraulic classifier. The spigot product was tabled on a Butchart table making a concentrate, a middling, and a final tailing. The overflow from the classifier was thickened and then tabled on a large Wilfley table.

The middlings obtained from the jigs were recrushed by rolls and sized, then rejigged and the middling from the rejigging was again crushed and rejigged until no +10-mesh middling remained. In practice these middlings after crushing would be returned back with the original feed to the screens.

The system of jigging was as follows: two jigs were operated in tandem and a third jig re-treated the low-grade concentrate from the second jig, which produced finished concentrate and a middling for recrushing. The first jig made concentrate; the second jig a low-grade concentrate and a final tailing, the low-grade concentrate being rejigged as described above.

The following tables give the results of the test.

Products	Weight		Assay Fe, %	Content Fe, lb.	Fe content, %
	lb.	%			
Screen Products—					
−¾+½ size	5,835	26·6	34·2	1,995·6	
−½+¼ "	6,100	27·8	32·6	1,988·6	
−¼+⅛ "	5,008	22·8	32·6	1,958·7	
−⅛	5,019	22·8	29·6	1,485·6	
Totals	21,962		32·45	7,128·5	
−¾+½ jig concentrate	1,815	31·5	50·0	907·5	47·3
" middling	2,744	47·7	29·7	815·0	42·5
" tailing	1,193	20·7	16·3	194·5	10·1
Totals	5,752			1,917·0	99·9
−½+¼ jig concentrate	1,591	26·1	54·8	871·9	54·8
" middling	2,200	37·1	32·5	734·5	38·2
" tailing	2,246	36·8	14·14	316·7	16·5
Totals	6,097	100·0		1,923·1	99·9
−¼+⅛ jig concentrate	1,455	30·0	52·93	769·7	52·8
" middling	1,780	36·7	27·70	493·1	33·8
tailing from No. 2 jig	1,210	25·0	12·04	145·2	13·4
tailing from No. 3 jig	407	8·4	12·06	49·2	
Totals	4,852	100·1		1,457·2	100·0

Products	Weight		Assay, Fe, %	Content, Fe, lb.
	lb.	%		
−¾+½ middling recrushed gives	2,744		29·70	815·0
−½+¼ from recrushing −¾+½ middling	505	18·9	32·17	162·6
−¼+⅛ " " −¾+½ "	1,354	50·7	30·40	411·6
−⅛ " " −¾+½ "	810	30·3	28·44	230·0
Totals	2,669	99·9	30·1	804·2
Products from −¾+½ middling—				
−½+¼ concentrates from ¾+½ middling	91	18·6	51·06	46·5
−½+¼ middling	296	60·7	29·88	88·5
−½+¼ tailing	101	20·7	13·18	13·3
Totals	488	100·0	30·4	148·3
Summary of −½+¼ middlings to be recrushed—				
(1) −½+¼ middling	2,260		32·5	734·5
(2) −½+¼ middling from jig recrushing −¾+½ middling	296		29·88	88·5
Totals	2,556		32·2	823·0
Results recrushed −½+¼ middling—				
−¼+⅛ from recrushing middling	1,628		32·49	529·1
−⅛ " "	914		30·1	275·1
Totals	2,512		31·6	804·2
Summary of −¼+⅛ middling to be rejigged—				
−¼+⅛ from recrushing −¾+½ middling	1,354		30·4	411·6
−¼+⅛ " " −¾+½ "	1,628		32·49	529·1
Totals	2,982		31·5	940·7
−¼+⅛ concentrate	528	16·1	47·84	252·4
middling	1,857	56·7	30·00	557·2
tailing	892	27·2	14·74	131·1
Totals	3,277	100·0		940·7
Summary of −⅛-inch material—				
−⅛ original	5,019		29·58	1,485·6
−⅛ from recrushing −¾+½ middling (A)	810		28·44	230·0
−⅛ from middling rejigging −½+¼ from middling (A)			30·1	275·1
−¾+½ recrushed middling plus −½+¼ recrushed middling (B)	914			
−⅛ from recrushing middling from jigging −¼+⅛ middling	1,857		30·00	557·2
−⅛ from recrushing original −¼+⅛ middling	1,759		27·70	487·2
Totals	10,359			3,035·1

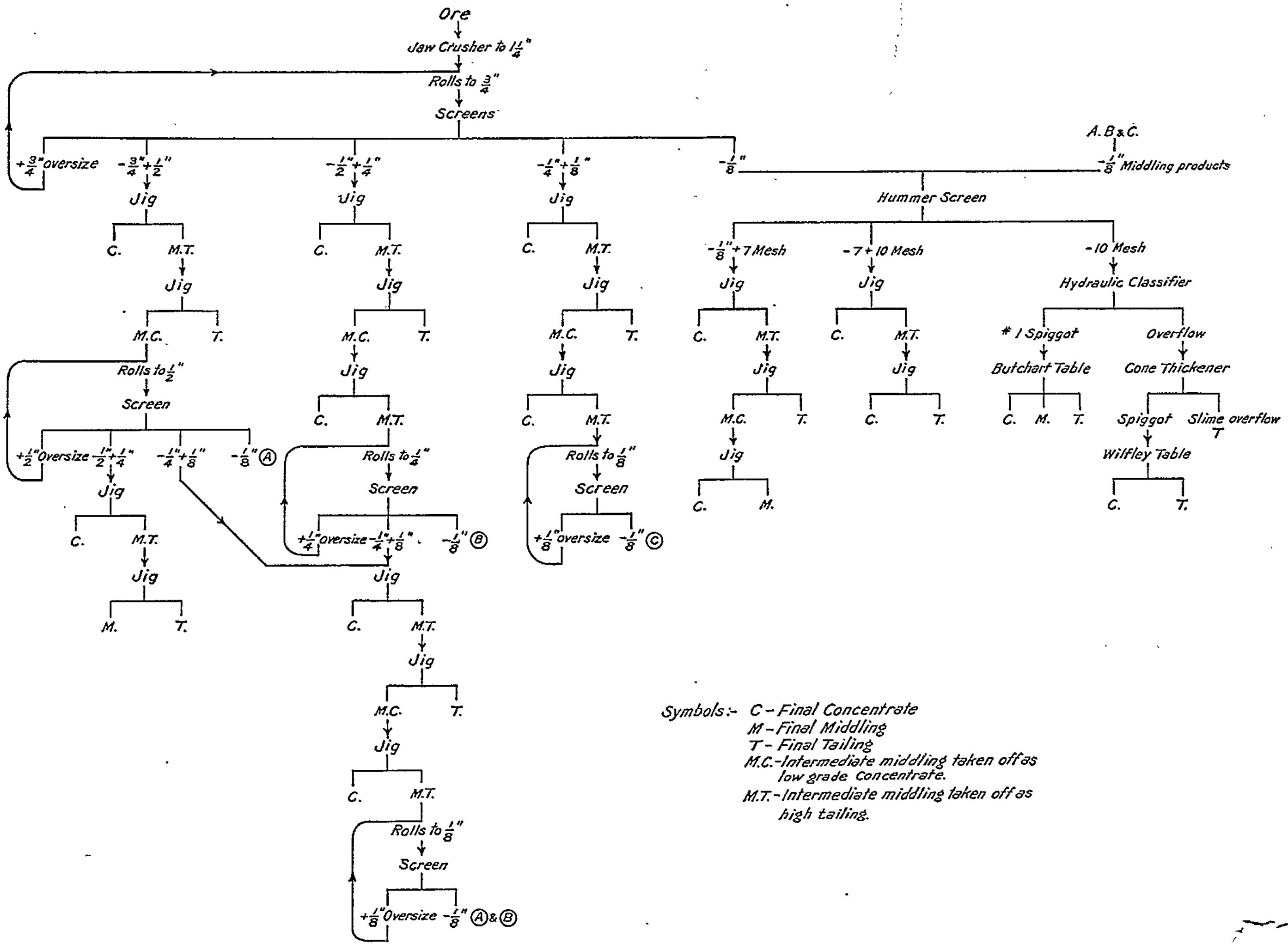

FIGURE 1. Flow-sheet of gravity concentration, Test No. 1, on Kaministikwia ore.

Products	Weight, lb.	Assay, Fe, %	Content, Fe, lb.
Hummer screen −⅛+7 mesh	5,379·0	29·62	1,593·3
" " −7+10 mesh	1,488·0	28·44	423·2
" " −10 mesh	3,492·0		1,018·6
Totals	10,359·0		3,035·1
−⅛+7 mesh jig concentrate	1,907·0	48·5	925·5
−⅛+7 mesh middling	409·0	27·1	101·8
−⅛+7 mesh tailing	2,851·0	16·7	476·1
Totals	5,167·0		1,503·4
−7+10 mesh concentrates	412·0	49·37	203·5
tailing	1,076·0	19·54	209·8
Totals	1,488·0		413·3
−10 mesh Butchart table concentrate	548·0	49·37	270·7
" " middling	664·0	29·93	198·5
Wilfley table concentrate	200·0	54·05	108·2
Total table tailing	2,080·0	16·6	345·3
Totals	3,492·0		922·7

Summary of Test No. 1

—	Weight		Assay, Fe, %	Content, lb. Fe	Recovery, %
	lb.	%			
Concentrates—					
−¾+½	1,815·0	21·2	50·0	907·5	
−½+¼	1,591·0	18·6	54·8	871·9	
−¼+⅛	1,455·0	17·0	52·9	769·7	
−⅛+7 mesh	1,907·0	22·3	48·5	925·5	
−7+10 mesh	412·0	4·8	49·4	203·5	
Butchart table	548·0	6·4	49·4	270·7	
Wilfley table	200·0	2·3	54·1	108·2	
Concentrate from rejigging −¼+½ middling	528·0	6·2	47·8	252·4	
Concentrate from recrushing −¾+½ middling	91·0	1·1	51·1	46·5	
Total concentrates	8,547·0	99·9	51·0	4,355·9	66·6
Middlings—					
Middling from −⅛+7 mesh	409·0		27·1	101·8	
Middling from Butchart table	664·0		29·9	198·5	
Totals	1,073·0		28·0	300·3	4·6
Tailings—					
−¾+½	1,193·0		16·3	194·5	
−½+¼	2,246·0		14·14	316·7	
−¼+⅛ No. 1 tailing	1,210·0		12·04	145·2	
No. 2 "	407·0		12·06	49·2	
Tailing from recrushing −¾+½ middling	101·0		13·18	13·3	
" " −½+¼ "	892·0		14·74	131·1	
Tailing from tables	2,080·0		16·60	345·3	
−⅛+7 mesh	2,851·0		16·7	476·1	
−7+10 mesh	1,076·0		19·5	209·8	
Totals	12,056·0		15·6	1,881·2	28·8
Final totals—ore feed	21,676·0		30·2	6,537·4	100·0

Recapitulation of Test No. 1

—	Weight		Assay, Fe %	Content, lb. Fe	Recovery, %
	lb.	%			
Concentrates	8,547·0	39·4	51·0	4,355·9	66·6
Middlings	1,073·0	5·0	28·0	303·3	4·6
Tailings	12,056·0	55·6	15·6	1,881·2	28·8
Totals	21,676·0	100·0	30·2	6,537·4	100·0

Disregarding middlings, ratio of concentration, 2·53:1.

Complete Analysis of Concentrates

	Per cent
Fe (total)	50·7
FeO	10·21
Fe_2O_3	61·14
Al_2O_3	0·38
SiO_2	26·30
CaO	0·40
MgO	0·18
Mn	0·01
P	0·08
*S	0·10

*(Probably high from salting.)

Discussion of Results. The banded structure of the ore causes it when crushed to break along the parting between the bands, and slab-like pieces of ore are produced which have a large flat surface area with but little volume. In recrushing the middling, by rolls, this effect is more pronounced than with the original ore.

These slab-like pieces of ore are much more difficult to concentrate than particles which approach the shape of cubes. In the screening operations these flat thin flakes report with pieces which have twice their mass. This is due to the latter having greater thickness. When the pieces of mixed shapes are jigged the flat thin scales of iron ore tend to pass into the tailing with the heavier pieces of jasper-quartz. These flat pieces also give trouble on the tables and tend to slide across the surface of the table into the tailing. The net result is that not only a lower recovery is obtained but that the capacities of the jigs and tables are lowered considerably.

Because a much greater proportion of flat pieces was produced during the recrushing of the middling considerable attention was given to the possible use of the ball or rod mill for recrushing in place of rolls.

This first gravity test is more of the nature of a preliminary test and, therefore, does not represent the best possible results.

Test No. 2

In this test (*see* Figure 2) the ore was crushed in the same manner as in Test No. 1. The screening, on the other hand, was slightly different, one size less being made on the Hummer screen. Instead of making two sizes + ($\frac{1}{8}$ x $\frac{1}{2}$) + 7 mesh, only one size was made — ($\frac{1}{8}$ x $\frac{1}{2}$) + 10 mesh.

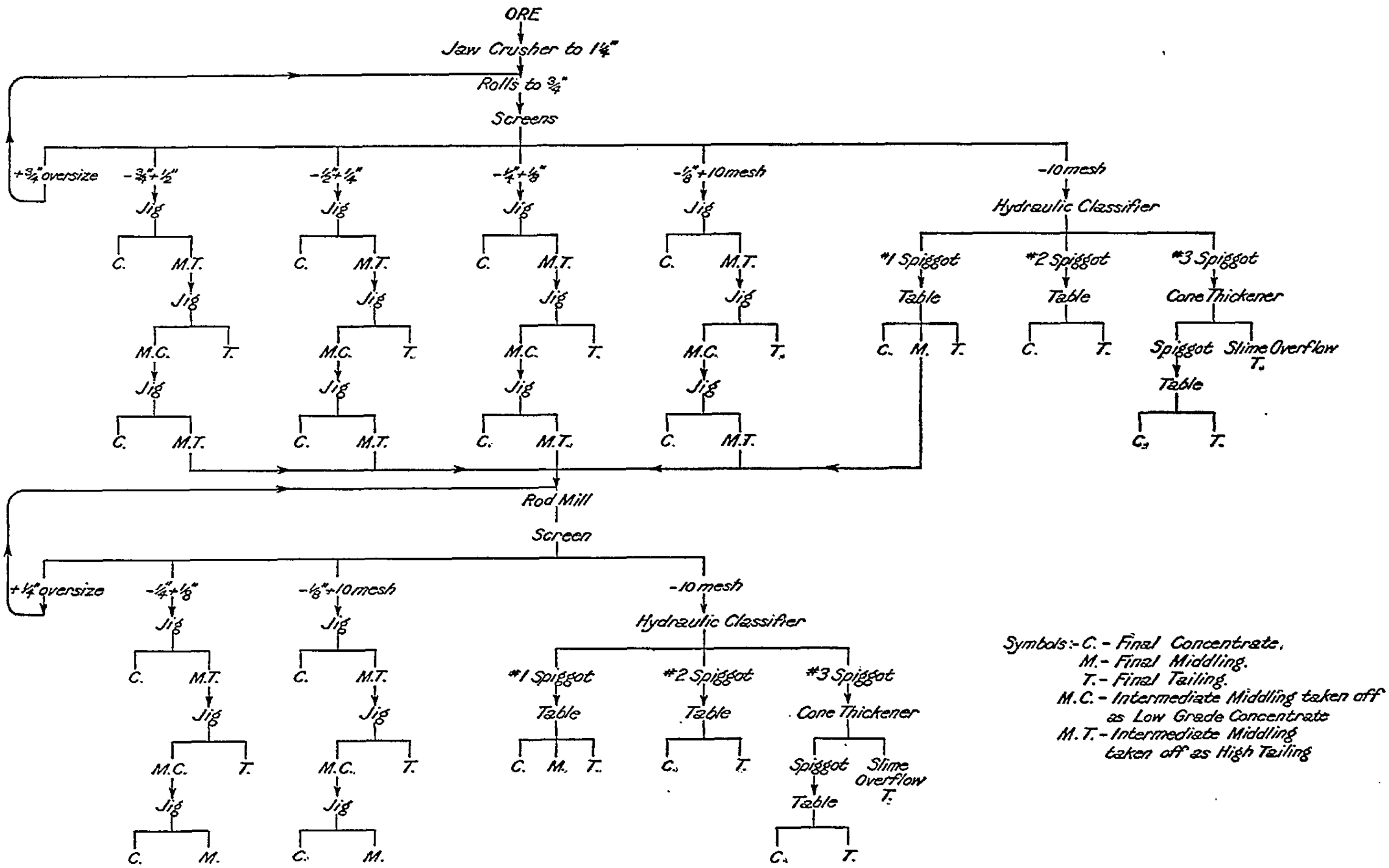

FIGURE 2. Flow-sheet of gravity concentration, Test No. 2, on Kaministikwia ore.

All the sizes coarser than −10 mesh were jigged as in Test No. 1. The −10 mesh from the wet operated Hummer screen was classified into three products instead of two as in Test No. 1. No. 1 spigot from the Richards classifier went to a Butchart table, No. 2 to the large Wilfley table, and No. 3, the classifier overflow, was thickened and tabled on a small Wilfley table.

The middling from all the operations was collected and recrushed in a 6- x 3-foot Marcy rod mill. The discharge of the mill was then sized over the same system of screens as the original ore, and the products rejigged and tabled.

The following tables give the results of the test.

Summary of Results

Products	Weight		Assay, Fe %	Content, lb. Fe	Content, %
	lb.	%			
Heads—Screen products—					
−¾+½	4,681·0	31·7	30·9	1,540·8	23·2
−½+¼	5,354·0	24·9	31·3	1,678·1	25·2
−¼+⅛	5,208·0	24·2	30·2	1,572·9	23·6
−⅛+10 mesh	4,597·0	21·3	29·5	1,354·6	20·4
−10 mesh	1,701·0	7·9	29·7	505·6	7·6
Totals	21,541·0	100·0	30·9	6,652·0	100·0
Jigging −¾+½—					
Concentrates	2,424·8	51·8	47·2	1,144·5	74·3
Middling	1,427·7	30·3	18·7	267·0	17·2
Tailing	828·5	17·7	15·6	129·3	8·5
Totals	4,681·0	100·0	31·3	1,540·8	100·0
Jigging −½+¼—					
Concentrates	1,705·0	31·8	52·3	891·4	56·4
Middling	1,423·0	26·6	32·1	457·2	23·2
Tailing	2,226·0	41·6	14·8	329·5	20·3
Totals	5,354·0	100·0	31·3	1,678·1	99·9
Jigging −¼+⅛—					
Concentrates	1,606·0	30·8	53·0	850·9	54·1
Middling	1,253·0	24·1	31·4	393·1	25·0
Tailing	2,348·0	45·1	14·1	328·9	20·9
Totals	5,208·0	100·0	30·2	1,572·9	100·0

Summary of Results—Continued

Products	Weight		Assay, Fe %	Content, lb. Fe
	lb.	%		
Jigging −⅛+10 *mesh*—				
Concentrate	1,352·0	29·4	53·9	728·7
Middling	900·0	19·6	31·5	283·5
Tailing	2,345·0	51·0	14·6	342·4
Totals	4,597·0	100·0	29·5	1,354·6
Tabling −10 *mesh*—				
Butchart table concentrate	305·0	17·9	50·9	155·3
Butchart table middling	230·0	13·5	26·7	61·4
Large Wilfley table concentrate	272·0	16·0	46·3	125·9
Small Wilfley table concentrate	51·0	3·0	60·0	30·6
Total tailing	843·0	49·6	15·7	132·4
Totals	1,701·0	100·0	29·7	505·6
Summary of middling for recrushing—				
−¾+½	1,427·7	27·3	18·7	267·0
−½+¼	1,423·0	27·2	32·1	457·2
−¼+⅛	1,253·0	23·9	31·4	393·1
−⅛+10 mesh	900·0	17·2	31·5	283·5
−10 mesh	230·0	4·4	26·7	61·4
Totals	5,333·7	100·0	27·9	1,462·2
Results of crushing middling in rod mill—				
All −¼+⅛	1,371·2	26·2	30·0	411·4
−⅛+10 mesh	1,298·0	24·8	28·7	372·5
−10 mesh	2,564·5	49·0	25·2	646·0
Totals	5,233·7	100·0	27·3	1,429·0
Loss				33·2
				1,462·2
Results of concentration of recrushed middling—				
Jigging −¼+⅛—				
Concentrates	548·5	40·0	46·4	254·5
Middling	292·1	21·3	27·7	80·9
Tailing	530·6	38·7	14·4	76·4
Totals	1,371·2	100·0	30·0	411·8
Jigging −⅛+10 *mesh*—				
Concentrates	394·6	30·4	48·3	190·6
Middling	358·2	27·6	29·5	105·7
Tailing	545·2	42·0	14·0	76·3
Totals	1,298·0	100·0	28·7	372·6
Tabling −10 *mesh*—				
Butchart table concentrate	377·0	14·7	49·0	184·7
Butchart table concentrate	79·5	3·1	23·1	18·4
Large Wilfley table concentrate	159·0	6·2	49·7	79·0
Small Wilfley table concentrate	138·5	5·4	55·8	77·3
Total tailing	1,810·5	70·6	15·8	286·1
Totals	2,564·5	100·0	25·2	645·5

Condensed Summary of Results

Products	Weight, lb.	Assay, Fe %	Content, lb. Fe	Recovery, %
Concentrates—				
−¾+½	2,424·8	47·2	1,144·5	
−½+¼	1,705·0	52·3	891·4	
−¼+⅛	1,606·0	53·0	850·9	
−⅛+10 mesh	1,352·0	53·9	728·7	
Butchart table	305·0	50·9	155·3	
Large Wilfley table	272·0	46·3	125·9	
Small Wilfley table	51·0	60·0	30·6	
−¼+⅛	548·5	46·4	254·5	
−⅛+10 mesh	394·6	48·3	190·6	
Butchart table	377·0	49·0	184·7	
Large Wilfley table	159·0	49·7	79·0	
Small Wilfley table	138·5	55·8	77·3	
	9,333·4	50·5	4,713·4	71·2
Middling remaining—				
−¼+⅛	292·1	27·7	80·9	
−⅛+10 mesh	358·2	29·5	105·7	
Table	79·5	23·1	18·4	
Totals	729·8	28·1	205·0	3·1
Tailings (all)—				
−¾+½	828·5	15·6	129·3	
−½+¼	2,226·0	14·8	329·5	
−¼+⅛	2,349·0	14·1	328·9	
−⅛+10 mesh	2,345·0	14·6	342·4	
Tables	843·0	15·7	132·4	
−¼+⅛	530·6	14·4	76·4	
−⅛+10 mesh	545·2	14·0	76·3	
Table	1,810·5	15·8	286·1	
Totals	11,477·8	14·8	1,701·3	25·7
Final totals	21,541·0	30·7	6,619·7	100·0

Recapitulation of Test No. 2

—	Weight		Assay, Fe %	Content, lb. Fe	Recovery, %
	lb.	%			
Concentrate	9,333 4	43·3	50·5	4,713·4	71·2
Middling	729·8	3·4	28·1	205·0	3·1
Tailing	11,477·8	53·3	14·8	1,701·3	25·7
Totals	21,541·0	100·0	30·7	6,619·7	100·0

Disregarding middling, ratio of concentration, 2·31:1

Complete Analysis of Concentrates

		Per cent
Fe (total)	50·6	
FeO		10·20
Fe_2O_3		61·01
SiO_2		25·73
Al_2O_3		0·32
CaO		0·46
MgO		0·27
Mn		0·013
P		0·07
*S		0·09

*(Probably high from salting.)

Discussion of Results. The results of this test were much better than from Test No. 1. Better adjustments were obtained on the jigs and tables which resulted in less middling and a slightly lower tailing being produced. The grade of the total concentrate was slightly lower chiefly due to the fact that a little too much concentrate was pulled when treating the coarser sized −¾ + ½-inch material.

The recrushing of the middling by the rod mill instead of by rolls as in Test No. 1, produced a much better product for retreatment. The particles of ore were more nearly the shape of cubes and very little material of slabby shape was produced. Owing to this fact it was unnecessary to make the intermediate size on the Hummer screen.

The additional classified product improved the table operations and it is believed that even another step in classification would further improve the operation.

Data on the Gravity Concentration. Jigs: The jig used was the movable sieve type known as the James jig. The machine had a large capacity per square foot of sieve area. The following table gives the capacity at which the jigs were operated on the various sizes. The jigs were run in tandem, that is, the second jig re-treated the tailing produced in the first jig.

Size	Rate of feed per hour, lb.	Stroke, inch	Head of water, feet	Capacity tons per sq. ft. of screen per hour
−¾+½	1,500	¾	5½	0·347
−½+¼	1,200	½	3	0·278
−¼+⅛	1,000	⅜	3	0·231
−⅛+10 mesh	800	¼	3	0·185

Rod Mill Capacity. One ton of mixed middlings per hour was crushed per ton of rods in mill in open circuit, all through ¼-inch screen. In practice this mill would be in closed circuit with a ½-inch screen and the capacity would be greatly increased.

Test No. 3

In this test the same method of crushing was used as in Test No. 1. The product was sized on $(\frac{3}{4} \text{ x } \frac{1}{2})$; $-\frac{1}{2} + (\frac{1}{4} \text{ x } \frac{1}{2})$; $-(\frac{1}{4} \text{ x } \frac{1}{2}) + (\frac{1}{8} \text{ x } \frac{1}{2})$; and $-(\frac{1}{8} \text{ x } \frac{1}{2})$. Each of these sizes was run over a Gröndal magnetic cobbing belt with magnetic drum pulley. A number of runs were made in each size for adjustments, such as speed of travel of the belt, amperage on the magnets, etc. The best results are given in the following table.

Results from Magnetic Cobber Test No. 1

Product	Weight		Assay, Fe %	Content, Fe lb.	Content, %
	lb.	%			
Screen products—					
$-\frac{3}{4}+\frac{1}{2}$	4,854	21·5	33·44	1,623·0	23·4
$-\frac{1}{2}+\frac{1}{4}$	5,603	24·8	30·85	1,728·5	25·0
$-\frac{1}{4}+\frac{1}{8}$	5,546	24·6	30·02	1,665·0	24·0
$-\frac{1}{8}$	6,556	29·1	29·1	1,908·0	27·6
Totals	22,559	100·0	30·7	6,924·0	100·0
$-\frac{3}{4}+\frac{1}{2}$ Concentrates No. 1	329	7·0	52·72	173·0	10·8
“ No. 2	759	16·2	48·76	370·0	23·0
Middling No. 1	1,830	39·1	35·12	642·0	39·9
“ No. 2	501	10·7	43·67	219·0	13·6
Tailing	1,262	27·0	16·20	204·0	12·7
Totals	4,681			1,608·0	100·0
$-\frac{1}{2}+\frac{1}{4}$ Concentrates No. 1	453	8·3	51·0	231·0	13·5
“ No. 2	1,012	18·6	46·7	467·0	27·2
Middling No. 1	1,251	23·0	39·2	490·0	28·5
“ No. 2	252	4·6	34·41	87·0	5·1
Tailing	2,466	45·4	17·9	441·0	25·7
Totals	5,434	99·9		1,716·0	100·0
$-\frac{1}{4}+\frac{1}{8}$ Concentrates No. 1	694	12·9	50·6	351·0	21·2
“ No. 2	64	1·2	48·5	31·0	1·9
Middling No. 1	1,413	26·2	43·06	609·0	36·9
“ No. 2	290	5·4	34·5	100·0	6·0
“ No. 3	322	6·0	34·3	110·0	6·7
Tailing	2,612	48·4	17·3	452·0	27·3
Totals	5,395	100·1		1,653·0	100·0
$-\frac{1}{8}$ Concentrates	1,206	18·4	51·0	615·0	31·9
Middling	2,059	31·4	37·1	764·0	39·7
Tailing	3,291	50·2	16·6	546·0	28·4
Totals	6,556	100·0		1,925·0	100·0

Discussion of Results. These magnetic separation results are not very satisfactory. In comparing them with the results of the jig and table tests the first point observed is that the amount of concentrate assaying over 50 per cent iron obtained by magnetic separation is very much less than by the gravity methods.

The second point is that the amount of tailing produced is less than by gravity methods and it contains more iron.

Considered from the point of view of the production of a marketable grade of iron ore, magnetic cobbing is neither feasible nor practical. It is possible that magnetic cobbing could be used as a preliminary step to eliminate a portion of the waste. The low-grade iron product thus obtained could then be either concentrated on jigs or finely ground and concentrated by wet magnetic separators, the tailing from the magnetic separators being passed to tables for the purpose of saving a portion of the hematite. This would involve sintering the fine concentrate, which is an expensive operation and would cost between 60 cents and a dollar per ton of concentrate. This method would be very similar to the practice attempted at Babbitt, Minnesota, but it would be complicated by the presence of hematite which is non-magnetic and which is not present in the Babbitt ore in appreciable quantities.

FINAL SUMMARY AND CONCLUSIONS

Magnetic cobbing cannot be used to produce marketable grades of iron ore.

Gravity concentration on jigs and tables produced concentrates containing over 50 per cent iron with a concentration ratio of 2·31 : 1, but very high in silica. The analysis follows:—

		Per cent
Fe (total)	50·6	
FeO		10·20
Fe_2O_3		61·01
SiO_2		25·73
Al_2O_3		0·32
CaO		0·46
MgO		0·27
Mn		0·013
P		0·07
*S		0·09

*(Probably high from salting.)

The concentrate must be classed as a high silica ore exceptionally free from impurities such as phosphorus and sulphur. To partly offset the objection to the high silica content the concentrate has certain physical properties such as the hardness and graded size of the ore particles which tend to make it an ideal burden for the blast furnace.

When examining the commercial aspects of mining, milling, and marketing low-grade ore of this character it must be kept in mind that in order to determine the profit, if any, the mining and milling costs must be multiplied by the number of tons of ore required to produce one ton of concentrate, and this sum subtracted from the price received per ton of concentrate plus the freight and handling to the user.

There is another feature to be considered in this regard, namely the character of the deposit, whether there is sufficient ore of the grade and character as represented by the sample tested to allow mining to be carried out on a large-scale basis. It is obvious that any change in the grade or character of the ore will affect the concentration and also that comparatively large-scale operations will be necessary in order to bring the operating costs within the limits of profitable operation. This particularly applies to the costs of mining.

The ore contains practically only a trace of sulphur, the sulphur in the samples being from a previous test on a high sulphide ore containing pyrite and other sulphides.

Report No. 323

CONCENTRATION OF LEAD-ZINC ORE FROM GENEVA, ONTARIO

C S. Parsons

Shipment. A shipment of 60 bags of lead-zinc ore approximately 3,000 pounds was received in March, 1929, from the Geneva Lake property of the Towagmac Exploration Company, Geneva, Ontario.

Characteristics of Ore. The ore consists of a siliceous gangue, containing galena, zinc blende, and pyrite together with low values in both gold and silver. The minerals are disseminated throughout the gangue, but very fine grinding is not necessary in order to free them.

Analysis of Sample

Lead... :	3·44	per cent
Zinc	9·41	"
Gold	0·01	oz./ton
Silver	1·25	"

EXPERIMENTAL TESTS

Three continuous small-scale selective flotation tests were made. The ore was treated at the rate of 100 pounds per hour. The flow-sheet used was as follows. The ore was crushed to $\frac{1}{8}$ inch and sampled, then fed to a small rod mill in closed circuit with a classifier. The classifier overflow went direct to a small 6-cell flotation machine. The lead was floated in this machine and the rougher concentrate was recleaned in one cell. The tailing from the lead flotation cell was pumped to an 8-cell flotation machine where the zinc was floated. The zinc rougher concentrate was recleaned in a 3-cell machine, the cleaner tailing being returned to the head of the zinc rougher cell.

The following tables give the reagents used and the results of the tests.

SUMMARY AND CONCLUSIONS

Owing to the excellent results obtained it is hardly necessary to discuss them in detail.

The grade of the concentrates can be varied between quite wide limits without affecting the recoveries of the lead or zinc. The silver and gold recoveries fall off slightly as the grade of the lead concentrate is raised. The most profitable grade to make will have to be determined from a study of the smelter charges and freight rates.

Test No. 1

Reagents Used (lb./ton)

Time	Lead reagents					Zinc reagents			Density	Remarks
	Soda ash	Cyanide	Potassium xanthate	Cresylic acid	Zinc sulphate	Copper sulphate	Potassium xanthate	Pine oil	Lead cell discharge	
10.00 a.m.......	0·99	0·11	0·01	0·4	1·5	1·0	0·12	0·15	1	Ore feed rate—110 lb./hour.
10.30 "	0·99	0·11	0·01	0·4	1·5	1·0	0·12	0·15		
11.00 "	0·99	0·12	0·01	0·4	1·6	1·0	0·12	0·15	1:2·3	
11.30 "	1·12	0·099	0·01	0·4	1·5	1·0	0·11	0·15	1:1·6	
12.00 p.m.......	1·12	0·099	0·01	0·4	1·6	1·0	0·097	0·15	1:1·6	
12.30 "	1·09	0·10	0·01	0·4	1·5	1·0	0·1	0·15	1:1·8	
1.00 "	1·09	0·106	0·01	0·4	1·4	1·0	0·1	0·15	1:1·2	Started taking samples every 5 minutes.
1.30 "	1·09	0·106	0·01	0·4	1·5	1·0	0·1	0·15	1:1·6	
2.00 "	1·2	0·13	0·01	0·4	1·7	1·0	0·12	0·15		
2.30 "	1·2	0·13	0·01	0·4	1·8	1·0	0·097	0·15		
3.00 "	1·12	0·12	0·01	0·4	1·7	1·0	0·097	0·15	1:1·6	
3.30 "	1·19	0·12	0·01	0·4	1·6	1·0	0·11	0·15		
3.40 "	1·19	0·12	0·01	0·4	1·6	1·0	0·045	0·15		Reduced amount of xanthate for zinc flotation. Results O.K.
4.00 "	1·2	0·12	0·01	0·4	1·7	1·0	0·045	0·15		
4.30 "	1·2	0·12	0·01	0·4	1·7	1·0	0·045	0·15	1:1·85	

Soda ash was fed to ball mill.
Cyanide was fed to ball mill.
Zinc sulphate to classifier overflow.
Xanthate to head of lead cells.
Cresylic acid to head of lead cells.
Copper sulphate to pump between lead and zinc cells.
Potassium xanthate to head of zinc cells.
Pine oil to pump between lead and zinc cells.
Densities were taken of the tailing discharges from lead cells.

Results—Test No. 1

Heads: Pb, 3·44 per cent; Zn, 9·41 per cent; Au, 0·01 oz./ton; Ag, 1·75 oz./ton.

Test	Product	Weight, %	Assay				Recovery, %			
			Pb %	Zn %	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
No. 1 Samples taken 1 p.m. to 4.30 p.m.	Pb concentrate......	6·81	48·80	12·06	0·12	15·75	96·6	8·7	82·1	92·1
	Pb tailing...........		0·10	3·00	tr	0·27				
	Zn concentrate......	15·45	0·25	53·61	0·01	0·90	1·1	88·1	15·0	1·2
	Zn tailing...........	77·74	0·10	0·39	trace	0·10	2·3	3·2	2·9	6·7
No. 1 Samples taken of entire run.	Pb concentrate......		48·00	12·60						
	Zn concentrate......		0·15	54·00						
	Tailing.............		0·12	0·39						

Screen Test on Tailing

Mesh	Weight %	Cumulative %
+ 65	2·5	2·5
− 65+100	8·4	10·9
−100+150	9·2	20·1
−150+200	13·9	34·0
−200	66·0	

Test No. 2

Reagents Used (lb./ton)

Time	Lead reagents					Zinc reagents			Density	Remarks
	Soda ash	Cyanide	Zinc sulphate	Sodium Aerofloat	Cresylic acid	Copper sulphate	Potassium xanthate	Pine oil	Lead cell discharge	
9.30 a.m.	0·9	0·09		0·014	0·4	0·91	0·04	0·08		Ore feed rate—110 lb./hour.
10.00 "	0·9	0·19		0·014	0·4	0·96	0·04	0·08		
10.30 "	0·88	0·19		0·014	0·4	0·96	0·04	0·08	1:2·3	
11.00 "	0·85	0·19		0·011	0·4	1·04	0·04	0·08	1:2·4	
11.30 "	0·85	0·19		0·011	0·5	0·88	0·039	0·08	1:1·95	
12.00 p.m.	0·85	0·2		0·014	0·5	0·9	0·039	0·08	1:1·78	Changed lead samples 12.30 p.m.
12.30 "	0·85	0·19		0·008	0·5	0·89	0·039	0·08	1:1·9	
1.00 "	0·85	0·18		0·008	0·5	0·88	0·039	0·08	1:1·6	
1.30 "	0·83	0·2		0·008	0·5	0·85	0·044	0·08	1:1·7	
2.00 "	0·85	0·19		0·008	0·5	0·88	0·044	0·08	1:2·3	
2.30 "	0·85	0·19	0·91	0·008	0·5	0·08	0·04	0·08	1:2·4	
3.00 "	0·9	0·2	0·91	0·008	0·5	0·88	0·04	0·08	1:2·3	Changed lead samples at 3 p.m.
3.30 "	0·9	0·12	0·9	0·008	0·5	0·89	0·04	0·08	1:2·1	
4.00 "	0·9	0·2	0·93	0·008	0·5	0·89	0·04	0·08	1:2·0	Changed zinc samples at 3.30 p.m
4.30 "	0·9	0·2	0·93	0·008	0·5				1:2·0	

Soda ash was fed to ball mill.
Cyanide was fed to ball mill.
Zinc sulphide was fed to classifier overflow.
Sodium Aerofloat was fed to lead cells.
Cresylic acid was fed to lead cells.
Copper sulphate was fed to pump to zinc cells.
Potassium xanthate was fed to zinc cells.

Densities taken on lead cells discharge.

Same grinding as in Test No. 1.

Test No. 3

Reagents Used (lb./ton)

Time	Lead reagents					Zinc reagents			Density	Remarks
	Soda ash	Cyanide	Zinc sulphate	Sodium Aerofloat	Cresylic acid	Copper sulphate	Potassium xanthate	Pine oil	Lead cell discharge	
10.00 a.m........	0·96	0·48	2·7	0·008	0·53					Ore feed rate—110 lb./hour.
10.30 "	0·96	0·48	2·6	0·008	0·53					
11.00 "	0·94	0·44	2·6	0·008	0·53	0·88	0·044	0·08	1:3·3	
11.30 "	0·94	0·44	2·8	0·008	0·53	0·88	0·04	0·08	1:2·3	
12.00 p.m........	0·88	0·44	2·6	0·114	0·53	0·92	0·047	0·08	1:2·4	
12.30 "	0·88	0·44	2·8	0·114	0·53	0·91	0·047	0·08	1:2·4	
1.00 "	0·96	0·46	2·8	0·114	0·53	0·92	0·047	0·08	1:2·3	
1.30 "	1·0	0·41	3·0	0·114	0·53	0·85	0·047	0·08	1:2·5	
2.00 "	1·0	0·42	2·8	0·114	0·53	0·88	0·047	0·08	1:2·5	
2.30 "	0·96	0·42	2·7	0·114	0·53	0·93	0·047	0·08	1:2·5	NOTE.—Xanthate feed line plugged.
3.00 "	0·96	0·44	2·8	0·114	0·66	0·94	0·09	0·08	1:1·78	
3.30 "	0·94	0·44	2·7	0·114	0·66	1·9	0·09	0·08	1:1·6	
4.00 "	0·82	0·41	2·6	0·114	0·66	1·9	0·09	0·08	1:1·33	
4.30 "	0·82	0·41	2·6	0·114	0·66	1·9	0·09	0·08	1:1·4	

Soda ash was fed to ball mill.
Cyanide was fed to ball mill.
Zinc sulphate was fed to ball mill.
Sodium Aerofloat was fed to lead cells.
Cresylic acid was fed to lead cells.

Copper sulphate was fed to pump to zinc cells.
Potassium xanthate was fed to zinc cells.
Pine oil was fed to pump to zinc cells.
Densities were on lead cell tailings.
Same grinding as in Test No. 2.

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Results: Tests Nos. 2 and 3

Heads: Pb, 3·44 per cent; Zn, 9·41 per cent; Au, 0·01 oz./ton; Ag, 1·75 oz./ton.

Test	Product	Weight, %	Assay				Recovery, %			
			Pb, %	Zn, %	Au, oz./ton	Ag, oz./ton	Pb	Zn	Au	Ag
No. 2 Sample No. 1 10.00 p.m. to 12.30 p.m.	Pb concentrate......	7·00	47·51	12·06	0·10	14·95	96·6	9·0	70·0	80·5
	Pb tailing...........		0·10	8·37	trace	0·22				
	Zn concentrate......	15·30	0·15	54·90	0·01	1·10	0·7	89·1	15·0	
	Zn tailing...........	77·70	0·12	0·23	trace	0·11	2·7	1·3	15·0	
No. 2 Sample No. 2 12.30 p.m. to 3.30 p.m.	Pb concentrate......	6·25	53·18	11·07	0·12	16·05	36·6	8·9	75·0	81·6
	Pb tailing...........		0·10	9·04	trace	0·27				
	Zn concentrate......	12·50	0·15	54·90	0·01	1·10	0·6	88·7		
	Zn tailing...........	81·25	0·12	0·23	trace	0·11	2·8	2·4		
No. 2 Sample No. 3 3.30 p.m. to 4.30 p.m.	Pb concentrate......	6·19	54·06	10·55	0·10	16·20	97·3	6·9	62·0	80·6
	Pb tailing...........		0·09	9·50	trace	0·31				
	Zn concentrate......	15·48	0·15	55·25	0·01	1·00	0·7	91·8		
	Zn tailing...........	78·33	0·09	0·16	trace	0·11	2·0	1·3		
No. 3 10.00 a.m. to 4.30 p.m.	Pb concentrate......	5·76	57·77	10·40	0·12	16·90	96·9	6·4	63·0	77·9
	Pb tailing...........		0·17	9·49	trace	0·21				
	Zn concentrate......	15·34	0·20	55·32	0·01	1·08	0·8	90·1		13·3
	Zn tailing...........	78·30	0·10	0·42	trace	0·14	2·3	3·5		8·8

Report No. 324

EXPERIMENTAL TESTS ON GOLD ORE FROM THE McMILLAN GOLD MINES, LTD., MONGOWIN TOWNSHIP, SUDBURY DISTRICT, ONTARIO

J. S. Godard

Shipment. A shipment of 50 pounds of gold ore consisting of a composite sample from 107 channel samples was received May 4, 1929, from the McMillan Gold Mines, Ltd., Sudbury, Ont., submitted by John H. Banks & Company, Consulting Engineers, New York.

Characteristics of the Ore. The ore contains sulphides of iron, both pyrrhotite and pyrite, arsenical pyrites, and a small amount of chalcopyrite. The gangue is siliceous.

Analysis. Analysis showed it to contain.

Gold....................	0·50 oz./ton	Arsenic.................	0·87 per cent
Silver..................	0·08 "	Copper..................	0·04 "
Iron....................	10·35 per cent	Insoluble...............	75·17 "
Sulphur.................	4·73 "		

Purpose of Experimental Tests. The purpose of the experimental tests was to determine whether straight cyanidation or concentration and cyanidation of the concentrates would be the more economical method of treating this ore.

Experimental Tests

Cyanidation Tests

Test No.	Mesh	Heads Au, oz./ton	Average tailing, Au, oz./ton	Extraction, %	Reagents, lb./ton	
					KCN	CaO
1................................	− 48	0·52	0·029	94·4	1·57	4·08
2................................	− 48		0·026	95·0	1·49	3·84
3................................	− 48		0·033	93·6	1·76	4·80
4................................	− 65	0·46	0·023	95·1	1·58	4·16
5................................	− 65		0·025	94·6	1·51	4·14
6................................	−100	0·44	0·013	97·1	1·81	4·30
7................................	−100		0·014	96·8	1·62	4·24
8................................	−150	0·48	0·020	95·9	1·59	4·05
9................................	−150		0·015	96·9	1·59	4·21

Screen Test on Tailings

Test No.	Mesh	Weight, %	Assay	Per cent of values	Average tailing, Au, oz./ton
1	+100	33·7	0·04	46·1	0·029
	+200	25·6	0·03	26·3	
	−200	40·7	0·02	27·6	
2	+100	31·3	0·05	61·6	0·026
	+200	29·4	0·02	23·1	
	−200	39·3	0·01	15·3	
3	+100	35·4	0·05	53·6	
	+200	24·0	0·03	21·8	0·033
	−200	40·6	0·02	24·6	
4	+150	33·3	0·03	43·0	0·023
	−150	66·7	0·02	57·0	
5	+150	30·1	0·035	42·8	0·025
	−150	69·9	0·02	57·2	
6	+200	28·2	0·02	43·7	0·013
	−200	71·8	0·01	56·3	
7	+200	29·2	0·025	50·7	0·014
	−200	70·8	0·01	49·3	

Tailings from Test Nos. 8 and 9 were not screened.
All samples dry-crushed to required mesh in Braun pulverizer.
Agitation in Winchester bottles.
Time of agitation, 48 hours.
Dilution, 1:2.
KCN, 0·075 per cent.
Cyanide used, Cassel's 96·3 per cent NaCN.

Concentration Tests—Flotation and Tabling

Test No.	Products	Weight, %	Assay		Per cent of values	
			Au, oz./ton	Ag, oz./ton	Au	Ag
10	Flotation concentrate	10·9	4·20	0·50	85·9	73·3
	Table concentrate	8·1	0·74	0·11	11·2	12·0
	Table tailing	55·7	0·023	0·015	2·4	11·3
	Slimes	25·3	0·01	0·01	0·5	3·4
	Average tailing including slimes		0·019			
	Screen test on table tailing					
	+100 mesh	13·0	0·04	0·02	23·0	17·6
	+200 "	35·6	0·02	0·02	31·4	48·0
	−200 "	51·4	0·02	0·01	45·6	34·4
11	Flotation concentrate	13·6	2·79	0·33	90·0	85·7
	Table concentrate	7·1	0·42	0·04	7·1	5·3
	Table tailing	46·5	0·012	0·01	1·3	9·0
	Slimes	32·8	0·02	trace	1·6	
	Average tailing including slimes		0·015			
	Screen test on table tailing					
	+200 mesh	20·3	0·02	0·01	33·9	20·3
	−200 "	79·7	0·01	0·01	66·1	79·7
12	Flotation concentrate	11·5	3·35	0·34	75·8	
	Table concentrate	9·5	1·14	0·16	21·3	
	Table tailing	56·8	0·022		2·5	
	Slimes	22·2	0·01	0·01	0·4	
	Average tailing including slimes		0·019			
	Screen test on table tailing					
	+100 mesh	11·9	0·04	0·01	21·4	
	+200 "	34·1	0·02		30·4	
	−200 "	54·0	0·02		48·2	

Flotation Reagents

Test No. 10.—Na_2CO_3, 2·0 lb./ton; coal-tar creosote, 0·07 lb./ton; potassium xanthate, 0·20 lb./ton; $CuSO_4$, 0·40 lb./ton; pine oil No. 5, 0·05 lb./ton.

Test No. 11.—Na_2CO_3, 3·0 lb./ton; P.T. & T. Co. No. 1580, 0·09 lb./ton; Na_2S, 1·0 lb./ton; potassium xanthate, 0·30 lb./ton; pine oil No. 5, 0·05 lb./ton.

Test No. 12.—Na_2CO_3, 3·0 lb./ton.; P. T. & T. Co. No. 1580, 0·9 lb./ton; Na_2S, 1·0 lb./ton; potassium xanthate, 0·30 lb./ton; pine oil No. 5, 0·05 lb./ton.

Concentration by Flotation and Tabling and Cyaniding of the Concentrates

Test No. 13

The same procedure was followed and reagents used as in Test No. 12 for the concentration of the ore. The combined flotation and table concentrates were dewatered and cyanided.

Cyanidation Results

Test No.	Concentrate, grammes	Solution, c.c.	KCN, %	Agitation, hours	Heads, Au, oz./ton	Tailing, Au, oz./ton	Extraction, %	Consumption per ton of concentrate		Consumption per ton of ore	
								KCN, lb.	CaO, lb.	KCN, lb.	CaO, lb.
13—1	259·2	614	0·10	50	2·88	0·22	92·3	6·33	24·2	1·27	4·84
13—2	260·6	612	0·20	50	2·88	0·065	97·3	7·90	22·0	1·58	4·40
13—3	263·7	609	0·15	74	2·88	0·17	94·3	10·20	32·0	2·04	6·40

Screen Tests on Cyanide Tailing

Test No.	Mesh	Weight, %	Assay, Au, oz./ton	Per cent of values	Average tailing, Au, oz./ton
13—1	+150	18·9	0·56	48·3	0·22
	−150	81·1	0·14	51·7	
13—2	+150	23·5	0·09	29·1	0·065
	−150	76·5	0·06	70·9	
13—3	+150	16·5	0·62	60·5	0·17
	−150	83·5	0·08	39·5	

Cyanidation of the concentrates made in Winchester bottles.
Cassel's cyanide, 96·3 per cent NaCN used.
Cyanide consumption based on ratio of concentration of 5:1.

Test No. 14

The same procedure was followed and reagents used as in Tests Nos. 12 and 13. The combined flotation and table concentrates were dewatered and reground for cyanidation.

Cyanidation Results

Test No.	Concentrate, grammes	Solution, c.c.	KCN, %	Agitation, hours	Heads, Au, oz./ton	Tailing, Au, oz./ton	Extraction, %	Consumption per ton of concentrate		Consumption per ton of ore	
								KCN, lb.	CaO, lb.	KCN, lb.	CaO, lb.
14—1	237·0	638	0·10	46	2·13	0·05	97·6	5·9	20·3	1·18	4·06
14—2	245·5	630	0·20	46	2·13	0·05	97·6	9·2	22·3	1·84	4·46
14—3	248·0	627	0·15	67	2·13	0·06	97·2	10·3	28·7	2·06	5·74

Screen test on cyanide tailing was not made.
Cyanidation of the concentrates made in Winchester bottles.
Cassel's cyanide, 96·3 per cent NaCN used.
Cyanide consumption based on ratio of concentration of 5:1.

Summary of Tests Nos. 13 and 14

Test No.	Recoveries, %		
	Concentration	Cyanidation	Net
13—1	97·1	92·3	89·5
13—2	97·1	97·3	94·5
13—3	97·1	94·3	91·5
14—1	97·1	97·6	94·7
14—2	97·1	97·6	94·7
14—3	97·1	97·2	94·3

CONCLUSIONS

Cyanidation of the Ore. Extractions of 94 to 97 per cent of the gold were obtained by cyanidation with reagent consumption of 1·5 pounds KCN per ton and 4 pounds lime per ton.

Concentration. Concentration by flotation alone gave varying recoveries, but when the flotation was followed by tabling the combined concentrates contained 97·1 per cent of the gold in each test. The ratio of concentration averaged about 5 : 1.

Cyanidation of the Concentrates. Cyanidation of the concentrates gave extractions of 92·3 to 97·6 per cent of the gold.

The reagents consumption varied with the strength of cyanide solutions used and the duration of the period of agitation. Regrinding of the concentrates previous to cyanidation resulted in increased extractions and more uniform results.

General. The cyanidation results showed that 95 per cent of the gold may be extracted by grinding to 40 per cent −200 mesh with moderate reagents consumption. The samples were ground dry and the advantage of closed circuit grinding in cyanide solution was lost. In larger scale tests or under actual milling conditions a better extraction might reasonably be expected.

The concentration results were good, and the cyanidation of the concentrates gave good extractions, especially where reground. There is a slight saving in cyanide consumption when based on pounds per ton of original ore; this, however, is offset by the cost of flotation reagents. The ratio of concentration, 5 : 1, might be slightly increased by cleaning the flotation rougher concentrate and retaining the middlings in the flotation circuit. The reduced cyanide consumption per ton of ore would hardly be sufficient to effect a saving when the cost of the flotation reagents is included plus an increased loss in the combined tailings.

Little or no saving in grinding costs could be expected, regardless of which method is employed. If the combined method were used the advantage of grinding in cyanide solution, where the greatest dissolution takes place, would be lost. When this fact together with complications of the flow-sheet and losses due to spills and rich pregnant solutions is considered, the straight cyanidation flow-sheet appears to be the more efficient of the two methods.

Report No. 325

EXPERIMENTAL TESTS ON GOLD ORE OF THE PERRIER MINE, NELSON, B.C.

A. K. Anderson

Shipments. Two bags of ore, gross weight 135 pounds, were received at the Ore Testing Laboratories, May 11, 1929, from the Perrier Syndicate, Nelson, British Columbia.

Characteristics of the Ore. The material received consisted of quartz gangue, carrying a small percentage of lead, zinc, and iron sulphides. The ore was not in large pieces, and included a large amount of finely-divided, semi-oxidized material, brownish red in colour.

Purpose of Experimental Tests. The shipment was made with the request that amalgamation, grinding, and concentration tests be made on the ore to ascertain the best process to apply on a daily tonnage of from 10 to 20 tons.

Sampling and Analysis. The two bags of ore marked 1P and 2P were each crushed to pass 10 mesh, thoroughly mixed, and a representative sample cut out by passing through a Jones riffle sampler.

The two lots on analysis were found to contain:—

—	Au, oz./ton	Ag, oz./ton	Pb, %	Zn, %
Lot 1P	0·29	0·51	0·50	0·94
Lot 2P	0·29	0·43	0·38	0·67

Due to the similarity of the two, as found by analysis and by the first amalgamation tests, all other work was done on Lot 1P.

EXPERIMENTAL TESTS

The tests were conducted on three sizes of material; first, finely ground ore, 85 per cent passing through a 200-mesh screen; second, on material coarsely ground to pass 48 mesh, with 38·8 per cent passing 200 mesh; and third, on ore ground to 150 mesh.

These tests included amalgamation, amalgamation followed by table concentration, concentration by flotation and by tabling, and lastly by cyanidation.

SERIES I: FINE GRINDING, 85 PER CENT—200 MESH

Test No. 1: Amalgamation

Samples of 1,000 grammes each of Lots 1P and 2P were ground 40 minutes in porcelain mills containing iron balls, and 1,000 grammes of water. After grinding and removing the balls, the pulp was amalgamated with 100 grammes of mercury. After removal of the amalgam, the tailing was assayed.

Product	Lot 1P				Lot 2P			
	Assay		Recovery, %		Assay		Recovery, %	
	Au, oz./ton	Ag, oz./ton	Au	Ag	Au, oz./ton	Ag, oz./ton	Au	Ag
Heads	0·29	0·51			0·29	0·43		
Amalgamation tailing	0·04	0·36	86·2	29·4	0·045	0·35	84·5	18·6

The tests show that a large part of the gold is free and can be recovered by amalgamation. The low recovery of silver is probably due to its presence as a sulphide or association with the lead sulphides in the ore.

Test No. 2: Amalgamation and Table Concentration

A sample of 1,000 grammes of Lot 1P was ground under conditions similar to those of Test No. 1. After removing the amalgam, the pulp was passed over a small Wilfley concentrating table and a concentrate and middling product removed.

Product	Weight, %	Assay				Per cent of values			
		Au, oz./ton	Ag, oz./ton	Pb, %	Zn, %	Au	Ag	Pb	Zn
Heads................		0·29	0·51	0·50	0·94	100·0	100·0	100·0	100·0
Table concentrates....	7·5	0·90	1·82	1·95	2·60	23·3	26·8	38·8	28·6
" middling........	1·3	0·11	0·20			0·5	0·5	1·5	1·8
" tailing..........	91·2	0·04	0·21	0·28	0·52	12·6	37·6	59·7	69·6
Amalgamation.........	....	....	....	......	......	63·6	35·1		

The recovery by amalgamation in this test does not coincide with that shown in Test No. 1, being considerably lower, 63·6 per cent of the gold as against 86·2 per cent. The table concentrate, assaying $18 per ton in gold and 1·82 ounces silver, has not a high lead and zinc content. The recovery by amalgamation and concentration was 86·9 per cent of the gold and 61·9 per cent of the silver.

Test No. 3: Concentration by Flotation

A sample of 1,000 grammes of ore was ground in a porcelain mill with 1,000 grammes of water and soda ash equivalent to 5·0 pounds per ton of ore. The pulp was then transferred to a Ruth flotation machine, and floated with potassium xanthate, 0·20 pound per ton, and pine oil, 0·04 pound per ton.

Product	Weight, %	Assay				Per cent of values			
		Au, oz./ton	Ag, oz./ton	Pb, %	Zn, %	Au	Ag	Pb	Zn
Heads................	....	0·29	0·51	0·50	0·94	100·0	100·0	100·0	100·0
Flotation concentrate..	5·2	4·26	7·06	7·42	9·62	69·5	80·8	90·5	70·5
" middling....	2·6	2·67	1·94	......	......	21·8	11·1	3·0	3·4
" tailing.......	92·2	0·03	0·04	0·03	0·20	8·7	8·1	6·5	26·1

As was the case in Test No. 2 not enough middling product was obtained to make a complete analysis. The lead and zinc assays of this product were, therefore, assumed equal to the head assay for calculation purposes.

This test shows that a high recovery of all the metals can be secured by flotation. Without cleaning the rougher concentrate, a total recovery is obtained of 91·3 per cent of the gold, 91·9 per cent of the silver, 93·5 per cent of the lead, and 73·9 per cent of the zinc, with a concentration ratio of 12·8 : 1.

SERIES II: COARSE GRINDING—48 MESH

For these tests, the ore was ground to pass 48 mesh. The following screen test shows the percentages on the finer sizes.

	Per cent
− 48+ 65	11·9
− 65+100	27·3
−100+150	12·1
−150+200	9·9
−200	38·8

Test No. 1: Amalgamation

A sample of 1,000 grammes of ore, −48 mesh, was amalgamated with 100 grammes of mercury.

Product	Assay		Recovery, %	
	Au, oz./ton	Ag, oz./ton	Au	Ag
Heads	0·29	0·51		
Amalgamation tailing	0·09	0·44	69·0	13·8

The recovery of gold at this mesh shows a slightly higher recovery than that of Test No. 2, Series I, 69·0 per cent as against 63·6 per cent. It would appear, therefore, that the amalgamable gold is freed at 48 mesh.

Test No. 2: Table Concentration

A sample of 1,000 grammes of ore, −48 mesh, was passed over a laboratory-size Wilfley table.

Product	Weight, %	Assay				Per cent of values			
		Au, oz./ton	Ag, oz./ton	Pb, %	Zn, %	Au	Ag	Pb	Zn
Heads	....	0·29	0·51	0·50	0·94	100·0	100·0	100·0	100·0
Table concentrate	11·15	1·60	2·24	2·15	2·76	56·1	49·8	50·0	45·4
" middling	12·32	0·14	0·24	0·21	0·39	5·4	5·9	5·4	7·1
" tailing	76·53	0·16	0·29	0·28	0·42	38·5	44·3	44·6	47·5

This test indicates that high recoveries at this mesh, on an unclassified feed are not to be expected.

Test No. 3: Amalgamation and Table Concentration

A sample of 1,000 grammes of ore, −48 mesh, was amalgamated, and after removal of amalgam, was passed over the Wilfley table.

Product	Weight, %	Assay				Per cent of values			
		Au, oz./ton	Ag, oz./ton	Pb, %	Zn, %	Au	Ag	Pb	Zn
Heads	....	0·29	0·51	0·50	0·94	100·0	100·0	100·0	100·0
Table concentrate	7·87	0·50	1·82	2·10	2·55	13·6	28·2	32·3	26·0
" middling	10·43	0·10	0·32	0·26	0·62	3·6	6·5	5·3	8·4
" tailing	81·70	0·05	0·29	0·39	0·62	14·1	46·4	62·4	65·6
Amalgamation	....	....	....	......	......	68·7	18·9		

The recovery by amalgamation in this test checks that of Test No. 1, Series II, 68·7 per cent and 69·0 per cent respectively. An additional recovery of 17·2 per cent is obtained in the table concentrate and middling. A total recovery of 85·9 per cent of the gold is secured by the two processes. The lead and zinc content of the table concentrate is low.

SERIES III: FINE GRINDING TO 150 MESH

Cyanidation

Ore ground to pass 150 mesh was agitated for 48 hours, 1 : 2 dilution with sodium cyanide equivalent in strength to 2·0 pounds KCN per ton. Two tests were run, one for 24 hours and the other for 48 hours. Lime was added at the rate of 5 pounds and 10 pounds per ton of ore in Tests Nos. 1 and 2 respectively.

Time of agitation	Titrations, lb./ton				Consumption, lb./ton ore		Assay		Extraction, %	
	Start		Finish		KCN	CaO	Au, oz./ton	Ag, oz./ton	Au	Ag
	KCN	CaO	KCN	CaO						
24 hours..............	2·0	......	0·95	Nil	2·1	5·0	0·012	0·205	95·9	60·0
48 hours..............	2·0	......	0·80	0·85	2·4	8·3	0·012	0·29	95·9	43·1

This test shows that the gold is readily soluble in cyanide solution, maximum extraction being obtained within 24 hours. However, a consumption of from 2·1 to 2·4 pounds cyanide per ton of ore is noted. Also, there is a strong tendency toward fouling of solutions. Five pounds of lime per ton of ore is not sufficient to maintain sufficient protective alkalinity, as indicated in the 24-hour test.

CONCLUSIONS

A comparison of the results secured by coarse and fine grinding as shown in Series I and II, indicates that fine grinding is not necessary to liberate the gold. Grinding to −65 mesh should yield as high a recovery as can be secured. Due to the similarity of results secured by amalgamation in all other tests, the high recovery indicated in Test No. 1 can be discounted and a figure of from 63 to 69 per cent may be expected.

A combination of amalgamation and table concentration should yield a total recovery of from 82·87 per cent of the gold, with from 13 to 18 per cent of this contained in a concentrate assaying approximately $10 per ton in gold and 1·5 to 2·0 ounces silver. The lead and zinc content will not be high, from 2 to 2½ per cent. A concentration ratio of 12 : 1 can be expected.

The highest recoveries indicated are those in Series I, Test No. 3, secured by flotation, where the following recoveries are recorded in a combined concentrate and middling: gold, 91·3 per cent; silver, 91·9 per cent; lead, 93·5 per cent; and zinc, 73·9 per cent; with a corresponding assay of gold, 3·72 ounces per ton; silver, 5·35 ounces per ton; lead, 5·11 per cent, and zinc 6·72 per cent. By preceding flotation with amalgamation and assuming a recovery by this operation of 65 per cent of the gold and 14 per cent of the silver, a shipping product containing 1·2 ounces gold, 4·5 ounces silver, 5·1 per cent lead, and 6·7 per cent zinc can be expected.

However, due to the small tonnage, 10 to 20 tons per day, to be treated, the installation and operating cost of a flotation unit will be much higher than a grinding mill equipped with amalgamation and followed by concentrating tables.

The low gross value of the ore, $5.80 in gold and 0·5 ounce per ton silver, makes it very doubtful whether a profit can be shown on a small tonnage.

The factors governing the plant to be installed will be the initial cost and subsequent milling charges together with freight and treatment costs on concentrates shipped.

The choice between a flow-sheet incorporating amalgamation and table concentration with a total recovery of 90·9 per cent, representing a saving of 29 cents per ton or $5.80 per day on a 20-ton basis, will be determined by the cost of equipment and operation. Whether this saving of $5.80 a day will pay for the interest and depreciation on the additional cost of a flotation unit over that of a table concentrator, and take care of the additional cost of operation and reagents, will be a point for careful consideration by the management.

As stated above, the ore is of such a low grade that operating on a very small tonnage will not be profitable.

Report No. 326

THE RECOVERY OF GOLD AND SILVER FROM THE ORE OF THE GEM LAKE MINE, EAST CENTRAL MANITOBA

A. K. Anderson

Shipment. Sixty-five bags of ore, gross weight 5,700 pounds were received at the Ore Testing Laboratories on April 11, 1929, from the Gem Lake mine, east central Manitoba, near the Manitoba-Ontario boundary. The shipment was submitted by the Gem Lake Mines, Ltd., 252 Fort Street, Winnipeg. Victor James of Wright, Boydell, James and Associates, 310 McKinnon Building, Toronto, is consulting engineer.

Characteristics of the Ore. The ore consisted of white and blue quartz carrying a low percentage of iron sulphides. Fine free gold was visible. Approximately 15 per cent of the shipment was barren schist wall-rock.

Purpose of Experimental Tests. The shipment was made for the purpose of determining the assay value of the ore in gold and silver, whether any refractory minerals were present, and by test work, to find the most suitable metallurgical process to apply for the recovery of contained values.

Sampling and Analysis. The ore as received was crushed by jaw breakers and rolls until all was reduced to $\frac{1}{8}$ inch. One-tenth of the total weight was cut out by an automatic sampler. By stage crushing and grinding to and quartering at 10 mesh, 35 mesh, 65 mesh, and finally 100 mesh duplicate samples were obtained from this portion which assayed as follows:—

1. Gold	3·23 oz./ton	Silver	0·35 oz./ton
2. Gold	3·26 "	Silver	0·35 "

The analysis of these head samples for arsenic, antimony, bismuth, and copper showed them to be not present in the ore.

EXPERIMENTAL TESTS

Four classes of test work were undertaken.

1. Grinding in water and amalgamating with mercury.
2. Grinding in cyanide solution and amalgamating, followed by cyanidation of the amalgamation tailing.
3. Concentration by flotation followed by cyanidation of the flotation tailing.
4. Straight cyaniding at different degrees of grinding.

1. AMALGAMATION

Test No. 1

A sample, 1,000 grammes of ore, was ground to pass 48 mesh and amalgamated with 10 per cent weight of mercury.

Results:

Heads	Au 3·24 oz./ton	Ag 0·35 oz./ton
Amalgamation tailing	Au 1·22 "	Ag 0·19 "
Recovery	Au 62·3 per cent	Ag 45·7 per cent

This shows that at 48 mesh, a high recovery of the gold and silver is not obtained by amalgamation alone.

2. AMALGAMATION AND CYANIDATION

Test No. 2

A sample, 1,000 grammes of ore, was ground in a porcelain mill containing iron balls with 1,000 grammes of sodium cyanide solution equivalent in strength to 2·0 pounds KCN per ton. Lime was added at the rate of 5·0 pounds per ton of ore. After grinding, the pulp was filtered and the solution retained for the following cyanide agitation. The ground ore was then amalgamated and after removal of the amalgam was agitated for 48 hours in 2 : 1 dilution with the original cyanide solution retained after grinding. The strength of this was brought to 2·0 pounds KCN per ton by the addition of fresh sodium cyanide.

Results:

Heads	Au 3·24 oz./ton	Ag 0·35 oz./ton
Amalgamation tailing	Au 0·49 "	Ag 0·12 "
Recovery	Au 84·9 per cent	Ag 65·7 per cent

Screen analysis of tailing after grinding in cyanide solution and amalgamating.

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Tailing +150 mesh	4·9	0·68	0·12	6·8	4·9
+200 mesh	15·8	0·56	0·12	18·2	15·8
−200 mesh	79·3	0·46	0·12	75·0	79·3

Cyanidation of Amalgamation Tailing

(Agitation—24 hours)

	Au	Ag
Heads (amalgamation tailing)	0·49 oz./ton	0·12 oz./ton
Cyanide tailing	0·11 "	0·05 "
Recovery	77·6 per cent	58·3 per cent

(Agitation—48 hours)

	Au	Ag
Heads (amalgamation tailing)	0·49 oz./ton	0·12 oz./ton
Cyanide tailing	0·05 "	0·01 "
Recovery	90·0 per cent	92·0 per cent
Recovery on original feed	13·6 "	31·6 "

Reagent consumption	KCN, 1·4 lb./ton
	CaO, 7·0 "

The tailing from the 48-hour cyanide test showed the following:—

—	Weight, %	Assay, oz./ton	
		Au	Ag
+200 mesh	20·5	0·12	0·03
−200 mesh	79·5	0·03	trace

Recovery by grinding in cyanide solution and amalgamating	Au 84·9 per cent	Ag 65·7 per cent
Recovery by cyaniding amalgamation tailing	Au 13·6 "	Ag 31·6 "
Total recovery	Au 98·5 "	Ag 97·3 "

An examination of this test shows that grinding to 79 per cent −200 mesh in cyanide solution and amalgamating, leaves a tailing containing 0·49 ounce gold and 0·12 ounce silver per ton. The +150-mesh portion of this assays 0·68 ounce and the −150+200-mesh part, 0·56 ounce gold per ton.

These results confirm the conclusions arrived at from Test No. 1, that a high recovery by amalgamation alone is not to be expected.

Cyanidation of the residue shows that a high recovery can be obtained by this method.

3. CONCENTRATION AND CYANIDATION

Test No. 3

A sample, 1,000 grammes of the ore, was ground in 1 : 1 pulp with water, and soda ash equal to 6·0 pounds per ton of ore. Flotation was performed in a Ruth mechanically agitated machine. Amyl xanthate, 0·2 pound per ton, and pine oil sufficient to froth, were added to the cell. After removing a pyrite concentrate, the flotation tailing was passed over a small Wilfley table. A complete recovery of the sulphides by flotation was indicated, as no concentrate was secured by tabling.

The tailing from the table was cyanided for 24 and 48 hours. A 2 : 1 pulp dilution was used with cyanide solution equivalent in strength to 2·0 pounds KCN per ton. Lime was added at the rate of 5·0 pounds per ton of ore.

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Flotation concentrate	1·28	192·06	19·82	86·2	68·2
" tailing	98·72	0·40	0·12	13·8	31·8

Cyanidation of Flotation Tailing

(Agitation—24 hours)

	Au	Ag
Heads (flotation tailing)	0·40 oz./ton	0·12 oz./ton
Cyanide tailing	0·19 "	0·09 "
Recovery	52·5 per cent	25 per cent

(Agitation—48 hours)

	Au	Ag
Heads (flotation tailing)	0·40 oz./ton	0·12 oz./ton
Cyanide tailing	0·05 "	0·04 "
Recovery	87·5 per cent	66·7 per cent
Recovery on original feed	12·1 "	21·2 "
	Au	Ag
Recovery by flotation	86·2 per cent	68·2 per cent
Recovery by cyanidation	12·1 "	21·2 "
Total recovery	98·3 "	89·4 "

An analysis of the tailing from the 48-hour cyanide test shows:—

—	Weight, %	Assay	
		Au	Ag
Tailing +200 mesh	18·5	0·11	0·08
Tailing −200 "	81·5	0·04	0·03

A high recovery of the values can be secured by this method. No tests were made on the very high-grade concentrate which assays 192·06 ounces per ton. The residue after cyaniding the tailing from flotation again shows a marked difference between the +200- and −200-mesh portions. Fine grinding to secure maximum recoveries is indicated.

4. STRAIGHT CYANIDATION

Test No. 4

Ore ground to pass 48 mesh with 43 per cent through 150 mesh was agitated with sodium cyanide solution (2·0 pounds KCN per ton), 1 : 2 dilution for 24 and 48 hours. Lime was added at the rate of 5·0 pounds per ton of ore.

Cyanidation Results

(Agitation—24 hours)

	Au	Ag
Heads	3·24 oz./ton	0·35 oz./ton
Tailing +150 mesh, weight 56·6 per cent	0·49 "	9·14 "
Tailing −150 mesh, weight 43·4 per cent	0·15 "	0·03 "
Average tailing	0·25 "	0·09 "
Recovery	89·5 per cent	74·3 per cent

Reagent consumption	KCN, 0·20 lb./ton CaO, 3·10 "

(Agitation—48 hours)

	Au	Ag
Heads	3·24 oz./ton	0·35 oz./ton
Tailing +150 mesh, weight 53·6 per cent	0·38 "	0·09 "
Tailing −150 mesh, weight 46·4 per cent	0·11 "	0·02 "
Average tailing	0·25 "	0·06 "
Recovery	92·3 per cent	82·8 per cent

Reagent consumption	KCN, 0·20 lb./ton CaO, 3·3 "

This test shows that with coarse grinding to 48 mesh, 92·3 per cent of the gold and 82·8 per cent of the silver is extracted in 48 hours' agitation with a cyanide consumption of 0·2 pound per ton of ore.

Test No. 5

The same procedure was followed as in Test No. 4, with the exception that the ore was ground to pass 100 mesh instead of 48 mesh.

Cyanidation Results

(Agitation—24 hours)

	Au	Ag
Heads	3·24 oz./ton	0·35 oz./ton
Tailing +150 mesh, weight, 28·0 per cent	0·26 "	0·08 "
Tailing −150 mesh, weight 72·0 per cent	0·10 "	0·04 "
Average tailing	0·15 "	0·05 "
Recovery	95·4 per cent	85·7 per cent

Reagent consumption	KCN, 0·30 lb./ton CaO, 3·3 "

(Agitation—48 hours)

	Au	Ag
Heads	3·24 oz./ton;	0·35 oz./ton
Tailing +150 mesh, weight 29·9 per cent	0·24 "	0·06 "
Tailing −150 mesh, weight 70·1 per cent	0·07 "	0·03 "
Average tailing	0·12 "	0·04 "
Recovery	96·3 per cent;	88·6 per cent

Reagent consumption.................................... KCN, 0·50 lb./ton
CaO, 3·5 "

This test shows that grinding to increase 100 mesh, with 75 per cent through 150 mesh, 96·3 per cent of the gold and 88·6 per cent of the silver is in solution within 48 hours. The cyanide consumption is 0·5 pound per ton of ore.

Test No. 6

This test was made on ore ground to pass 150-mesh screen. Other conditions were the same as in Tests Nos. 4 and 5.

Cyanidation Results

(Agitation—24 hours)

	Au	Ag
Heads	3·24 oz./ton;	0·35 oz./ton
Tailing +200 mesh, weight 14·0 per cent	0·16 "	0·04 "
Tailing −200 mesh, weight 86·0 per cent	0·05 "	0·02 "
Average tailing	0·065 "	0·02 "
Recovery	98·0 per cent;	94·3 per cent

(Agitation—48 hours)

	Au	Ag
Heads	3·24 oz./ton;	0·35 oz./ton
Tailing +200 mesh, weight 17·2 per cent	0·12 "	0·06 "
Tailing −200 mesh, weight 82·8 per cent	0·05 "	0·03 "
Average tailing	0·06 "	0·04 "
Recovery	98·1 per cent;	88·6 per cent

Reagent consumption.................................... KCN, 0·60 lb./ton
CaO, 3·80 "

This test indicates that with fine grinding 86 per cent −200 mesh, an extraction of 98 per cent of the gold and 94 per cent of the silver is obtained within 24 hours. No advantage is obtained in prolonging the time of agitation as the −200-mesh portions of the two lots, agitated for 24 and 48 hours respectively, show no difference in the amount of gold that goes into solution.

A comparison of the cyanide results at the different degrees of grinding is given in the following table:—

Test No.	Degree of grinding	Tailing assay		Recovery, %	
		Au, oz./ton	Ag, oz./ton	Au	Ag
4	− 48	0·25	0·06	92·3	82·8
5	−100	0·12	0·04	96·3	88·6
6	−150	0·06	0·04	98·1	88·6

SUMMARY

The following tabulation shows the comparative tailing assay and recovery obtained by the four different methods of tests.

Test No.	Method	Tailing assay		Recovery, %	
		Au, oz./ton	Ag, oz./ton	Au	Ag
1	Amalgamation..................................	1·22	0·19	62·3	45·7
2	Amalgamation and cyanidation.................	0·05	0·01	98·5	97·3
3	Concentration and cyanidation.................	0·05	0·04	98·3	89·4
4	Straight cyanidation..........................	0·06	0·04	98·1	88·6

In Test No. 3, concentration by flotation followed by cyanidation of the flotation tailing, the recoveries recorded do not take into consideration the additional treatment necessary to recover the gold in the flotation concentrate. As this product is worth over $1.90 a pound it would not be practical to recover the gold by cyanidation and would best be sold to the smelter. The cost of marketing this product would have to be considered.

The tests show that fine grinding in cyanide solution to at least 80 per cent through 200 mesh, followed by amalgamation and agitation gives the highest recoveries. Test No. 2 shows a recovery of 98·5 per cent of the gold and 97·3 per cent of the silver by this method as against 98·1 per cent of the gold and 88·6 per cent of the silver by straight cyanidation in Test No 4. Authorities differ as to the practical application of amalgamation of a cyanide pulp; some claiming that the action of the lime necessary to maintain the alkalinity of the pulp for cyaniding has a hardening effect on the amalgam, that cyanide has a solvent effect on mercury and the copper plates are slowly corroded underneath the coating of amalgam, making it necessary to renew them in from six months to two years, depending largely on the attention they receive and other local factors. Others claim no difficulties from these causes and have amalgamated successfully a cyanide pulp.

If amalgamation in water is practised the amalgamation tailing will have to be thickened and filtered before cyanidation to remove excess water and would require additional plant equipment. As the average grade of milling ore will, in all probability, not contain the high values of this shipment on which the tests were conducted, it will probably not be necessary to amalgamate. It is therefore recommended that straight cyanidation of the ore be adopted as being the simpler process. The tests show that the grinding should be at least 80 per cent −200 mesh for good results. The cyanide consumption is moderate, 0·6 pound per ton, and the lime consumption under 4 pounds per ton.

Report No. 327

THE CONCENTRATION OF THE COPPER ORE OF THE WINDSOR MINE, LA SARRE, QUE.

A. K. Anderson

Shipment. One piece of ore, weight 28 pounds, was received on June 25, 1929, from W. J. Blainey, 33 Adelaide Street West, Toronto, Ontario. The ore was from the property of the Windsor Mines, Ltd., La Sarre, Quebec.

Characteristics of the Ore. The piece consisted of heavy, massive sulphides, pyrite and pyrrhotite carrying chalcopyrite and small amounts of zinc blende.

Purpose of Tests. The sample was sent in for assay and also to determine the behaviour of the sulphides when treated by flotation for the concentration of the copper and zinc.

Sampling and Analysis. The piece was crushed; ground to pass through 10 mesh, and a representative portion cut out by passing through a Jones riffle sampler. It contained the following assay values:—

Copper..................	4·70 per cent	Gold....................	0·02 oz./ton
Zinc.....................	1·78 "	Silver...................	0·34 "

EXPERIMENTAL TESTS

Due to the small shipment (and probably not representative of the orebody), one test was conducted to ascertain what recovery could be obtained by standard flotation practice.

A 1,000-gramme sample of the ore was finely ground with an equal weight of water containing 3·0 grammes soda ash, and 0·1 gramme sodium cyanide. It was then floated with 0·06 gramme potassium xanthate and two drops of pine oil. The copper concentrate secured was cleaned, producing a middling product. Copper sulphate was then added at the rate of 1 pound per ton of ore, and 0·04 gramme xanthate added, and a zinc concentrate removed. Due to the small bulk of the zinc concentrate, it was not cleaned. The results were as follows:—

Product	Weight, %	Assay				Per cent of values			
		Cu, %	Zn, %	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Heads..................	100·00	4·70	1·78	0·02	0·34	100·00	100·00	100·0	100·0
Copper concentrate....	16·16	27·50	1·25	0·08	1·56	96·29	11·29	65·0	73·5
Copper middling.......	9·63	0·31	2·09	0·01	0·59	0·65	11·25	5·0	17·7
Zinc concentrate.......	3·01	0·91	39·68	....	....	0·59	67·11		
Zinc tailing............	71·20	0·16	0·26	....	....	2·47	10·35		

Reagents used:

Copper circuit—		Zinc circuit—	
Soda ash.............	6·0 lb./ton	Pot. xanthate.........	0·08 lb./ton
Cyanide..............	0·20 "	Pine oil..............	0·02 "
Pot. xanthate.........	0·12 "	Copper sulphate.......	1·0 "

The test shows that the ore responds quite readily to concentration by flotation, 96 per cent of the copper is recovered in a product assaying 27·50 per cent copper. The zinc shows a recovery of 67 per cent in a rougher concentrate assaying 40 per cent zinc. The gold and silver are largely recovered with the copper. No further work was undertaken on this sample.

Report No. 328

COMPARATIVE TESTS ON TWO TYPES OF ORE FROM THE SYLVANITE GOLD MINES, LTD., KIRKLAND LAKE, ONT.

J. S. Godard

Shipment. A shipment consisting of two samples, each weighing 500 pounds, was received May 31, 1929, from the Sylvanite Gold Mines, Ltd., Kirkland Lake, Ont.

Characteristics of the Ore. Both samples were those of gold ores. Sample No. 1 was taken from what is termed the main south vein. It is more highly schisted and appears to contain a greater percentage of silica than Sample No. 2, which represents the regular mill feed and is the typical reddish feldspar porphyry of the Kirkland Lake district.

Analysis:

Sample No. 1, Au	0·39 oz./ton
Sample No. 2, Au	0·47 "

Purpose of Experimental Tests. The purpose of the experimental tests was to determine any difference in the extractions, cyanide consumptions, and settling properties of the two types of ore.

EXPERIMENTAL TESTS

Cyanidation Tests in Winchester Bottles

Ore at −20 mesh, ground wet to 95 per cent −200 mesh, KCN 0·04 per cent, dilution 1 : 2·5, time 48 hours. The tailings were screened on 200 mesh with the following results:—

Sample No. 1—Test No. 1

Mesh	Weight, %	Assay, Au, oz./ton	Average tailing, Au, oz./ton
+200	5·0	0·07	0·023
−200	95·0	0·02	

Sample No. 1—Test No. 2

+200	5·2	0·14	0·026
−200	94·8	0·02	

Sample No. 2—Test No. 1

+200	5·5	0·13	0·045
−200	94·5	0·04	

Sample No 2—Test No. 2

+200	4·9	0·07	0·041
−200	95·1	0·04	

Summary: Samples Nos. 1 and 2—Tests Nos. 1 and 2

Sample No.	Test No.	Head	Tailing	Extraction, %	KCN, lb./ton
1	1	0·39	0·023	94·1	0·28
1	2	0·39	0·026	93·3	0·29
2	1	0·47	0·045	90·4	0·12
2	2	0·47	0·041	91·3	0·08

Cyanidation Tests in Glass Agitators

Ore at −20 mesh, ground wet to 95 per cent −200 mesh, KCN 0·04 per cent, dilution 1 : 2·5, time 48 hours. The tailings were screened on 200 mesh with the following results:—

Sample No. 1—Test No. 3

Mesh	Weight, %	Assay, Au, oz./ton	Average tailing, Au, oz./ton
+200	5·1	0·85	0·062
−200	94·9	0·02	

Sample No. 2—Test No. 3

+200	5·5	0·05	0·041
−200	94·5	0·04	

Summary: Samples Nos. 1 and 2 —Test No. 3

Sample No.	Test No.	Head, Au, oz./ton	Tailing, Au, oz./ton	Extraction, %	KCN, lb./ton
1	3	0·39	0·062	84·0	0·28
2	3	0·47	0·041	91·3	0·11

Cyanidation Tests in 100-pound Unit

Ore at −200 mesh, ground to 95 per cent −200 mesh in small rod mill in closed circuit with a classifier. Classifer overflow agitated in Pachuca tank, KCN 0·04 per cent for grinding circuit and agitation, dilution 1 : 2·5, time of agitation 34 hours. Tailings filtered in frame press and repulped.

Sample No. 1—Test No. 4

Mesh	Weight, %	Assay, Au, oz./ton	Average tailing, Au, oz./ton
+200	4·5	0·03	0·016
−200	95·5	0·015	0·016

Sample No. 2—Test No. 4

+200	6·0	0·04	0·031
−200	94·0	0·03	

Duplicate Sample

+200	5·7	0·045	0·026
−200	94·3	0·025	

Summary: Samples Nos. 1 and 2—Test No. 4

Sample No.	Test No.	Head, Au, oz./ton	Tailing, Au, oz./ton	Extraction, %
1	4	0·39	0·016	95·7
2	4	0·47	0·029	93·8

Settling Tests

Tests made in 4,000 c.c. cylinder, $4\frac{9}{16}$-inch diameter. Tailings from Sample No. 1, Test No. 4, and Sample No. 2, Test No. 4, were used. They were screened on 200 mesh and samples were made up containing 3 per cent +200 mesh. Solution from Sample No. 2, Test No. 4, was used. Readings were taken at 3-minute intervals for a 30-minute period.

For results *see* Figure 3.

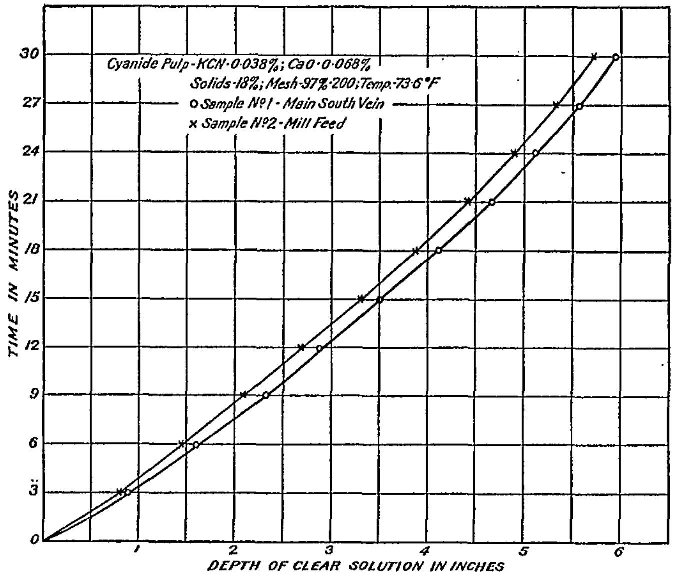

FIGURE 3. Graph showing rate of settlement of solids in cyanide pulp from gold ores of Sylvanite mine.

CONCLUSIONS

Extraction. Sample No. 1 proved to be a less refractory ore than No. 2. A higher extraction and a lower tailing was obtained on Sample No. 1 though the head was of lower grade. In Sample No. 1 (Test No. 3) the +200-mesh tailing was very high. This was due to free gold settling out in the agitator.

Cyanide Consumption. Sample No. 1 consumes more cyanide than No. 2. The cyanide consumptions were in the following proportion:—

$$\frac{\text{Sample No. 1}}{\text{Sample No. 2}} = \frac{28}{12}$$

Settling. In the same solution, Sample No. 1 showed a slightly faster settling rate.

Report No. 329

THE RECOVERY OF GOLD FROM THE ORE OF THE EVANGELINE GOLD MINES, LTD., LISCOMB MILLS, NOVA SCOTIA

A. K. Anderson

Shipment. One bag of gold ore, approximately 100 pounds in weight, was received at the Ore Dressing Laboratories, May 30, 1929, from the Evangeline Gold Mines, Ltd., Liscomb Mills, N.S.

Characteristics of the Ore. The ore consisted of quartz gangue carrying free gold and approximately 2 per cent arsenopyrite.

Purpose of Experimental Tests. The shipment was made for the purpose of determining what recovery of gold could be obtained by amalgamation followed by concentration.

Sampling and Analysis. The ore was crushed to pass 10 mesh, cut through a Jones riffle sampler with intervening further grinding to pass 35, 65, and 100 mesh, until a final sample was secured for assay. This showed the shipment to contain 0·42 ounce gold and 0·09 ounce silver per ton. The arsenic content amounted to 0·94 per cent.

Two tests were made on the ore. Test No. 1 was conducted on material more coarsely ground than Test No. 2. Both were amalgamated with mercury and then passed over a Wilfley concentrating table.

Test No. 1

A sample, 1,000 grammes of ore, −10 mesh, was ground with an equal weight of water in a porcelain mill containing iron balls for 10 minutes, after which the balls were removed and the ore amalgamated with mercury. After removal of the amalgam, the pulp was passed over a laboratory-size Wilfley concentrating table. The middling product was cleaned and then added to the concentrate.

Product	Weight, %	Assay, Au, oz./ton	Per cent of values
Heads	....	0·42	
Recovery by amalgamation	....		84·9
Amalgamation tailing	....	0·06	
Table concentrate	3·58	0·80	6·8
Table tailing + 65 mesh	24·51	0·02	1·2
" − 65+100 "	23·59	0·03	1·7
" −100+150 "	9·17	0·07	1·5
" −150+200 "	8·40	0·05	1·0
" −200	30·75	0·04	2·9
Average table tailing	....	0·035	

In this test, as shown by the weights of the various sizes in the above table, 24·5 per cent of the ore remains on 65 mesh with 30·7 per cent passing 200 mesh. The assays of these different sizes show that the gold is freed by crushing at coarse sizes. A recovery of 84·9 per cent is secured by amalgamation along with an additional recovery of 6·8 per cent in the table concentrate or a total of 91·7 per cent.

Test No. 2

In this test grinding was carried on for double the length of time of Test No. 1. Other conditions and procedure were the same.

Product	Weight, %	Assay, Au, oz./ton	Per cent of values
Heads	....	0·42	
Recovery by amalgamation	....		96·4
Amalgamation tailing	....	0·015	
Table concentrate	2·5	0·38	2·3
Table tailing + 65 mesh	4·4	0·025	0·2
" − 65+100 "	22·5	0·005	0·3
" −100+150 "	11·3	0·005	0·1
" −150+200 "	10·9	0·005	0·1
" −200 "	48·4	0·005	0·6
Average table tailing	....	0·005	

In this test, grinding so that 4·4 per cent remains on 65 mesh, a recovery by amalgamation of 96·4 per cent of the gold is obtained, or a total of 98·7 per cent when followed by table concentration.

CONCLUSIONS

It appears that a large part of the gold in the ore is recoverable by amalgamation. An additional recovery is made on concentrating the arsenopyrite. In Test No. 2, where a total recovery of 98·7 per cent is secured, grinding is finer than would ordinarily be done by stamp mills. The screen analysis of Test No. 1 would more closely parallel that secured in practice.

Crushing in a stamp battery to 35 mesh with inside and plate amalgamation followed by table concentration of the amalgamation tailings will prove the best practice. Finer grinding would give higher amalgamation results but the loss in stamp duty would probably counteract the increased recovery.

The table concentrate could be returned to the stamp battery, or better still to a small regrinding mill, the discharge of which could be returned to the stamp battery. A balance will be obtained when the finely reground sulphides will pass out in the table tailing thus recovering as much as possible of the amalgamable gold liberated by the regrinding of the concentrates.

Report No. 330

THE RECOVERY OF GOLD FROM THE ORE OF THE BELLEDAT-GOUDREAU MINE, GOUDREAU, ONTARIO

A. K. Anderson

Shipment. Ten bags of gold ore, gross weight 1,070 pounds, were received April 6, 1929, at the Ore Dressing Laboratories, from the Belledat-Goudreau mine of the Towagmac Exploration Company, Goudreau, Ontario.

Characteristics of the Ore. The ore consisted of quartz gangue, carrying a small amount of iron sulphides and a small amount of graphite as carbonaceous material. Lead sulphide was present in very small amounts.

Purpose of Experimental Tests. The shipment was made to determine the best method to apply for the recovery of the precious metals.

Sampling and Analysis. The entire lot was passed through jaw crushers and rolls until reduced to $\frac{1}{4}$-inch and a tenth portion cut out by an automatic sampler. This portion was crushed to 10 mesh and cut through a Jones riffle sampler. Further grinding to pass the finer meshes and passes through the sampler yielded a final sample for assay. This showed the following values: gold, 1·92 ounces, silver, 0·82 ounce per ton.

EXPERIMENTAL TESTS

Three distinct methods were employed in making the examination: first, cyanidation; second, concentration by flotation followed by cyanidation of the concentrate and tailing; and third, amalgamation followed by cyanidation of the tailing.

CYANIDATION

Test No. 1

Ore ground to pass 65 mesh, with 50 per cent through 150 mesh, was agitated for 48 hours with sodium cyanide solution equivalent in strength to 2·0 pounds KCN per ton. Lime was added at the rate of 5·0 pounds per ton of ore. Ratio of ore to solution was 1 : 2.

Head assay	Au, 1·92 oz./ton
Tailing assay	Au, 1·19 "
Extraction	38·0 per cent
Cyanide consumed	0·30 lb./ton ore

Screen Analysis of Tailing

Screen size	Weight, %	Assay, Au, oz./ton
+150 mesh	50·6	1·02
−150 "	49·4	1·36

This test shows that cyanidation at this mesh is not successful. As the −150-mesh portion of the tailing is quite high, extremely fine grinding may be necessary.

Test No. 2

Ore ground to pass 100 mesh was cyanided for 48 hours under conditions similar to those of Test No. 1.

Head assay, Au	1·92 oz./ton
Tailing assay, Au	1·29 "
Recovery	32·8 per cent
Cyanide consumed	0·50 lb./ton ore

Screen Analysis of Tailing

Screen size	Weight, %	Assay, Au, oz./ton
+150 mesh	31·3	1·45
−150 "	68·7	1·22

This test again shows that the ore does not respond to treatment by cyanidation.

Test No. 3

Ore ground to pass 150 mesh was agitated with cyanide solution under the same conditions as Tests Nos. 1 and 2.

Head assay, Au	1·92 oz./ton
Tailing, Au	0·47 "
Extraction	75·6 per cent
Cyanide consumed	0·70 lb./ton ore

Screen Analysis of Tailing

Screen size	Weight, %	Assay, Au, oz./ton
+200 mesh	17·7	0·53
−200 "	82·3	0·46

This test shows that with fine grinding, 82 per cent −200 mesh, an extraction of 75·6 per cent is obtained. The −200-mesh portion of the tailing still contains considerable gold, $9.20 a ton.

From the above tests, it appears that straight cyanidation of the ore will not be successful.

FLOTATION

Test No. 4

A sample, 1,000 grammes of ore, was ground in a porcelain ball mill containing iron balls, and 1,000 grammes of water, together with soda ash equivalent to 3 pounds per ton of ore.

The pulp was then floated in a Ruth laboratory flotation machine with 0·20 pound potassium xanthate and 0·06 pound pine oil per ton of ore. The concentrate was not cleaned.

Flotation Results

Product	Weight, %	Assay, Au, oz./ton	Per cent of values
Flotation concentrate	3·97	30·80	84·7
" tailing	96·03	0·23	15·3

Screen Analysis of Tailing

Product	Weight, %	Assay, Au, oz./ton
+100 mesh	0·5	14·13
−100+150	1·8	1·52
−150+200	16·2	0·24
−200	81·5	0·12

This test shows that the greater part of the gold is associated with the sulphides and by fine grinding can be concentrated by flotation. The assay value of the −200-mesh portion would indicate that fine grinding is important. The high values of the +100-mesh portion possibly is due to the presence of free gold.

CONCENTRATION AND CYANIDATION

Test No. 5

In this test concentrating the values by flotation, followed by cyanidation of the concentrate and of the tailing was studied.

The ore was ground in water, 1 : 1 dilution together with soda ash equal to 2·0 pounds per ton of ore, until approximately 80 per cent passed 200 mesh.

Flotation was carried on in a Denver Sub-A type machine, using 0·20 pound amyl xanthate, 0·10 pound wood creosote, 0·10 pound wood tar, and 0·08 pound pine oil per ton. The concentrate was not cleaned.

After flotation, the concentrate and tailing were dewatered and cyanided for 48 hours. The concentrate was agitated with a 5·0 pound per ton cyanide solution and 5·5 pounds lime per ton added. The tailing was cyanided with a 2·0 pound cyanide solution together with 5·0 pounds lime.

Flotation Results

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Heads	....	1·92	0·82		
Flotation concentrate	4·2	28·66	16·18	95·4	92·2
" tailing	95·8	0·06	0·06	4·6	7·8

Cyanidation of Flotation Concentrate

Product	Assay, oz./ton		Extraction, %	
	Au	Ag	Au	Ag
Heads	28·66	16·18		
Tailing	6·88	7·20	75·9	55·5

Cyanidation of Flotation Tailing

Product	Assay, oz./ton		Extraction, %	
	Au	Ag	Au	Ag
Heads	0·06	0·06	91·7	75·0
Tails	0·005	0·015		

This test shows that a high recovery of the gold and silver in the ore can be secured by flotation, 95·4 per cent of the gold and 92·2 per cent of the silver in a concentrate assaying 28·66 ounces and 16·18 ounces respectively.

However, on attempting to recover the values contained in this product by cyanidation, a low percentage of extraction is recorded, 75·9 per cent of the gold and 55·5 per cent of the silver.

The cyanidation of the flotation tailing is quite worthy of notice. Although the value is low, $1.20 per ton, no difficulty is experienced in reducing this to 10 cents, or a recovery of 91·7 per cent of the gold contained in this product.

Test No. 6

From observations made in previous tests, especially those employing straight cyanidation, it was suspected that the carbon present in the ore was causing trouble. This seems to be substantiated by the results secured in Test No. 5, in which the flotation tailing yields to cyanidation quite readily, while the concentrate containing the carbon does not.

To investigate this point, the ore was ground with water, 1 : 1 dilution, to pass approximately 80 per cent through 200 mesh. Kerosene at the rate of 0·5 pound per ton of ore was added and ground with the ore, together with soda ash equivalent to 3·0 pounds per ton. After grinding the pulp was floated with 0·04 pound pine oil per ton.

The froth secured was of a black oily nature, very small in bulk and carrying visible sulphides.

After removal of this graphitic concentrate, the tailing was dewatered and cyanided, 1 : 2 dilution, with 2·0 pounds cyanide solution for 48 hours. Lime was added at the rate of 5 pounds per ton of ore.

Flotation Results

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Heads		1·92	0·82		
Flotation concentrate	1·1	31·14	36·86	25·5	46·1
" tailing	98·9	1·02	0·48	74·5	53·9

Cyanidation of Flotation Tailing

—	Assay, oz./ton		Extraction, %	
	Au	Ag	Au	Ag
Heads	1·02	0·48		
Cyanide tailing	0·015	0·025	98·5	94·8

Cyanide consumption, 0·60 pound per ton of ore.

Recoveries on Flotation Feed

—	Gold, %	Silver, %
Recovery by flotation	25·5	46·1
" cyanidation, calculated	73·4	51·1
Total recovery	98·9	97·2

The results secured on cyaniding the tailing after removal of the graphite indicate that the presence of this material is harmful to the cyanide process. After this carbon has been removed, the flotation tailing having a gold value of $20.40 is reduced to 30 cents, representing a recovery of 98·5 per cent of the gold in it.

The graphite concentrate, while representing 1·1 per cent of the weight of the ore, is of very high value, 31·14 ounces gold and 36·86 ounces silver per ton. This product contains 25·5 per cent of the gold in the mill feed.

A total recovery by flotation and cyanidation of 98·9 per cent of the gold and 97·2 per cent of the silver is recorded.

An analysis of the graphite concentrate shows it to contain 20·5 per cent lead and 5·5 per cent carbon.

Test No. 7

As the concentrate secured in the preceding test carried very high gold values, an attempt was made in this test to produce a low-grade product containing all the graphite and to leave the gold in the flotation tailing for subsequent recovery by cyanidation.

The ore was ground to 86 per cent −200 mesh in the presence of lime instead of soda ash to depress the gold and sulphides. The graphite was floated with 1 pound kerosene and 0·04 pound pine oil per ton of ore.

The tailing was cyanided for 48 hours with a 2·0 pound per ton cyanide solution. Lime for protective alkalinity was added at the rate of 5·0 pounds per ton of ore.

Flotation Results

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Heads		1·92	0·82		
Flotation concentrate	1·2	22·56	25·20	21·2	37·1
" tailing	98·8	1·02	0·52	78·8	62·9

Cyanidation of Flotation Tailing

Product	Assay, oz./ton		Extraction, %		Reagent consumption, lb./ton	
	Au	Ag	Au	Ag	KCN	CaO
Heads	1·02	0·52				
24-hour tailing	0·05	0·06	95·1	88·5	0·80	4·0
48-hour tailing	0·01	0·07	99·0	86·5	0·80	4·0

Recovery of Flotation Feed

—	Gold, %	Silver, %
Recovery in flotation concentrate	21·2	37·1
" " tailing	78·0	54·4
Total recovery	99·2	91·5

The flotation concentrate in this test employing lime in the grinding mill, results in a lower-grade concentrate, 22·56 ounces as compared with 31·14 ounces secured when grinding with soda ash. A slightly higher total recovery is also noted.

Test No. 8

In this test a further attempt was made to keep the gold out of the concentrate. Water-glass at the rate of 0·5 pound per ton of ore was added to the pulp after grinding with lime and kerosene. The froth secured was brittle and apparently carried little values. The flotation tailing was cyanided under the same conditions as Test No. 7.

Flotation Results

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Flotation concentrate	1·4	1·48	10·42	1·4	14·5
" tailing	98·6	1·46	0·87	98·6	85·5

Cyanidation of Flotation Tailing

Product	Assay, oz./ton		Extraction, %		Reagent consumption, lb./ton	
	Au	Ag	Au	Ag	KCN	CaO
Heads........................	1·46	0·07				
24-hour tailing..................	0·52	0·34	64·4	60·9	1·0	3·0
48-hour tailing..................	0·62	0·32	57·5	63·2	1·2	3·0

While the use of water-glass in the flotation circuit produces no concentration of values in the froth taken off, the carbonaceous material is apparently not completely removed from the tailing. This is shown by the results secured on cyaniding the flotation tailing. A poor recovery is obtained with a higher tailing loss after 48 hours' agitation than after 24 hours. This points to precipitation of the dissolved gold by carbon remaining in the tailing.

Test No. 9

In this test the effect of grinding with lime and kerosene in cyanide solution, and floating the graphite prior to agitation was studied.

The ore was ground to 80 per cent −200 mesh, 1 : 1 dilution with a 2·0 pound per ton cyanide solution. Lime at the rate of 5 pounds, kerosene at 1 pound per ton of ore, were added while grinding. After grinding the pulp was floated with 0·04 pound pine oil per ton. The concentrate secured did not appear to be high in carbon and was of a dark grey colour.

The flotation tailing was dewatered and cyanided in 1 : 2 dilution with the same solution with which grinding and flotation were conducted. Lime equal to 2·5 pounds per ton of ore was added, the solution brought up in strength to 2·0 pounds cyanide per ton, and agitation continued for 48 hours.

Flotation Results

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Heads....................................	....	1·92	0·82		
Flotation concentrate......................	1·59	0·84	3·06	1·1	8·2
" tailing..........................	98·41	1·12	0·55	98·9	91·8

Cyanidation of Flotation Tailing

Product	Assay, oz./ton		Extraction, %		Reagent consumption, lb./ton	
	Au	Ag	Au	Ag	KCN	CaO
Heads..........................	1·12	0·55				
24-hour tailing..................	0·44	0·22	60·7	60·0	0·50	3·3
48-hour tailing..................	1·02	0·30	9·0	45·4	0·50	3·5

It appears that floating the graphite in cyanide solution is not successful. A low-grade concentrate is secured but the carbon apparently remains in the tailing resulting in a low recovery of the gold on cyaniding. Precipitation of dissolved gold takes place, as an extraction of 60·7 per cent is secured with 24 hours and only 9·0 per cent in 48 hours.

AMALGAMATION

Test No. 10

A sample, 100 grammes, of the ore was ground to pass 65 mesh, and amalgamated with mercury.

Heads	Au, 1·92 oz./ton
Amalgamation tailing	0·34 "
Recovery	82·3 per cent

This test indicates that a large part of the gold is free and can be recovered by amalgamation.

AMALGAMATION AND CONCENTRATION

Test No. 11

As it appears that a very considerable portion of the gold can be amalgamated, the results of concentrating the tailing after removal of free gold was investigated.

A 1,000-gramme sample of the ore was ground with an equal weight of water in a porcelain mill containing iron balls. Soda ash at the rate of 4 pounds per ton of ore was added. Grinding was such that 70 per cent passed 200 mesh. After amalgamating and removing the amalgam the pulp was floated using the original alkaline solution in which it was ground, to which was added an additional 2 pounds soda ash per ton; 0·20 pound potassium xanthate and 0·06 pound pine oil per ton were used as flotation reagents.

Amalgamation and Flotation Results

Product	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
Heads	....	1·92	0·82		
Recovery by amalgamation	....	....	....	90·0	27·3
Amalgamation tailing	....	0·19	0·53	10·0	72·7
Flotation concentrate	3·36	4·48	14·52	7·8	59·5
" middling	4·54	0·45	1·59	1·1	8·8
" tailing	92·10	0·022	0·039	1·1	4·4

Screen Analysis of Tailing

Product	Weight, %	Assay, oz./ton	
		Au	Ag
+100	5·8	0·03	0·04
−100+150	7·8	0·03	0·05
−150+200	16·1	0·025	0·03
−200	70·3	0·02	0·04

This test shows that 90 per cent of the gold is recovered by amalgamation and an additional 8·9 per cent by flotation, a total of 98·9 per cent by the two processes.

Test No. 12

The conditions established in Test No. 11, where a total recovery of 98·9 per cent of the gold was obtained by amalgamation followed by flotation, were checked on a larger scale.

The ore was fed at the rate of 100 pounds per hour to a 12- by 24-inch trunnion discharge rod mill. The mill discharge at approximately 50 per cent solids was allowed to flow through a slowly revolving agitator containing mercury and thence over amalgamating plates. The pulp after passing these plates was elevated to an Aikens classifier (12 inches by 5 feet 6 inches) discharging its oversize to the rod mill for regrinding. The overflow, approximately 70 per cent −200 mesh, together with 0·20 pound potassium xanthate per ton, flowed to a 6-cell, Denver Sub-A flotation machine. This was so arranged that the feed entered at the second cell. The rougher concentrate taken off cells Nos. 2 and 3 flowed to cell No 1 for cleaning, producing a finished product. The middling from this cleaner cell flowed to cell No. 2. The concentrate secured from cells Nos. 4, 5, and 6, acting as scavenger cells, flowed by gravity to cell No. 2 thus uniting with the stream of fresh pulp and the middling from the cleaner cell. Pine oil at the rate of 0·20 pound per ton of ore was fed to the first scavenger cell or cell No. 4. Adding the frothing reagent at this point was found to yield the best flotation conditions. If fed directly to the rougher cells too great a volume of low-grade froth was produced.

Amalgamation and Flotation Results

Product	Weight, %	Assay, oz./ton		Recovery, %	
		Au	Ag	Au	Ag
Heads		1·92	0·82		
Recovery by amalgamation				64·1	37·8
Amalgamation tailing		0·69	0·51	35·9	62·2
Flotation concentrate	6·18	10·78	16·62	34·7	55·1
" tailing	93·82	0·025	0·06	1·2	7·1

Reagents used:

Soda ash	6·3 lb./ton
Potassium xanthate	0·19 "
Pine oil	0·20 "

During this test trouble was encountered in keeping the amalgamating plates clean. The graphite in the ore coated the amalgam with a greasy black film which prevented the gold from adhering to the plates. This is shown in the recovery secured, 64·1 per cent of the gold as against 90 per cent in the small-scale test (No. 11), where the ore was amalgamated in a rotating porcelain jar.

The gold not recovered by amalgamation, however, is recovered in the flotation concentrate which assays 10·78 ounces gold and 16·62 ounces silver per ton. A total recovery, by the combined methods, of 98·8 per cent of the gold and 92·9 per cent of the silver is obtained.

CONCLUSIONS

Carbon in the ore undoubtedly is responsible for the low recoveries secured by cyanidation. When this is removed prior to cyanidation no difficulty is experienced in recovering the gold remaining in the flotation tailing.

Flotation alone, yields a high recovery of the precious metals. The concentrate, however, is of very high grade and does not yield to cyanidation very readily. While this high-grade concentrate could be shipped to a smelter, trouble would be experienced in obtaining check assays.

The ideal process would be to produce a concentrate containing all the graphite and no gold, leaving the values in the tailing for subsequent recovery by cyanidation. However, this condition is not realized. Grinding and floating in a circuit alkaline with lime gives a graphitic concentrate assaying 22·56 ounces gold and 25·2 ounces silver per ton. Soda ash used instead of lime gives a still higher grade product. The bulk of concentrate recovered is the same in both cases, 1·1 per cent of the weight of the feed.

The gold in the flotation tailing is extracted by cyanide with ease.

Tests show that over 80 per cent recovery can be made by amalgamation. When followed by concentration a total recovery of 98·9 per cent of the gold is obtained as shown in Test No. 11.

However, when the ore is treated continuously employing rod-mill grinding followed by amalgamation, the carbon in the ore interferes seriously, coating the plates with a graphite scum which impairs their efficiency. In continuous practice constant attention to keep the plates clean would not be practical. In the small-scale tests, where the ore after grinding was rotated in a jar with mercury, the abrasive action produced doubtless kept the amalgam clean, thus securing a high recovery.

As the carbon present interferes with both cyanidation of the raw ore and amalgamation, it appears from the above tests that the only practical method to employ is that shown in Test No. 7, namely, concentration by flotation and cyanidation of the flotation tailing.

Applying this method the ore is ground in water together with lime sufficient to produce an alkaline pulp. Kerosene is also added to the grinding circuit at the rate of from ½ to 1 pound per ton of ore. Pine oil sufficient to froth is added to the flotation cells and the graphite removed. After dewatering, the flotation tailing is repulped and agitated with cyanide solution for approximately 48 hours, when a recovery of 99 per cent of the gold is secured, or a total of 98·9 per cent by the two processes.

The graphite concentrate 1·2 per cent of the weight of the ore fed, containing 21·2 per cent of the total gold, is of quite high value—22·56 ounces per ton or $4.51. This product may be shipped to a smelter. As the bulk of this material secured in the tests was too small for experimental work no research was conducted on it. It would seem, however, that by amalgamating in grinding pans, or amalgamating barrels, much of the contained gold would be recovered.

Report No. 331

CONCENTRATION OF COPPER ORE FROM ARNO MINES, LTD., COXHEATH, NOVA SCOTIA

G. B. O'Malley

Shipment. A shipment of 380 pounds of ore was received on May 21, 1929, from Arno Mines, Ltd., Coxheath, N.S. The shipment consisted of two lots, No. 1, 250 pounds of newly broken ore, No. 2, 130 pounds of dumped material from the preliminary mine workings.

Characteristics of the Ore. The ore consists of massive chalcopyrite with pyrite in a siliceous gangue. There are small gold and silver values. Sample No. 2 contains a certain amount of partly oxidized sulphides.

Analyses of Samples.

	No. 1	No. 2
Copper	4·89 per cent	5·54 per cent
Gold	0·01 oz./ton	0·03 oz./ton
Silver	0·72 "	0·52 "
Iron	8·63 per cent	6·16 per cent
Sulphur	8·62 "	
Insolubles	77·98 "	61·92 "

EXPERIMENTAL TESTS

Batch flotation tests on 2,000-gramme samples gave results as follows:

Sample No. 1

Test No.	Product	Weight, %	Assay			Per cent of values		
			Cu, %	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
1	Concentrate..	18·2	26·40	0·04	3·44	94·4	81·0	72·8
	Tailing.......	81·8	0·35	trace	0·18	5·6	19·0	27·2

Reagents:

Soda ash, 5·0; sodium cyanide, 0·15; thiocarbanilide, 0·1 lb./ton to ball mill. Pine oil, 0·06 lb./ton to cell.

Test No.	Product	Weight, %	Assay			Per cent of values		
			Cu, %	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
2	Concentrate..	28·0	17·05	0·02	2·86	98·8	56·0	96·2
	Tailing.......	72·0	0·08	trace	0·04	1·2	44·0	3·8
	+ 65 mesh.	0·8	0·22	"	0·09	2·2	....	1·6
	+100 " ..	1·6	0·14	"	0·06	2·8	....	2·1
	+150 " ..	1·3	0·16	"	0·10	2·6	....	2·9
	+200 " ..	7·1	0·15	"	0·09	13·5	....	14·2
	−200 " ..	89·2	0·07	"	0·04	78·8	....	79·2

Reagents:

Lime, 5·0; thiocarbanilide, 0·1 lb./ton to ball mill. Potassium xanthate, 0·1; pine oil, 0·06 lb./ton to cell.

Sample No. 1

Test No.	Product	Weight, %	Assay			Per cent of values		
			Cu, %	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
3	Concentrate..	16·6	27·95	....	....	96·2		
	Middling.....	7·9	1·79	....	....	2·9		
	Tailing.......	75·5	0·06	....	....	0·9		

Reagents:

Lime, 5·0 lb./ton to ball mill. Potassium xanthate, 0·2; pine oil, 0·06 lb./ton to cell.

Test No.	Product	Weight, %	Assay			Per cent of values		
			Cu, %	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
4	Concentrate..	20·3	23·25	....	....	98·7		
	Tailing.......	79·7	0·08	....	....	1·3		

Reagents:

Lime, 5·0 lb./ton to ball mill. Barnac, 0·18; frothing agent B.F. No. 1, 0·12 lb./ton to cell.

Sample No. 2

Test No.	Product	Weight, %	Assay			Per cent of values		
			Cu, %	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
5	Concentrate..	18·1	29·62	....	....	95·8		
	Middling.....	4·3	2·19	....	....	1·7		
	Tailing.......	77·6	0·18	....	....	2·5		

Reagents:

Lime, 8·0 lb./ton to ball mill. Amyl xanthate, 0·1; pine oil, 0·06 lb./ton to cell.

SUMMARY

Owing to the simplicity of the ore and the small amount of pyrite, there is no difficulty in obtaining excellent extraction of the copper content in a high-grade concentrate, whilst the recoveries of the small silver and gold values are sufficiently good. The somewhat higher recovery of gold by means of the soda ash circuit is not enough to justify the greater reagent cost as compared with the lime circuit using potassium xanthate and pine oil.

The test in which Barnac was used indicates that this reagent gives good results with this type of ore.

The test on the oxidized ore shows that sulphidizing is not necessary, and the treatment of the dumped material need be no different in essentials from that of the newly mined ore.

Report No. 332

THE RECOVERY OF GOLD FROM THE ORE OF THE ST. ANTHONY GOLD MINES, LTD., SAVANT LAKE, ONT.

A. K. Anderson

Shipment. Twenty-nine bags of ore, gross weight 2,280 pounds, were received by freight at the Laboratories on June 11, 1929, shipped from the St. Anthony Gold Mines, Ltd., Savant Lake, Ontario.

Characteristics of the Ore. The shipment consisted of dense white quartz carrying less than 1 per cent iron pyrite, a quartz porphyry containing fine flakes of white mica. Both porphyry and quartz were cut by sericite stringers. A dark-coloured fine-grained ferruginous carbonate schist was also included in the shipment.

Purpose of Experimental Tests. This shipment was made for the purpose of determining the conditions yielding the highest recovery of the gold by amalgamation.

Sampling and Analysis. The lot was crushed through jaw crushers and rolls to pass a ¼-inch screen and a tenth part cut out by an automatic sampler. This was then crushed to pass 8 mesh and cut through a Jones riffle sampler. By grinding to pass still smaller meshes, with accompanying passes through the sampler, a representative portion, – 100 mesh, was obtained for assay. This sample showed the shipment to contain 0·30 ounce gold and 0·22 ounce silver per ton.

EXPERIMENTAL TESTS

Two series of tests were undertaken, the first, amalgamation, and the second, cyanidation.

The first included amalgamation of material ground to pass different meshes. This procedure was followed in order to determine at what mesh the gold was freed from the gangue.

The second series, that employing cyanidation for the extraction of the gold, consisted of tests run on different sizes of material.

AMALGAMATION

Representative 1,000-gramme portions of the ore were ground dry to pass 28, 35, and 48 mesh, rotated for one hour in a porcelain mill with an equal weight of water and 100 grammes of mercury. After removal of the amalgam, the residue was filtered, dried, and assayed.

Head assay: Au, 0·30 oz./ton; Ag, 0·22 oz./ton.

Test No.	Mesh size	Tailing assay, oz./ton		Recovery, %	
		Au	Ag	Au	Ag
1	−28	0·134	0·118	55·6	46·4
2	−35	0·092	0·122	69·3	44·6
3	−48	0·02	0·12	90·3	45·5

Screen Analysis of Test No. 2 Tailing, —35-mesh Grinding

Mesh size	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
+ 48	25·18	0·13	0·07	35·4	14·5
− 48+ 65	25·16	0·10	0·08	27·2	16·5
− 65+100	14·09	0·11	0·11	16·8	12·7
−100+150	6·36	0·06	0·16	4·1	8·4
−150+200	6·14	0·06	0·16	4·0	8·1
−200	23·07	0·05	0·21	12·5	39·8

The above tests show that the size of screen used is important. Using 28 or 35 mesh results in low recoveries. A great difference in recovery is apparent when the material is ground to 48 mesh, 90·3 per cent of the gold being obtained as against 69·3 per cent when a 35-mesh screen is used.

CYANIDATION

To determine what results would be obtained by treatment with cyanide solution, representative portions of the ore were ground to pass 48, 100, and 150 mesh. These were then agitated for 48 hours in a 1 : 2 dilution with a 2·0 pound per ton cyanide solution. After the agitation period, the residues were washed, dried, and assayed.

Head assay: Au, 0·30 oz./ton; Ag, 0·22 oz./ton.

Test No.	Mesh size	Tailing assay, oz./ton		Extraction, %	
		Au	Ag	Au	Ag
4	− 48	0·01	0·05	90·7	77·3
5	−100	0·016	0·05	90·6	77·3
6	−150	0·01	0·03	90·7	86·4

Screen Analysis of Test No. 4, —48-mesh Grinding

Mesh size	Weight, %	Assay, oz./ton		Per cent of values	
		Au	Ag	Au	Ag
− 48+ 65	4·2	0·035	0·07	13·3	5·5
− 65+100	22·2	0·01	0·06	20·1	25·1
−100+150	13·7	0·01	0·05	12·4	12·9
−150+200	17·6	0·01	0·05	15·9	16·6
−200	42·3	0·01	0·05	38·3	39·9

In these tests, assays made on samples taken after 24 hours showed the same gold content as those after 48 hours.

The results secured indicate that fine grinding is not essential for highest recovery. Material passing 65 mesh, as indicated by the screen analysis of the tailing from Test No. 4, is as low in value as the −200-mesh portion. In this test, 4·2 per cent of the weight remains on 65 mesh and has an assay value of 0·035 ounce gold per ton. This would indicate that for maximum results grinding should be at least −48 mesh with a low percentage remaining on the next finer screen.

CONCLUSIONS

Tests Nos. 1, 2, and 3 show that to obtain a maximum recovery of gold by amalgamation, the ore should be crushed to pass 48 mesh. As this size of screen on stamp batteries will result in a low stamp duty, the next larger size, 35 mesh, will in all probability prove to be the most satisfactory to use. Milling operations may show that stamping at this size will produce a product containing more fines than indicated in Test No. 2 where 25 per cent of the material remained on 48 mesh.

The tests in which cyanidation was investigated show the ore to be readily amenable to this treatment. Maximum results are secured on material crushed to pass 48 mesh with an extraction practically the same as that secured by amalgamation, 90·7 per cent as against 90·3 per cent.

The indicated treatment for the ore is crushing in stamp batteries to 48 mesh. Higher recoveries will be secured at this mesh, but the increase in stamp duty obtained by substituting 35-mesh screens may yield more commercial results. Inside and plate amalgamation will recover as high a percentage of the gold as can be obtained by an all-slime cyanide installation.

Report No. 333

THE RECOVERY OF GOLD FROM THE ORE OF THE MALARTIC MINES, LTD., AMOS, QUE.

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

Shipments. Two lots of ore were received by freight at the Ore Dressing Laboratories. The first, called Class 2 ore, contained in 35 bags, gross weight 5,470 pounds, arrived February 16, 1929. The second shipment, Class 1 ore, a 65-bag lot, gross weight 8,720 pounds, reached the Laboratories on March 8, 1929.

Characteristics of the Ore. The ore comprising these shipments was of a siliceous nature, containing finely disseminated iron pyrite. No copper or other sulphides were visible.

Purpose of Experimental Tests. The shipments were made to determine by test work the best process to apply for the recovery of the values contained in the ore.

Sampling and Analysis. Both lots were crushed in jaw crushers and rolls to ½ inch, and a tenth part cut out by an automatic sampler. This sample was then crushed to pass 10 mesh, cut through a Jones riffle sampler and again stage-ground to pass increasing finer meshes, with accompanying passes through the sampler until a representative portion, −100 mesh, was secured. This showed the ore to contain:—

Class 2 ore	Au	0·29 oz./ton
Class 1 ore	Au	0·23 "

EXPERIMENTAL TESTS

As both shipments are of the same nature all test work was done on the first shipment called Class 2 ore.

The investigation included tests made by flotation, flotation preceded by amalgamation, and by cyanidation. The cyanide tests were carried out on material ground to varying degrees of fineness to determine the coarsest crushing necessary to free the gold from the gangue.

FLOTATION

Test No. 1

In this test, the result obtained by grinding the ore to pass 95 per cent through 200 mesh and concentrating the values by flotation was studied.

A 1,000-gramme sample of ore, with an equal weight of water was ground in a porcelain mill containing iron balls. Soda ash at the rate of 7 pounds per ton of ore was added to the mill.

The pulp was then floated in a laboratory-size Ruth machine with 0·20 pound amyl xanthate and 0·04 pound pine oil per ton of ore.

Product	Weight, %	Assay, Au, oz./ton	Per cent of values
Heads	....	0·29	
Flotation concentrate	5·56	2·88	55·2
" middling	5·62	1·15	22·3
" tailing	88·82	0·07	22·5

A screen analysis of the tailing shows the following:—

Mesh size	Weight, %	Assay, Au, oz./ton
+150	0·9	1·06
−150+200	3·7	0·73
−200	95·5	0·04

The great difference between the −200-mesh portion and the coarser sizes points to the presence of free gold. This, doubtless, is the cause of the low recovery by flotation, a recovery of 77·5 per cent in the concentrate and middling being secured.

AMALGAMATION AND FLOTATION

Test No. 2

The difference between this test and Test No. 1 is that, after grinding under the same conditions, the pulp was amalgamated with mercury prior to flotation. Reagents were the same in each case.

Product	Weight, %	Assay, Au, oz./ton	Per cent of values
Heads	....	0·29	
Amalgamation tailing	....	0·10	67·2
Flotation concentrate	4·27	1·08	15·9
" middling	4·16	0·38	5·4
" tailing	91·57	0·036	11·5

Screen Analysis of Flotation Tailing

Mesh size	Weight, %	Assay, Au, oz./ton
+100	0·6	0·06
−100+150	2·1	0·05
−150+200	5·4	0·05
−200	91·9	0·035

This test shows that removing the free gold prior to concentration is of benefit; 67·2 per cent being recovered by amalgamation, and a further saving of 21·3 per cent being obtained by flotation, or a total of 88·5 per cent by the combined processes. The screen analysis of the tailing shows much more uniform conditions than those of Test No. 1.

Test No. 3

Conditions established in Test No. 2 were checked on a larger scale. The ore was fed at the rate of 100 pounds per hour to a 12- by 24-inch trunnion discharge rod mill. The mill discharge, at approximately 60 per cent solids, passed through a slowly revolving agitator containing mercury and then over amalgamating plates. Grinding was such that 9 per cent remained on 100 mesh and 68 per cent passed 200 mesh.

After amalgamation the pulp was elevated to a 12-inch by 5-foot 6-inch Aikens classifier, discharging the oversize to the rod mill for regrinding. The overflow passed to a 6-cell, Denver Sub-A flotation machine so arranged that the feed entered the second cell. This cell and the following one, No. 3, produced a rougher concentrate which was pumped to cell No. 1, giving a finished concentrate.

The middling from this flowed to cell No. 1. The last three cells, Nos. 4, 5, and 6, were used as scavenger cells, the concentrate given off flowing to cell No. 2 where it united with the flotation feed and middling from the cleaner cell.

Soda ash at the rate of 5 pounds per ton of ore was added to the grinding mill. Amyl xanthate, 0·15 pound, and pine oil, 0·14 pound per ton, were added to the flotation circuit. The run was continuous over six hours. Two samples of the products were taken, the first four hours represented by Sample No. 1 and the last two by Sample No. 2.

Sample No. 1

Product	Weight, %	Assay, Au, oz./ton	Per cent of values
Heads	....	0·29	
Amalgamation tailing	....	0·09	69·0
Flotation concentrate	5·13	1·20	21·2
" tailing	94·87	0·03	9·8

Sample No. 2

Product	Weight, %	Assay, Au, oz./ton	Per cent of values
Heads	....	0·29	
Amalgamation tailing	....	0·09	69·0
Flotation concentrate	7·23	0·86	21·4
" tailing	92·77	0·03	9·6

This test run checks the results secured in small-scale Test No. 2, a slightly higher recovery of gold being obtained, 90·4 per cent as against 88·5 per cent.

CYANIDATION

Test No. 4

The effect of cyaniding the ore at different finenesses of grinding was studied. Representative portions of the ore were ground dry to pass 28, 35, 48, 65, 100, and 150 mesh. Each lot was then cyanided for 48 hours, 1 : 2 dilution with a 2·0 pound cyanide solution. Lime for protective alkalinity was added at the rate of 6 pounds per ton of ore. Samples were taken at the end of 24- and 48-hour agitation.

Grinding—28 mesh

Sample	Titration, lb./ton				Consumption, lb./ton ore		Assay, Au, oz./ton	Extraction %
	Start		Finish					
	KCN	CaO	KCN	CaO	KCN	CaO		
24-hour tailing	2·0	....	1·9	1·1	0·20	3·6	0·06	79·3
48-hour tailing	2·0	....	1·9	1·0	0·20	4·0	0·05	82·8

Screen Analysis of 48-hour Tailing

Mesh	Weight, %	Assay, Au, oz./ton	Per cent of values
− 28+ 35	21·32	0·07	28·7
− 35+ 65	27·16	0·06	31·3
− 65+100	7·66	0·06	8·8
−100+150	5·07	0·06	5·8
−150+200	5·23	0·06	6·0
−200	33·56	0·03	19·4

At this mesh an extraction of 82·5 per cent of the gold is obtained within 48 hours. The tailing, however, is rather high, $1.00 remaining in the residues.

Grinding—35 mesh

Sample	Titration, lb./ton				Consumption, lb./ton ore		Assay, Au, oz./ton	Extraction, %
	Start		Finish					
	KCN	CaO	KCN	CaO	KCN	CaO		
24-hour tailing......	2·0	....	1·9	1·0	0·20	4·0	0·04	86·2
48-hour tailing......	2·0	....	1·9	1·0	0·20	4·0	0·05	82·8

Screen Analysis of 48-hour Tailing

Mesh	Weight, %	Assay, Au, oz./ton	Per cent of values
− 35+ 65	40·17	0·05	37·9
− 65+100	10·56	0·11	21·9
−100+150	4·65	0·06	5·3
−150+200	5·71	0·12	12·9
−200	38·91	0·03	22·0

Cyaniding at the size on which this test was carried out does not show much improvement over the results secured on the coarser sizing.

Grinding—48 mesh

Sample	Titration, lb./ton				Consumption, lb./ton ore		Assay, Au, ton./oz.	Extraction, %
	Start		Finish					
	KCN	CaO	KCN	CaO	KCN	CaO		
24-hour tailing......	2·0	....	1·9	0·85	0·20	4·3	0·03	89·7
48-hour tailing......	2·0	....	1·85	0·65	0·30	4·7	0·018	93·8

Screen Analysis of 48-hour Tailing

Mesh	Weight, %	Assay, Au, oz./ton	Per cent of values
− 48+ 65	5·4	0·03	9·0
− 65+100	16·3	0·03	27·1
−100+150	8·3	0·03	13·8
−150+200	10·1	0·03	16·8
−200	59·9	0·01	33·3

This shows that with increasing fineness of grinding the extraction improves. Within 48 hours, 93·8 per cent of the gold is in solution. This increase is apparently due to the increase in the −200-mesh portion, as shown in the screen analysis of the tailing.

Grinding —65 mesh

Sample	Titration, lb./ton				Consumption, lb./ton ore		Assay, Au, oz./ton	Extraction, %
	Start		Finish					
	KCN	CaO	KCN	CaO	KCN	CaO		
24-hour tailing......	2·0	....	1·90	0·85	0·20	4·3	0·03	89·7
48-hour tailing......	2·0	....	1·80	0·85	0·40	4·3	0·019	93·4

Screen Analysis of 48-hour Tailing

Mesh	Weight, %	Assay, Au, oz./ton	Per cent of values
− 65+150..	16·6	0·03	26·6
−150+200..	12·2	0·025	16·3
−200..	71·2	0·015	57·1

Grinding so that the ore passes 65 mesh instead of 48 mesh does not seem to be of any benefit.

Grinding—100 mesh

Sample	Titration, lb./ton				Consumption, lb./ton ore		Assay, Au, oz./ton	Extraction, %
	Start		Finish					
	KCN	CaO	KCN	CaO	KCN	CaO		
24-hour tailing......	2·0	....	1·85	0·90	0·30	4·2	0·02	93·1
48-hour tailing......	2·0	....	1·80	0·85	0·40	4·3	0·013	95·5

Screen Analysis of 48-hour Tailing

Mesh	Weight, %	Assay, Au, oz./ton	Per cent of values
−100+200..	16·6	0·03	37·4
−200..	83·4	0·01	62·6

Grinding to pass 100 mesh which yields a large percentage of −200-mesh material shows a slightly higher extraction than the coarser material.

Grinding —150 mesh

Sample	Titration, lb./ton				Consumption, lb./ton ore		Assay, Au, oz./ton	Extraction, %
	Start		Finish					
	KCN	CaO	KCN	CaO	KCN	CaO		
24-hour tailing......	2·0	....	1·80	0·65	0·40	4·7	0·02	93·1
48-hour tailing......	2·0	....	1·75	0·65	0·50	4·7	0·018	93·8

Screen Analysis of 48-hour Tailing

Mesh	Weight, %	Assay, Au, oz./ton	Per cent of values
−150+200	9·6	0·045	24·1
−200	90·4	0·015	75·9

Very fine grinding, 90 per cent −200 mesh, does not show higher recoveries than that secured by cyaniding −48-mesh material.

As seen from the preceding tests, the ore is amenable to cyanidation. A larger scale test was undertaken which would more closely parallel conditions met in continuous operation than those under which the ore was treated in batches. As these small-scale tests show that the most economical conditions are those grinding to pass 48 mesh, an attempt was made to grind the ore to this size.

Ore crushed to pass 10 mesh together with a 1·0 pound cyanide solution was fed at the rate of 27½ pounds per hour to a small rod mill in closed circuit with a drag classifier. The rod load and solution were adjusted to what gave on a previous siliceous ore 1·9 per cent +48 mesh with 49·5 per cent passing 200 mesh. However, due to the nature of the ore, much finer grinding was secured, 77·5 per cent passing 200 mesh.

The classifier overflow passed to an agitator fitted with a cone bottom. The pulp was drawn through a bottom outlet by a centrifugal pump and returned to the top of the tank. This ensured positive agitation of the coarse particles. After 6 hours' operation the agitator was filled containing approximately 165 pounds of ore with a ratio of solution to ore of 1·9: 1. Agitation was continued for 46 hours with additions of cyanide to maintain a solution strength of 1·0 pound per ton. Lime was added from time to time as indicated by titrations to maintain approximately 0·5 pound lime per ton of solution.

After agitation, the pulp was passed through a filter press, washed, and sampled.

Sample No. 1—Taken when Agitator was Full

Product	Assay, Au, oz./ton	Extraction, %
Heads	0·29	
Tailing	0·051	82·4

This shows that during the grinding and agitation period while filling the tank a large part of the gold is dissolved.

Sample No. 2—After 24 Hours' Agitation

Product	Assay, Au, oz./ton	Extraction, %
Heads	0·29	
Tailing	0·051	82·4

Screen Analysis of Tailing

Mesh	Weight, %	Assay, Au, oz./ton	Per cent of values
+100	2·0	0·07	2·7
−100+200	6·8	0·06	8·0
−200	91·2	0·05	89·3

Sample No. 3—After 46 Hours' Agitation

Product	Assay, Au, oz./ton	Extraction, %
Heads	0·29	
Tailing	0·011	96·2

Screen Analysis of Final Tailing

—	Weight, %	Assay, Au, oz./ton	Per cent of values
+100	5·0	0·02	8·8
−100+150	4·7	0·015	6·1
−150+200	12·8	0·015	16·7
−200	77·5	0·010	68·4

It is seen from the above test that the maximum extraction is obtained with 46 hours with a high percentage of gold dissolved in the grinding circuit. The conditions and results of the run and the small test on ore ground to pass 100 mesh are alike. The small one has somewhat finer grinding, with a slightly lower recovery, 83 per cent −200 mesh as against 77·5 per cent and recoveries of 95·5 per cent and 96·2 per cent respectively.

The small test on −48-mesh material shows a recovery of 93·8 per cent as against 95·5 per cent on −100-mesh grinding. It is reasonable, therefore, to assume that in practice this would be improved as indicated in the results of the large and small tests on −100-mesh material.

Flotation alone yields a recovery of 77·5 per cent of the gold. When preceded by amalgamation a 90·4 per cent recovery is made but with the disadvantage that 21·4 per cent is in the pyritic concentrate. This would entail further treatment for the recovery of the gold.

From the above, it appears that the most economical system to employ is that incorporating coarse grinding to approximately 48 mesh followed by a suitable agitation system to keep the coarse particles of ore in suspension. A cyanide loss of approximately ½ pound per ton of ore milled can be expected.

The ore does not present any unexpected difficulties nor does it contain deleterious minerals likely to interfere with cyanidation. It grinds more easily than a hard quartz ore. This factor is worthy of attention as much can be gained by data secured prior to the installation of grinding mills.

Report No. 334

AMALGAMATION AND FLOTATION TESTS ON AN ORE FROM THE GOUDREAU-LOCALSH AREA, ONTARIO

G. B. O'Malley

Shipment. A shipment of 22 pounds of ore was received on July 29, 1929, from R. A. Hedley, Sault Ste. Marie, Ont. The ore was from the Goudreau-Localsh area.

Characteristics of the Ore. The sample consists of pyrite carrying gold and silver, in a gangue of quartz and schistose material.

Sampling and Analysis. The whole shipment was crushed to −14 mesh and a head sample cut. It assayed as follows:—

Au	15·52 oz./ton
Ag	1·91 "
Fe	5·87 per cent

Purpose of Tests. Tests by amalgamation and flotation were requested. In particular it was desired to know the ultimate possible extraction by amalgamation.

AMALGAMATION TESTS

Two tests were made in exactly similar manner. The ore was dry-ground to −100 mesh and amalgamated during grinding in a pebble mill, the charge to which was:—

Ore at −100 mesh	1,000 grammes
Water	1,000 c.c.
Mercury	100 grammes
Pebbles	

The period of grinding and amalgamation was 1 hour.

Results:

Test No. 1—Screen Analysis of Tailing

Product	Weight, %	Au, oz./ton	Ag, oz./ton	Per cent of values	
				Au	Ag
+150	0·3	1·17	Nil	9·5	
+200	8·6				
−200	91·1	1·09	0·13	90·5	100·0
Average tailing		1·10	0·12		

Recovery:

Au	92·9 per cent
Ag	93·8 "

Test No. 2—Screen Analysis of Tailing

Product	Weight, %	Au, oz./ton	Ag, oz./ton	Per cent of values	
				Au	Ag
+150	0·4	5·59	0·62	15·4	15·4
+200	3·3				
−200	96·30	1·18	0·13	84·6	84·6
Average tailing		1·34	0·15		

Recovery:

Au	91·4 per cent
Ag	92·2 "

FLOTATION TESTS

Flotation tests on 1,000-gramme samples gave results as follows:—

Test No.	Product	Weight, grms.	Weight, %	Assay			Per cent of values		
				Au, oz./ton	Ag, oz./ton	Fe, %	Au	Ag	Fe
3	Head	....	....	15·52	1·91	5·87			
	Concentrate	110·0	10·9	116·00	8·66	32·54	86·1	82·2	59·3
	Tailing	896·5	89·1	2·29	0·23	2·73	13·9	17·8	40·
		1,006·5							

Reagents:	Water-gas tar	0·16 lb./ton added to cell.
	Amyl xanthate	0·3 " "
	Pine oil No. 5	0·1 " "

Test No.	Product	Weight, grms.	Weight, %	Au, oz./ton	Ag, oz./ton	Fe, %	Au	Ag	Fe
4	Head	....	....	15·52	1·91	5·87			
	Concentrate	108·4	10·9	113·26	9·08	32·54	71·6	66·5	54·7
	Tailing	888·5	89·1	5·51	0·56	3·30	28·4	33·5	45·3
		996·9							

Reagents:	Soda-ash	10·0 lb./ton added to ball mill.
	Water-gas tar	0·16 " "
	Amyl xanthate	0·3 " "
	Pine oil No. 5	0·1 " "

Test No.	Product	Weight, grms.	Weight, %	Au, oz./ton	Ag, oz./ton	Fe, %	Au	Ag	Fe
5	Head	....	....	15·52	1·91	5·87			
	Concentrate	118·0	11·8	113·80	8·00	30·39	94·2	86·3	57·2
	Tailing	879·2	88·2	0·94	0·17	3·04	5·8	13·7	42·8
		997·2							

Reagents:	Soda ash	4·0 lb./ton added to ball mill.
	Water-gas tar	0·16 lb./ton added to cell.
	Amyl xanthate	0·3 " "
	Pine oil No. 5	0·1 " "

SUMMARY

It will be seen that good recoveries of the gold and silver contents of the ore can be made by either amalgamation or flotation. The flotation concentrate in the small-scale test carried over 40 ounces gold per ton. The high-grade nature of the sample accounts for certain irregularities in the assays, and for the presence of 1 ounce gold per ton in the tailings.

Report No. 335

THE RECOVERY OF SILVER FROM THE TAILING OF THE CASTLE-TRETHEWEY MILL AT BESTEL, ONTARIO

C. S. Parsons and D. S. Halford

Shipments. Two shipments of tailing were received. Shipment No. 1 was received April 19, 1929, and Shipment No. 2 on July 9, 1929.

Characteristics and Analysis of Shipments. Both shipments consisted of reground sand tailing from the mill of the Castle-Trethewey Mines, Limited.

Shipment No. 1 was packed in carbide tins, which had not been thoroughly cleaned out. The carbide remaining in the tins interfered with the concentration tests by flotation, and a second shipment had to be requested in order to carry out the investigation.

Analysis:

Shipment No. 1	2,255 pounds Ag 2·92 oz./ton
Shipment No. 2	1,500 pounds Ag 4·50 oz./ton

Purpose of Experimental Tests. In the present flow-sheet of the mill these tailings are reground and concentrated on tables. A high-grade concentrate is obtained assaying over 500 ounces per ton in silver. Although the recovery obtained is fairly high there was still the possibility that it could be increased. These shipments were made in order to determine if the recovery could be increased, and a grade of concentrate produced which would yield a greater profit by using flotation to replace the tables.

EXPERIMENTAL TESTS

One small-scale continuous test and eleven small-scale batch tests were made. The continuous test was made with a feed rate of 100 pounds of tailings per hour.

A screen test was made on the head sample cut from the feed for the continuous test.

Mesh	Weight, %	Cumulative, %
− 14+ 35	0·1	0·1
− 35+ 48	1·0	1·1
− 48+ 65	7·5	8·6
− 65+100	24·8	33·4
−100+150	15·9	49·3
−150+200	21·8	71·1
−200	28·9	100·0
Totals	100·0	

Small-batch Flotation Tests

Test No.	Products	Weight, %	Assay, Ag, oz./ton	Recovery, %	Reagents
1	Concentrate No. 1	4·5	83·72	83·06	Ore 2,000 grms. First concentrate floated with mixture of coal tar 40 per cent and coal-tar creosote 60 per cent 1 lb./ton, and pine oil. Recleaned making concentrate No. 1 and middling. Refloated tailing with 0·10 lb./ton amyl xanthate making concentrate No. 2.
	Middling	3·56	7·28	5·71	
	Concentrate No. 2	1·06	23·12	5·41	
	Tailing	90·88	0·29	5·82	
	Totals	100·00		100·00	
2	Concentrate	2·73	144·12	89·84	Ore 2,000 grms. Floated with 0·10 lb./ton Barnac, 3 drops froth stabilizer, 3 drops of pine oil.
	Middling	2·95	8·38	5·64	
	Tailing	94·32	0·21	4·52	
	Totals	100·00		100·00	
3	Concentrate	2·29	182·14	91·47	Ore 2,000 grms. Floated with 0·20 lb./ton potassium xanthate, 4 drops of pine oil.
	Middling	2·30	4·86	2·46	
	Tailing	95·41	0·29	6·07	
	Totals	100·00		100·00	
4	Concentrate	2·59	153·18	87·71	Ore 2,000 grms. Purpose of test to note the effect of soda ash. Reagents: 1 lb./ton soda ash, 0·1 lb./ton potassium xanthate, and 4 drops of pine oil.
	Middling	4·57	5·86	5·92	
	Tailing	92·84	0·31	6·37	
	Totals	100·00		100·00	
5	Concentrate	3·03	136·30	89·51	Ore 2,000 grms. Floated with Aerofloat No. 25, 0·10 lb./ton; 0·02 lb. added to cleaner cell.
	Middling	2·74	10·44	6·20	
	Tailing	94·23	0·21	4·29	
	Totals	100·00		100·00	
6	Concentrate	4·32	102·0	94·45	Ore 2,000 grms. Ground 10 minutes in ball mill. Same reagents as in Test No. 5.
	Middling	5·56	1·74	2·08	
	Tailing	90·12	0·18	3·47	
	Totals	100·00		100·00	
7	Concentrate	0·73	511·6	84·56	Ore 1,000 grms. Floated with lime 1 lb./ton, Flotagen 0·2 lb./ton, Fumol No. 25, 0·10 lb./ton; to cleaner cell 0·02 lb./ton Flotagen and 0·02 lb./ton Fumol No. 25.
	Middling	3·37	7·72	5·89	
	Tailing	95·90	0·44	9·55	
	Totals	100·00		100·00	
8	Concentrate	3·43	115·68	90·9	Ore 2,000 grms. Floated with 1 lb./ton sulphonated coal-tar creosote and 0·06 lb./ton pine oil and 0·06 lb./ton Fumol.
	Tailing	96·57	0·41	9·1	
	Totals	100·00		100·00	
9	Concentrate	3·05	118·6	80·4	Ore 2,000 grms. Floated with 1 lb./ton sulphonated oil and 0·08 lb./ton pine oil.
	Tailing	96·95	0·91	19·6	
	Totals	100·00		100·00	

Small-batch Flotation Tests—Continued

Test No.	Products	Weight, %	Assay, Ag, oz./ton	Recovery, %	Reagents
10	Concentrate........	0·79	457·06	82·2	Ore 2,000 grms. Floated with lime 1 lb./ton. Other reagents as in Test No. 9.
	Tailing............	99·21	0·79	17·8	
	Totals.........	100·00		100·0	
11	Concentrate........	2·15	181·68	90·1	Ore 2,000 grms. Floated with amyl xanthate 0·10 lb./ton, 0·06 lb./ton pine oil, 0·06 lb./ton Fumol.
	Middling...........	1·15	14·14	3·5	
	Tailing............	96·70	0·29	6·4	
	Totals.........	100·00		100·0	

Small-scale Continuous Flotation Test

Feed rate, 100 pounds an hour.

Reagents:

Lime..	1·0 lb./ton
Amyl xanthate..	0·10 "
Fumol..	0·05 "

One cleaning only on the rougher concentrate.

Results:

Products	Weight, %	Assay, Ag, oz./ton	Recovery, %	Remarks
Feed............	....	4·74	....	
Concentrate......	2·90	154·20	94·05	Sample taken during first half of test.
Tailing...........	97·10	0·29	5·95	
Totals........	100·00		100·00	
Feed............	....	4·74	....	
Concentrate......	2·26	198·24	94·43	Sample taken during last half of test.
Tailing...........	97·74	0·27	5·57	
Totals........	100·00		100·00	

DISCUSSION OF RESULTS

The small-batch tests indicate that it is possible to make a 400- to 500-ounce concentrate with a recovery of better than 80 per cent. In the small, continuous test the highest grade of concentrate produced was 188·24 ounces, but the recovery was 94·4 per cent. It required very careful control of the operation of the small unit to produce even this grade of concentrate.

This delicate control required can be better appreciated when the fact is realized that in eight hours' running time, only 20 pounds of concentrate was floated out of 800 pounds of ore. Owing to the small amount of concentrate floated it was not practical to use more than one cleaning operation. Taking all the factors into consideration the results of the small-scale continuous test are exceptionally good.

The use of a small amount of lime has a decided effect on the grade of concentrate. In Test Nos. 7 and 10, one pound was used and a short period of conditioning allowed before flotation. In practice this could be obtained by adding the lime to the ball mill while regrinding the sands. The same effect might be obtained by adding milk of lime to the cleaner cells.

The other reagents which gave the best results were either amyl xanthate with Fumol and pine oil or Flotagen with Fumol and pine oil. Straight phospho-cresylic acid (Aerofloat No. 25) also gave fair results.

CONCLUSIONS

It is quite evident from the results of these tests that the silver can be floated and a high recovery obtained. The only point in doubt is the grade of concentrate which can be produced. Taking into consideration the experience at Cobalt there would seem to be little doubt that in large-scale operation, where two or more stages of cleaner cells could be used, that at least 300-ounce silver concentrate can be made with a recovery of from 80 to 90 per cent.

The reagent cost should be low and not exceed 10 cents per ton of feed.

Report No. 336

FLOTATION TESTS ON COPPER ORE FROM McGINN CLAIMS, BALDWIN TOWNSHIP, SUDBURY DISTRICT, ONTARIO

J. S. Godard

Shipments. A shipment of 90 pounds of copper ore was received October 7, 1929, from the McGinn claims in Baldwin township, Sudbury district, Ontario. The shipment was submitted by W. P. Murdock, Espanola station, Ontario.

Characteristics of the Ore. The ore is a copper one, containing chalcopyrite, pyrite, and pyrrhotite in a siliceous gangue.

Analysis of Head Sample:

Analysis showed the head sample to contain:

Copper	2·72 per cent	Nickel	None
Iron	4·35 "	Gold	0·01 oz./ton
Zinc	0·10 "	Platinum group metals	None

Purpose of Experimental Tests. The purpose of the experimenta tests was to determine if the ore were amenable to flotation.

EXPERIMENTAL TESTS

Test No. 1

Reagents:

CaO, 3·0 lb./ton; thiocarbanilide, 0·01 lb./ton; pine oil, 0·04 lb./ton.

Tailing screened on 200 mesh—

Product	Weight, %	Assay	Per cent of values
		Cu, %	Cu
+200	22·7	0·13	22·7
−200	77·3	0·13	77·3

Results—Flotation—

Product	Weight, %	Assay	Per cent of values
		Cu, %	Cu
Concentrate	11·9	21·50	95·7
Tailing	88·1	0·13	4·3

Test No. 2

Reagents:

CaO, 3·0 lb./ton; thiocarbanilide, 0·10 lb./ton; pine oil, 0·04 lb./ton.

Tailing screened on 200 mesh—

Product	Weight, %	Assay	Per cent of values
		Cu, %	Cu
+200	9·5	0·16	11·4
−200	90·5	0·13	88·6

Results—Flotation—

Concentrate	12·9	19·80	95·7
Tailing	87·1	0·13	4·3

Test No. 3

Reagents:

Na_2CO_3, 3·0 lb./ton; NaCN, 0·15 lb./ton; No. 305 0·02 lb./ton; pine oil No. 5, 0·12 lb./ton

Tailing screened on 200 mesh—

Product	Weight, %	Assay	Per cent of values
		Cu, %	Cu
+200	22·6	0·08	19·0
−200	77·4	0·10	81·0

Results—Flotation—

Concentrate	14·9	17·66	97·0
Tailing	85·1	0·096	3·0

Test No. 4

Reagents.

Na_2CO_3, 3·0 lb./ton; NaCN 0·10 lb./ton; No. 208 0·02 lb./ton; amyl xanthate, 0·03 lb./ton; pine oil, 0·16 lb./ton; F.P. No. 1, 0·04 lb./ton.

Tailing screened on 200 mesh—

Product	Weight, %	Assay	Per cent of values
		Cu, %	Cu
+200	7·2	0·05	7·2
−200	92·8	0·05	92·8

Results—Flotation—

Concentrate	21·25	11·80	98·5
Tailing	78·5	0·05	1·5

CONCLUSIONS

The ore is amenable to flotation. No difficulty was experienced in obtaining a good grade of copper concentrates with high recoveries.

Report No. 337

THE CONCENTRATION OF COPPER-TIN ORE FROM THE SNOWFLAKE MINING CO., ALBERT CANYON, BRITISH COLUMBIA

A. K. Anderson

Shipments. Three bags of ore were received on June 26, 1929, shipped by the Snowflake Mining Company, from Albert Canyon, British Columbia. Two of these, gross weight 173 pounds, were marked milling ore, and the third, weighing 115 pounds, zinc ore.

Characteristics of the Ore. The milling ore consisted of alternate bands of carbonaceous slates, quartz, iron pyrite, and stannite. Crystals of iron pyrite were also disseminated throughout the bands of other minerals. Occasional small spots of zinc and lead sulphides were noted. The copper-tin mineral, stannite, was massive while the quartz and iron pyrite tended toward a crystalline structure.

The lot designated zinc ore was of massive sulphides, stannite and iron pyrite predominating. A small amount of quartz in the form of stringers and inclusions in the sulphides was present. No appreciable amount of zinc sulphide was noticeable.

Purpose of Experimental Tests. The shipment was forwarded for the purpose of ascertaining what separation of the various mineral constituents can be secured by flotation concentration.

Sampling and Analysis. The two lots, milling ore and zinc ore, were sampled separately. They were crushed to pass 10 mesh, passed through a Jones riffle sampler several times until a portion weighing approximately 10 pounds was obtained. This was then crushed to finer sizes with alternate passes through the sampler until a final representative portion was secured for analysis.

The following results were obtained on these head samples:—

—	Milling ore	Zinc ore
Copper	5·40 per cent	18·60 per cent
Tin	4·35 "	16·30 "
Lead	2·76 "	0·82 "
Zinc	3·25 "	7·14 "
Iron	15·3 "	12·30 "
Silver	33·13 oz./ton	98·25 oz./ton

EXPERIMENTAL TESTS

All test work undertaken was along the lines of concentration by flotation, endeavouring to separate the various minerals into products of marketable value. Tests Nos. 1 to 6 were made to determine the effects of various reagents on the total recovery. The later tests investigated the separation of lead from copper and tin.

Milling Ore

Test No. 1

This test was made under conditions which, when applied to a normal ore containing copper, zinc, and iron sulphates, result in good separations.

A sample, 1,000 grammes of ore, −10 mesh, was ground for 40 minutes in a porcelain mill containing iron balls and 1,000 grammes of water, 3·0 grammes soda ash, and 0·10 gramme sodium cyanide.

After grinding, the pulp was transferred to a laboratory-size Ruth flotation machine and 0·10 gramme potassium xanthate added together with sufficient pine oil to froth. A dark-coloured, fast floating froth was removed.

Copper sulphate, 0·5 gramme was then added to the residue in the machine and the pulp agitated for three minutes after which a second concentrate was removed by the addition of 0·04 gramme potassium xanthate and pine oil. This second product contained a large percentage of iron pyrite.

Product	Weight, %	Assay					
		Cu, %	Sn, %	Pb, %	Zn, %	Fe, %	Ag, oz./ton
Heads	....	5·40	4·35	2·76	3·25	15·30	33·13
Concentrate No. 1	32·63	16·20	12·25	8·70	9·60	11·35	94·50
Concentrate No. 2	26·19	0·18	0·38	0·12	0·88	44·05	6·32
Tailing	41·18	0·06	0·14	0·26	0·15	1·80	1·19
	100·00						

Product	Weight, %	Per cent of values					
		Cu	Sn	Pb	Zn	Fe	Ag
Heads	....	....	....	....	....	....	....
Concentrate No. 1	32·63	98·7	96·2	95·4	91·5	23·2	93·5
Concentrate No. 2	26·19	0·9	2·4	1·0	6·7	72·2	5·0
Tailing	41·18	0·4	1·4	3·6	1·8	4·6	1·5
	100·00	100·0	100·0	100·0	100·0	100·0	100·0

The results secured show that the minerals in the ore can be concentrated by flotation. Iron sulphide is depressed by soda ash and cyanide but the zinc minerals are not. Concentrate No. 1 contains the major amounts of all the valuable minerals. Under normal conditions the zinc should have appeared in the second concentrate. However, only 6·7 per cent is in this product.

Test No. 2

In this test lime at the rate of 6·0 pounds per ton of ore was added to the grinding mill instead of soda ash as in the preceding test. Other conditions were the same.

Product	Weight, %	Assay					
		Cu, %	Sn, %	Pb, %	Zn, %	Fe, %	Ag, oz./ton
Heads	....	5·40	4·35	2·76	3·25	15·30	33·13
Concentrate No. 1	5·67	7·08	2·97	5·48	5·06	9·51	112·90
Concentrate No. 2	4·64	7·36	2·60	3·48	9·40	11·92	63·66
Tailing	89·69	5·18	4·4	2·66	3·13	16·32	27·21
	100·00						

Product	Weight, %	Per cent of values					
		Cu	Sn	Pb	Zn	Fe	Ag
Head	....	....	....	....	....	....	....
Concentrate No. 1	5·67	7·45	3·98	10·87	8·13	3·43	18·96
Concentrate No. 2	4·64	6·34	2·85	5·65	12·36	3·52	8·75
Tailing	89·69	86·21	93·17	83·48	79·51	93·05	72·29
	100·00	100·00	100·00	100·00	100·00	100·00	100·00

These results show that lime is detrimental to the flotation of this ore. Practically no recovery of any of the minerals is obtained, even the iron being so strongly depressed that the addition of 1 pound per ton of copper sulphate fails to reactivate it.

Test No. 3

The effect of reducing the soda ash from 6 pounds per ton as used in Test No. 1 to 2 pounds was investigated. Other conditions in this test were the same as those in Tests Nos. 1 and 2.

Product	Weight, %	Assay					
		Cu, %	Sn, %	Pb, %	Zn, %	Fe, %	Ag, oz./ton
Heads	....	5·40	4·35	2·76	3·25	15·30	33·13
Concentrate No. 1	28·31	17·70	15·45	9·62	8·30	8·90	107·08
Concentrate No. 2	32·96	0·72	0·30	0·25	3·13	40·66	7·94
Tailing	38·73	0·21	0·20	0·25	0·18	1·80	1·96
	100·00						

Product	Weight, %	Per cent of values					
		Cu	Sn	Pb,	Zn	Fe	Ag
Heads	....	....	....	....	....	....	....
Concentrate No. 1	28·31	94·0	96·0	93·8	68·1	15·2	89·98
Concentrate No. 2	32·96	4·5	2·2	2·8	29·9	80·6	7·77
Tailing	38·73	1·5	1·8	3·4	2·0	4·2	2·25
	100·00	100·0	100·0	100·0	100·0	100·0	100·00

This test shows that soda ash added at the rate of 2 pounds per ton instead of 6 pounds, results in a slightly lower recovery of all minerals, excepting copper.

Test No. 4

This test was made under conditions similar to those of Test No. 2, with the exception that 2 pounds of lime was used in place of 6 pounds. This did not give an alkaline reaction on testing the pulp with phenolphthalein.

Product	Weight, %	Assay					
		Cu, %	Sn, %	Pb, %	Zn, %	Fe, %	Ag, oz./ton
Heads	....	5·40	4·35	2·76	3·25	15·30	33·13
Concentrate No. 1	32·23	15·54	13·53	8·00	10·12	8·80	96·46
Concentrate No. 2	13·27	0·64	0·30	0·31	0·42	41·20	7·70
Tailing	54·50	0·28	0·20	0·10	0·15	13·40	2·71
	100·00						

Product	Weight, %	Per cent of values					
		Cu	Sn	Pb	Zn	Fe	Ag
Heads	....	....	....	....	....	....	....
Concentrate No. 1	32·23	95·5	96·7	96·4	96·0	18·2	92·6
Concentrate No. 2	13·27	1·6	0·8	1·5	1·6	35·0	3·0
Tailing	54·50	2·9	2·5	2·1	2·4	46·8	4·4
	100·00	100·0	100·0	100·0	100·0	100·0	100·0

The results secured with 2 pounds of lime used in the grinding circuit are much higher than those in Test No. 2 where 6 pounds of lime was used. As the pulp was not alkaline to phenolphthalein, it may be assumed that flotation in a slightly alkaline circuit will give good results.

A comparison between Tests Nos. 3 and 4 shows that lime has a tendency to depress the silver minerals more strongly than soda ash. A lower grade concentrate is also produced with a tailing higher in copper and silver. It therefore appears that it is preferable to use soda ash instead of lime for an alkaline reagent.

Test No. 5

In this test the soda ash was increased to 4 pounds per ton, the ore ground with 0·10 pound thiocarbanilide and 0·20 pound cyanide per ton.

It was then floated with 0·16 pound potassium xanthate and pine oil. The concentrate secured was cleaned once.

Copper sulphate at the rate of 1 pound per ton of ore was then added and a second concentrate taken off. This was chiefly iron pyrite.

Product	Weight, %	Assay					
		Cu, %	Sn, %	Pb, %	Zn, %	Fe, %	Ag, oz./ton
Concentrate No. 1	22·26	23·04	18·52	12·34	9·18	5·18	123·80
Middling	8·73	2·60	0·38	0·28	9·45	22·94	35·48
Concentrate No. 2	31·69	0·30	0·14	0·15	1·36	38·91	5·30
Tailing	37·32	0·13	0·10	0·28	0·13	2·05	1·22
	100·00						

Product	Weight, %	Per cent of values					
		Cu	Sn	Pb	Zn	Fe	Ag
Concentrate No. 1	22·26	93·3	97·3	94·0	61·0	7·1	84·0
Middling	8·73	4·1	0·8	0·8	24·6	12·3	9·5
Concentrate No. 2	31·69	1·7	1·0	1·6	12·9	75·9	5·1
Tailing	37·32	0·9	0·9	3·6	1·5	4·7	1·4
	100·00	100·0	100·0	100·0	100·0	100·0	100·0

It appears that a high-grade concentrate can be made by cleaning the rougher concentrate. A recovery in the combined concentrate and middling of the first flotation shows results closely paralleling those of Test No. 1.

The second concentrate again shows that only small amounts of the valuable minerals, apart from silver, are depressed by soda ash and cyanide.

Test No. 6

In this test the effect of cyanide and zinc sulphate as a depressant was investigated.

The ore was ground in 1 : 1 pulp with 0·10 pound thiocarbanilide, 0·30 pound cyanide, 0·30 pound zinc sulphate, and 4·0 pounds soda ash per ton.

The pulp was then floated with 0·16 pound cresylic acid, and the concentrate cleaned.

After removing this concentrate, 1·0 pound copper sulphate was added and a second concentrate taken off by the addition of 0·16 pound Aerofloat No. 25. This concentrate was high in iron pyrite.

Product	Weight, %	Assay					
		Cu, %	Sn, %	Pb, %	Zn, %	Fe, %	Ag, oz./ton
Concentrate No. 1	19·79	22·70	19·77	11·16	8·56	4·14	126·25
Middling	6·57	7·27	5·52	4·55	7·52	15·64	73·22
Concentrate No. 2	10·94	0·72	0·19	0·46	9·19	33·77	13·04
Tailing	62·70	0·21	0·12	0·17	0·10	15·95	2·28
	100·00						

Product	Weight, %	Per cent of values					
		Cu	Sn	Pb	Zn	Fe	Ag
Concentrate No. 1	19·79	86·7	88·9	82·9	52·0	5·3	76·5
Middling	6·57	9·2	8·2	11·2	15·2	6·6	14·7
Concentrate No. 2	10·94	1·5	0·5	1·9	30·9	23·8	4·4
Tailing	62·70	2·6	2·4	4·0	1·9	64·3	4·4
	100·00	100·0	100·0	100·0	100·0	100·0	100·0

The use of zinc sulphate results in recoveries lower than those secured in previous tests as indicated by the values remaining in the tailing.

No sharp separation of the minerals was made in the preceding tests. Good recoveries of copper, tin, lead, and silver were secured in the first concentrate.

Test No. 7

In this test, iron pyrite was depressed by cyanide added in the grinding mill and a flotation concentrate made containing the copper, tin, lead, and silver. This product was then cleaned twice to raise the grade before subsequent treatment.

This high-grade concentrate was then conditioned with lime and cyanide to note if the stannite would be depressed, thus allowing a lead concentrate to be removed.

A sample, 1,000 grammes of the ore, −10 mesh, was ground in a porcelain mill containing iron balls and 1,000 grammes of water; 5·0 pounds soda ash, 0·12 pound cyanide, 0·16 pound thiocarbanilide per ton were also added.

After grinding, the pulp was transferred to a Ruth flotation machine and after adding 0·14 pound cresylic acid, a concentrate was removed. This was cleaned twice, yielding 1st and 2nd middling. The high-grade concentrate was then agitated for 20 minutes with 0·05 gramme lime and 0·20 gramme cyanide. Without further addition of reagents, a concentrate called lead concentrate was then removed. The portion that did not float was called copper-tin concentrate.

Product	Weight, %	Assay					
		Cu, %	Sn, %	Pb, %	Zn, %	Fe, %	Ag, oz./ton
Lead concentrate	12·97	21·57	20·15	14·00	8·48	3·62	125·72
Copper-tin concentrate	5·78	23·32	20·54	4·76	10·74	5·44	84·20
1st middling	9·40	2·15	1·34	1·74	8·86	22·06	36·26
2nd middling	3·75	12·50	7·72	10·85	11·17	12·06	130·54
Tailing	68·10	0·11	0·09	0·21	0·56	18·12	2·50
	100·00						

Product	Weight, %	Per cent of values					
		Cu	Sn	Pb	Zn	Fe	Ag
Lead concentrate	12·97	57·2	61·1	64·8	32·8	2·5	52·3
Copper-tin concentrate	5·78	27·6	27·8	9·8	18·5	17·0	15·6
1st middling	9·40	4·1	2·9	5·8	24·8	11·2	10·9
2nd middling	3·75	9·6	6·8	14·5	12·5	2·5	15·7
Tailing	68·10	1·5	1·4	5·1	11·4	66·8	5·5
	100·00	100·0	100·0	100·0	100·0	100·0	100·0

It appears that the following recoveries were obtained by the initial concentration: copper 98·5 per cent, tin 98·6 per cent, lead 94·9 per cent, and silver 94·5 per cent in a product 31·9 per cent of the total weight. The two cleaning operations raised the grade, dropping 28·2 per cent of the total iron.

Treating this cleaned concentrate with lime and cyanide did not have the expected result. Stannite was not depressed as shown by the assays of the two concentrates. The lead concentrate has the higher values in lead and silver, indicating that the minerals carrying these float more readily than the copper-tin mineral.

Test No. 8

This test was made to determine if an increase in the amounts of lime and cyanide used in conditioning the concentrate would depress the stannite. Other details of the test were the same as those of Test No. 7. The initial flotation was made on pulp ground with 2·5 pounds soda ash, 0·12 pound cyanide, and 0·16 pound thiocarbanilide per ton of ore.

After cleaning twice, 0·10 gramme lime and 0·4 gramme cyanide were added, doubling the amounts used in the preceding test. On conditioning for the same length of time, 20 minutes, a product called lead concentrate was taken off.

Product	Weight, %	Assay					
		Cu, %	Sn, %	Pb, %	Zn, %	Fe, %	Ag, oz./ton
Lead concentrate	17·57	23·42	20·34	12·64	8·65	3·78	123·72
Copper-tin concentrate	2·35	20·00	17·94	2·15	14·50	8·13	85·40
1st middling	9·83	1·33	0·57	1·28	7·08	24·00	26·58
2nd middling	3·79	10·33	4·80	9·52	11·97	15·02	111·24
Tailing	66·46	0·22	0·10	0·23	0·56	18·03	2·24
	100·00						

Product	Weight, %	Per cent of values					
		Cu	Sn	Pb	Zn	Fe	Ag
Lead concentrate	17·57	78·3	83·1	76·3	44·9	30·5	67·8
Copper-tin concentrate	2·35	9·0	9·8	1·7	10·1	0·9	6·3
1st middling	9·83	2·5	1·3	4·3	20·6	10·9	8·2
2nd middling	3·79	7·4	4·2	12·4	13·4	2·6	13·2
Tailing	66·46	2·8	1·6	5·3	11·0	55·1	4·5
	100·00	100·0	100·0	100·0	100·0	100·0	100·0

No separation of the lead from the copper-tin minerals was obtained, the results agreeing closely with those of the preceding test.

Test No. 9

The same procedure was followed in this test as in Nos. 7 and 8. The cyanide and lime were again increased to 1·0 gramme and 0·5 gramme and the time from 20 to 45 minutes.

Product	Weight, %	Assay			Per cent of values		
		Cu, %	Pb, %	Ag, oz./ton	Cu	Pb	Ag
Lead concentrate	19·80	23·68	11·47	120·00	86·5	84·2	78·7
Copper-tin concentrate	0·81	20·22	3·80	84·08	3·0	1·1	2·3
1st middling	10·25	1·16	0·92	17·24	2·2	3·5	5·9
2nd middling	4·16	4·58	2·86	61·48	3·5	4·4	8·5
Tailing	64·98	0·40	0·28	2·20	4·8	6·8	4·6
	100·00				100·0	100·0	100·0

As found by the preceding tests, agitation of the cleaned concentrate in contact with lime and cyanide does not depress stannite. The copper content of the lead product is even higher than that of the copper-tin concentrate.

Test No. 10

To determine if cyanide would depress stannite, a large amount of cyanide was added to the ball mill and the ore ground as in former tests. A concentrate was removed using potassium xanthate and pine oil. The material thus obtained was cleaned twice and the products of the test analysed to determine where the values were.

A sample, 1,000 grammes of ore, —10 mesh, was ground for 40 minutes with 1,000 grammes water, 0·05 gramme thiocarbanilide, 4·0 grammes soda ash, 1·0 gramme cyanide.

Flotation reagents added to the cell were 0·08 gramme potassium xanthate and 2 drops of pine oil. No additional reagents were used in the cleaning operations.

Product	Weight, %	Assay			Per cent of values		
		Cu, %	Pb, %	Ag, oz./ton	Cu	Pb	Ag
Concentrate	21·07	22·52	11·36	120·68	90·4	90·1	82·9
1st middling	8·79	1·22	0·71	13·90	2·0	2·4	4·0
2nd middling	3·96	5·32	2·25	55·60	4·0	3·4	7·2
Tailing	66·18	0·28	0·17	2·76	3·6	4·1	5·9
	100·00				100·0	100·0	100·0

There is no evidence that stannite is depressed by grinding with 2·0 pounds cyanide per ton of ore. The rougher concentrate, 33·8 per cent of the total weight contains 96·4 per cent of the copper in the ore.

Test No. 11

This test was made to note the effect of zinc sulphate added to the grinding mill.

A sample, 1,000 grammes of the ore at —10 mesh, was ground for 40 minutes with 4·0 grammes soda ash, 1·0 gramme zinc sulphate, and 1,000 grammes of water.

The pulp was then floated by the addition of 0·22 pound T.T. mixture per ton of ore. The concentrate was cleaned once.

Product	Weight, %	Assay			Per cent of values		
		Cu, %	Pb, %	Ag, oz./ton	Cu	Pb	Ag
Concentrate	17·15	22·30	5·94	103·48	75·5	39·1	58·1
Middling	16·26	6·72	7·88	63·68	21·6	49·2	33·9
Tailing	66·59	0·22	0·46	3·70	2·9	11·7	8·0
	100·00				100·0	100·0	100·0

It is apparent that stannite is not depressed by zinc sulphate as 97·1 per cent of the copper is recovered in the rougher concentrate.

Test No. 12

Grinding with large amounts of zinc sulphate and cyanide was then tried.

A sample, 1000 grammes of ore at – 10 mesh, was ground for 40 minutes with 1000 c.c. of water, 4·0 grammes of soda ash, 1·0 gramme each of zinc sulphate and cyanide.

T.T. mixture at the rate of 0·30 pound per ton was added and a concentrate removed. As flotation was rather slow, a second concentrate was removed by the addition of potassium xanthate. The concentrates were not cleaned.

Product	Weight, %	Assay			Per cent of values		
		Cu, %	Pb, %	Ag, oz./ton	Cu	Pb	Ag
Concentrate No. 1	15·10	15·94	17·40	147·92	46·9	92·3	67·8
Concentrate No. 2	9·80	19·80	0·46	65·08	37·8	1·6	19·3
Tailing	75·10	1·05	0·23	5·66	15·3	6·1	12·9
	100·00				100·0	100·0	100·0

No benefit is derived from the use of zinc sulphate and cyanide. A tailing high in copper results, but no marked effect that would be of value in separating stannite from lead minerals is evident.

Test No. 13

Sodium sulphite as a depressant was next added to the grinding mill.

Soda ash, 8·0 pounds, and sodium sulphite 1·0 pound per ton were added to the ore which was ground in 1 : 1 dilution with water.

T.T. mixture, 0·30 pound per ton, was added to the flotation machine and a concentrate removed. This product apparently contained all the sulphides in the ore, iron pyrite included.

This low-grade product was then agitated for 5 minutes with 0·1 gramme cyanide and 3 grammes of lime. T.T. mixture, xanthate, and pine oil were added and a concentrate and middling produced.

Product	Weight, %	Assay			Per cent of values		
		Cu, %.	Pb, %	Ag, oz./ton	Cu	Pb	Ag
Concentrate	23·46	18·64	9·82	113·80	83·7	80·9	80·6
Middling	28·14	2·66	1·48	18·48	14·3	14·6	15·7
Tailing	48·40	0·21	0·26	2·50	2·0	4·5	3·7
	100·00				100·0	100·0	100·0

Sodium sulphite has no depressing effect on stannite as shown by the fact that 98 per cent of the copper is carried in the first concentrate. It is again demonstrated that agitation with lime and cyanide does not separate stannite from the lead minerals.

It is apparent from a study of the foregoing tests that stannite is not depressed to any great extent by any of the reagents used. The object of the following tests was to depress the lead contained in the rougher concentrate thus securing a concentration of the copper-tin minerals in the froth.

Test No. 14

A sample, 1,000 grammes of ore at −10 mesh, was ground in a porcelain jar containing iron balls and 1,000 grammes of water, 4·0 grammes soda ash, 0·10 gramme cyanide.

After transferring to the flotation machine, 0·10 gramme potassium xanthate and pine oil were added and a rougher concentrate removed. This product was then cleaned twice to remove as much of the iron and zinc as possible. These operations gave a high-grade concentrate and two middling products.

The concentrate was returned to the flotation machine and 3 grammes of lime added. The pulp was then agitated for three minutes after which a drop of pine oil was added and a concentrate removed.

This was called the copper-tin concentrate; the material that did not float being the lead concentrate.

Product	Weight, %	Assay			Per cent of values		
		Cu, %	Pb, %	Ag, oz./ton	Cu	Pb	Ag
Copper-tin concentrate	15·98	25·28	4·81	99·88	77·3	27·3	47·6
Lead concentrate	2·75	9·10	44·54	150·20	4·8	43·4	12·3
1st middling	10·44	0·95	1·07	22·20	1·9	4·0	6·9
2nd middling	9·24	8·11	5·88	99·10	14·3	19·3	27·3
Tailing	61·59	0·14	0·28	3·14	1·7	6·0	5·9
	100·00				100·0	100·0	100·0

This shows that the rougher concentrate contains 98·3 per cent of the copper, 94·0 per cent of the lead, and 94·1 per cent of the silver. The lead concentrate is much higher in value than any produced in previous tests. The recovery of lead in this is low due to the small weight of concentrate produced. There is no sharp separation of the silver minerals, the copper-tin concentrate assaying 99·8 ounces silver per ton, and the lead concentrate 150·20 ounces.

It appears from these results that there is a possibility of depressing the lead minerals with lime, enabling a separation of that metal from copper and tin to be made.

Test No. 15

This test was run under the same general conditions as Test No. 14 with the exception that the quantity of lime added to depress the lead minerals was cut in half, 1½ grammes of lime instead of 3 grammes.

Product	Weight, %	Assay					
		Cu, %	Sn, %	Pb, %	Zn, %	Fe, %	Ag, oz./ton
Copper-tin concentrate	16·7	25·54	23·12	4·92	9·66	4·08	128·60
Lead concentrate	3·2	8·82	7·77	44·64	7·46	6·43	173·00
1st middling	9·9	1·02	0·30	0·56	4·88	27·20	19·80
2nd middling	4·4	5·44	4·80	2·87	12·94	22·16	72·80
Tailing	65·8	0·17	0·05	0·10	0·48	16·20	2·60
	100·0						

Product	Weight, %	Per cent of values					
		Cu	Sn	Pb	Zn	Fe	Ag
Copper-tin concentrate	16·7	85·3	88·1	32·9	50·1	4·5	63·4
Lead concentrate	3·2	5·6	5·7	57·1	7·4	1·4	16·3
1st middling	9·9	2·0	0·7	2·2	15·0	17·7	5·8
2nd middling	4·4	4·8	4·8	5·1	17·7	6·4	9·4
Tailing	65·8	2·3	0·7	2·6	9·8	70·0	5·1
	100·0	100·0	100·0	100·0	100·0	100·0	100·0

The results secured here agree closely with those of Test No. 14. Any differences in the recoveries obtained in the finished concentrate may be attributed to manipulation of the flotation machine. The two middling products especially show this, being of lower grade than those of the preceding test. The amount of lime used to depress the lead does not appear to be critical.

Test No. 16

In this test the time element was investigated. Agitation of the mixed copper-tin and lead concentrate was increased from 5 to 10 minutes. Other conditions were the same as those of Test No. 14, 6 pounds of lime per ton of original feed being added.

Product	Weight, %	Assay					
		Cu, %	Sn, %	Pb, %	Zn, %	Fe, %	Ag, oz./ton
Copper-tin concentrate	16·2	27·26	24·57	2·04	9·82	4·12	124·20
Lead concentrate	3·2	7·28	2·88	55·40	5·00	3·26	174·40
1st middling	8·6	1·59	0·91	0·82	7·03	24·77	30·00
2nd middling	2·9	10·35	9·12	5·83	12·89	15·86	114·16
Tailing	69·1	0·18	0·07	0·23	0·64	17·20	3·38
	100·0						

Product	Weight %	Per cent of values					
		Cu	Sn	Pb	Zn	Fe	Ag
Copper-tin concentrate	16·2	84·8	89·2	13·2	50·2	4·4	59·3
Lead concentrate	3·2	4·5	2·1	70·9	5·0	0·6	16·4
1st middling	8·6	2·6	1·8	2·8	19·1	14·0	7·6
2nd middling	2·9	5·7	5·9	6·8	11·8	3·0	9·8
Tailing	69·1	2·4	1·0	6·3	13·9	78·0	6·9
	100·0	100·0	100·0	100·0	100·0	100·0	100·0

These results show that increased time of contact is of advantage. This is indicated clearly in a comparison of Tests Nos. 14 and 16.

Test No.	Time, minutes	Copper-tin concentrate					
		Assay			Per cent of values		
		Cu, %	Pb, %	Ag, oz./ton	Cu	Pb	Ag
14	5	25·54	4·92	128·60	85·3	32·9	63·4
16	10	27·26	2·04	124·20	84·8	13·2	59·3

Test No.	Time, minutes	Lead concentrate					
		Assay			Per cent of values		
		Cu, %	Pb, %	Ag, oz./ton	Cu	Pb	Ag
14	5	8·82	44·64	173·00	5·6	57·1	16·3
16	10	7·28	55·40	174·40	4·5	70·9	16·4

It is seen that increasing the time of agitation from 5 to 10 minutes reduces the lead in the copper-tin concentrate from 4·92 per cent to 2·04 per cent. This results in a recovery of 70·9 per cent of the lead in the finished lead concentrate as against 57·1 per cent, with an assay value of 55·40 per cent lead instead of 44·64 per cent.

The results secured indicate that the initial concentration effects the following recoveries: copper 97·6 per cent, tin 99·0 per cent, lead 93·7 per cent, zinc 86·1 per cent, and silver 93·1 per cent.

The combined middling products contain 8·3 per cent of the copper, 7·7 per cent of the tin, 9·6 per cent of the lead, and 17·4 per cent of the silver. As these products would be re-treated in practice, the total recovery will be materially increased over that shown in the finished products, without lowering the assay values.

Zinc Ore

Test No. 1

As this ore was of such a high-grade nature and presumably not representative of the run-of-mine, two tests only were made on it.

This material was treated in much the same manner as was the milling ore in the preceding tests.

A sample, 1,000 grammes of ore at −10 mesh, was ground for 40 minutes in a porcelain mill containing iron balls and 1,000 grammes of water, 4·0 grammes soda ash, 0·06 gramme cyanide, 0·06 gramme zinc sulphate, and 0·05 gramme thiocarbanilide.

After transferring to the flotation machine, 0·16 pound cresylic acid per ton was added, and a concentrate taken off. This was cleaned giving a middling and a concentrate.

The residue remaining in the machine after the initial flotation was conditioned with 0·5 gramme copper sulphate and a second concentrate, called zinc concentrate, removed. This showed no characteristic colour of zinc sulphide but was heavily charged with iron pyrite.

The concentrate secured from the cleaning of the first product was agitated with 3 grammes of lime for 5 minutes and then floated with a drop of pine oil, giving a copper-tin concentrate and a residue called lead concentrate.

Product	Weight, %	Assay					
		Cu, %	Sn, %	Pb, %	Zn, %	Fe, %	Ag, oz./ton
Heads	....	18·6	16·30	0·82	7·14	12·30	98·25
Lead concentrate	3·7	20·38	16·22	1·35	7·78	10·86	105·60
Copper-tin concentrate	56·0	27·16	23·60	0·96	9·23	4·90	140·60
Copper-tin middling	13·9	15·44	13·73	0·46	8·26	15·55	80·52
Zinc concentrate	6·3	2·40	1·56	0·56	5·70	24·10	21·48
Tailing	20·1	0·62	0·07	0·27	0·34	26·80	4·70
	100·0						

Product	Weight, %	Per cent of values					
		Cu	Sn	Pb	Zn	Fe	Ag
Heads	....	....	....	....	....	....	....
Lead concentrate	3·7	4·1	3·8	6·7	4·0	3·7	3·7
Copper-tin concentrate	56·0	82·7	83·4	72·5	72·1	25·6	75·4
Copper-tin middling	13·9	11·7	12·1	8·6	17·9	6·3	18·7
Zinc concentrate	6·3	0·8	0·6	4·9	5·0	14·2	1·3
Tailing	20·1	0·7	0·1	7·3	1·0	50·2	0·9
	100·0	100·0	100·0	100·0	100·0	100·0	100·0

As in the case of the milling ore, zinc tends to float with the copper-tin minerals.

The only apparent benefit derived is dropping approximately 50 per cent of the iron. This reduction in bulk automatically raises the metal content in the concentrate.

Test No. 2

Aerofloat No. 25 was used as a collecting agent in this test.

A sample, 1,000 grammes of ore at −10 mesh, was ground 40 minutes; 1,000 grammes of water, 4·0 grammes soda ash, 0·06 gramme cyanide, 0·06 gramme zinc sulphate, and 0·24 pound Aerofloat No. 25 per ton.

Flotation was performed without further addition of reagents. The concentrate was cleaned once.

The addition of copper sulphate to the residue did not bring up any zinc so no second concentrate was taken.

Three grammes of lime was added to the cleaned concentrate and agitation continued for 5 minutes after which a copper-tin concentrate was removed.

Product	Weight, %	Assay					
		Cu, %	Sn, %	Pb, %	Zn, %	Fe, %	Ag, oz./ton
Heads	....	18·60	16·30	0·82	7·14	12·30	98·25
Lead concentrate	20·8	21·06	15·48	1·56	9·13	10·50	110·00
Copper-tin concentrate	50·0	26·62	24·64	0·73	9·07	4·80	141·20
Copper-tin middling	20·0	1·23	0·72	0·41	2·25	24·20	6·92
Tailing	9·2	4·50	2·86	0·54	3·06	26·10	27·08
	100·0						

Product	Weight, %	Per cent of values					
		Cu	Sn	Pb	Zn	Fe	Ag
Heads	....	....	....	....	....	....	....
Lead concentrate	20·8	23·9	20·2	39·5	26·5	22·6	23·5
Copper-tin concentrate	50·0	72·5	77·3	44·4	63·3	24·8	72·5
Copper-tin middling	20·0	1·3	0·9	10·0	6·3	50·1	1·4
Tailing	9·2	2·3	1·6	6·1	3·9	2·5	2·6
	100·0	100·0	100·0	100·0	100·0	100·0	100·0

No satisfactory results were achieved. The rougher concentrate contains 90·8 per cent of the total weight, and the tailing assays are high. This class of material does not respond to the treatment given the milling ore, or to conditions which under normal ores give a separation of zinc from the other minerals. It would appear that the zinc is combined with the copper-tin mineral and not present as zinc sulphide.

SUMMARY AND CONCLUSIONS

As the tests on the shipment termed zinc ore did not yield satisfactory results, the milling ore only will be discussed.

The tests show that high recoveries of the valuable minerals can be made quite easily using any of the popular flotation reagents. The use of lime in the grinding circuit as shown in Test No. 2 is highly detrimental, resulting in almost a total loss of the minerals. Soda ash sufficient to produce an alkaline reaction in the flotation circuit is therefore indicated. This reagent together with cyanide is necessary to depress iron pyrite present in the ore, thus removing this valueless mineral. The small amount of zinc present does not appear to be present as zinc sulphide. If it were, this metal would appear in concentrate No. 2 in Tests Nos. 1, 3, 4, 5, and 6. However, in all these tests, most of the zinc is in concentrate No. 1. Apparently this metal is in combination with the copper-tin mineral, stannite, replacing part of the iron in it.

Test No. 6 shows that the use of zinc sulphate is not advisable as low recoveries result.

All these tests, with the exception of Test No. 2 where the ore was ground with lime, show that in a rougher concentrate, average recoveries of 98 per cent of the copper, 98 per cent of the tin, 95 per cent of the lead, and 95 per cent of the silver are obtained with a ratio of concentration of 3 : 1.

Tests Nos. 5 and 6 indicate that this rougher concentrate can be cleaned in one operation to approximately 23 per cent copper, 19 per cent tin, 12 per cent lead, and 125 ounces silver per ton.

Tests Nos. 7 to 13 indicate that stannite is not depressed by the application of any of the known reagents and that a separation of copper and tin from lead is not to be expected from this procedure.

This separation, however, can be effected by reversing the process, depressing the lead and floating the copper-tin mineral. Tests Nos. 14, 15, and 16 show the results of conditioning with lime the combined copper-tin and lead concentrate resulting from the cleaning of the rougher concentrate. The time of contact is important as shown by comparisons of Tests Nos. 15 and 16. By increasing the time from 5 to 10 minutes, the grade of lead concentrate is raised from 44·64 per cent lead to 55·40 per cent, with a corresponding increase in recovery of from 57·1 to 70·9 per cent. The grade and per cent recovery of copper and tin are approximately the same in both cases. No difficulty is to be expected in making a high-grade concentrate with good recoveries of copper, tin, lead, and silver. A separation of the lead in a marketable concentrate can be made, leaving a concentrate high in copper and tin. The silver is not segregated in any one product, being found in both. Further investigation might possibly result in a concentration of this metal in either one or the other.

The process indicated is one employing flotation, producing a bulk concentrate containing the copper, tin, lead, and silver, striving for maximum recoveries. As there is appreciable iron pyrite present in the ore, the use of soda ash and cyanide is necessary to depress this mineral. Further increase in grade is secured by subjecting the rougher concentrate to two cleaning operations. The middling products may either be returned to the circuit or treated in a separate unit. This point can best be determined in the mill itself. Any of the popular flotation reagents give high recoveries, potassium xanthate and pine oil will be found to be satisfactory.

After the high-grade concentrate is secured, it is then to be agitated with lime in a conditioning tank. The indicated amount of lime is from 1½ to 3 pounds per ton of original feed and a time contact of 10 minutes will be found necessary. Finer adjustments of these factors will be determined in the plant itself, doubtless resulting in recoveries better than those recorded in the tests.

As there are several factors governing the treatment of this ore, it is recommended that a small test mill be erected so that concentration practice can be stabilized prior to embarking on any large program of construction.

Subsequent treatment of the copper-tin concentrate is a problem not touched on in this investigation.

Report No. 338

EXPERIMENTAL TESTS ON GOLD-COPPER ORE FROM THE GREENE-STABELL MINE

C. S. Parsons

Shipment. A shipment of 3,000 pounds of ore was received July 15, 1929, from the mine of Greene-Stabell Mines, Limited, located in the township of Dubuisson, Abitibi district, Que.

Characteristics of the Ore. The ore is a gold-copper one and contains gold, 0·38 ounce per ton, and copper 0·50 per cent. Though no gold is visible 50 per cent is amalgamable when ground to 10 per cent +200 mesh. The remainder is presumed to be associated with the sulphides, which consitute about 5 per cent of the ore. The sulphides present are pyrite, pyrrhotite, and chalcopyrite, and are found in fractures in the quartz which constitutes 80 per cent of the gangue.

Analysis of Head Sample.

The analysis is as follows:—

Gold	0·38 oz./ton	Lime	0·90 per cent
Silver	0·18 “	Magnesia	1·35 “
Copper	0·50 per cent	Sulphur	2·00 “
Iron	6·80 “	Silica	79·68 “
Alumina	2·86 “		

Purpose of Experimental Tests. The following experimental tests were made to determine the percentage of gold which could be recovered by flotation or by amalgamation and flotation for the purpose of designing a mill, which could treat the ore in the most economical manner.

EXPERIMENTAL TESTS

FLOTATION

Test No. 1

Ore 2,000 grammes at −14 mesh, water 1,500 c.c., soda ash 2·5 pounds per ton, sodium cyanide 0·2 pound per ton. Ground 25 minutes and floated using water-gas tar 3 drops, potassium xanthate 0·15 pound per ton, cresylic acid 4 drops, pine oil 2 drops. After flotation of No. 1 concentrate, a second concentrate was made by adding copper sulphate 0·1 pound per ton, potassium xanthate 0·05 pound per ton.

Results:

Product	Weight, %	Assay			Per cent of values		
		Au, oz./ton	Cu, %	Ag, oz./ton	Au	Cu	Ag
Concentrate No. 1	4·9	5·62	10·22	2·62	71·4	93·8	71·4
Concentrate No. 2	1·8	0·96	1·34	0·70	4·5	4·5	7·0
Tailing	93·3	0·10	0·01	....	24·1	1·7	21·6

A screen test was made on the tailings:—

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+ 65	0·4	0·93	0·02	3·7	1·0
+100	1·8	0·27	0·02	4·9	3·9
+150	8·3	0·16	0·01	13·2	7·8
+200	5·1	0·22	0·01	11·1	4·9
−200	84·4	0·08	0·01	67·1	82·4

Average tailing: Au 0·10 oz./ton, Cu 0·01 per cent.

Test No. 2

Ore 2,000 grammes at −14 mesh, water 1,500 c.c. Ground 25 minutes; added soda ash 0·3 pound per ton, sodium cyanide 0·15 pound per ton, water-gas tar 3 drops, potassium xanthate 0·15 pound per ton, and pine oil 3 drops.

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	5·3	5·08	8·52	76·0	97·9
Tailing	94·7	0·09	0·01	24·0	2·1

Flotation concentrate assayed: Ag. 2·26 oz./ton.

Test No. 3

Ore 2,000 grammes −14 mesh, water 1,500 c.c. Ground 25 minutes. Rougher concentrate floated using water-gas tar 3 drops, amyl xanthate 0·15 pound per ton, pine oil 3 drops. Rougher concentrate was dewatered agitated 15 minutes in 1 : 2 pulp, containing soda ash 1·0 gramme, sodium cyanide 0·20 gramme, then floated using potassium xanthate 0·05 gramme and pine oil 1 drop.

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	1·83	10·88	23·3	56·9	89·1
Middling	3·30	1·70	1·3	16·0	9·0
Tailing	94·87	0·10	0·01	27·1	1·9

Test No. 4

Ore 2,000 grammes at −14 mesh, water 1,500 c.c., soda ash 2·5 pound per ton, water-gas tar 3 drops. Ground 10 minutes; amyl xanthate 0·15 pound per ton, pine oil 6 drops added to cell.

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	7·9	3·40	5·96	73·2	95·3
Tailing	92·1	0·107	0·025	26·8	4·7

A screen test was made on the tailing.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+100	10·3	0·23	0·02	22·3	8·4
+150	9·2	0·17	0·01	14·6	3·6
+200	10·9	0·17	0·01	17·4	4·4
−200	69·6	0·07	0·03	45·7	83·6

Average tailing: Au 0·107 oz./ton, Cu 0·025 per cent.

Test No. 5

Ore 2,000 grammes at −14 mesh, water 1,500 c.c., soda ash 2·5 pounds per ton, water-gas tar 3 drops. Ground 5 minutes; amyl xanthate 0·15 pound per ton, pine oil 4 drops added to cell.

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	10·4	2·80	4·70	76·8	97·3
Tailing	89·6	0·098	0·015	23·2	2·7

A screen test was made on the tailing.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+100	14·0	0·19	0·005	27·2	4·7
+150	11·4	0·14	0·005	16·4	4·0
+200	14·6	0·13	0·01	19·5	10·1
−200	60·0	0·06	0·02	36·9	81·2

Average tailing: Au 0·098 oz./ton, Cu 0·015 per cent.

Test No. 6

A two-stage flotation test was made as follows:—

Ore 2,000 grammes at −14 mesh, water 1,500 c.c., soda ash 2·5 pounds per ton, water-gas tar 3 drops. Ground 2 minutes. Added to cell, amyl xanthate 0·15 pound per ton, pine oil 3 drops. First concentrate removed. Tailing dewatered and reground 15 minutes. Added amyl xanthate 0·15 pound per ton, pine oil 3 drops. Second concentrate removed.

A sizing test was made to determine approximately the size of the ore particles at the first flotation.

Product	Weight, %	Cumulative weight, %
− 14+ 20	2·37	2·37
− 20+ 28	7·73	10·10
− 28+ 35	7·83	17·93
− 35+ 48	7·52	25·45
− 48+ 65	11·90	37·35
− 65+100	19·10	56·45
−100+150	8·50	64·95
−150+200	8·55	73·50
−200	26·50	100·00

Results:

Product	Weight, %	Assay		Per cent of values		Remarks
		Au, oz./ton	Cu, %	Au	Cu	
First concentrate	8·8	3·08	5·68	68·6	94·6	Coarse flotation.
Second concentrate	2·4	2·96	0·86	17·9	3·9	Fine flotation.
Final tailing	88·8	0·06	0·009	13·5	1·5	

A screen analysis on final tailing gives:—

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+100	1·2	0·17	0·005	3·4	0·6
+150	3·2	0·12	0·005	6·4	1·7
+200	8·6	0·12	0·005	17·3	4·6
−200	87·0	0·05	0·01	72·9	93·0

Average tailing: Au 0·06 oz./ton, Cu 0·009 per cent.

AMALGAMATION

Four amalgamation tests were made on samples, which were dry-ground through given meshes.

Test No.	Mesh	Heads, oz./ton	Tails, oz./ton	Per cent recovery
8	−14	0·38	0·27	29·0
9	−35	0·38	0·28	26·3
10	−48	0·38	0·21	44·7
11	−65	0·38	0·20	47·4

AMALGAMATION AND FLOTATION OF THE AMALGAMATION TAILING

Test No. 7

Ore 2,000 grammes at −14 mesh, water 2,000 grammes. Ground 25 minutes in ball mill, then amalgamated with 200 grammes of mercury for 1 hour. After amalgamation the mercury was separated and the pulp dewatered for flotation. The reagents used for the flotation of the amalgamation tailing were 3 drops of water-gas tar, 0·15 pound per ton amyl xanthate, and 3 drops of pine oil.

Results:

Product	Weight, %	Assay			Per cent of values		
		Au, oz./ton	Cu, %	Ag, oz./ton	Au	Cu	Ag
Heads	100·0	0·38	0·5	0·18	100·0	100·0	100·0
Recovery by amalgamation	....	....	....	....	72·5		
Amalgamation tailing	100·0	0·105	0·5	....	27·5	100·0	
Flotation concentrate	5·6	0·81	8·66	1·46	11·9	97·0	
Flotation tailing	94·4	0·063	0·02	....	15·6	3·0	
Totals	100·0				100·0	100·0	

A screen test on final flotation tailing gives:—

Product	Weight, %	Assay, Au, oz./ton
+150	2·5	0·11
−150+200	3·8	0·12
−200	93·7	0·16
Totals	100·0	0·063

Recoveries: Au 84·4 per cent, Cu 97 per cent.

Large-scale Continuous Tests

Three tests, Nos. 12, 13, and 14, were made using amalgamation followed by flotation of the amalgamation tailings.

The flow-sheet in each test was as follows. The ore, dry-crushed to −10 mesh, was fed at the rate of about 110 pounds per hour into a small rod mill in closed circuit with an Akins classifier. The classifier overflow was passed through an amalgamator and over an amalgamation plate, 12 inches wide by 11 feet long. The plate tailings were pumped into a small contact tank, where the reagents were added, then floated in a small 6-cell Fahrenwald flotation machine arranged as follows. The pulp entered No. 2 cell and a concentrate was taken from cells Nos. 2 and 3. This concentrate was pumped into No. 1 cell from which the final concentrate was taken. The rougher concentrates from cells Nos. 4, 5, and 6 flowed into the bottom of No. 3 cell. Therefore, tailing was discharged from No. 6 cell.

Test No. 12

The flotation tailing was screened on 200 mesh.

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	8·3	0·07	0·01	11·2	4·2
−200	91·7	0·05	0·02	88·8	95·8

Average flotation tailing: Au 0·052 oz./ton, Cu 0·019 per cent.
Assay of flotation concentrate: Au 4·04 oz./ton, Cu 9·45 per cent.

Recoveries:

I. *Gold*

(*a*) *By amalgamation*—
Head 0·38 oz./ton; plate tailing 0·20 oz./ton; Au recovered 47·4 per cent.

(*b*) *By concentration*—
Head (plate tailing) 0·20 oz./ton; concentrate, 4·04 oz./ton; flotation tailing 0·052 oz./ton; recovery 74·7 per cent.

(*c*) *Total recovery*—
Amalgamation 47·4 per cent; concentration 39·3 per cent; total 86·7 per cent.

II. *Copper*

Heads (plate tailing) 0·47 per cent; concentrate 9·45 per cent; flotation tailing 0·019 per cent; recovery 96·4 per cent.
Ratio of concentration calculated from copper assays 21:1.
Reagents: amyl xanthate 0·20 lb./ton; pine oil 0·05 lb./ton.

Test No. 13

The plate tailing was screened on 200 mesh.

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	25·9	0·20	0·10	23·0	4·5
−200	76·1	0·21	0·67	77·0	95·5

Average plate tailing: Au 0·21 oz./ton, Cu 0·53 per cent.

The flotation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	25·0	0·11	0·027	34·3	18·5
−200	75·0	0·07	0·040	65·7	81·5

Average flotation tailing: Au 0·08 oz./ton, Cu 0·037 per cent.

Tabling the flotation tailing gives:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Table concentrate	12·9	0·133	0·011	21·9	3·0
Table tailing +200	12·2	0·11	0·03	17·0	7·8
Table tailing −200	35·5	0·08	0·02	36·1	22·6
Table slimes	39·4	0·05	0·08	25·0	66·6

The table concentrate was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	74·1	0·13	0·011	72·7	75·9
−200	25·9	0·14	0·01	27·3	24·1

Assay of flotation concentrate: Au 3·16 oz./ton, Cu 12·37 per cent.

Recoveries:

I. *Gold*

(a) *By amalgamation—*
Head 0·38 oz./ton; plate tailing 0·21 oz./ton; Au recovered 44·75 per cent.

(b) *By concentration—*
Head (plate tailing) 0·21 oz./ton; concentrate 3·16 oz./ton; flotation tailing 0·08 oz./ton; recovery 63·1 per cent.

(c) *Total recovery—*
Amalgamation 44·75 per cent; concentration 34·9 per cent; total 79·65 per cent.

II. *Copper*

Head 0·48 per cent; concentrate 12·37 per cent; flotation tailing 0·04 per cent; recovery 92·0 per cent.
Ratio of concentration calculated from copper assays 28:1.
Reagents: soda ash 1·5 lb./ton, cyanide 0·1 lb./ton, xanthate 0·2 lb./ton, pine oil 0·05 lb./ton

Test No. 14

The plate tailing was screened on 200 mesh.

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	27·2	0·19	0·13	26·2	9·0
−200	72·8	0·20	0·49	73·8	91·0

Average plate tailing: Au 0·20 oz./ton, Cu 0·39 per cent.

The flotation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	21·2	0·11	0·005	33·0	12·2
−200	78·8	0·06	0·01	67·0	87·8

Average flotation tailing: Au 0·071 oz./ton, Cu 0·01 per cent.

Tabling the flotation tailing gives:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Table concentrate	10·5	0·127	....	18·6	....
Tailing +200	8·7	0·10	....	12·1	....
Tailing −200	45·9	0·07	....	44·8	....
Slimes	34·9	0·05	....	24·5	....

The table concentrate was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	67·0	0·13	....	68·8	....
−200	33·0	0·12	....	31·2	....

Assay of flotation concentrate: Au 3·20 oz./ton, Cu 9·50 per cent.

Recoveries.

I. *Gold*

(*a*) *By amalgamation*—
Head 0·38 oz./ton, plate tailing 0·20 oz./ton, recovery 47·4 per cent.

(*b*) *By concentration*—
Head (plate tailing) 0·20 oz./ton, concentrate 3·20 oz./ton, flotation tailing 0·071 oz./ton, recovery 63·1 per cent.

(*c*) *Total recovery*—
Amalgamation 47·4 per cent, concentration 34·7 per cent, total 82·1 per cent.

II. *Copper*

Head 0·39 per cent, concentrate 9·50 per cent, flotation tailing 0·01 per cent; recovery 98·4 per cent.
Ratio of concentration calculated from copper assays 24·7:1.
Reagents: sodium sulphide 0·30 oz./ton, xanthate 0·20 lb./ton, pine oil 0·05 lb./ton.

SUMMARY OF RESULTS

Amalgamation. A series of amalgamation tests, Nos. 8, 9, 10, was made on ore ground dry to different sizes giving a minimum recovery of 29 per cent at −14 mesh; and at −65 mesh a maximum recovery of 47·4 per cent. One batch test, No. 7, was made on ore ground in a ball mill with water to pass 100 mesh and 93·7 per cent −200 mesh. The recovery in this test was high amounting to 72·5 per cent. In Tests Nos. 12, 13, and 14, which were continuous tests made using a feed rate of 100 pounds of ore an hour; the ore was ground in a ball mill and passed over stationary plates. With the feed to the plates 91·7 per cent −200 mesh, 47·4 per cent of the gold was recovered on the plates and in the amalgamator, but with the feed 76·1 per cent −200 mesh, only 44·75 per cent was recovered.

Flotation. By straight flotation, Test No. 1, grinding 84·1 per cent −200 mesh, 71·4 per cent of the gold was recovered in a concentrate containing 93·8 per cent of the copper and assaying 10·2 per cent copper with a recovery of 93·8 per cent of the copper. Ratio of concentration, 20·4: 1. By floating 1·8 per cent more as concentrate the ratio of concentration was 14·93 : 1, and the recovery of gold was increased to 75·9 per cent. The same grinding in Test No. 2 gave 76·0 per cent recovery of the gold and 97·9 per cent of the copper in a concentrate assaying 8·52 per cent copper and 5·08 ounces gold per ton. The ratio of concentration was 18·7 : 1.

In Test No. 3 coarse grinding was used and only 72·9 per cent of the gold was recovered, in a combined middling and concentrate. In Test No. 4 grinding 69·6 per cent −200 mesh and floating a dirty concentrate, 73 per cent of the gold was recovered with a ratio of concentration 12·7 : 1.

In Test No. 5 grinding 60 per cent −200 mesh, 76·8 per cent of the gold was recovered but the ratio of concentration was 9·6 : 1 and the grade of the copper concentrate was 2·80 ounces gold per ton and 4·7 per cent copper.

Flotation Test No. 6 was run in a different manner. The ore was first floated at 20 mesh and the tailing then reground and again floated. The results were 86·6 per cent recovery of the gold and 98·3 per cent of the copper in a concentrate assaying 4·65 per cent copper and 3·05 ounces in gold per ton. The ratio of concentration was approximately 8·93 : 1.

Combination of Amalgamation and Flotation. In the small-scale batch test (No. 7) the ore was ground very fine, 94·7 per cent −200 mesh, amalgamated and then floated. Amalgamation recovered 72·5 per cent of the gold. Flotation recovered 11·9 per cent additional gold, making a total recovery of 84·4 per cent of the gold. The recovery of copper was 97 per cent in a concentrate containing 8·60 per cent copper and 0·81 ounce gold per ton. Ratio of concentration was 17·85 : 1.

Three continuous tests were made. In each test the reagents were different. The amalgamation results varied somewhat with the grinding. In Test No. 12 the grinding was 91·7 per cent −200 mesh and 47·4 per cent of the gold was recovered on the plates. In Test No 13 the grinding was 75 per cent −200 mesh and only 44·75 per cent of the gold was recovered on the plates. In Test No. 14 the grinding was 78 per cent −200 mesh and 47·4 per cent was recovered. The flotation results were also affected by the degree of grinding but to a greater extent by the reagents used. In Test No. 12 with xanthate and pine oil in a natural pulp 39·3 per cent additional gold was recovered making a total recovery of 86·7 per cent of the gold. The copper concentrate assayed 9·45 per cent copper and 4·04 ounces gold per ton. The ratio of concentration was

21 : 1. In Test No. 13 with the reagents changed, using cyanide and soda ash only 34·9 per cent additional gold was recovered thus making a total recovery per cent of 79·65. The copper concentrate was higher in grade than in the preceding test and assayed 12·37 per cent copper and 3·16 ounces gold per ton. Copper recovery was 92 per cent and ratio of concentration 28 : 1.

In Test No. 14 the reagents were again changed and sodium sulphide was used. The gold recovery fell off a little as compared with the Test No. 12. Only 34·7 per cent additional gold was recovered, making a total of 82·1 per cent of the gold recovered. The copper concentrate assayed 9·5 per cent copper and 3·20 ounces gold per ton. The recovery of copper was 98·4 per cent and the ratio of concentration 24·7 : 1.

Product	Test No. 12	Test No. 13	Test No. 14
Recovery of gold by—			
Amalgamation	47·4%	44·75%	47·4%
Copper concentration	Cu 9·45% Au 4·04 oz/ton	Cu 12·37% Au 3·16oz/ton	Cu 9·5% Au 3·2 oz./ton
Recovery of copper in concentrate	96·4%	92·0%	98·4%
Recovery of gold in concentrate	39·3%	34·9%	34·7%
Total recovery of gold by amalgamation and flotation	86·7%	79·65%	82·1%
Ratio of concentration	21:1	28:1	24·7:1

CONCLUSIONS

(1) Fine grinding is essential (see flotation tests, Nos. 1 to 6). The fact is the finer the ore is ground the higher the recovery of gold.

(2) Straight flotation does not recover sufficient gold.

(3) Straight amalgamation does not recover sufficient gold.

(4) Two-stage flotation, coarse and fine, gives good recoveries of gold but the ratio of concentration is small, and the freight and treatment charges on the concentrate would be high.

(5) Amalgamation followed by flotation gives the best recoveries and the highest ratio of concentration.

RECOMMENDATIONS

The writer recommends grinding to at least 80 per cent −200 mesh, preferably 90 per cent; amalgamation followed by flotation of the amalgamation tailing. By this method over 80 per cent of the gold can be recovered, 45 per cent of which will be obtained as amalgam. A concentrate will be made containing over 9 per cent copper and about 4·0 ounces gold per ton. The copper recovery will be 95 per cent and the ratio of concentration 21 : 1.

The difficulty in this flow-sheet will be the control of the dilution for flotation. The pulp for flotation should not exceed 3 : 1. It would also be useless to try to amalgamate the classifier overflow, as nothing but very fine gold would find its way over the classifier, and it is a well known fact that fine gold is almost impossible to catch on amalgamation plates. Amalgamation will, therefore, have to be done between the ball mill and the classifier. In this way the circulating load on the grinding circuit of the ball mill and classifier will pass over the plates. In order to keep the coarse material in the ball mill discharge from scouring the plates, it would be well to pass the ball mill discharge over a 14-mesh vibrating screen, and return the oversize to the mill. The undersize is passed over the plates and then to a bowl classifier, the returns from the bowl going back to the mill, the overflow to flotation.

The difficulty in maintaining the proper dilution for flotation is now apparent. However, by using a very steep slope on the plates, and in view of the fact that owing to the plates being in closed circuit with ball mill and classifier, a particle of gold would have more than one chance of being caught by the plates, it should be possible to amalgamate in a 2½ : 1 pulp. This leaves one-half part of water to be used as back wash on the classifier. It is possible that this may not be sufficient water, therefore space should be allowed for the installation of a thickener to dewater the pulp for flotation. It is suggested that Allen cones be used and the overflow returned to the grinding circuit. A Genter thickener would be better, but is much more expensive.

If any other type of flotation machine other than the mechanical agitation cell be used, a mechanical agitator for mixing the pulp and reagents should precede the flotation.

Report No. 339

EXPERIMENTAL TESTS ON COPPER-BEARING GOLD ORE FROM THE CENTRAL MANITOBA MINES, LIMITED, WADHOPE, MAN.

J. S. Godard

Shipment. A shipment of approximately one ton of ore was received April 11, 1929, from I. M. Marshall, Superintendent of the Central Manitoba Mines, Ltd. It consisted of 1,500 pounds from the Kitchener vein, 100 pounds from Rogers shaft, 100 pounds from Tene 6 shaft, and 200 pounds of No. 2 ore—a mixture of ores from Rogers and Tene 6 shafts in the proportion in which it was proposed to mine them.

Characteristics of the Ores. There is very little difference in the characteristics of any of the ores. All are copper-bearing gold ores in which the gold is largely in the free form, some, however, is associated with chalcopyrite and pyrite which are present in small amounts in all the samples. While some pyrrhotite is also present indications are that it does not carry gold. The gangue is siliceous.

Analyses of the Samples.	Gold, oz./ton	Copper, per cent
Kitchener ore	0·64	0·42
Rogers ore	0·31	0·49
Tene 6 ore	0·30	0·51
No. 2 ore	0·35	0·60

Purpose of Experimental Tests. In the present mill, the cyanide process is used to extract the gold. Good extractions are obtained, but with a relatively high consumption of cyanide.

Fouling of the cyanide solution occurs, due to the solubility of the copper and this necessitates running to waste some of the solution as well as causing some difficulties in precipitation and subsequent refining of the gold.

Considering the existing difficulties due to the presence of copper and the possibilities of an increase in the copper content of the various ores as mining proceeds to greater depths, it was thought that other milling methods, than that at present in use, should be investigated.

Experimental Tests

FLOTATION—TABLING THE FLOTATION TAILING

Test No. 1

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Flotation concentrate	4·0	8·36	10·80	79·8	97·5
Table "	1·7	3·32	0·11	13·5	0·4
Table tailing	57·9	0·03	0·01	4·1	1·3
Slimes	36·4	0·03	0·01	2·6	0·8

Ore ground: 12 per cent +200 mesh.
Head sample: Au 0·42 oz./ton, Cu 0·44 per cent.

Test No. 2

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Flotation concentrate	7·9	3·72	5·85	57·3	97·8
Table "	2·2	8·16	0·04	34·9	0·2
Table tailing +200	7·4	0·10	0·01	1·4	0·2
Table tailing −200	55·2	0·04	0·01	4·3	1·2
Slimes	27·3	0·04	0·01	2·1	0·6

Head sample: Au 0·51 oz./ton, Cu 0·4 per cent.

AMALGAMATION

Test No. 3

The amalgamation tailing was screened on 100 and 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+100	6·6	0·20	....	17·8	....
+200	25·1	0·08	....	27·0	....
−200	68·3	0·06	....	55·2	....

Head sample: Au 0·50 oz./ton, tailing 0·074, recovery 85·3 per cent.

Test No. 4

The amalgamation tailing was screened on 200 mesh.

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu; %	Au	Cu
+200	17·1	0·09	....	21·0	....
−200	82·9	0·07	....	79·0	....

Head sample: Au 0·50 oz./ton, tailing 0·073, recovery 85·4 per cent.

AMALGAMATION, FLOTATION, AND TABLING

Test No. 5

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Flotation concentrate	3·7	1·02	11·87	52·4	96·7
Table "	6·2	0·22	0·01	18·9	0·1
Table tailing	55·2	0·028	0·02	21·5	2·4
Slimes	34·9	0·015	0·01	7·2	0·8

Amalgamation tailing from products: Au 0·072 oz./ton, Cu 0·45 per cent.

Test No. 6

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Flotation concentrate	3·4	1·18	12·62	49·6	97·1
Table "	5·4	0·19	0·04	12·7	0·5
Table tailing	51·2	0·036	0·013	22·8	1·5
Slimes	40·0	0·03	0·01	14·9	0·9

Amalgamation tailing from products: Au 0·081 oz./ton, Cu 0·44 per cent.

Test No. 7

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Flotation concentrate	3·2	1·26	13·26	32·6	94·5
Table "	3·7	1·38	0·07	41·4	0·6
Table tailing	50·1	0·03	0·01	12·1	1·1
Slimes	43·0	0·04	0·04	13·9	3·8

Amalgamation tailing from products: Au 0·12 oz./ton, Cu 0·45 per cent.

Distribution of gold in per cent—Tests Nos. 5, 6, and 7

(Results based on: Au 0·50 oz/ton head)

No.	Amalgamation	Flotation concentrate	Table concentrate	Total recovery
5	85·6	7·5	2·7	95·8
6	83·8	8·0	2·1	93·9
7	75·2	8·1	10·3	93·6

FLOTATION, AND CYANIDATION OF THE FLOTATION TAILING

Test No. 8

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	2·9	12·76	14·57	69·9	93·7
Tailing	97·1	0·164	0·029	30·1	6·3

Head sample from products: Au 0·53 oz./ton, Cu 0·45 per cent.

The cyanidation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	13·2	0·30	....	82·0	....
−200	86·8	0·01	....	18·0	....

Cyanide tailing: Au 0·048 oz./ton.

Test No. 9

Results:

Product	Weight, %	Assay		Per cent of values	
		Au,oz./ton	Cu, %	Au	Cu
Concentrate	3·8	12·36	11·86	70·7	97·9
Tailing	96·2	0·21	0·011	29·3	2·1

Head sample from products: Au 0·66 oz./ton, Cu 0·46 per cent.

The cyanidation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	11·1	0·02	....	19·8	....
−200	88·9	0·01	....	80·2	....

Cyanide tailing: Au 0·011 oz./ton.

Test No. 10

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	2·7	10·08	15·00	36·8	95·4
Tailing	97·3	0·48	0·02	63·2	4·6

Head sample: Au 0·74 oz./ton, Cu 0·42 per cent.

The cyanidation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	12·7	0·08	....	54·0	....
−200	87·3	0·01	....	46·0	....

Cyanide tailing: Au 0·019 oz./ton.

Test No. 11

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	3·6	7·44	12·18	36·1	97·9
Tailing	96·4	0·49	0·01	63·9	2·1

Head sample: Au 0·74 oz./ton, Cu 0·45 per cent.

The cyanidation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	9·4	0·24	....	62·4	....
−200	90·6	0·015	....	37·6	....

Cyanide tailing: Au 0·36 oz./ton.

Summary:

No.	Heads to cyanidation	Cyanidation tailing	KCN %	KCN consumption, lb./ton	Time, hrs.	Dilution	Extraction, %	Recovery		Total
								Flotation	Cyanidation	
8	0·164	0·048	0·05	0·68	41	1:2	70·7	69·9	21·3	91·2
9	0·210	0·011	0·05	0·85	41	1:2	94·7	70·7	27·7	98·4
10	0·480	0·019	0·05	0·80	41	1:2	95·9	36·8	60·6	97·4
11	0·49	0·036	0·05	0·60	41	1:2	92·0	36·1	58·8	94·9

AMALGAMATION AND FLOTATION OF THE AMALGAMATION TAILING

Test No. 12

Ore 100 pounds, −18 mesh, fed to rod mill in closed circuit with a classifier; 1,000 grammes of mercury poured into the well of the classifier. The classifier overflow ran down an amalgamation plate, 77 inches long, with a ⅞-inch cascade in the centre, and a slope of 1 inch in 3 inches. The plate tailing was floated in batch lots in a Fahrenwald machine, and the rougher concentrates cleaned in a Ruth machine. Middlings were dewatered and added to Fahrenwald machine.

The amalgamation tailing was screened on 200 mesh.

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	8·4	0·08	0·25	10·9	5·2
−200	91·6	0·06	0·42	89·1	94·8

Amalgamation tailing from products: Au 0·062 oz./ton, Cu 0·41 per cent.

A sample of the flotation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	11·3	0·05	0·03	17·6	27·6
−200	88·7	0·03	0·01	82·4	72·4

Flotation tailing from products: Au 0·032 oz./ton, Cu 0·012 per cent.

A second sample of flotation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	14·5	0·06	....	33·7	....
−200	85·5	0·02	....	66·3	....

Flotation tailing from products: Au 0·026 oz./ton.

A grab sample of the flotation tailing was tabled.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	3·5	0·10	0·08	18·4	15·3
Tailing +200	5·9	0·03	0·03	9·5	9·9
Tailing −200	44·8	0·01	0·01	23·7	24·6
Slimes	45·8	0·02	0·02	48·4	50·2

Flotation tailing from products: Au 0·019 oz./ton, Cu 0·018 per cent.

A sample of the flotation middling was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	1·6	0·41	1·30	3·2	4·4
−200	98·4	0·20	0·46	96·8	95·6

Flotation middling from products: Au 0·20 oz./ton, Cu 0·47 per cent.

Summary of Tests:

Analysis of head sample: Au 0·51 oz./ton, Cu 0·40 per cent.
Analysis of flotation concentrate: Au 3·50 oz./ton, Ag 5·80 oz./ton.
Cu 27·96 per cent, Fe 29·22 per cent, and SiO_2 7·02 per cent.
Heads to amalgamation: Au 0·51 oz./ton.
Amalgamation tailing: Au 0·062 oz./ton.
Au amalgamated: 87·8 per cent.
Recoveries, calculated from assays, from amalgamation tailing: Au 53·7 per cent, Cu 97·0 per cent.
Total gold recovery: 94·3 per cent.
Ratio of concentration—calculated from copper assays: 72:1.

Rogers Ore

Head sample: Au 0·31 oz./ton, Ag 0·20 oz./ton, Cu 0·49 per cent, Fe 3·44 per cent, Insol. 85·62 per cent.

FLOTATION—TABLING THE FLOTATION TAILING

Test No. 1

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Flotation concentrate	4·1	2·80	12·46	47·2	95·8
Table “	3·8	2·54	0·23	39·7	1·6
Table tailing +200	7·4	0·33	0·02	10·0	0·3
Table “ −200	65·9	0·01	0·01	2·7	1·2
Slimes	18·8	0·005	0·03	0·4	1·1

Head sample: Au 0·24 oz./ton, Cu 0·53 per cent.

Test No. 2

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Flotation concentrate	9·9	1·74	5·14	53·7	97·9
Table “	3·6	3·93	0·07	44·0	0·5
Table tailing +200	9·1	0·04	0·01	1·1	0·2
Table “ −200	49·2	0·005	0·01	0·8	0·9
Slimes	28·2	0·005	0·01	0·4	0·5

Head sample: Au 0·32 oz./ton, Cu 0·52 per cent.

AMALGAMATION

Test No. 3

The amalgamation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	12·0	0·11	0·27	27·3	6·1
−200	88·0	0·04	0·53	72·7	93·9

Head sample: Au 0·31 oz./ton.
Amalgamation tailing: Au 0·048 oz./ton.
Au amalgamated: 84·5 per cent.

AMALGAMATION—FLOTATION OF THE AMALGAMATION TAILING

Test No. 4

The flotation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	11·1	0·02	0·04	33·3	7·6
−200	88·9	0·005	0·06	66·7	92·4

Flotation tailing: Au 0·007 oz./ton, Cu 0·058 per cent.

Results of Flotation:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	8·0	0·18	5·34	69·2	88·9
Tailing	92·0	0·007	0·058	30·8	11·1

Amalgamation tailing: Au 0·021 oz./ton, Cu 0·48 per cent.
Recovery—amalgamation 93·2 per cent; flotation 4·7 per cent, total 97·9 per cent.

Test No. 5

Flotation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	12·9	0·15	0·02	81·5	23·0
−200	87·1	0·005	0·01	18·5	77·0

Flotation tailing: Au 0·024 oz./ton, Cu 0·011 per cent.

Results of Flotation:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	7·4	0·26	6·40	46·4	97·9
Tailing	92·6	0·024	0·011	53·6	2·1

Amalgamation tailing: Au 0·041 oz./ton, Cu 0·48 per cent.
Recovery—amalgamation 86·8 per cent, flotation 6·2 per cent, total 93 per cent.

Tene 6 Ore

Head sample: Au 0·30 oz./ton, Ag 0·21 oz./ton, Cu 0·51 per cent, Fe 4·04 per cent. Insoluble 79·60 per cent.

FLOTATION—TABLING THE FLOTATION TAILING

Test No. 1

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Flotation concentration	6·7	1·82	8·10	47·5	97·4
Table "	3·6	3·38	0·08	47·4	0·5
Table tailing +200	4·4	0·20	0·01	3·4	0·1
Table " −200	59·7	0·005	0·01	1·2	1·1
Slimes	25·6	0·005	0·02	0·5	0·9

Head sample: Au 0·26 oz./ton, Cu 0·56 per cent.

Test No. 2

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Flotation concentrate	10·6	1·76	4·88	74·1	98·1
Table "	3·7	1·58	0·04	23·2	0·3
Table tailing +200	4·3	0·01	0·01	0·2	0·1
Table " −200	42·2	0·01	0·01	1·7	0·8
Slimes	39·2	0·005	0·01	0·8	0·7

Head sample: Au 0·25 oz./ton, Cu 0·53 per cent.

AMALGAMATION

Test No. 3

Amalgamation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	7·6	0·07	0·37	53·5	5·2
−200	92·4	0·005	0·55	46·5	94·8

Head sample: Au 0·31 oz./ton.
Amalgamation tailing: Au 0·01 oz./ton.
Au amalgamated: 96·6 per cent.

AMALGAMATION AND FLOTATION OF THE AMALGAMATION TAILING

Test No. 4

The flotation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	8·5	0·09	0·02	62·6	15·6
−200	91·5	0·005	0·01	37·4	84·4

Flotation tailing: Au 0·012 oz./ton, Cu 0·011 per cent.

Results of Flotation:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	5·0	0·40	10·67	63·7	98·2
Tailing	95·0	0·012	0·011	36·3	1·8

Amalgamation tailing: Au 0·031 oz./ton, Cu 0·59 per cent.
Recovery: amalgamation 89·7 per cent, flotation 6·6 per cent, total 96·3 per cent.

Test No. 5

The flotation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	9·0	0·15	0·04	60·0	28·3
−200	91·0	0·01	0·01	40·0	71·7

Flotation tailing: Au 0·023 oz./ton, Cu 0·013 per cent.

Results of Flotation:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	11·6	0·26	4·59	59·8	97·8
Tailing	88·4	0·023	0·013	40·2	2·2

Amalgamation tailing: Au 0·051 oz./ton, Cu 0·53 per cent.
Recovery: amalgamation 83 per cent, flotation 10 per cent, total 93 per cent.

No. 2 Ore

Head sample: Au 0·35 oz./ton, Ag 0·7 oz./ton, Cu 0·60 per cent, Fe 5·57 per cent. Insoluble 76·89 per cent.

FLOTATION—TABLING THE FLOTATION TAILING

Test No. 1

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Flotation concentrate	4·7	2·44	12·64	57·0	98·9
Table "	4·9	1·52	0·03	37·0	0·3
Table tailing +200	3·2	0·15	0·005	2·4	
Table " −200	56·3	0·01	0·005	2·8	0·5
Slimes	30·9	0·005	0·005	0·8	0·3

Head sample: Au 0·20 oz./ton (low), Cu 0·60 per cent.

Test No. 2

Results:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Flotation concentrate	9·5	2·68	6·31	65·5	99·0
Table "	5·8	2·20	0·03	31·6	0·3
Table tailing +200	4·7	0·16	0·005	1·9	
Table " −200	43·8	0·005	0·005	0·6	0·4
Slimes	36·2	0·005	0·005	0·4	0·3

Head sample: Au 0·40 oz./ton, Cu 0·61 per cent.

AMALGAMATION

Test No. 3

The amalgamation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	11·6	0·08	....	23·1	....
−200	88·4	0·035	....	76·9	....

Amalgamation tailing: Au 0·04 oz./ton.
Au amalgamated: 88·5 per cent.

AMALGAMATION AND FLOTATION OF THE AMALGAMATION TAILING

Test No. 4

The flotation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	10·0	0·16	0·01	78·0	3·6
−200	90·0	0·005	0·03	22·0	96·4

Flotation tailing: Au 0·021 oz./ton, Cu 0·028 per cent.

Results of Flotation:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	10·5	0·25	5·80	58·3	97·6
Tailing	89·5	0·021	0·028	41·7	2·4

Amalgamation tailing: Au 0·045 oz./ton, Cu 0·62 per cent.
Recovery: amalgamation 87·0 per cent; flotation 7·6 per cent, total 94·6 per cent.

Test No. 5

The flotation tailing was screened on 200 mesh.

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
+200	9·0	0·07	0·01	57·8	9·0
−200	91·0	0·005	0·01	42·2	91·0

Flotation tailing: Au 0·011 oz./ton, Cu 0·01 per cent.

Results of Flotation:

Product	Weight, %	Assay		Per cent of values	
		Au, oz./ton	Cu, %	Au	Cu
Concentrate	7·4	0·26	8·14	65·3	98·5
Tailing	92·6	0·011	0·01	34·7	1·5

Amalgamation tailing: Au 0·029 oz./ton, Cu 0·61 per cent.
Recovery: amalgamation 91·7 per cent, flotation 5·4 per cent, total 97·1 per cent.

CONCLUSIONS

Other than the application of the cyanide process direct to these ores, there are three main methods of treating such ores. These methods are:

1. Amalgamation—flotation of the amalgamation tailing.
2. Flotation followed by tabling the flotation tailing.
3. Flotation followed by cyanidation of the flotation tailing.

Amalgamation—Flotation of the Amalgamation Tailing. By this method the recoveries of gold were from 90 to 95 per cent, of which about 85 per cent was recovered by amalgamation.

In Test No. 12 on Kitchener ore, which may be considered as typical of the results obtainable in practice on ore of this grade, the flotation concentrate after amalgamation assayed, gold 3·50 ounces per ton, silver 5·80 ounces per ton, copper 27 per cent, iron 29 per cent, and silica 7 per cent.

In obtaining such a concentrate the ratio of concentration was 72 : 1.

Flotation—Tabling the Flotation Tailing. This method of treatment showed an average recovery of 94 per cent of the gold.

Owing to the erratic flotative property of free gold great differences occurred in the recoveries of the gold by flotation. The free gold dropped during flotation was later recovered in the table concentrate.

As the minus 200-mesh material contained very little gold after flotation it is possible that classification of the flotation tailing might be practised before tabling. Such a step would lower the costs.

The table concentrates contained very little copper and might possibly be profitably treated by cyanidation in the mill.

Flotation—Cyanidation of the Flotation Tailing. This method of treatment was tried on Kitchener ore only.

In this method the purpose of flotation is to remove the chalcopyrite, with as little gold as possible, previous to cyanidation thus allowing the greater part of the gold output of the mill to be in the form of bullion.

As in all tests where flotation precedes either tabling or cyanidation erratic recoveries of the gold were made in the flotation concentrate. In Tests Nos. 10 and 11 less than 37 per cent of the gold was removed by flotation and about 60 per cent was recovered in the cyanidation.

After the removal of the chalcopyrite the cyanide consumption was reduced to about three-quarters of a pound of KCN per ton of tailings.

The successful application of this method requires a rather complicated flow-sheet and good control of the flotation circuit, otherwise the feed to the cyanidation unit might be too low in gold to permit economical operation of that unit.

Report No. 340

THE SELECTIVE FLOTATION OF A LEAD-ZINC ORE FROM THE HASLAT-DUCK LAKE MINES, LTD., SCHREIBER, ONT.

A. K. Anderson

Shipment. One box of ore, net weight 120 pounds, was received by freight on November 21, 1929, from the Haslat-Duck Lake Mines, Ltd., a property about 15 miles from Schreiber, Ont. The Head Office of the company is at 202 Hamilton Trust Bldg., Toronto, Mr. N. F. Stewart, Secretary-Treasurer.

Characteristics of the Ore. The ore consisted of massive, fine-grained, intimately associated sulphides of lead, zinc, and iron contained in a siliceous gangue.

Purpose of Experimental Tests. The shipment was made for the purpose of ascertaining whether the ore is amenable to concentration by flotation.

Sampling and Analysis. The lot was passed through jaw crushers and rolls and crushed to pass 14 mesh. The material was then cut by a Jones riffle sampler and a representative portion secured for analysis. This showed the shipment to contain:—

Lead....................	4·94 per cent	Tin......................	nil
Zinc....................	8·05 "	Gold....................	0·11 oz./ton
Copper..................	0·19 "	Silver..................	2·06 "

EXPERIMENTAL TESTS

All test work was along the lines of selective flotation, endeavouring to secure a lead concentrate carrying the maximum amount of the precious metals, and a marketable zinc concentrate.

Three tests were made investigating the effects of various reagents.

Test No. 1

Ore, 2,000 grammes −14 mesh, 4·0 pounds soda ash and 0·15 pound cyanide per ton were ground with 2,000 grammes water in a ball mill to pass 87 per cent −200 mesh.

After grinding, the pulp was transferred to a mechanically agitated flotation machine, and potassium xanthate at the rate of 0·10 pound per ton of ore and pine oil 0·07 pound were added, and a lead concentrate removed. This product was cleaned, yielding concentrate and middling.

Copper sulphate 1·0 pound per ton of ore was then added and the pulp conditioned for three minutes after which a zinc concentrate was removed by the addition of 0·05 pound potassium xanthate and 0·07 pound pine oil per ton. This product also was cleaned.

Test No. 1

Product	Weight, %	Assay					Per cent of values				
		Pb, %	Zn, %	Cu, %	Au, oz./ton	Ag, oz./ton	Pb	Zn	Cu	Au	Ag
Heads............	....	4·94	8·05	0·19	0·11	2·06	.....	.....	.....	.....	.....
Lead concentrate.	5·53	59·50	5·90	2·46	0·55	19·95	69·1	4·2	67·4	24·7	53·6
Lead middling....	7·45	16·82	11·86	0·52	0·40	6·96	26·3	11·4	19·2	24·2	25·2
Zinc concentrate..	11·46	0·17	48·33	0·14	0·02	0·38	0·4	71·4	7·9	1·9	2·1
Zinc middling....	10·93	0·73	4·94	0·07	0·08	0·76	1·7	6·9	3·8	7·1	4·0
Tailing..........	64·63	0·18	0·73	0·005	0·08	0·48	2·5	6·1	1·7	42·1	15·1
	100·00	......	......	......	....	....	100·0	100·0	100·0	100·0	100·0

This test shows that from 100 tons of ore there is produced 12·98 tons of rougher lead concentrate assaying 34·93 per cent lead, 9·32 per cent zinc, $9.20 gold, and 12·49 ounces silver per ton, with recoveries of 95·4 per cent of the lead, 48·9 per cent of the gold, and 78·8 per cent of the silver.

Cleaning operations left too great a bulk of middling. In actual mill operation, this product would be returned to the circuit, resulting in the recovery of the contained values in the finished concentrate.

Conditions in this test produced a final lead concentrate assaying 59·5 per cent lead, 5·90 per cent zinc, 2·46 per cent copper, $11.00 in gold, and 19·95 ounces silver per ton.

The zinc flotation shows that from every 100 tons of ore milled there is produced 22·39 tons of rougher concentrate assaying 0·44 per cent lead, 27·14 per cent zinc, $1.00 in gold, and 0·56 ounce silver per ton, with a recovery of 78·3 per cent of the zinc.

Test No. 2

The reagents in this test were changed from those of the preceding test.

Reagents, lb./ton—

To ball mill		To flotation cell			
		Lead		*Zinc*	
Soda ash	4·0	Cresylic acid	0·20	Copper sulphate	1·00
Cyanide	0·20			Potassium xanthate	0·10
Thiocarbanilide	0·08			Pine oil	0·06

Results:

Product	Weight, %	Assay					Per cent of values				
		Pb, %	Zn, %	Cu, %	Au, oz./ton	Ag, oz./ton	Pb	Zn	Cu	Au	Ag
Lead concentrate	3·87	69·45	3·23	1·97	0·65	23·60	54·7	1·6	36·4	23·7	44·1
Lead middling	7·00	27·86	10·72	1·24	0·60	11·36	39·7	9·6	41·4	39·6	38·4
Zinc concentrate	5·02	0·17	58·31	0·21	0·02	0·70	0·2	37·4	5·0	0·9	1·7
Zinc middling	10·60	0·62	28·88	0·20	0·08	0·88	1·3	39·1	10·1	8·0	4·5
Tailing	73·51	0·27	1·31	0·02	0·04	0·32	4·1	12·3	7·1	27·8	11·3
	100·00	......	......	......	....	....	100·0	100·0	100·0	100·0	100·0

This test shows that from 100 tons of feed, there is produced 10·87 tons of rougher lead concentrate assaying 42·67 per cent lead, 8·05 per cent zinc, $12.40 in gold, and 15·72 ounces silver per ton, with recoveries of lead 94·4 per cent, gold 63·3 per cent, and silver 82·5 per cent.

Too dilute a pulp in the cleaning operations resulted in a large amount of middling, and a finished concentrate assaying 69·45 per cent lead, $13.00 in gold, and 23·60 ounces silver per ton.

The zinc flotation shows a recovery of 76·5 per cent of the zinc in a rougher concentrate assaying 0·47 per cent lead, 38·34 per cent zinc, $1.20 in gold, and 0·82 ounce silver per ton. This concentrate on cleaning shows that the grade can be raised to 58·31 per cent zinc.

Test No. 3

The reagents were again changed for this test.

Reagents, lb./ton—

To ball mill		To flotation cell			
		Lead		*Zinc*	
Soda ash	4·0	Aerofloat No. 15	0·14	Copper sulphate	1·00
Cyanide	0·20			Potassium xanthate	0·10
				Pine oil	0·06

Results:

Product	Weight, %	Assay					Per cent of values				
		Pb,%	Zn, %	Cu, %	Au, oz./ton	Ag, oz./ton	Pb	Zn	Cu	Au	Ag
Lead concentrate.	6·78	60·58	3·83	0·85	0·85	21·25	82·7	3·3	25·6	60·1	69·4
Lead middling....	3·09	12·74	8·00	0·30	0·22	5·08	7·9	3·2	4·1	7·1	7·6
Zinc concentrate..	10·35	0·62	52·00	1·00	0·10	1·16	1·3	69·3	46·0	10·8	5·8
Zinc middling....	4·30	1·41	26·47	0·57	0·14	1·64	1·2	14·6	10·9	6·3	3·4
Tailing..........	75·48	0·45	0·98	0·04	0·02	0·38	6·9	9·6	13·4	15·7	13·8
	100·00	......	......	......	....	....	100·0	100·0	100·0	100·0	100·0

Under these conditions, 100 tons of feed produces 9·87 tons of rougher lead concentrate assaying 45·6 per cent lead, 5·13 per cent zinc, $13.00 in gold, and 16·19 ounces silver per ton, with recoveries of 90·6 per cent of the lead, 67·2 per cent of the gold, and 77·0 per cent of the silver.

Cleaning operations were conducted under more satisfactory conditions resulting in a decrease in the weight of middling produced and a concentrate of good grade.

The zinc flotation produced 14·65 tons of rougher zinc concentrate assaying 44·51 per cent zinc and containing 83·9 per cent of the total amount.

The following comparison of the results secured in the rougher concentrates made in the three tests is of interest.

Test No.	Lead concentrate						Zinc concentrate	
	Assay			Per cent of values			Assay	Per cent of values
	Pb %	Au, oz./ton	Ag, oz./ton	Pb	Au	Ag	Zn %	Zn
1......................	34·93	0·46	12·49	95·4	48·9	78·8	27·14	78·3
2......................	42·67	0·62	15·72	94·4	63·3	82·5	38·34	76·5
3......................	45·60	0·65	16·19	90·6	67·2	77·0	44·51	83·9

Converting the above results to dollars and cents with lead at 6·25 cents, zinc at 5·62 cents, gold at $20.00 an ounce, and silver at 50 cents, the following is seen.

	Test No. 1	Test No. 2	Test No. 3
Lead....................................	$5.88	$5.83	$5.59
Zinc....................................	7.08	6.92	7.59
Gold....................................	1.07	1.39	1.48
Silver..................................	0.81	0.85	0.79
	14.84	14.99	15.45

These gross values indicate that under the conditions of Test No. 3, the highest over-all recovery is obtained. This recovery, however, is attributable to the increased amount of zinc recovered in the zinc concentrate, due to a lesser amount lost in the lead concentrate. Test No. 2 shows a higher recovery of lead with approximately an equal recovery of the precious metals as in Test No. 3.

Other factors affect the comparison such as freight and treatment charges on the products. As lead yields a higher return than zinc this metal, therefore, is the important constituent of the ore.

Flotation conditions that will concentrate the precious metals in the lead product are to be striven for, as in subsequent smelting operations these will be recovered.

These tests indicate that the ore is amenable to flotation under ordinary conditions. The ultimate highest monetary return from the ore can best be determined by actual mill operation where the most suitable reagents to apply can be determined.

Report No. 341

AMALGAMATION TESTS ON GOLD ORE FROM THE GRANADA ROUYN MINES, LTD., ROUYN DISTRICT, QUEBEC

J. S. Godard

Shipments. A shipment of 440 pounds was received from the Granada Rouyn Mines, Limited, on November 8, 1929. The shipment consisted of two samples, No. 1 weighing 110 pounds and No. 2, 330 pounds.

Characteristics of the Ore. Both samples were free milling gold ores. No. 1 sample was of comparatively low grade, while No. 2 was a very high-grade ore.

The free gold in No. 1 sample is in a much finer state than in No. 2 sample where it occurred in very coarse flakes, about 25 per cent being large enough to remain on a 10-mesh screen.

The percentage of sulphides in both samples is small, No. 1, however, containing more than No. 2. Iron pyrites and a small amount of arsenopyrite are the chief sulphides in No. 1 sample, and while these two sulphides are the predominating ones in No. 2 sample the relationship of iron to arsenic is more nearly equal than in No. 1 sample.

The gangue material of each sample is silica. The gangue in No. 1 sample is darker in colour than in No. 2, suggesting a higher percentage of basic rock material.

Analysis.

No. 1 sample: Au, 0·34 oz./ton.
No. 2 sample: Au, 7·89 "

EXPERIMENTAL TESTS

Although No. 2 sample was very high-grade ore it is considered to be typical of the mode of occurrence of the gold in the ore which is expected will constitute the mill feed. This type of ore is so obviously an ideal one for amalgamation that no other method for recovery of the gold was attempted.

Sample No. 1

Test No. 1

Amalgamation on Sample Dry Crushed to pass 18 Mesh

Screen Test on Tailing

Mesh	Weight, %	Assay	Per cent of values
+ 28	25·3	0·25	59·8
+ 35	17·1	0·116	18·7
+ 48	9·8	0·06	5·6
+ 65	7·5	0·04	2·8
+100	5·2	0·05	2·5
+150	3·9	0·04	1·5
+200	3·6	0·04	1·3
−200	27·6	0·03	7·8

Average tailing: 0·106 oz./ton.

Test No. 2

Amalgamation on Sample Dry Crushed to pass 28 Mesh

Mesh	Weight, %	Assay	Per cent of values
+ 35	13·5	0·06	18·2
+ 48	18·0	0·05	20·3
+ 65	15·9	0·04	14·4
+100	10·3	0·07	16·2
+150	2·4	0·06	3·2
+200	3·6	0·04	3·2
−200	36·3	0·03	24·5

Average tailing: 0·044 oz./ton.

Test No. 3

Amalgamation on Sample Dry Crushed to pass 35 Mesh

Mesh	Weight, %	Assay	Per cent of values
+ 48	11·6	0·07	17·0
+ 65	24·1	0·05	25·3
+100	10·5	0·05	11·1
+150	4·2	0·04	3·6
+200	7·3	0·05	7·7
−200	42·3	0·04	35·3

Average tailing: 0·048 oz./ton.

Test No. 4

Amalgamation on Sample Dry Crushed to pass 48 Mesh

Mesh	Weight, %	Assay	Per cent of values
+ 65	18·8	0·05	22·6
+100	17·0	0·05	20·5
+150	10·9	0·03	8·0
+200	10·0	0·03	7·2
−200	43·3	0·04	41·7

Average tailing: 0·042 oz./ton.

Test No. 5

Amalgamation on Sample Dry Crushed to pass 65 Mesh

Mesh	Weight, %	Assay	Per cent of values
+100	16·0	0·07	21·5
+150	10·3	0·05	10·0
+200	12·2	0·04	9·4
−200	61·5	0·05	59·1

Average tailing: 0·052 oz./ton.

Test No. 6

Amalgamation on Sample Dry Crushed to pass 100 Mesh

Mesh	Weight, %	Assay	Per cent of values
+150	11·7	0·08	19·5
+200	17·5	0·06	21·8
−200	70·8	0·04	58·7

Average tailing: 0·048 oz./ton.

Summary of Amalgamation Tests on Dry Crushed Samples

Head sample: Au 0·34 oz. / ton.

Test No.	Mesh	Tailing, oz./ton	Recovery, %
1	− 18	0·106	68·9
2	− 28	0·044	67·1
3	− 35	0·048	85·9
4	− 48	0·042	87·7
5	− 65	0·052	84·8
6	−100	0·048	85·9

Test No. 7

Amalgamation—Classification of the Amalgamation Tailing and Tabling the Classifier Oversize

The ore was ground wet to 48 mesh.

Results:

I. Classifier overflow screened on 100 mesh.

Mesh	Weight, %	Assay, oz./ton	Per cent of values
+100	6·2	0·02	4·1
−100	93·8	0·03	95·9

Average assay classifier overflow: Au 0·029 oz./ton.

II. Tabling:

Product		Weight, %	Assay, oz./ton	Per cent of values
Concentrate	+ 65	12·6	0·12	31·1
	− 65	16·3	0·10	33·5
Tailing	+ 48	13·0	0·02	5·5
	+ 65	18·5	0·02	7·6
	+100	16·6	0·01	3·5
	−100	23·0	0·04	18·9

Average assay classifier oversize: Au 0·049 oz./ton.
Average assay of table concentrates: Au 0·11 oz./ton.
Amalgamation tailing (calculated): Au 0·039 oz./ton.

Test No. 8

Amalgamation and Tabling the Amalgamation Tailing

The ore was ground wet to 48 mesh.

Results: Tabling.

Product		Weight, %	Assay, oz./ton	Per cent of values
Concentrate	+ 65	2·2	0·16	4·2
	− 65	5·2	0·20	12·6
Tailing	+ 48	2·7	0·02	0·6
	+ 65	8·0	0·015	1·5
	+100	13·7	0·015	2·6
	−200	36·2	0·02	8·7
Slimes		32·0	0·18	69·8

Average assay of table concentrates: Au 0·19 oz./ton.
Amalgamation tailing (calculated): Au 0·083 oz./ton.

Test No. 9

Amalgamation—Flotation of the Amalgamation Tailing

The ore was ground wet to 48 mesh.

Results:

I. Screening the flotation tailing.

Product	Weight, %	Assay, oz./ton	Per cent of values
+ 48	2·9	0·015	3·0
+ 65	9·5	0·01	7·5
+100	16·1	0·03	35·8
−100	71·5	0·01	53·7

Average assay of flotation tailing: Au 0·013 oz./ton.

II. Flotation.

Product	Weight, %	Assay, oz./ton	Per cent of values
Concentrate	1·9	1·24	64·5
Middling	3·4	0·02	1·9
Tailing	94·7	0·013	33·6

Amalgamation tailing (calculated): Au 0·037 oz./ton.
Flotation concentrate contained: As 4·39 per cent, Fe 37·13 per cent, Pb 0·16 per cent.

Summary of Results of Amalgamation Tests Nos. 7, 8, and 9

The ore was ground wet to 48 mesh.

Head sample: Au 0·34 oz./ton.

Test No.	Tailing	Recovery, %
7	0·039	88·6
8	0·083	75·6
9	0·037	89·1

Tailing in No. 8 evidently high. Refer to table slimes Test No. 8.

Sample No. 2

Head Sample: Au 0·42 oz./ton.

Test No. 1

Amalgamation on Sample Dry Crushed to pass 18 Mesh

Screen test on tailing.

Mesh	Weight, %	Assay, oz./ton	Per cent of values
+ 28	28·8	1·47	62·1
+ 35	13·4	0·68	13·4
+ 48	10·4	0·67	10·2
+ 65	11·8	0·35	6·1
+100	5·2	0·28	2·1
+150	5·4	0·20	1·6
+200	6·5	0·13	1·2
−200	18·5	0·12	3·3

Average tailing: Au 0·68 oz./ton.

Test No. 2

Amalgamation on Sample Dry Crushed to pass 28 Mesh

Mesh	Weight, %	Assay, oz./ton	Per cent of values
+ 35	9·0	0·83	17·2
+ 48	17·8	0·66	27·1
+ 65	17·4	0·31	12·4
+100	16·3	0·22	8·3
+150	3·5	0·16	1·3
+200	9·9	0·15	3·4
−200	26·1	0·50	30·3

Average tailing: Au 0·43 oz./ton.

Test No. 3

Amalgamation on Sample Dry Crushed to pass 35 Mesh

Mesh	Weight, %	Assay, oz./ton	Per cent of values
+ 48	8·3	0·51	7·4
+ 65	13·7	0·45	10·8
+100	17·5	0·27	8·3
+150	9·7	0·19	3·2
+200	14·5	0·16	4·1
−200	36·3	1·04	66·2

Average tailing: Au 0·57 oz./ton.

Test No. 4

Amalgamation on Sample Dry Crushed to pass 48 Mesh

Mesh	Weight, %	Assay, oz./ton	Per cent of values
+ 65	18·8	0·48	13·7
+100	23·9	0·28	10·2
+150	9·8	0·22	3·3
+200	13·2	0·24	4·8
−200	34·3	1·30	68·0

Average tailing: Au 0·66 oz./ton.

Test No. 5

Amalgamation on Sample Dry Crushed to pass 65 Mesh

Mesh	Weight, %	Assay oz./ton	Per cent of values
+100	24·8	0·19	12·4
+150	15·0	0·19	7·5
+200	17·2	0·16	7·3
−200	43·0	0·64	72·8

Average tailing: Au 0·38 oz./ton.

Test No. 6

Amalgamation on Sample Dry Crushed to pass 100 Mesh

Mesh	Weight, %	Assay, oz./ton	Per cent of values
+150	16·9	0·28	4·9
+200	28·8	0·28	8·4
−200	54·3	1·54	86·7

Average tailing: Au 0·964 oz./ton.

Summary of Amalgamation Tests on Dry Crushed Samples

Head sample: Au 4·42 oz./ton.

Test No.	Mesh	Tailing, oz./ton	Recovery, %
1	− 18	0·68	84·7
2	− 28	0·43	90·4
3	− 35	0·57	87·2
4	− 48	0·66	85·2
5	− 65	0·38	91·5
6	−100	0·964	78·2

With the exception of Test No. 1, the −200-mesh amalgamation tailings were all high and to determine if these high tailings were due to the presence of free gold, they were re-amalgamated with the following results:

Test No.	Heads, oz./ton	Tails, oz./ton	Recovery, %
2	0·50	0·24	52·0
3	1·04	0·75	27·9
4	1·30	0·82	36·9
5	0·64	0·37	42·2
6	1·54	1·01	34·4

By increasing the time of amalgamation recoveries of 28 to 52 per cent were made on the −200-mesh material.

Using the assays shown above for the −200-mesh material and recalculating the average tailing for Tests Nos. 2 to 6 inclusive, a new summary of the results is as follows.

Test No.	Mesh	Tailing, oz./ton	Recovery, %
2	− 28	0·37	91·7
3	− 35	0·47	89·5
4	− 48	0·49	88·9
5	− 65	0·26	94·1
6	−100	0·68	84·7

Test No. 7

Amalgamation—Classification of the Amalgamation Tailing and Tabling the Classifier Oversize

Ore ground wet to 48 mesh.
Classifier overflow screened on 100 mesh.

Product	Weight, %	Assay, oz./ton	Per cent of values
+100	11·3	0·07	7·5
−100	88·7	0·11	92·5

Average assay of classifier overflow: Au 0·106 oz./ton.

Results: Tabled.

Product		Weight, %	Assay, oz./ton	Per cent of values
Concentrate	+ 65	4·2	2·06	32·4
	− 65	12·6	0·90	42·5
Tailing	+ 48	10·7	0·10	4·0
	+ 65	26·9	0·06	10·1
	+100	23·4	0·07	5·2
	−100	22·2	....	5·8

Average assay of classifier oversize: Au 0·267 oz./ton.
Average assay of table concentrates: Au 1·19 oz./ton.
Amalgamation tailing (calculated): Au 0·186 oz./ton.

Test No. 8

Amalgamation—Tabling the Amalgamation Tailing

Ore ground wet to 48 mesh.

Results:

Product		Weight, %	Assay, oz./ton	Per cent of values
Concentrate	+ 65	2·2	2·30	23·2
	− 65	5·6	1·18	30·2
Tailing	+ 48	4·0	0·84	15·4
	+ 65	12·4	0·12	6·8
	+100	17·7	0·07	5·7
	−100	41·0	0·07	13·2
Slimes		17·1	0·07	5·5

Average assay of table concentrates: Au 1·50 oz./ton.
Amalgamation tailing (calculated): Au 0·218 oz./ton.

Test No. 9

Amalgamation—Flotation of the Amalgamation Tailing

Screening the flotation tailing.

Results:

Product	Weight, %	Assay, oz./ton	Per cent of values
+ 48	6·0	0·14	10·5
+ 65	15·5	0·14	27·2
+100	21·0	0·10	26·3
−100	57·5	0·05	36·0

Average assay of flotation tailing: Au 0·08 oz./ton.

Flotation.

Product	Weight, %	Assay, oz./ton	Per cent of values
Concentrate	0·6	11·68	38·3
Middling	1·5	2·31	19·0
Tailing	97·9	0·08	42·7

Amalgamation tailing (calculated): Au 0·18 oz./ton.
Flotation concentrate assayed: As 24·87 per cent, Fe 26·52 per cent, Pb 6·75 per cent.

Summary of Results of Amalgamation Tests Nos. 7, 8, and 9

Ore ground wet to 48 mesh.

Head sample: Au 4·42 oz./ton.

Test No.	Tailing, oz./ton	Recovery, %
7	0·19	95·7
8	0·22	95·0
9	0·18	95·9

Tests Nos. 10, 11, and 12

Amalgamation—Flotation of the Amalgamation Tailings

In Tests Nos. 10, 11, and 12 the ore was crushed dry to pass 10 mesh and the metallics on 10 mesh removed. The −10-mesh ore was fed into a small rod mill in closed circuit with a classifier, 1,000 grammes of mercury was poured into the well of the classifier. The classifier overflow passed over an amalgamation plate, 66 inches long, sloped 1 inch in 3 inches, with a $\frac{7}{8}$-inch cascade at the middle. The plate tailings were floated in batch lots in a Fahrenwald machine, and when the concentrates were cleaned a small Ruth machine was used.

Test No. 10

Screening the plate tailing.

Product	Weight, %	Assay, oz./ton	Per cent of values
+ 35	0·9	0·47	2·9
+ 48	3·8	0·26	7·2
+ 65	12·7	0·22	19·3
+100	22·9	0·16	25·3
+200	26·3	0·11	19·9
−200	33·4	0·11	25·4

Average assay of plate tailing: Au 0·145 oz./ton.

Flotation of the amalgamation tailing.

Product	Weight, %	Assay, oz./ton	Per cent of values
Concentrate	0·27	25·08	43·8
Middling	0·38	6·62	16·3
Tailing	99·35	0·062	39·9

Assay of plate tailing (calculated): Au 0·15 oz./ton.

Flotation concentrate assayed: Ag 37·42 oz./ton, Fe 25·61 per cent, As 15·50 per cent, Cu 0·34 per cent, Pb 7·21 per cent, Zn 2·45 per cent.

Screening the flotation tailing.

Product	Weight, %	Assay , oz./ton	Per cent of values
+ 35	0·8	0·08	1·0
+ 48	3·2	0·15	7·8
+ 65	11·1	0·18	32·3
+100	22·4	0·06	21·6
+200	24·6	0·04	15·8
−200	37·9	0·035	21·5

Assay of flotation tailing (calculated): Au 0·062 oz./ton.

Test No. 11

Screening the plate tailing.

Product	Weight, %	Assay, oz./ton	Per cent of values
+ 35	1·1	0·21	2·1
+ 48	4·6	0·19	7·9
+ 65	11·7	0·12	12·6
+100	23·0	0·12	24·8
+200	24·4	0·08	17·6
−200	35·2	0·11	35·0

Average assay of plate tailing: Au 0·11 oz./ton.

Flotation of the amalgamation tailing.

Product	Weight, %	Assay, oz./ton	Per cent of values
Concentrate	0·81	11·86	62·5
Tailing	99·19	0·058	37·5

Assay of plate tailing (calculated): Au 0·15 oz./ton.

Flotation concentrate assayed: Ag 23·04 oz./ton, Cu 0·16 per cent, Zn 1·32 per cent, Pb 4·21 per cent, Fe 12·04 per cent, As 6·52 per cent.

Screening the flotation tailing.

Product	Weight, %	Assay, oz./ton	Per cent of values
+ 35	1·2	0·87	18·1
+ 48	4·6	0·08	6·4
+ 65	14·2	0·06	14·8
+100	19·7	0·055	18·8
+200	24·1	0·04	16·7
−200	36·2	0·04	25·2

Assay of flotation tailing (calculated): Au 0·058 oz./ton.

Test No. 12

Screening the plate tailing.

Product	Weight, %	Assay, oz./ton	Per cent of values
+ 35	1·5	0·19	2·4
+ 48	5·1	0·19	8·0
+ 65	14·8	0·14	17·2
+100	23·0	0·11	21·0
+200	23·7	0·10	19·7
−200	31·9	0·12	31·7

Average assay of plate tailing: Au 0·12 oz./ton.

Flotation.

Product	Weight, %	Assay, oz./ton	Per cent of values
Concentrate	0·95	6·18	47·3
Tailing	99·05	0·066	52·7

Assay of plate tailing (calculated): Au 0·12 oz./ton.

Flotation concentrate assayed: Ag 15·30 oz./ton, Cu 0·20 per cent, Pb 3·02 per cent, Zn 0·93 per cent, Fe 11·22 per cent, As 5·62 per cent.

Screening the flotation tailing.

Product	Weight, %	Assay, oz./ton	Per cent of values
+ 35	1·3	0·12	2·4
+ 48	4·8	0·10	7·3
+ 65	14·5	0·10	22·1
+100	21·2	0·09	29·1
+200	23·2	0·05	17·7
−200	35·0	0·04	21·4

Assay of flotation tailing (calculated): Au 0·066 oz./ton.

Summary—Tests Nos. 10, 11, and 12

Head sample: Au 5·70 oz./ton.

Test	Amalgamation tailing, oz./ton	Flotation tailing, oz./ton	Per cent of total gold recovered		
			Amalgamation	Flotation	Total
10	0·15	0·062	97·3	1·4	98·7
11	0·11	0·058	98·1	1·1	99·2
12	0·12	0·066	97·8	1·0	98·8

CONCLUSIONS

No. 1 Sample. Amalgamation at −35 mesh recovered about 86 per cent of the gold in this sample.

No. 2 Sample. Amalgamation at −35 mesh recovered about 90 per cent of the gold when the samples were crushed dry, and 95 per cent on ore ground wet to −48 mesh. In the larger scale tests (Nos. 10, 11, and 12) 97 to 98 per cent of the gold was recovered by amalgamation. However, this latter figure is high, and considering the grade of ore on which the tests were made, if the calculation were based on 90 to 92 per cent recovery they would more nearly approximate the figures obtainable in good amalgamation practice.

Concentration tests were made on each sample after amalgamation. Both tabling and flotation indicated good recoveries of the gold remaining after amalgamation. In the design of the mill allowance should be made for the possible installation of concentrating equipment, but the actual construction might well be left until the mill has been in operation for some time and tests conducted on the tailings from the ore milled.

It is suggested that stamp milling, using inside and outside amalgamation, be practised on these ores, and the feed to the stamps be about one-inch size and the battery be equipped with 35-mesh screens. Using 1,250-pound stamps the stamp duty should be about 3 tons per 24 hours.

Some criticism will doubtlessly be made of the suggestion that stamp milling be practised rather than ball milling. However, good amalgamation results are necessary and as an amalgamation device the stamp mill has no superior, hence its recommendation, especially so when consideration is given to the size of the proposed mill, 60 to 100 tons per day, and that construction and installation charges are to be kept as low as possible.

Report No. 342

EXPERIMENTAL TESTS ON ORES FROM THE GIANT MINE, SPILLAMACHEEN, GOLDEN MINING DIVISION, B.C.

J. S. Godard

Shipment. One shipment, weighing 363 pounds, was received from the Giant mine, Spillamacheen, B.C., on November 1, 1929. The shipment consisted of three ores: barite 108 pounds, lead-barite 135 pounds, copper 120 pounds; and it was made at the suggestion of V. L. Eardley-Wilmot of the Mineral Resources Division of the Mines Branch of the Department of Mines.

Characteristics of the Ore—

Barite Ore. This is a crude barite ore, containing about 94 per cent $BaSO_4$ and small quantities of the sulphides of lead, zinc, and iron, and under 3 per cent of silica.

Lead-barite Ore. This sample consists of galena in a gangue of barite and silica. Some silver and small quantities of the sulphides of iron, zinc, and copper are present.

Copper Ore. This ore consists of chalcopyrite and some pyrite in a siliceous gangue.

Analyses:

Ore	Pb, %	Cu, %	Fe, %	Zn, %	$BaSO_4$ %	SiO_2, %	Ag, oz./ton	Au, oz./ton
Barite	0·05	Nil	0·10	0·05	93·26	2·72	0·14	Nil
Lead-barite	17·55	0·04	0·41	0·11	43·46	32·74	4·66	"
Copper	0·11	4·30	8·36	0·13	2·25	70·54	1·14	"

Purpose of Experimental Tests. The purpose of the experimental tests was to concentrate the valuable mineral constituents of the ores.

EXPERIMENTAL TESTS

BARITE ORE

Results:

Test No. 1

Product	Weight, %	Assay, $BaSO_4$	Per cent of values
Concentrate	90·1	97·46	91·5
Tailing	9·9	82·12	8·5

Test No. 2

Concentrate	93·0	95·56	94·2
Tailing	7·0	79·00	5·8

Test No. 3

Rough concentrate	2·7	87·04	0·6
$BaSO_4$ concentrate	86·1	98·02	89·3
Tailing	11·2	84·82	10·1

Assay of rougher concentrate: Pb 2·84 per cent, Ag 4·64 oz./ton.

Reagents:

Test No. 1—To ball mill: water glass 0·10 lb./ton. To cell: oleic acid 0·60 lb./ton, pine oil 0·08 lb./ton.

Test No. 2—To ball mill: water glass 0·08 lb./ton. To cell: oleic acid 0·60 lb./ton, pine oil 0·08 lb./ton.

Test No. 3—To ball mill: soda ash 3·0 lb./ton, thiocarbanilide 0·10 lb./ton. To cell: Aerofloat No. 15, 0·04 lb./ton, pine oil 0·12 lb./ton, oleic acid 0·40 lb./ton, water glass 0·10 lb./ton.

CONCLUSIONS

A commercial grade of barite with good recovery can be obtained by flotation. When a small quantity of a rougher lead concentrate is removed before the flotation of the barite, a slight improvement in grade as well as colour is noted in the barite concentrate.

LEAD-BARITE ORE

Test No. 1

Product	Weight, %	Assay			Per cent of values		
		Pb, %	Ag, oz./ton	$BaSO_4$, %	Pb	Ag	$BaSO_4$
Lead concentrate	12·0	82·02	22·50	7·50	62·7	61·9	1·8
Lead middling	10·6	32·14	9·90	45·21	21·7	24·2	9·8
Barite concentrate	29·6	3·45	0·62	87·64	6·5	4·2	53·1
Barite middling	20·2	3·94	0·96	59·37	5·1	4·5	24·6
Tailing	27·6	2·30	0·82	18·85	4·0	5·2	10·7

Test No. 2

Lead concentrate	23·4	72·68	17·80	6·72	95·2	93·6	3·4
Lead middling	9·0	7·71	2·30	46·06	4·1	4·7	8·9
Barite concentrate	34·3	0·05	Nil	88·96	0·1	....	65·3
Barite middling	17·4	0·50	0·20	45·28	0·5	0·8	16·9
Tailing	15·9	0·15	0·26	16·25	0·1	0·9	5·5

Test No. 3

Lead concentrate	23·1	72·90	18·50	7·46	94·5	91·9	3·7
Lead middling	16·2	5·25	1·76	48·45	4·8	6·1	16·8
Barite concentrate	29·3	0·15	0·12	92·78	0·2	0·8	58·4
Barite middling	16·2	0·19	0·18	53·43	0·2	0·6	18·6
Tailing	15·2	0·41	0·20	7·51	0·3	0·6	2·5

Reagents:

Test No. 1. To ball mill: soda ash 3·0 lb./ton, thiocarbanilide 0·20 lb./ton. To cell: Aerofloat No. 15, 0·04 lb./ton. Lead rougher cleaned using Aerofloat No. 15, 0·04 lb./ton. For floating $BaSO_4$, water glass 0·08 lb./ton, oleic acid 0·16 lb./ton, pine oil 0·02 lb./ton. $BaSO_4$ rougher cleaned using water glass 0·05 lb./ton, oleic acid 0·025 lb./ton, pine oil 0·02 lb./ton. Note.—$BaSO_4$ concentrate grey in colour.

Test No. 2. To ball mill: soda ash 5·0 lb./ton, thiocarbanilide 0·20 lb./ton, amyl xanthate 0·05 lb./ton. To cell: Aerofloat No. 15, 0·02 lb./ton. After removal of lead concentrate a middling product was floated using $CuSO_4$ 0·40 lb./ton, potassium xanthate 0·10 lb./ton, Aerofloat No. 15, 0·04 lb./ton.

For floating $BaSO_4$: water glass 0·06 lb./ton, oleic 0·16 lb./ton. $BaSO_4$ rougher concentrate cleaned using water glass 0·06 lb./ton, oleic acid 0·05 lb./ton, and recleaned using water glass 0·05 lb./ton, oleic acid 0·025 lb./ton. $BaSO_4$ middling combined.

Test No. 3. To ball mill: soda ash 4·0 lb./ton, thiocarbanilide 0·25 lb./ton, amyl xanthate 0·08 lb./ton. To cell: AerofloatNo. 15, 0·04 lb./ton. After removal of the lead concentrate a clean middling was floated using Na_2S 1·0 lb./ton, amyl xanthate 0·10 lb./ton, and Aerofloat No. 15, 0·04 lb./ton. $BaSO_4$ rougher concentrate floated using water glass 0·15 lb./ton, oleic acid 0·25 lb./ton, and pine oil 0·04 lb./ton.

$BaSO_4$ rougher cleaned using water glass 0·12 lb./ton, oleic 0·10 lb./ton, pine oil 0·04 lb./ton. $BaSO_4$ recleaned using water glass 0·08 lb./ton, oleic acid 0·10 lb./ton, pine oil 0·04 lb./ton. This concentrate was again cleaned using water glass 0·05 lb./ton, oleic 0·05 lb./ton, pine oil 0·04 lb./ton.

All middling products were combined.

CONCLUSIONS

Good recoveries of the lead and silver were obtained without difficulty.
The best grade $BaSO_4$ obtained was 93 per cent, with a recovery of 58 per cent. Three cleaning operations were made to secure this grade, which is below the commercial standards. Only under the most favourable marketing conditions should the concentration of the barite be attempted.

COPPER ORE

Test No. 1

Product	Weight, %	Assay		Per cent of values	
		Cu, %	Ag, oz./ton	Cu	Ag
Concentrate	21·4	18·96	4·88	96·0	91·4
Middling	6·6	1·46	0·84	2·3	4·8
Tailing	72·0	0·10	0·06	1·7	3·8

Test No. 2

Concentrate	20·8	18·92	4·80	95·2	87·5
Middling	7·6	2·08	0·92	3·8	6·2
Tailing	71·6	0·06	0·10	1·0	6·3

Test No. 3

Concentrate	10·5	30·38	6·20	75·0	56·3
Middling	8·2	8·81	3·80	17·0	26·9
Tailing	81·3	0·42	0·24	8·0	16·8

Test No. 4

Concentrate	19·6	20·54	5·34	94·1	86·9
Middling	7·4	1·56	0·94	2·7	5·8
Tailing	73·0	0·19	0·12	3·2	7·3

Test No. 5

Concentrate	20·6	18·88	4·96	94·7	90·7
Middling	3·5	2·54	1·32	2·2	4·1
Tailing	75·9	0·17	0·077	3·1	5·2

Tailing Screened:

+100	13·1	0·37	0·10	28·2	16·9
+200	32·5	0·23	0·08	43·4	33·7
−200	54·4	0·09	0·07	28·4	49·4

Average tailing: Cu 0·17 per cent, Ag 0·077 oz./ton.

Test No. 6

Product	Weight, %	Assay		Per cent of Values	
		Cu, %	Ag, oz./ton	Cu	Ag
Concentrate	15·2	25·22	6·44	93·5	86·6
Middling	4·6	1·59	1·04	1·8	4·2
Tailing	80·2	0·24	0·13	4·7	9·2
Tailing Screened:					
+200	20·5	0·38	0·16	32·9	25·8
−200	79·5	0·20	0·12	67·1	74·2

Test No. 7

Concentrate	15·2	24·88	5·88	90·4	73·3
Middling	7·1	2·62	1·56	4·4	15·2
Tailing	77·7	0·28	0·18	5·2	11·5

Test No. 8

Concentrate	15·8	24·40	6·40	91·6	82·8
Middling	5·7	3·47	2·04	4·7	9·5
Tailing	78·5	0·20	0·12	3·7	7·7

Reagents:

Test No. 1. To ball mill: lime 4·0 lb./ton. To cell: amyl xanthate 0·10 lb./ton, Aerofloat No. 15, 0·08 lb./ton, pine oil 0·08 lb./ton. Rougher concentrate cleaned using pine oil 0·04 lb./ton.

Test No. 2. To ball mill: lime 4·0 lb./ton. To cell: potassium xanthate 0·10 lb./ton, pine oil 0·08 lb./ton, Aerofloat No. 15, 0·04 lb./ton. Rougher concentrate cleaned without additional reagents.

Test No. 3 To ball mill: soda ash 3·0 lb./ton, cyanide 0·15 lb./ton, thiocarbanilide 0·04 lb./ton. To cell: Aerofloat No. 15, 0·04 lb./ton, amyl xanthate 0·01 lb./ton, pine oil 0·04 lb./ton. Rougher concentrate cleaned without additional reagents.

Test No. 4. To ball mill: lime 5·0 lb./ton. To cell: amyl xanthate 0·06 lb./ton, pine oil 0·12 lb./ton, cresylic acid 0·04 lb./ton. Hard to froth. Rougher concentrate cleaned using pine oil 0·04 lb./ton.

Test No. 5. To ball mill: lime 4·0 lb./ton. To cell: amyl xanthate 0·08 lb./ton, Aerofloat No. 25, 0·04 lb./ton, pine oil 0·04 lb./ton. Rougher concentrate cleaned using lime 0·20 lb./ton, and pine oil 0·02 lb./ton.

Test No. 6. To ball mill: lime 4·0 lb./ton. To cell: amyl xanthate 0·04 lb./ton, Aerofloat No. 25, 0·08 lb./ton, cresylic acid 0·02 lb./ton. Rougher concentrate cleaned using Aerofloat No. 25, 0·04 lb./ton.

Test No. 7. To ball mill: lime 4·0 lb./ton. To cell: American Cyanamid Co. No. 208, 0·10 lb./ton, pine oil 0·12 lb./ton. Rougher concentrate cleaned without additional reagents.

Test No. 8. To ball mill: lime 2·0 lb./ton. To cell: Aerofloat No. 15, 0·10 lb./ton, pine oil 0·04 lb./ton. Rougher concentrate cleaned using pine oil 0·04 lb./ton.

CONCLUSIONS

Very good results were obtained on this ore by flotation. Recoveries of 93 per cent of the copper and 90 per cent of the silver were obtained in the concentrates. Copper concentrates assaying 25 per cent Cu and 6 ounces per ton Ag are about as high grade as may be obtained without too high losses in the tailings

No difficulty need be anticipated in the flotation of this ore.

III

REPORT OF INVESTIGATIONS: NON-METALLIC MINERALS SECTION

Report No. 343

THE TESTING OF AN ASBESTOS-BEARING ROCK FROM VILLE MARIE, QUEBEC

R. K. Carnochan

Shipment. A sample of asbestos-bearing rock, shipping weight 55 pounds, was received January 8, 1929. The sample was sent in by L. J. Jodouin of Sudbury, Ontario, and was taken from a property one mile from the C.P.R. station at Lorraineville in Duhamel township, 6 miles from Ville Marie, Témiscamingue county, Quebec.

Purpose of Experimental Tests. Test work to determine the quantity and quality of fibre that could be recovered from the asbestos-bearing rock was desired.

Characteristics of the Sample. The gangue is mostly greenstone with small amounts of serpentine and calcite. The fibre occurs both with the fibre parallel to the face of the rock to which it adheres and in veins with the fibre at an angle of about 45 degrees to the sides of the vein. The fibre is somewhat brittle generally and parts of it are very brittle, it is mostly white in colour but some of it is tinged amber.

EXPERIMENTAL TESTS

The whole sample, 53 pounds net, was screened on 1 inch and the oversize was crushed in a small jaw crusher and screened again. This crushing and screening was repeated twice more which was sufficient to reduce all the sample to −1 inch. The −1 inch was cut into two portions by means of a Jones riffle.

One portion of the −1 inch was screened on 2 mesh and the oversize was hand-picked. This hand-picking gave 1 pound 1½ ounces of fibre. The oversize rock was crushed in the jaw crusher to −2 mesh and combined with the other −2 mesh. In crushing and screening 1 ounce of fibre was obtained on the screen.

The −2 mesh was screened on 4 mesh and the oversize was put over a small suction table to remove fibre. This gave 2 pounds 5 ounces of fibre; the rock from the table was crushed and screened on a 4-mesh screen until it was all −4 mesh when it was put with the other −4 mesh; this crushing consisted of three passes through the jaw crusher and one through the rolls. In screening after the second pass through the jaw crusher 1 pound 4 ounces of fibre was picked off the screen, after the third pass 7 ounces, and after the pass through the rolls, 1 ounce.

The −4 mesh was screened on 8 mesh. The oversize was put over a suction table and 1 pound 7 ounces of fibre was obtained. The +8-mesh rock was crushed three times in the rolls and screened after each crushing on 8 mesh. After the three crushings the rock was all −8 mesh and it was put with the other −8 mesh. In crushing with the rolls 10 ounces of fibre was obtained on the screen after the second pass and 2 ounces after the third pass.

The −8 mesh was screened on 14 mesh and the +14 mesh was run over a suction table. This operation gave 15 ounces of fibre. The 14-mesh rock was put through the rolls three times, the product from the rolls being screened on 14 mesh and only the oversize returned. After the three passes all the rock was −14 mesh and it was put with the other −14 mesh. In crushing with the rolls 6 ounces of fibre was obtained on the screen after the second pass.

The −14 mesh was screened on 35. The −14 +35 and the −35 were tabled separately on a small Wilfley table. This gave in each case a fibre product and a rock product. The slimes from tabling both sizes were run into the same settling-tank. This gave a slime fibre product. The products from the tabling were dried and weighed.

The following table gives the weights of the different products obtained in the test and the per cent that each one is of the feed:—

	lb.	oz.	per cent of feed
Coarse fibre	8	11½	32·90
−14+35 fibre		11	2·59
−35 fibre		15	3·54
Slime fibre	1	3	4·48
−14+35 rock	8	8	32·08
−35 rock	5	12	21·70
Loss		11½	2·71
Feed	26	8	100·00

The total fibre recovered in the test work from 26·5 pounds of feed was 11 pounds 8½ ounces, which equals 43·51 per cent of the feed.

The coarse fibre recovered was 8 pounds 11½ ounces equal to 32·90 per cent of the feed. Two tests were made on this coarse fibre by means of a standard testing machine. The average test of the fibre was 0·3 −4·3 −6·4 −5·0.

The −14+35 fibre weighed 11 ounces and equalled 2·59 per cent of the feed. This fibre contains a small amount of sand.

The −35 fibre weighed 15 ounces and was equal to 3·54 per cent of the feed. This product contains a lot of fine sand.

The slime fibre weighed 1 pound 3 ounces and equals 4·48 per cent of the feed. It contains a great deal of very fine rock slimes.

Samples of the different fibres produced and of the products from testing the coarse fibre were sent to Mr. J. G. Ross of Montreal, an expert on asbestos, who reported as follows:—

> The samples of asbestos fibre submitted as from Ville Marie, Quebec, appear to be of the hornblende variety, and have some of the characteristics of amosite.
>
> The material while fibrous, is weak, harsh, and brittle, and the longer grades would not be suitable for spinning, not having the requisite strength.
>
> On account of weakness of fibre a smaller percentage, under regular milling treatment, would be held on the coarser screens than in the laboratory results reported.
>
> I do not think that manufacturers would use this except perhaps as cement stock, testing 0-0-6-10, and it would not pay to develop for this grade only. Some might be used as roofing material in the same manner as actinolite from Hastings, Ont., but transportation costs would put it out of consideration.
>
> J. A. Jacobs has shown me fair chrysotile fibre said to have come from within a few miles of Ville Marie. As there is serpentine in that country, prospecting might open up a deposit of good material.

CONCLUSIONS

The test work shows—

(1) A large quantity of fibre in the rock submitted.
(2) The fibre is very poor in quality.

Report No. 344

GRINDING AND WASHING OF SAND FROM GUIGUES TOWNSHIP, QUEBEC

R. K. Carnochan

Shipments. On September 15, 1928, 124 sacks of sand, shipping weight 40,000 pounds, were received from a property in Guigues township, Quebec, on the east side of lake Timiskaming. The shipment was sent in by Mr. Fred W. Arnott of Fabre, Quebec.

Purpose of Experimental Tests. Tests were desired to determine what sand products could be obtained from this shipment and the best methods of making these products.

Characteristics of the Shipment. The shipment, as received, was divided into 4 lots. Lots Nos. 1 and 2 were white sand, rather coarse grained, and the sand of Lots Nos. 3 and 4 was of the same size grain but was brownish in colour. A few bags were unmarked and these appeared to be the same as Lots Nos. 3 and 4. A number of the bags were lettered. Some of the lettered bags contained white sand similar to Lots Nos. 1 and 2, and some were filled with brown sand the same as Lots Nos. 3 and 4.

	Bags
Lot 1	21
Lot 2	30
Lot 3	16
Lot 4	33
Unmarked	14
Lettered	10
Total	124

EXPERIMENTAL TESTS

Test No. 1

Two bags of Lot No. 1, 3 bags of Lot No. 2, 2 bags of Lot No. 3, 3 bags of Lot No. 4, and 1 bag of the unmarked sand were screened wet on a 4-mesh Hummer screen. The undersize from the screen was put through an Akins classifier four times in order to wash it. The results of this test were as follows:—

Product	Weight		Fe_2O_3	TiO_2	SiO_2
	lb.	%	%	%	%
+4 mesh	55·5	2·95			
Washed sand	1,482·5	78·82	0·07	0·03	97·62
Washings	343·0	18·23			
Feed	1,881·0	100·00			

The washed sand contained 0·07 per cent Fe_2O_3, which is too high in iron to allow the sand to be used for making glass.

Test No. 2

Two bags of Lot No. 1, 3 bags of Lot No. 2, 2 bags of Lot No. 3, 3 bags of Lot No. 4, and 1 bag of the unmarked sand were ground wet in a chaser mill fitted with a 20-mesh outside revolving screen which returned the oversize to the mill. The −20 from the screen was run through an Akins classifier four times in order to wash it. The results of this test were as follows:—

Product	Weight		Fe_2O_3	TiO_2	SiO_2
	lb.	%	%	%	%
Washed sand	1,241	65·98	0·08	0·04	98·16
Washings	640	34·02			
Feed	1,881	100·00			

Test No. 2 gave a product running 0·08 per cent Fe_2O_3, which is too high to allow the sand to be used for glass-making.

Test No. 3

It was decided to run a test on Lots Nos. 1 and 2 alone as these two lots contain less iron than Lots Nos. 3 and 4.

Four bags of Lot No. 1 and 6 bags of Lot No. 2 were crushed wet in a chaser mill fitted with an outside 20-mesh revolving screen. The −20 mesh was run four times through an Akins classifier in order to wash it. The results of this test are:—

Product	Weight		Fe_2O_3	TiO_2	SiO_2
	lb.	%	%	%	%
Washed sand	1,358·5	73·63	0·06	0·04	98·38
Washings	486·5	26·37			
Feed	1,845·0	100·00			

This test gave a product lower in iron than either Tests Nos. 1 or 2, but as glass sand should contain 0·06 per cent or less combined iron and titanium, and the washed sand runs 0·10 per cent combined iron and titanium, it is evident that a cleaner product must be made before the sand can be used for glass-making.

With the purpose of making a cleaner sand by tabling out the titanium 456 pounds of the washed sand were run over a large Wilfley table:—

Results:

Product	Weight	Per cent of original unwashed sand	Fe_2O_3	TiO_2
	lb.		%	%
Table concentrate	6·5	1·05	0·54	1·96
Table tailing	430·0	69·43	0·06	0·02
Loss	19·5	3·15		
Feed	456·0	73·63		

In an effort to improve the tabling by removing some of the coarse sand that had passed the 20-mesh revolving screen of the chaser mill, 384·5 pounds of the washed sand was screened on 10 mesh and the −10 mesh tabled on the large Wilfley table.

Results:

Product	Weight	Per cent of original unwashed sand	Fe_2O_3	TiO_2
	lb.		%	%
+10	5·5	1·05	0·04	0·01
−10	379·0			
Table concentrate	23·5	4·50	0·21	0·30
Table tailing	349·0	66·83	0·06	0·02
Loss	6·5	1·25		
Feed	384·5	73·63		

Thinking that screening the washed sand on 20 mesh before tabling might make some improvement, a test carrying out this idea was made on 373 pounds.

Results:

Product	Weight	Per cent of original unwashed sand	Fe_2O_3	TiO_2
	lb.		%	%
+20	91	17·97	0·04	0·01
−20	282			
Table concentrate	26	5·13	0·59	0·18
Table tailing	253	49·94	0·06	0·02
Loss	3	0·59		
Feed	373	73·63		

These three tabling tests show that the titanium can be reduced from 0·04 per cent to 0·02 per cent and the iron stays the same (0·06 per cent). This tabled product is not suitable for glass sand.

Screen tests were made on the three lots of dried table tailings using 20- and 40-mesh screens. Analyses of the −20+40 in each case gave Fe_2O_3, 0·05 per cent, and TiO_2, 0·01 per cent. This shows that it is possible to prepare a sand suitable for making glass from Lots Nos. 1 and 2 by crushing in a chaser mill with a revolving screen to 20 mesh, washing the −20, tabling the washed sand, drying the table tails and screening them on 40 mesh. It would be possible to screen on 40 before drying or even before tabling but the method outlined seems to be the best as washed sand would be more easily and accurately screened dry than wet.

The following tables give the results of the screen tests on the three table tails, and indicate the per cent of glass sand that would be recovered from the original sand.

Tabling without Screening

Product	Screen test	Per cent of original unwashed sand	Fe_2O_3	TiO_2
	%		%	%
+20	13·0	9·03	0·05	0·01
−20+40	55·9	38·81		
−40	31·1	21·59		
Feed	100·0	69·43		

In actual practice the table tails would be screened only on 40 mesh, and therefore 9·03 per cent plus 38·81 per cent equals 47·84 per cent of the original sand would be recovered as glass sand.

Tabling after Screening on 10 mesh

Product	Screen test	Per cent of original unwashed sand	Fe_2O_3	TiO_2
	%		%	%
+20	9·5	6·35	0·05	0·01
−20+40	63·2	42·24		
−40	27·3	18·24		
Feed	100·0	66·83		

In practice the table feed would not be screened on 10 mesh and the table tailing would not be screened on 20 mesh, and therefore the glass sand that would be recovered would be the +10, +20, and −20+40 mesh or 1·05 plus 6·35 plus 42·24 equals 49·64 per cent.

Tabling after Screening on 20 mesh

Product	Screen test	Per cent of original unwashed sand	Fe_2O_3	TiO_2
	%		%	%
+40	65·8	32·86	0·05	0·01
−40	34·2	17·08		
	100·0	49·94		

In practice the table feed would not be screened on 20 mesh, and therefore the glass sand recovered from the original sand would be the +20 and the +40 or 17·97 plus 32·86 equals 50·83 per cent.

If the three figures for the per cent of glass sand recovered from the original sand are averaged a more reliable value is obtained. This average is 49·44 per cent.

Test No. 4

Tests Nos. 4, 5, and 6 were made solely to prepare samples of different sizes, both washed and unwashed, to submit to various users of sand products who would be willing to test the samples in actual commercial work.

A quantity of sand made up of 13 bags of Lot No. 1, 18 bags of Lot No. 2, and 4 bags marked with letters and containing white sand similar to Lots Nos. 1 and 2, was screened dry on a 4-mesh Hummer screen. The −4 mesh was washed by passing it four times through an Akins classifier. The washed sand was then dried and screened on 5½, 7, 10, 14, and 20 mesh. One-third of the −20 was screened on 26 mesh. The Hummer screen was used for all this screening. The following products were obtained:—

Product	Weight		Fe_2O_3	TiO_2
	lb.	%	%	%
+4 unwashed	2·0	0·03		
−4 +5½ washed	76·0	1·21	0·09	0·04
−5½+7 “	79·0	1·25	0·08	0·03
−7 +10 “	214·5	3·41	0·06	0·03
−10+14 “	370·5	5·89	0·05	0·02
−14+20 “	1,104·0	17·55	0·15	0·02
⅔ of −20 “	2,321·5	36·89	0·11	0·06
−20+26 “	419·0	6·66	0·05	0·02
−26 “	736·0	11·70	0·17	0·09
Washings and loss	969·5	15·41		
Feed	6,292·0	100·00	0·25	0·08

Test No. 5

Twenty-eight bags of sand made up of 6 bags of Lot No. 3, 13 of Lot No. 4, 6 of the unmarked lot, and 3 of the lot marked with letters and containing brown sand similar to Lots Nos. 3 and 4, were screened dry on a 4-mesh Hummer screen. The −4 was rescreened on 14 and the −4+14 was then run four times through an Akins classifier to wash it. The washed sand was dried and screened on 5½-, 7-, 10-, and 14-mesh screens by means of the Hummer. The following table gives the results of this test:—

Product	Weight		Fe_2O_3	TiO_2
	lb.	%	%	%
+4 unwashed	57·0	1·22	....	....
−14 “	3,072·0	65·52	0·33	....
−4+5½ washed	49·0	1·04	0·11	....
−5½+7 “	164·0	3·50	0·10	....
−7+10 “	415·5	8·86	0·08	....
−10+14 “	638·5	13·62	0·08	....
−14 “	232·5	4·96	0·15	....
Washings and loss	60·0	1·28	....	....
Feed	4,688·5	100·00	0·26	0·05

Test No. 6

Twenty-nine bags of sand made up of 6 bags of Lot No. 3, 14 of Lot No. 4, 6 of the unmarked lot, and 3 of the lot marked with letters and containing brown sand similar to Lots Nos. 3 and 4, were screened dry on 4 mesh. The −4 was screened on 20. The −4+20 was washed by putting it through an Akins classifier four times. The washed sand was dried and screened on 5½-, 7-, 10-, 14-, and 20-mesh screens. All the screening in this test was done by means of a Hummer screen. The following table gives the results of this test:—

Product	Weight		Fe_2O_3	TiO_2
	lb.	%	%	%
+4 unwashed	58·0	1·21		
−20 "	2,001·0	41·77	0·45	....
−4+5½ washed	38·0	0·79	0·11	....
−5½+7 "	199·5	4·16	0·09	....
−7+10 "	579·0	12·09	0·07	....
−10+14 "	516·0	10·77	0·08	....
−14+20 "	1,065·5	22·24	0·07	....
−20 "	203·5	4·25	0·14	....
Washings and loss	130·5	2·72		
Feed	4,791·0	100·00	0·26	0·05

CONCLUSIONS

Lots Nos. 1 and 2 are much cleaner and whiter than Lots Nos. 3 and 4, the last two lots being light brown in colour. It would be impossible to make glass sand out of Lots Nos. 3 and 4. Test No. 3 shows that Lots Nos. 1 and 2 can be reduced to 0·05 per cent Fe_2O_3 and 0·01 per cent TiO_2. This is low enough to allow the cleaned sand to be used for making clear glass.

The products produced by Tests Nos. 4, 5, and 6 would be suitable for various purposes, such as steel foundry sand, core sand, etc.

To sum up, the sand from this deposit could be used for several purposes.

Report No. 345

CONCENTRATION OF A RADIUM ORE FROM CARDIFF TOWNSHIP, HALIBURTON COUNTY, ONTARIO

R. K. Carnochan

Shipments. Three shipments of radium ore were sent in by the Ontario Radium Corporation from their property consisting of lots 4 and 5, concession 21, and lots 4, 5, and 6, concession 22, Cardiff township, Haliburton county, Ontario. The property is 2 miles from Wilberforce and 18 miles from Bancroft.

The first shipment, which was received on July 9, 1929, consisted of two lots of ore; each contained 40 bags and weighed about 2,000 pounds. The lots were marked Nos. 1 and 2 and were from the west and east ends respectively of a large dyke.

The second shipment, received August 22, 1929, was a bag containing 50 pounds of hand-picked uraninite. This was called Lot No. 3.

The third shipment, received November 12, 1929, and called Lot No. 4, contained 603 bags of ore, weight 71,595·5 pounds, and one bag containing 80 pounds 7 ounces of hand-picked uraninite. The 603 bags of ore and the bag of uraninite were mined together.

Purpose of Experimental Tests. Tests were desired on the different lots in order to determine a suitable method for concentrating the ore and also the amount of radium that could be recovered from each lot.

Characteristics of the Ore. The ore consists of uraninite in a gangue of purple fluorite, magnetite, green apatite, hornblende, biotite mica, orthoclase feldspar, and calcite. Except in the hand-picked material, very little uraninite could be seen in the shipments previous to cencentration.

Acknowledgments. Thanks are due H. V. Ellsworth, of the Geological Survey, for assistance in the test work, examination of head samples and products by means of the electroscope, and the furnishing of an analysis of uraninite from the same property; to R. A. Rogers, of the Mines Branch, for the analysis of concentrates, and to H. S. Spence, of the Mines Branch, for co-operation in the experimental work.

EXPERIMENTAL WORK

Test on Lot No. 1

The net weight of Lot No. 1 was 2,080 pounds, from which a representative sample was obtained by repeated crushing and cutting. The sample, which weighed 47 pounds, was crushed to −20 mesh and screened on 40 and 100 mesh. The three sizes obtained by screening were tabled on a small Wilfley table. The table concentrates were magnetite and fluorite containing uraninite; they were dried and run over the Ullrich magnetic separator using a low current in order to lift the magnetite only. The Ullrich magnetic products were very pure magnetite, and the non-magnetics were mostly fluorite with a little uraninite. These non-magnetic products were retabled to obtain clean uraninite concentrates.

The electroscope showed that none of the three concentrates obtained was above 1 per cent uraninite. This makes it quite evident that Lot No. 1 is entirely worthless as a radium ore.

Test on Lot No. 2

The net weight of Lot No. 2 was 2,562 pounds. This lot was treated similarly to Lot No. 1. The sample used for concentrating weighed 56 pounds. When the Ullrich non-magnetic products were retabled better uraninite concentrates were obtained. These concentrates were sampled and assayed for U_3O_8. The following data was obtained from the test:—

Product	Weight	U_3O_8	U_3O_8
	grms.	%	grms.
−20+40 concentrate from retabling	34·0	17·42	5·92
−40+100 concentrate from retabling	38·1	37·64	14·34
−100 concentrate from retabling	13·3	42·24	5·62
Total	85·4	30·30	25·88

A recovery of 85·4 grammes out of 56 pounds is equivalent to 6·72 pounds of concentrates from one ton of ore. These concentrates at 30·30 per cent U_3O_8 would contain 2·04 pounds U_3O_8, or 1·73 pounds uranium. Contained in this amount of uranium there would be 0·259 milligram of radium which at $70 a milligram would be worth $18.13.

Test on Lot No. 3

Lot No. 3 consisted of 50 pounds of hand-picked uraninite. This was repicked by hand and 25 pounds of very clean uraninite was obtained. The remaining 25 pounds was crushed to 20 mesh, screened on 40- and 100-mesh screens, and the three sizes so obtained were tabled on a small Wilfley table, making concentrates containing magnetite and uraninite. These concentrates were run through an Ullrich magnetic machine using a very weak current so that only the magnetite was lifted. The non-magnetic products from the Ullrich were very clean uraninite concentrates.

The following table summarizes the uraninite products obtained from Lot No. 3:—

Product	Weight	U_3O_8	U_3O_8
	lb.	%	lb.
Repicked uraninite	25·0	69·0	17·25
−20+40 Ullrich non-magnetic	5·51	65·70	3·62
−40+100 Ullrich non-magnetic	4·05	63·70	2·58
−100 Ullrich non-magnetic	2·82	65·96	1·86
Total uraninite products	37·38	67·71	25·31

It will be noticed in the above table that the repicked uraninite is put down as being 69 per cent U_3O_8. The reasons for doing this will be given in the description of the tests on Lot No. 4.

Out of the 50 pounds of Lot No. 3 there were obtained concentrates containing 25·31 pounds of U_3O_8. These concentrates would, therefore, contain 21·47 pounds of uranium. Contained in this amount of uranium would be 3·214 milligrams of radium which at $70 a milligram would be worth $225.

The above shows that Lot No. 3 is very valuable material but as it is hand-picked and not the ordinary run of ore that would be mined no deduction as to the probable worth of the deposit can be made from the results of the test work.

Tests on Lot No. 4

After the removal of some specimens the ore of Lot No. 4 was crushed in a gyratory and Symons disk crusher and then screened on a 12-mesh Newaygo screen. The +12 mesh was crushed in a small set of rolls in closed circuit with the screen. A head sample was taken out of the −12 mesh.

The balance of the −12 mesh was classified in a Richards launder classifier into three sizes. The coarsest size was tabled on a large Wilfley table, the medium size on a one-quarter size Butchart table, and the finest size was first thickened in a Callow cone and then tabled on a one-quarter size Wilfley table. Each of the three tables gave a magnetite concentrate containing uraninite.

The concentrates obtained were assayed for U_3O_8 with the following results:—

Product	U_3O_8 %
Coarse concentrate from retabling	44·44
Medium concentrate from retabling	52·62
Fine concentrate from retabling	56·14
−20+40 Ullrich non-magnetic	67·48
−40+100 Ullrich non-magnetic	67·28
−100 Ullrich non-magnetic	67·32

No assay for U_3O_8 was made on the repicked uraninite, but it was assumed that it ran 69 per cent. This figure was obtained by considering the repicked material to be 95 per cent uraninite, and taking uraninite to be 72·44 per cent U_3O_8, this being an assay made on a clean piece of uraninite from the same property from which the ore has been mined, by H. V. Ellsworth. This figure seems fair as it is reasonable to believe that the repicked uraninite would be slightly higher in U_3O_8 than the best concentrate obtained, that is slightly better than 67·48 per cent U_3O_8, which was the value obtained for the −20+40 Ullrich non-magnetic.

Using the above weights and assays and assuming the repicked uraninite to be 69 per cent U_3O_8 the following calculations were made:

Concentrates obtained from hand-picked uraninite.

Product	Weight	U_3O_8	U_3O_8
	lb.	%	lb.
Repicked uraninite	64·75	69·0	44·68
−20+40 Ullrich non-magnetic	3·69	67·48	2·49
−40+100 Ullrich non-magnetic	2·92	67·28	1·96
−100 Ullrich non-magnetic	1·80	67·32	1·21
Total uraninite products	73·16	68·81	50·34

The above concentrates were obtained from 69,214 pounds of ore. Therefore from a ton of ore there would be obtained 2·114 pounds of concentrates containing at 68·81 per cent 1·454 pounds of U_3O_8.

Concentrates obtained from milling ore.

Product	Weight	U_3O_8	U_3O_8
	lb.	%	lb.
Coarse concentrates from retabling	79·0	44·44	35·11
Medium concentrates from retabling	4·31	52·62	2·27
Fine concentrates from retabling	1·31	56·14	0·74
Total concentrates from retabling	84·62	45·05	38·12

These concentrates were obtained from 69,044 pounds. Had all the milling ore, 69,133·5 pounds, been milled there would have been obtained 84·73 pounds of concentrates. This amount would have really been obtained from 69,214 pounds of milling ore plus hand-picked uraninite. Therefore by milling a ton of ore, 2·448 pounds of concentrates would be obtained containing, at 45·05 per cent, 1·103 pounds of U_3O_8.

Concentrates obtained per ton of ore:

Source	Weight	U_3O_8	U_3O_8
	lb.	%	lb.
Hand-picked	2·114	68·81	1·454
Milling ore	2·448	45·05	1·103
Total concentrates	4·562	56·05	2·557

The amount of uranium contained in the concentrates produced from a ton of ore would be 2·557 x 0·8481 = 2·169 pounds.

The amount of radium contained in the concentrates produced from a ton of ore would be $\frac{2\cdot169 \times 453\cdot59 \times 1{,}000 \times 3\cdot3}{10{,}000{,}000} = 0\cdot3247$ milligrams.

The value of the radium, at $70 a milligram, contained in the concentrates produced from a ton of ore would be 0·3247 x 70 = $22.73.

CONCLUSIONS

The ore is easily concentrated and the concentrates obtained are of good grade.

Tests on the largest lot show $22.73 worth of radium in the concentrates obtained from a ton of ore, but before deciding whether or not this is commercial ore several things would have to be determined, some of these being, the amount of ore available, the cost of mining, milling and refining, the amount of radium and uranium lost in refining, and the price that could be obtained for radium and uranium. Thus it is obvious that a great deal of work must still be done before the mining of radium ore from this property is proved commercially feasible.

Report No. 346

GRINDING OF SCRAP MICA FROM LAKE GIRARD MINE, ST. PIERRE DE WAKEFIELD, QUE.

R. K. Carnochan

Shipments. Fourteen barrels of scrap mica, gross weight 3,160 pounds, were received June 10, 1929, from Mr. W. Ahearn, Ottawa, Ont. This mica came from the Lake Girard mine, St. Pierre de Wakefield, Quebec.

Purpose of Experimental Tests: Mr. Ahearn desired that the mica be ground up in order to obtain samples of ground mica to submit to prospective purchasers, and also wished to be advised upon the most suitable method of grinding this mica commercially.

Characteristics of the Ore. The mica, as received, consisted of flakes ranging in size up to pieces 8 or 10 inches long and 2 inches thick. The whole shipment was phlogopite or amber mica.

Experimental Tests

Test No. 1

Preliminary Crushing

The whole shipment, net weight 2,866 pounds was put through a large gyratory crusher in 40 minutes in order to break down the large pieces small enough to be fed to a small hammer mill.

It was then put through the hammer mill in 2 hours and 7 minutes. A batch, 1,400 pounds, of the hammer mill discharge was put through a small KEK grinder in 5 hours and 9 minutes, and 1,465 pounds was run a second time through the hammer mill in 1 hour and 23 minutes, and then through the KEK in 2 hours and 30 minutes. The screen tests on the products, from this work, were as follows:—

Size	Once through hammer mill	Twice through hammer mill
Inch	%	%
+0·525	8·5	
+0·371	9·1	2·2
+ 3	14·3	7·2
+ 4	15·3	15·7
+ 6	10·8	13·4
+ 8	9·0	11·9
+10	8·4	11·9
+14	6·2	9·0
+20	4·3	6·5
+28	2·6	4·0
+35	2·9	4·8
+48	2·0	3·4
+65	1·0	2·5
−65	5·6	7·5

Size	Once through hammer mill and once through KEK	Twice through hammer mill and once through KEK
	%	%
+ 4	0·1	0·2
+ 6	0·7	1·0
+ 8	2·8	4·2
+ 10	7·3	9·7
+ 14	10·5	12·2
+ 20	11·6	12·3
+ 28	8·9	8·8
+ 35	12·0	11·4
+ 48	9·7	9·0
+ 65	7·6	7·1
+100	8·2	7·6
+150	6·5	5·9
+200	3·7	2·7
−200	10·4	7·9

The table concentrates were dried and run separately over an Ullrich magnetic machine using a very low current in order to lift only magnetite. The Ullrich magnetic products were very clean magnetite and the non-magnetics were mostly fluorite with some uraninite.

The Ullrich non-magnetic products were retabled separately and good uraninite concentrates were obtained.

The hand-picked uraninite of Lot No. 4 was repicked and out of the 80 pounds 7 ounces received, 64 pounds 12 ounces of very clean uraninite was obtained. The discard, 15 pounds 11 ounces, was crushed in a small set of rolls to all pass 20 mesh and screened on 40 and 100. Each of the three sizes obtained was tabled on a small Wilfley table. The table concentrates were dried and run through the Ullrich machine to remove magnetite. Very clean uraninite concentrates were obtained in this way.

The weights of the different products were as follows:—

	lb.
Gross weight of 603 bags of milling ore, wet	71,595·5
Tare weight of 603 bags of milling ore, wet	1,205·0
Net weight of 603 bags of milling ore, wet	70,390·5
Moisture at 1·786 per cent	1,257·0
Net weight of milling ore, dry	69,133·5
Ore not milled, dry	89·5
Ore milled, dry	69,044·0
Milling ore received, dry	69,133·5
Hand-picked uraninite, dry	80·5
Total ore received, dry	69,214·0
Coarse table concentrates	1,702·0
Medium table concentrates	160·0
Fine table concentrates	42·0
Total table concentrates	1,904·0
Coarse Ullrich magnetic, magnetite	1,455·0
Medium Ullrich magnetic, magnetite	117·0
Fine Ullrich magnetic, magnetite	20·5
Total Ullrich magnetic, magnetite	1,592·5

	lb.	oz.
Coarse concentrate from retabling	79	
Medium concentrate from retabling	4	5
Fine concentrate from retabling	1	5
Total concentrate from retabling	84	10
Hand-picked uraninite	80	7
Repicked uraninite	64	12
Discard from picking	15	11
—20+40 Ullrich non-magnetic	3·69	
—40+100 Ullrich non-magnetic	2·92	
—100 Ullrich non-magnetic	1·80	

The capacities of the machines were as follows:—

	Pounds per hour
Gyratory	4,300
Hammer mill, 1st pass	1,355
Hammer mill, 2nd pass	1,059
KEK, put through hammer mill once	272
KEK, put through hammer mill twice	586

This test showed that it is necessary to put the material through the gyratory in order to get it small enough to feed to the hammer mill and that it is better to crush twice in the hammer mill so that it will readily go through the KEK. In actual operation a larger hammer mill would be used making it unnecessary to crush in the gyratory, and producing a fine enough product to feed directly to the KEK or whatever fine grinder was used.

Test No. 2

Separating Ground Mica by means of the Gayco Air Separator

A lot of 131 pounds of mica ground in the KEK machine was put through the Gayco air separator in order to see how effective it would be in removing the fine mica from the coarse. During the run two samples were taken of the fines produced by the Gayco. These samples ran as follows:—

Size	Sample A	Sample B
	Per cent	Per cent
+ 14	1·1	0·4
+ 20	1·5	0·6
+ 28	2·1	1·5
+ 35	4·5	4·3
+ 48	7·1	8·2
+ 65	9·8	11·7
+100	15·7	17·1
+150	16·9	19·0
+200	10·5	10·3
−200	30·8	26·9

This test shows that the Gayco separator is unsuitable for this work, as when set at its finest it only produced a product all −10 mesh.

Test No. 3

Grinding Wet in Pebble Mill and Separating in Dorr Bowl Classifier

A lot of 1,400 pounds of the KEK discharge was used in a grinding test with a 4-foot by 8-foot mill, lined with porcelain and carrying 2,865 pounds of porcelain balls, and a 7-foot Dorr bowl classifier. The feed rate was kept at 100 pounds per hour.

It was found that the greatest capacity obtained was 36 pounds per hour which is too low to allow this method of grinding to be used. It was also found that the bowl classifier separated very well at 200 mesh, giving a product that ranged from 98·3 to 99·9 per cent −200 mesh.

Test No. 4.

Part of the KEK discharge, 1,391 pounds in weight, was fed to the Keedy sizer, running time was 11 hours and 55 minutes. Products were as follows:—

Size	Pounds
+ 20	461¾
+ 24	81
+ 28	109¼
+ 34	79
+ 42	74
+ 50	112½
+ 62	94¼
+ 74	60¾
+ 86	54
+109	72
+125	18¾
+150	48¾
+200	31
−200	94

This machine makes an excellent separation into a great number of sizes.

Test No. 5. Grinding by means of KEK and Keedy to −200 mesh

A lot of 200 pounds of +20-mesh mica from Test No. 4 was ground in the KEK and run through the Keedy. All the sizes except the −200 mesh were mixed together, brought up to 200 pounds in weight, ground in the KEK and fed again to the Keedy. This was repeated five times, the period of grinding varying with each pass. The results of this test were as follows:—

−200 from Keedy	Method of grinding in KEK	Time in Keedy
12 pounds	1 pass, 20 minutes	2 hours 10 minutes.
11·5 pounds	1 pass, 30 minutes	2 hours 30 minutes.
30·5 pounds	2 passes, 20 minutes each and 4 passes, 15 minutes each.	2 hours 30 minutes.
19 pounds	6 passes, 15 minutes each	2 hours 30 minutes.
28 pounds	10 passes, 15 minutes each	4 hours 30 minutes.

This test shows that the Keedy sizer is a poor machine for separating out one size and that the KEK grinder has a rather small capacity in grinding to 200 mesh, being only about 36 pounds per hour.

Test No. 6. Grinding by means of KEK and Hand Screen

Twenty pounds of +20 mesh from Test No. 4 was ground in the KEK in two minutes and screened on 150 mesh by hand, the oversize brought up to 20 pounds, and ground and screened again. This process was repeated three times using the 150-mesh screen and then three times each using the 100, 65, 48, 35, 28, and 20. The results of this test are given below:—

Screen used	Fines from 3rd pass, lb.	Fines, lb. per hour	Screen test of fines							
			+28	+35	+48	+65	+100	+150	+200	−200
150	1·0	30·0	......	......	......	......	......	......	10	90
100	1·75	52·5	......	......	......	......	......	13	22	65
65	2·75	82·5	......	......	......	......	16	25	13	46
48	3·75	112·5	......	......	......	12	26	17	8	37
35	4·5	135·0	......	......	22	17	18	14	7	22
28	6·5	195·0	......	16	21	14	11	11	6	21
20	9·0	270·0	12	22	14	9	10	9	5	19

This test shows that the capacity of the KEK grinder increased as the size to which the mica is being ground increases. On the fine sizes the capacity is very low but on the coarser sizes it is fair.

Test No. 7. KEK Grinder and Hummer Screen

Approximately 82 pounds of the +20 mica from Test No. 4 was ground in the KEK and then screened on a 60-mesh, 5-by 3-foot Hummer screen at 20 degrees slope. The oversize from the screen was again ground in the KEK and screened a second time. Results of this test are as follows:—

KEK time, min.	Hummer time, min.	+60 lb.	−60 lb.	KEK capacity, −60, lb./hr.	Hummer capacity, −60, lb./hr.
9	24	71	7	47	17
8	35	62	11	82	19

This test shows that the Hummer screen is only fair at separating out the −60-mesh material.

CONCLUSIONS

As the mica submitted is of the amber variety and cannot be used for making wallpaper, it is unnecessary to grind it wet. Dry grinding would be cheaper than wet grinding besides giving a greater capacity for the same horse-power and using smaller machines. In view of this it is recommended that this mica be ground dry using for preliminary crushing, a hammer mill capable of taking a very large piece of mica; and for the finer grinding either hammer mills, large KEK machines, or similar grinders operating in conjunction with rotary screens.

Report No. 349

THE TESTING OF GYPSUM FROM THE MAGDALEN ISLANDS

R. A. Rogers

Shipments. Three samples of crude gypsum rock were received from L. H. Cole, Mines Branch, Ottawa, on July 10, 1929. These were from Amherst, Grindstone, and Alright islands of the Magdalen islands, Quebec.

The samples were taken from the following localities:—

Amherst island—from deposit one-half mile from Government wharf. This outcrop is 25 feet high by 150 yards long and extends inland for some distance.

Grindstone island—from deposit two miles from Government wharf. At this locality there were no outcrops on the surface, but sample was taken from under 5 feet of overburden and the deposit seems quite extensive.

Alright island—from deposit three-quarters of a mile from Government wharf. This outcrop on the shore where sample was taken is about 100 feet high and is exposed for 60 yards. The deposit appears to run inland for about three miles and can be traced by outcrops and sinkholes for that distance.

Purpose of Tests: The purpose of the tests was to determine the suitability of the gypsums for making plaster, and to obtain comparative data.

Sampling and Analysis: The samples as received were in lumps averaging about 5 inches in diameter.

The gypsum from Amherst island was faint pink in colour, and had a fibrous, banded texture; that from Grindstone island was also faint pink in colour, but had a fibrous texture with thin lenses of reddish clay. The gypsum from Alright island was mottled grey-white in colour, and had a massive texture.

The samples were first washed to remove clay and sand, and then dried at a low temperature. They were then crushed in a Sturtevant swing-sledge mill, and finally ground in a Munson burr mill to 95 per cent through 100 mesh.

The products from the burr mill were sampled in a Jones riffle sampler, and the samples chemically analysed by the writer. The results of the analyses follow:—

—	Amherst	Grindstone	Alright
Insoluble	0·68	1·06	0·48
Ferric oxide (Fe_2O_3)	0·10	0·07	0·11
Alumina (Al_2O_3)	0·08	0·13	0·11
Lime (CaO)	32·51	32·51	33·09
Magnesia (MgO)	0·04	0·04	0·17
Sulphur trioxide (SO_3)	46·31	46·94	46·40
Water (H_2O)	20·07	19·43	18·19
Carbon dioxide (CO_2)	0·10	0·04	0·65
	99·89	100·22	99·20

EXPERIMENTAL TESTS

Experimental Tests. Fifty-pound lots of each sample were calcined in a small electric batch kettle to the first settle. The calcined gypsum was tested. The water remaining after calcination, the testing consistency, and the time of setting were determined. Briquettes and cylinders were made, dried in the air to constant weight, and broken in Riehle and Olsen testing machines. This gave the tensile and compressive strengths.

Summary of Tests

—	Amherst	Grindstone	Alright
$\frac{\text{Water after calcination}}{\text{Water before calcination}}$	0·283	0·302	0·328
Final temperature of calcination, °F	298	310	290
Water after calcination, per cent	5·68	5·88	5·98
Testing consistency, c.c. per 100 grms	69	70	74
Time of setting in minutes	13	17	14
Tensile strength, lb./sq. in	343	383	253
Compressive strength, lb./sq. in	1,647	1,563	1,149
Colour of calcined product after set	Very faint pink	Very faint pink	White

CONCLUSIONS

The tests show that these gypsums are suitable for the manufacture of structural materials having a gypsum base. The Amherst and Grindstone deposits would not be satisfactory where a white finishing plaster is required as they do not make white plasters.

IV

REPORT OF INVESTIGATIONS: HYDROMETALLURGICAL AND ELECTROCHEMICAL SECTION

R. J. Traill

Engineer-in-charge

The following investigations were undertaken during the year:—

I. *Preliminary Tests on Titaniferous Magnetite from Mine Centre, Ont.* This ore contains in addition to iron some values in titanium and vanadium and the purpose of the test work requested was to determine how this ore would react to hydrometallurgical treatment and what recoveries could be obtained. With this data at hand a comparison with the smelting process, as carried out by Mr. Goodwin, could be made prior to further work on the utilization of these ores on a commercial basis.

Method of procedure and results obtained are shown in the report of tests.

II. *Separation of Copper from Molybdenum in Molybdenite Concentrate.* The ore contains chalcopyrite and molybdenite. The molybdenite concentrate carries 3 to 5 per cent copper, which reduces the marketable value of the product.

The purpose of the tests was to determine if a simple economic method of eliminating the copper could be devised so that a marketable molybdenite could be produced. A brief outline of the methods used and results obtained is reported. The method giving the best results is not adaptable to the location and size of the property.

III. *Hydrometallurgical Methods of Treatment for Zinc Concentrates from Various Canadian Ores.* Zinc is produced hydrometallurgically mainly by three processes, namely the Trail-Anaconda, the Tainton, and that of the Electrolytic Zinc Co. of Australasia. Another, that of the Base Metals Extraction Co., or the Harris-Field process, has been suggested and is at present in the predevelopment stage. These processes differ in various details and each claim certain advantages. The purpose of the investigation was to make a comparison of the three processes and determine, if possible, on a laboratory-scale basis, whether there is any particular advantage in any one of them relative to Canadian ores.

The investigation embraces a study of roasting, leaching, purification, and electrodeposition, and takes into consideration ultimate recovery of zinc, presence and effect of impurities, and grade of zinc obtained.

Preliminary tests on roasting were underway when owing to pressure of other work and lack of accommodation, further work in this investigation had to be postponed to some future date.

IV. *Treatment of Mixed (Bulk) Concentrate from Base Metal Sulphide Ore by the Base Metals Extraction Co. Process Coupled with the Waelz Process.* This test work was carried out under a co-operative agreement between the Base Metals Extraction Co. and this Division. The special features of the Base Metals Extraction Co. process have been outlined in the Report of the Laboratory for the years 1927 and 1928.

The main objective of the test work was to demonstrate the principle of the Waelz process, which is a volatilization process, and which would appear adaptable to the retreatment of zinc plant leach residues for the recovery of the remaining zinc and lead.

The test work embraced roasting, leaching, and purification, with retreatment of the plant residue in a rotary kiln under conditions as closely approximating, as was possible with the equipment at hand, those of the Waelz process.

The concentrates were roasted in this laboratory, the calcine going to the plant of the Base Metals Extraction Co. for leaching, purification, and electrodeposition. The leach residue was returned to this laboratory for treatment by the Waelz process of zinc volatilization.

V. *Preliminary Tests on the Use of Coking Coals in Metallization of Iron Ores.* These tests were made on an iron concentrate obtained from low-grade banded iron ore from Kaministikwia, Ont., with coals from Sydney, N.S., and Minto, N.B.

Report No. 347

I. PRELIMINARY TESTS ON TREATMENT OF TITANIFEROUS MAGNETITE FROM MINE CENTRE FOR RECOVERY OF IRON, TITANIUM, AND VANADIUM

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

Shipment. A shipment of 100 pounds of titaniferous magnetite marked M. C. 43, originally from a deposit near Mine Centre, Ont., was received from W. M. Goodwin, Ste. Anne de Bellevue, Que. The ore was shipped from Kingston, Ont., and was similar to that used in experiments on smelting carried out by W. M. Goodwin and reported by him in Report No. 8 (1921) to the Honorary Advisory Council for Scientific and Industrial Research. It contains small quantities of vanadium.

Purpose of Test Work. In the smelting process mentioned above, a pig iron containing 90 per cent of the vanadium value was obtained, while the titanium value was lost in the slag. The possibility of recovering this titanium value in addition to the iron and vanadium was suggested by Mr. Goodwin and as a preliminary to a further and more complete study of the utilization of these ores to be undertaken by him, the following tests were carried out.

The process suggested was the one that has been already carried out in this laboratory on ilmenite ores, embracing the low-temperature reduction of the oxide iron to metallic iron, the separation of the metallic iron by dissolution in ferric chloride, recovering the iron as electrolytic iron and obtaining the titanium and vanadium in the insoluble residue from the leaching. The main point to be determined is the behaviour of the vanadium and its concentration in the insoluble residue, followed by the separation and recovery of the titanium and vanadium values.

EXPERIMENTAL TESTS

A 20-pound sample was taken and crushed to pass an 8-mesh screen and a sample cut for screen and chemical analysis.

Screen Analysis

Mesh	Weight	Per cent	
	grms.		
+10	81·8	16·36	81·32 per cent
+20	143·4	28·68	
+35	118·4	23·68	
+65	63·0	12·60	
+80	0·9	0·18	18·68 “
+100	18·4	3·68	
+200	24·4	4·88	
−200	49·7	9·94	
	500·0	100·00	

Approximate Chemical Analysis

	Per cent
Iron	48·38
Titanium dioxide	12·82
Vanadium pentoxide	0·86
Silica	6·40

The above analysis differs somewhat from that reported by Goodwin given as follows.

	Per cent
Iron	47·53
Titanium dioxide	10·34
Vanadium pentoxide	0·80
Silica	13·93

METALLIZATION

Metallization was carried out in an electric furnace of the heavy wire resistance type.

The containers for the charge were vitreosil muffles. Temperatures were measured within the furnace but not within the charge.

The charge consisted of:—

	Grms.
Ore	970
Coal	545
Coke	136

Four hours' heating was required to bring the furnace temperature up to 1,000° C. with the temperature within the charge probably around 750° C. The furnace temperature was raised to 1,050° C. for five hours, the highest temperature within the charge probably being 50 to 100° below the furnace temperature. The weight of the reduced product was 1,072 grammes. Magnetic separation was carried out on a Ball-Norton belt concentrator, the products being:—

	Grms.
Magnetics	723
Non-magnetics	347

Showing 67·5 per cent magnetic product.

Analysis of Magnetics

	Per cent
Total iron	60·26
Metallic iron	56·24
Titanium dioxide	16·52
Vanadium pentoxide	1·08
Metallization	93·3

The above test was duplicated, 1,814 grammes ore, 1,090 grammes coal, and 272 grammes coke being mixed and divided into two portions, placed in vitreosil muffles and metallized, using the same temperature range as previously. The results follow.

Weight of reduced product 1,051 and 1,060 grammes respectively. The products were combined and passed over the magnetic separator.

	Grms.
Magnetics	1,307
Non-magnetics	759

Showing 62 per cent magnetic product.

Analysis of Magnetics

	Per cent
Total iron	61·21
Metallic iron	58·34
Titanium dioxide	17·61
Vanadium pentoxide	1·07
Metallization	95·3

The non-magnetic tailing, analysed for V_2O_5, showed 0·11 per cent. These results indicate no difficulty in obtaining good metallization of iron oxide in this ore and also show that about 90 per cent of the vanadium value and practically all of the titanium value is retained in the magnetics.

LEACHING OF METALLIZED PRODUCT

Leaching tests were made and the following tests are given as examples of the results obtained.

A metallized product, analysing 60·26 per cent total iron, 56·24 per cent metallic iron, 16·52 per cent titanium dioxide, was leached with 4 litres of ferric chloride solution containing 416 grammes of iron. The charge of metallized product was 415 grammes. Time of leaching was 4 hours. After filtration the liquor showed a small amount of unreduced ferric chloride.

The leach residue weighed 180 grammes, showing 51·6 per cent of the magnetic product as being dissolved. Analysis of the leach residue was:—

	Per cent
Iron	13·59
Titanium dioxide	33·93

The extraction of metallic iron was approximately 90 per cent.

Some hydrolysis was evident, amounting by calculation to about 3·5 per cent.

Approximately 99 per cent of the titanium was found in the residue.

A second batch of 415 grammes of metallized product, analysing 61·21 per cent total iron, 58·34 per cent metallic iron, 17·61 per cent titanium dioxide and 1·07 per cent vanadium pentoxide, was leached with ferric chloride containing 483·6 grammes iron.

In this test reduction was complete in 30 minutes. The residue weighed 206 grammes showing 50·3 per cent of the charge had dissolved.

The residue analysis was:

	Per cent
Iron	20·82
Titanium dioxide	35·33
Vanadium pentoxide	2·10

The extraction of metallic iron was 82 per cent.

Hydrolysis occurred to a marked extent in this test, calculation showing an equivalent of 6 per cent. This would account for the high iron content of the residue. Over 99 per cent of the titanium and 97·7 per cent of the vanadium was found in the residue.

The interesting feature of these tests is the insolubility of the titanium and vanadium values in the ferric leach liquor and their retention in the residue. The value of this residue from the titanium recovery viewpoint is decidedly questionable. Even with higher iron extraction, or a decrease of iron in the residue, the value would not be materially increased as the undesired elements or gangue material amounting to approximately 33 per cent in the residue is too large.

Elimination of this gangue material from the ore would be beneficial, and would show increased titanium content in the residue. A separation is possible as is shown in the next part of this report.

The hydrolysis of the ferric liquor is probably due to the degree of neutrality of the liquor used and if so would be overcome with the addition of a small amount of hydrochloric acid in the leaching liquor.

MAGNETIC CONCENTRATION OF ORE

The possibility of magnetic concentration of the ore was investigated mainly to determine if some of the gangue material could be removed without serious loss of titanium or vanadium values.

In the event that such a scheme is practicable, there would result a higher grade product to work with, and the leach residue obtained, after dissolution of the iron, would contain higher vanadium and titanium values and lower interfering and troublesome gangue elements, with probably higher ultimate recoveries of titanium and vanadium.

The following is the result of a magnetic concentration test on the ore.

A sample, 1,000 grammes −8-mesh ore, was passed over the Ball-Norton belt separator.

Weight of magnetic concentrate	808 grms.	=80·8 per cent
Weight of non-magnetic tail	173 "	=17·3 "
Loss in concentrating	19 "	= 1·9 "

Analysis

	Head (raw ore) %	Magnetic concentrate %	Non-magnetic tailing %
Iron	48·38	55·9	21·00
Titanium dioxide	12·85	14·2	7·51
Vanadium pentoxide	0·86	1·0	0·27
Insoluble residue	10·50	4·06	30·76

The values for insoluble residue are only approximate, but form a basis of comparison.

The magnetic concentrate was screened to pass 14 mesh and the +14 mesh crushed sufficiently to pass 14 mesh and the whole again passed over the magnetic belt separator.

The result showed a further separation of 2·9 non-magnetic tailing. This result would indicate that some 20 per cent of the weight of the ore can be eliminated with losses of approximately 5 per cent vanadium, 10 per cent titanium, and 8 per cent iron.

This magnetic concentrate, metallized and leached as proposed, should theoretically furnish a product well over 50 per cent titanium dioxide and over 3·5 per cent vanadium pentoxide, the remainder being gangue with a small amount of iron oxide. There is, however, the point to be considered as to how the increased fineness of the ore, namely −14 mesh, would affect metallization. From experience with ilmenite it has been found that the finer particles of metallic iron while hot oxidize very readily in contact with air. Finer grinding would also increase dust losses in any type of rotary or continuous operation furnace.

The value of this method could be determined satisfactorily only on large-scale operation. The above test, however, indicates and illustrates a possibility worth consideration and examination.

VANADIUM RECOVERY FROM LEACHED RESIDUES

Various methods have been suggested for the recovery of vanadium from ores. In general the processes comprise roasting with, or without, the addition of chemical reagents, such as sodium chloride, sodium nitrate, sodium carbonate, sodium bicarbonate, fluorspar, etc., followed by leaching of the soluble vanadates and subsequent precipitation as calcium or iron vanadate.

The following table shows some results obtained by roasting with various chemical reagents and leaching the soluble vanadate. The residue used for the charge contained 2·10 per cent V_2O_5, vanadium pentoxide.

Tests Nos. 1 to 4 were roasted in open dishes in a muffle furnace and Test No. 5 was roasted in a combustion furnace with a slow current of oxygen passing over continuously. The latter result would indicate that excess oxygen is not a contributing factor in promoting the formation of soluble vanadate.

Test	Charge	Time	Temp.	Weight of roast	Charge to leach	Leaching medium	Volume leach	V_2O_5 in leach	Extraction
	grms.	hrs.	°C	grms.	grms.		c.c.	grms.	%
1	25—Residue.								
	4—NaCl	5	760–830	29·5	10	Water	200	0·1186	66·6
	2—$NaHCO_3$				5	4% NaOH	100	0·0589	66·2
	1—$NaNO_3$				5	10% Na_2CO_3	100	0·067	75·2
2	25—Residue	5	760–830	29·0	10	Water	200	0·006	3·3
	4—CaF				5	4% NaOH	100	0·0056	6·2
3	25—Residue.								
	4—CaF	5	750–830	30·5	10	Water	200	0·006	3·5
	1—$NaNO_3$				5	4% NaOH	100	0·006	7·0
4	Same charge as 1	7¾	710–830	28·5	10	Water	200	0·087	47·8
5	25—Residue.								
	4—NaCl	5	800–900	29·5	10	Water	200	0·1048	58·8
	2—$NaHCO_3$				10	5% NaOH	200	0·1108	62·2

A test to determine if time was a contributing factor to obtaining high solubility was run with the following results.

The charge consisted of 200 grammes residue, 24 grammes NaCl, 32 grammes $NaHCO_3$, and was roasted in an open dish and hand-rabbled. Samples were withdrawn at 90-minute intervals and the soluble vanadate determined.

Sample	Time roasted	Temp. °C.	Soluble V_2O_5 on 5-gramme sample
	hrs.	°C.	grms.
1	1	750	0·058
2	2½	800	0·071
3	4	860	0·074
4	5½	875	0·074
5	7	870	0·079

The final roast contained 1·86 per cent V_2O_5.

The above results indicate that after one hour's roasting 55 to 60 per cent of the V_2O_5 is soluble and this amount gradually increases with time until we find at the fifth sample some 85 per cent of the V_2O_5 soluble in water. The combined residue from the five separate 5-gramme samples treated with 4 per cent Na_2CO_3 solution showed only an additional 3 per cent of the total V_2O_5 dissolved.

A larger bulk leaching test of the above roast, consisting of 100 grammes roast treated with 400 c.c. water for 5 hours, gave 1·41 grammes V O_5 in solution or 75·8 per cent. The residue of 94·5 grammes analysed 0·47 per cent V_2O_5. Another sample of leach residue roasted with 20 per cent salt alone for 7 hours showed only 56·8 per cent extraction of V_2O_5 on leaching with water.

Present results, therefore, would indicate that the salt-sodium bicarbonate combination gives the better results in formation of soluble vanadate.

LEACHING RAW ORE

Some tests to determine the possibility of roasting the raw ore with various chemicals in order to obtain the vanadium in soluble form were tried with little success. Combinations of ore + NaCl + $NaHCO_4$ + $NaNO_3$, ore + CaF, ore + CaF + Na_2CO_3, ore + coal + NaCl + Na_2CO_3, ore + $NaHCO_3$ were used with varying results usually under 50 per cent extraction. The ore + coal + NaCl + $NaCO_3$ combination gave consistently higher results than the other combinations.

Finer grinding might assist in obtaining better results, but if the ore was crushed too fine it would be less suitable for further use in metallizing or other furnace treatment, and, therefore, further tests along this line were considered inadvisable.

COMPLETE TEST ON V_2O_5 EXTRACTION

The following compilation shows the vanadium and its occurrence in the various steps of the process under investigation.

The ore was metallized, the magnetics separated and leached with ferric chloride. The leached residue was next roasted with NaCl + $NaHCO_3$ and the soluble vanadate extracted with water.

Product	Weight, grms.		V_2O_5 %	V_2O_5 weight	
	Used	Ob-tained		Used	Ob-tained
Ore	1814		0·86	15·60	
Magnetic product		1307	1·07		13·98
Non-magnetic product		759	0·11		0·83
					14·81
Magnetics to leach	415		1·07	4·44	
Residue from leach		209	2·08		4·36
Leached residue to roast	200		2·08	4·16	
Roast		220	1·86		4·09
Roast product	100		1·86	1·86	
Water leach					1·41
Water leach residue		94·5	0·47		0·44
					1·85

CONCLUSIONS

Metallization of the iron content of this ore presents no difficulty. Muffle charges unrabbled have shown 95 per cent metallization and with more suitable equipment, such as a rotating kiln, higher results should be obtained.

Leaching of the metallic iron with ferric chloride is readily accomplished, although in the tests carried out there is a disturbing iron hydrolysis factor which has not been definitely traced to any special condition.

The ore carries a high gangue, which is found with the residue resulting from the dissolution of the iron with ferric chloride. The presence of this gangue materially reduces the value of the leach residue in terms of titanium and vanadium.

A magnetic concentration test on the ore shows that quite a large proportion of this gangue can be eliminated with but slight losses of titanium and vanadium. Such a magnetic concentration prior to metallization is, therefore, recommended in following out this process.

For recovery of vanadium from the ferric chloride leach residue, roasting with salt and sodium bicarbonate appears to give best results, 75 per cent or over of the vanadium being obtained in a soluble form. Its recovery would entail precipitation with iron or calcium salts.

Treatment of the final residue for recovery of titanium has not been attempted, but the sulphate process should be adaptable. Recovery of the iron as electrolytic iron should present no difficulty as the ore seems to be free from the usual disturbing impurities.

The tests herein described are intended solely to determine the adaptability of this process treatment from a chemical or hydrometallurgical viewpoint and should be considered only as a basis to any future study that may be undertaken.

II. THE SEPARATION OF COPPER FROM MOLYBDENUM IN MOLYBDENITE CONCENTRATES BY HYDROMETALLURGICAL METHODS

R. J. Traill

The Ore. The ore contains chalcopyrite and molybdenite, and assays 4·6 per cent copper and 2·8 per cent molybdenite.

The molybdenum is present as very fine-grained molybdenite; in some specimens it occurs in patches free from chalcopyrite and pyrite, while in others the chalcopyrite and pyrite are scattered throughout the molybdenite forming a complex mixture. The gangue is mostly siliceous with some calcite. Selective flotation tests show a recovery of 72 per cent of the molybdenite in a 50 per cent molybdenite concentrate carrying 3 to 5 per cent copper, and a recovery of 93 per cent of the copper in a 25 per cent copper concentrate.

The molybdenite concentrate having such a high copper content is not readily marketable.

Purpose of Tests. The purpose of the tests was to determine if a simple economic method of eliminating the copper could be devised so that a marketable molybdenum product could be produced.

Methods of Procedure. Hydrometallurgically two methods are possible.

1. Direct leaching with a suitable solvent to remove copper.
2. Roasting in air, followed by selective leaching with solvents for the removal of either the copper or the molybdenum.

Test Work. Under procedure (1) ferric chloride and ferric sulphate with and without acid were tried but their solvent effect was very poor, only 18 to 25 per cent of the copper being extracted. Direct leaching was therefore abandoned as being unsatisfactory.

The chalcopyrite in the concentrate appears to be badly oxidized as shown by a test in which the concentrate was roasted with excess sulphur in a non-oxidizing atmosphere, the roast product leached with ferric chloride. The result showed over 85 per cent copper extraction with about one-half per cent copper remaining with the molybdenite. The molybdenite is not appreciably soluble in the ferric chloride.

Under procedure (2) tests embracing the roasting at various temperatures and leaching with soda and other solvents gave results that were not satisfactory, due to the high percentage of alkaline earths present in the concentrate and the concurrent excessive sulphate formation in roasting. The molybdic oxide obtained by roasting is quite soluble in the soda solution, but in the re-precipitation of the molybdenum with lime to obtain calcium molybdate so much calcium sulphate is precipitated that the resulting product is too low grade for commercial use, and the presence of the sulphate entirely prohibits its use in alloy steel work.

Recourse was then made to first leaching the alkaline earths with hydrochloric acid, followed by roasting the unattached sulphides to oxides and leaching with soda solution. The molybdenum value was then recovered from the soda leach as calcium molybdate. Copper in the leach residue was readily recovered by leaching and precipitating as cement copper.

This method showed recoveries of 80 to 90 per cent of the molybdenum as high-grade calcium molybdate with copper recoveries of over 90 per cent.

The cost of chemical treatment by this method is approximately $100 per ton concentrate, or 22·8 cents per pound molybdenum on the concentrate used.

In the case of the small property concerned such a cost is probably prohibitive and quite unattractive.

III. HYDROMETALLURGICAL METHODS OF TREATMENT FOR ZINC CONCENTRATES FROM VARIOUS CANADIAN ORES

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

Introduction

The method of roasting zinc concentrates presents certain problems as carried out in the aforementioned hydrometallurgical processes. The roasting furnaces are of the Wedge or McDougall type or some modification of these. Differences in the characteristics and composition of the ore are the determining factors in the type of furnace to be employed. The fineness of the concentrates, the iron and lead content, and the mineralogical composition are the primary features which necessitate a study of roasting conditions. From these characteristics of the ore the time of roasting, the heat generated or required, the formation of zinc sulphate, and the formation of insoluble zinc compounds are governed.

The effect of the size of ore particles on the speed of roasting is important. Fine particles roast more quickly than coarse, but they tend to increase the losses owing to the fine dust being carried away with the flue gases. Sizing of the ore is an advantage in obtaining more perfect roasting conditions. Coarse particles will require a longer period of time in the furnace, during which the fines present may be over-roasted. It is recommended that the ore should be crushed to pass at least a 60-mesh screen.

The present general practice of using flotation concentrates in roasting precludes the possibility of having coarse particles in the roasting charge, as flotation concentrates usually contain over 90 per cent of material which passes a 200-mesh screen. The dust losses are overcome in this case by the installation of Cottrell precipitators. However, it is necessary to maintain a greater control of the temperature as the tendency towards the formation of insoluble zinc compounds is increased.

The presence of lead has a tendency to make the roast sticky.

Another feature which has an important bearing on the subsequent leaching is the quantity of zinc sulphate formed from the zinc sulphide. The amount of water-soluble zinc sulphate formed should not exceed the sulphuric acid mechanically lost during the subsequent leaching and electrolysis. This is most important as the presence of too great an excess of sulphate will increase the sulphuric acid in the electrolyte beyond the limit required for leaching fresh calcine, thus necessitating the running to waste of a certain part of the solution at some stage of the leaching-electrolysis cycle.

Zinc sulphate is more readily formed during low-temperature roasting. At the higher temperatures zinc sulphate is dissociated into zinc oxide, sulphur trioxide and sulphur dioxide, and oxygen. This dissociation is first apparent at about 575°C and the decomposition pressure increases with rise of temperature.

Formation of sulphate is further favoured by the presence of other metallic sulphides, especially sulphides of iron. The fact that iron sulphide is more quickly oxidized than zinc sulphide leads to the assumption that the $FeSO_4$ formed from the iron sulphide is decomposed liberating SO_3 which attacks ZnO and forms $ZnSO_4$ which can exist at temperatures at which iron sulphates are broken up.

The catalytic effect of iron oxide in the oxidation of sulphur dioxide to sulphur trioxide is well known. This fact has also been brought forward as an explanation for the need of iron as a contributing factor in the formation of zinc sulphate during the roasting of sulphides of zinc.

From the theoretical considerations discussed above, the temperature may be taken as the most important factor in the control of the formation of zinc sulphate. The amount of zinc sulphate in the calcine should be kept between 5 and 10 per cent, depending on the strength of acid required in the leaching. It must be borne in mind, however, that while high temperature minimizes the tendency towards the formation of sulphates, it increases the formation of ferrites (insoluble zinc compounds).

In the roasting of zinc sulphides a chemical combination of varying degrees occurs between the zinc and iron content of the ore, yielding a compound which is insoluble in the common acid leaching solutions employed. The formation of this compound usually referred to as zinc ferrite, ZnO, Fe_2O_3, is governed by the temperature of the roast and more or less on the mineralogical character of the ore. Much investigational work has been done in the study of ferrite formation, and while the conditions of its formation are fairly well defined it becomes necessary to subject each individual ore to special study in this connexion.

Zinc ferrite may be of the natural or artificial variety. When a mixture of ZnO and Fe_2O_3 in the proportion of 1 to 1·96 is subjected to a temperature of 600° C. or over, a synthetic or artificial ferrite is formed. This substance is yellowish brown, non-magnetic, and insoluble in solutions of ammonium chloride or weak sulphuric acid and only partly soluble in strong sulphuric acid. Ferrite so formed is analogous to that produced in the roasting of zinc blende with which are associated sulphides of iron. Natural ferrite is formed when the ore containing iron and zinc sulphides, isomorphously crystallized and mineralogically known as marmatites, is subjected to oxidation at roasting temperatures. Such ferrites are appreciably magnetic. The ratio of zinc to iron in the ferrite is a constant. By knowing the units of iron present in the marmatite, the units of insoluble zinc can be calculated. Providing that the ore is a true marmatite these conditions will hold regardless of the temperature at which roasting has been carried out.

From the above brief discussion it may be observed that the satisfactory roasting of zinc ores for hydrometallurgical treatment is by no means a simple operation, but is one that requires careful control and intelligent study of the several factors involved.

It is recognized that, while laboratory results may not in all cases be duplicated in commercial practice, the carrying out of laboratory tests is necessary and the results obtained have a decided value in anticipating what might be expected in the operation on a plant scale.

EXPERIMENTAL TESTS

A. ROASTING

The preliminary roasting tests herein described were conducted on zinc flotation concentrates of Sherritt-Gordon ore from the Sherritt-Gordon mine, concentrated in the Ore Dressing Laboratory of this Division.

A number of tests on 500-gramme lots were carried out in open vitreosil dishes in an electric wire resistance muffle furnace. These were followed by roasts in which the charge was increased to 5,000 grammes. A vitreosil muffle, 22 inches long, 8 inches wide and 5·5 inches high, closed at one end was used, the whole being placed in the electric furnace mentioned above. Air was admitted to the back end of the muffle by means of an iron pipe which was connected to an air line from a small blower. The charge was hand-rabbled throughout the duration of the roast. Temperatures were recorded by a thermocouple pyrometer, the couple being encased in a silica tube, embedded in the charge.

Roasting was carried out varying the conditions of air supply, either by enrichment with oxygen or by restriction. Very distinct stages were observed during the period of roasting. The initial stage, denoted by the presence of a blue flame over the charge, extended over a period from 10 to 15 minutes, during which time pyritic sulphur was burnt off. The next observed change was the glowing of the charge, followed by a fluid-like consistency of the roast. This condition lasted for several hours after which the charge assumed a pasty condition and gradually darkened. At the final stage no sulphur dioxide evolution could be detected. The time period of these stages varied according to the changes in conditions of roasting.

Test No. 1

A charge of zinc concentrate analysing 37·95 per cent Zn and 18·06 per cent Fe was placed in the muffle. (The zinc in this concentrate is not representative. Average concentrate would probably carry about 42 to 46 per cent Zn.) Air was blown in and the charge hand-rabbled. The temperature range during the active part of the roast was from 280° C. to 680° C.

The roast was apparently complete in about seven hours, no sulphur dioxide being detected in the gases from the muffle although roasting was continued for an additional two hours.

Test No. 2

A duplicate test to the above was run in which the results obtained were similar to Test No. 1.

Test No. 3

Enrichment of the air with oxygen was next tried. The charge consisted of 5,000 grammes and forced air was used as in the previous tests. After the completion of the first phase, i.e. the free burning of the sulphur from the pyrite, oxygen was added to the air supply. This was continued for 3¼ hours at the rate of 2½ to 3 cubic feet per 10-minute period. In all about 44½ cubic feet of oxygen was used. The effect of the oxygen-enriched air was to intensify and quicken the reactions. The roast was complete in about 5¼ hours but heating of the charge continued for another hour. The temperature ranged from 275° C. to 650° C. and during the greater part of the time did not exceed 620° C.

Test No. 4

A further test with slight modifications was carried out. The charge was heated to about 180° C. without applying the forced air. At the start of the first phase the air-oxygen mixture was turned on and continued for 10 minutes at which time the ignition of the sulphur was complete. The temperature rose in this period to 320° C. and the reaction was very vigorous. As soon as the action subsided, the oxygen was shut off and air alone admitted to the muffle. Roasting was carried out for 6¼ hours although complete in about 5½ hours. The maximum temperature was 676° C.

Test No. 5

A new head sample was used in this test analysing 42·02 per cent Zn and 15·92 per cent Fe. During the first 7½ hours of the run no forced air was employed. Oxidation was carried on by the air admitted through the open end of the muffle. For the final 1½ hours of the roast forced air was admitted to the muffle. The charge was apparently completely roasted in 8 hours, although continued for an additional hour. The temperature range was from 493° C. to 691° C.

The results of these five tests follow:

Test	Charge	Analysis head		Total Zn calcine	Per cent NH_4Cl soluble Zn calcine	Soluble Zn	S as SO_3	Total S	Time of roast in hours	
		Zinc	Iron						Apparent	Total
	grms.	%	%	%		%	%	%		
1.......	5000	37·95	18·06	43·33	32·73	75·5	2·88	3·10	7	9
2.......	5000	37·95	18·06	43·13	32·31	74·9	3·52	4·10	7	9
3.......	5000	37·95	18·06	43·03	32·99	76·6	4·10	4·59	5¼	6¼
4.......	5000	37·95	18·06	45·35	35·26	77·7	2·70	3·08	5½	6¼
5.......	5000	42·02	15·92	46·21	38·10	82·4	2·66	2·89	8	9

The effect of enriching the air with oxygen reduces the time of roasting. There is a small increase in the percentage solubility of the zinc. The longer period in which oxygenated air was used would appear to have increased the sulphate formation. The higher percentage of soluble zinc in Test No. 5 is due largely to the lower iron content and for the same reason the formation of sulphate is low. While the restricted air supply may also be a contributing factor, its effect is rather to increase the time of roasting than materially affect the nature of the roast.

B. FORMATION OF FERRITES

The results obtained in the roasting tests show that a certain proportion of the zinc in the calcine is insoluble in ammonium chloride solution. This insoluble zinc is due to the formation of the compound ferrite. From the foregoing discussion on ferrite formation it is shown that the temperature of roasting, the iron content, and the mineralogical nature of the ore are the determining factors. The Tainton process claims that a magnetic concentration of the ferrite in the calcine is possible and has embraced this feature in the carrying out of that process. In view of this some magnetic concentration tests were made on roasted Sherritt-Gordon concentrates. Initial tests carried out on a Ball-Norton belt magnetic separator of low magnetic intensity gave negative results. A further test was made on an Ullrich magnetic separator of high intensity in which at all higher intensities the entire sample of calcine was drawn out and only by operating at the minimum magnetic capacity of the machine was it possible to obtain two products. A 5,000-gramme sample was used and the results of the test are tabulated below.

Product	Weight		Analysis			Zn soluble in NH_4Cl	Soluble in 10% H_2SO_4		Soluble in 20% H_2SO_4	
			Total Zn	Fe	NH_4Cl Sol. Zn		Zn	Fe	Zn	Fe
	grms.	%	%	%	%	%	%	%	%	%
Magnetic............	2,678	53·5	41·92	21·63	31·0	74·0	83·7	26·6	88·3	44·9
Non-magnetic........	2,301	46·0	45·80	18·21	35·6	77·6	87·0	33·4	90·8	51·4

An examination of the above results shows that while the magnetic product contains more insoluble zinc and a greater iron content than the non-magnetic there is no complete separation of the ferrites by magnetic separation. There is conclusive evidence, however, to show that the iron content is an important factor in determining the solubility of the zinc. Insufficient data have as yet been obtained to determine the exact mineralogical nature of the ore. But it may be assumed that marmatite is present, although in what proportion remains to be ascertained. It is known that pyrrhotite is associated with the zinc sulphides and its presence would increase the tendency to form artificial ferrites. An examination of the results obtained in several of the roasting tests affords interesting information in the light of this discussion on the formation of insoluble zinc during roasting.

Zn in head	Fe in head	Ratio Fe : Zn	Total Zn in calcine	Zn soluble in NH_4Cl	Insoluble zinc
%	%	%	%	%	%
36·44	18·64	1 : 1·95	39·39	71·5	28·5
36·44	18·64	1 : 1·95	39·46	70·4	29·6
37·95	18·06	1 : 2·10	43·33	75·5	24·5
37·95	18·06	1 : 2·10	43·13	74·9	25·6
37·95	18·06	1 : 2·10	43·03	76·6	23·4
37·95	18·06	1 : 2·10	45·35	77·7	22·3
42·02	15·92	1 : 2·63	46·21	82·4	17·6

A definite relationship is shown between the iron content of the concentrate and the percentage of insoluble zinc in the calcine. A detailed knowledge of the ore would be necessary to determine how much of the ferrite is of the natural variety and how much artificial. The preliminary nature of this work has, however, established the relationship of the iron in the ore to the insoluble zinc in the calcine, and future investigations will endeavour to study this feature of zinc roasting to a greater degree.

Report No. 348

IV. REPORT ON THE TREATMENT OF MIXED (BULK) CONCENTRATE FROM BASE METAL SULPHIDE ORE

W. E. Harris,* R. J. Traill, and H. C. Mabee

The purpose of the test work was to furnish data concerning the suitability of the process of the Base Metals Extraction Co. for the recovery of base metals from a sulphide ore, coupled with the Waelz process for the elimination of the remaining zinc and other volatile metal elements from zinc plant residues. The work was carried out on a co-operative basis between the Base Metals Extraction Co., Ltd., and the Division of Ore Dressing and Metallurgy of the Mines Branch.

A shipment of 8 tons of bulk concentrates was furnished by Treadwell-Yukon Mines, Ltd., at Bradley, Ontario, and was received September, 1929.

The concentrate contained values in zinc, lead, copper, silver, and gold. It also contained carbonaceous material, part of which was present as graphite, a constituent of the ore, and part as coal which in some unexplainable manner had been introduced into the shipment. The presence of this carbonaceous material is unfortunate as it incurs an important and probably injurious effect in roasting. The concentrate was partly air-dried, screened to break up lumps and bagged preparatory to feeding to furnace.

The procedure followed embraced the usual flow-sheet of roasting the concentrate, leaching the calcine and neutralizing to remove dissolved iron and certain impurities, filtering, purification of electrolyte, and electro-deposition.

The leach residues were dried and re-treated by the Waelz process for volatilization of contained zinc and lead, the residue from the Waelz process furnishing a smelter product for recovery of copper, gold, and silver values.

An investigation of this latter phase was in reality the main objective of the test, the roasting and leaching being simply a necessary part of plant practice. The results obtained on the roasting and leaching, however, are of some interest in view of the complex nature of the concentrate under treatment.

ANALYSIS OF BULK CONCENTRATE

Each bag of concentrate fed to the furnace was sampled and the bulk sample so obtained riffle-sampled for analysis.

Zinc	29·89 per cent	Iron	19·39 per cent
Copper	5·53 "	Gold	0·16 oz./ton
Lead	3·44 "	Silver	7·4 "
Carbon	4·65 "		

ROASTING OF BULK CONCENTRATE

The roasting was carried out in a 6-hearth, 36-inch Herreshoff furnace heated by means of a gas (city gas) jet on each of the four lower hearths.

The temporary location of the furnace did not permit the use of a sufficient natural draught to carry off the gases of combustion and roasting,

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and a suction fan had to be connected between the furnace and stack. This arrangement had a tendency to draw the heat to the upper hearths and made temperature control on the upper hearths of the furnace practically impossible. This condition was only slightly improved by connecting part of the suction to the fifth hearth.

The temperature on hearths four and five could be fairly well regulated and controlled, but as the roasting operation progressed the temperature on the upper hearths, particularly the second hearth, would mount up until it equalled and occasionally went higher than on hearths four and five. The reason for this is probably accountable to several factors, namely, fan draught, extreme fineness of concentrate (approximately 300 mesh), high iron sulphide plus copper and lead sulphide and presence of carbon (4·65 per cent) in the form of graphite and coal.

The effect of temperature change on lower hearths, feed rate, and air supply to various hearths was tried with no very great change in calcine composition. The copper solubility was perhaps a little lower at the higher temperatures. On feed rates between 25 and 45 pounds per hour little difference was noted in composition, the low feed rate showing a slightly higher soluble zinc at similar temperatures. With air supply to hearths best results were obtained with the upper three hearths closed and the lower three slightly open. An overall sample of the calcine analysed as follows.

Total Zn	Soluble Zn	Sulphate Zn	Per cent of zinc soluble	Total Cu %	Soluble Cu %
34·0	27·4	3·74	80·5	6·37	4·04
Per cent copper soluble	Pb	Total Fe	Soluble Fe	Au	Ag
63·4	4·37	22·78	6·96	0·18	8·72

The following series of calcine analyses show in some degree the effect of temperature. The temperature shown is the average over a period of several hours on hearths four and five.

Temperature °C.	Total Zn	Soluble Zn	Sulphate Zn	Per cent of zinc soluble	Total Cu	Soluble Cu	Per cent of copper soluble
690 to 700	31·3	28·2	4·2	90·0	5·85	3·60	61·60
	33·8	28·6	4·6	84·6	6·00	3·90	65·00
	31·8	28·4	3·2	89·3	6·68	3·70	56·40
	32·8	27·8	3·5	84·7	6·37	4·17	65·48
	32·8	27·0	3·4	82·3	6·47	4·07	62·9
730 to 750	33·5	29·1	4·6	86·9	6·11	3·91	63·5
	33·9	27·9	4·2	82·4	6·46	3·99	61·8
	34·4	27·8	3·04	80·8	6·75	3·82	56·6
	34·6	28·1	3·49	81·2	5·93	3·66	61·7
	32·4	26·1	2·2	80·5	6·79	3·86	56·9

The above are the results obtained with a feed of 25 to 35 pounds per hour and a calcine discharge of 23 to 27½ pounds per hour. With an increased feed averaging 45 pounds and a calcine discharge of 35½ pounds, results in the neighbourhood of the following were obtained. The temperature was more difficult to control here and hand-rabbling was necessary on some of the hearths to break up lumps and keep the calcine from building up too much on the hearth.

Temperature °C.	Total Zn	Soluble Zn	Sulphate Zn	Per cent zinc soluble	Total Cu	Soluble Cu	Per cent copper soluble
700 to 760	35·87	31·26	7·77	87·2	4·97	3·14	63·2
	36·36	30·97	4·07	85·2	5·69	3·41	60·0
	37·63	30·87	3·99	82·0	6·26	3·18	60·4
	37·04	31·75	4·21	85·7	5·84	3·33	60·1

From these figures it is apparent that no very great difference exists in the results obtained on the solubilities of the zinc and the copper. There are, however, several factors that probably have a bearing on the results. As previously stated control of temperature on upper hearths was practically impossible. The temperature on the second hearth was never below 600° C. and frequently went as high as 700° C. and over. The charge on this hearth was at all times extremely sticky. Such high initial temperatures increase the possibility of $Fe_2O_3 + ZnO$ and $Fe_2O_3 + CuO$ combinations, resulting in ferrites, particularly where there is extreme fineness of ore particles. Lead sulphide fuses without roasting at high initial temperatures and tends to coat zinc particles, causing formation of lumps which pass through the furnace unroasted. These factors may account in some measure for the low zinc solubility obtained, and may possibly affect the copper solubility.

The presence of the carbonaceous matter should also be taken into consideration. It was noticed while the furnace was operating that a zinc reduction reaction was taking place. This was evidenced by a heavy white fume appearing on hearths three, four, and five, and a zinc oxide deposition on the iron hearth-openings.

This zinc oxide fume caused a loss of zinc in the calcine as most of it was carried out of the furnace with the gases to the dust collector.

It seems reasonable to assume that with this strong reducing action going on the copper would also be affected and that the less soluble cuprous oxide might be formed on the lower hearths.

The low copper solubility, however, is the more serious phase in the present instance. The question of the zinc solubility is of lesser importance as the insoluble zinc contained in the leach residues is subsequently recovered by the Waelz process. With regard to the copper, however, the low solubility obtained pre-empted the possibility of getting a solution of sufficiently high copper content to warrant preliminary electrolysis of copper as was originally intended.

Almost 6 tons of concentrate was roasted in the Herreshoff furnace, producing over 4 tons of calcine with dust losses averaging 10 to 12 per cent.

A screen analysis of the roasted calcine gave 27 per cent +30 mesh, 8·5 per cent +100 mesh, 7·25 per cent +200 mesh, and 57·25 per cent −200 mesh.

LEACHING OF CALCINE

The leaching of the calcine was simple and presented no difficulties. It was carried out in a lead-lined cylindrical vat, 3 feet 6 inches deep by 3 feet in diameter, equipped with a mechanical agitator, and containing lead coils for the purpose of heating the solution.

The actual leaches were carried out on the batch system with an 18 per cent acid basis solution, each charge being about 400 pounds of calcine.

The calcine was added to the continuously agitated acid solution and agitation continued until the acid was reduced to about one-half of one per cent. The leach was then heated (steam coil) to the boiling point for about ten minutes resulting in a further decrease of acidity to close to the neutral point.

At this stage about eight-tenths of one per cent (in relation to calcine) of manganese dioxide was added to oxidize any ferrous iron, followed by a gradual addition of powdered limestone to slight excess to neutralize any free acid present and to precipitate silica and iron. The charge was then re-heated to the boiling point to bring about coagulation of the precipitate. The resulting crude solution contained between 130 and 150 grammes per litre of zinc and between 12 and 15 grammes per litre in copper. The latter figure being too low to permit of direct electrolysis of copper in the filtered solution, it was decided to save time by precipitating the dissolved copper into the residue in the majority of the leaches made.

This was done during the coagulation period, zinc powder being added in slight excess to precipitate the copper. As soon as the solution tested copper-free it was again tested for iron and any iron found reprecipitated and the charge filtered. The effect of this boiling results in easy filtration even in the presence of the large proportion of fines occasioned by the fineness of the concentrates used. The filter used was a United Filters plate and frame press fitted with a Gould's triplex plunge pump. Filtration was easily carried out at a pressure of 30 to 40 pounds per square inch until just toward the end when the press became nearly full and the final packing of the press had to be completed at about 80 pounds per square inch.

The cake was dense, hard, and dry in appearance, yet showed an average of 21 per cent moisture on analysis. The average amount of residue obtained on the larger leaches was about 60 per cent of the calcine feed, but on a series of smaller controlled leaches, residues of 53 to 54 per cent were obtained. The following table shows analyses of residues in percentages.

—	Total zinc	Soluble zinc	Sulphate zinc	Total copper	Soluble copper	Total iron	Total lead	Gold, oz./ton	Silver, oz./ton
1	17·4	10·18	3·74	8·3	3·51	31·89	5·88	0·24	12·36
2	14·75	7·97	3·10	3·92	1·44	34·50	6·27	0·24	11·44
3	15·70	8·27	2·99	4·54	1·34	37·58	6·40	0·24	12·44
4	15·75	8·96	2·75	5·16	1·96	35·96	6·38	0·28	11·04
5	13·52	6·50	0·50	5·10	1·92	37·22	6·04	0·28	13·16
6	12·58	7·26	0·79	4·64	1·72	35·33	6·05	0·28	12·84

In the above table (1) represents the average analysis of seventeen large leaches where the copper has been precipitated during the coagulation period into the residue. Analyses (2) to (6) are of residues obtained from the regular plant practice method of filtering prior to the usual precipitation of the copper in the solution by zinc dust.

It should be stated that, with the Waelz process in mind, no serious effort was made to obtain maximum extraction of zinc in these leaches, calculations showed, however, that about 80 per cent of the total zinc and about 90 per cent of the soluble zinc were obtained in solution. With regard to the copper no definite figures on solubility were obtained inasmuch as the copper was precipitated with the insoluble residue. Samples of solution, however, were taken prior to the precipitation of the copper and coupled with samples taken in the regular method (2) to (6) it would appear that about 50 per cent of the total copper and 80 per cent of the soluble copper were obtained in solution.

The solution obtained after precipitation of the copper, filtration, and press washing, contained 120 to 130 grammes per litre of zinc and was free from iron and copper, but contained nickel, a highly undesirable impurity, in the approximate proportion of one part in two million.

This nickel impurity is readily removed by the mercury salt plus zinc dust method, a patented process of the Base Metals Extraction Co. The method consists of adding to the zinc solution a very small quantity of mercury sulphate solution together with a small quantity of acid. Zinc dust is then added and the whole agitated preferably in heated solution. In the purification of the above solution the amounts of the reagents used were 0·01 per cent mercury sulphate, 0·15 per cent acid, and 0·15 per cent zinc dust in quantitative relation to the solution. This method of purification will precipitate in one operation all metals electro-negative to zinc that may be in solution. In the treatment of this precipitated residue for recovery of metals such as cadmium, cobalt, or nickel about 70 per cent of the mercury can readily be recovered.

The purified zinc solution was electrolysed as follows for recovery of zinc.

ELECTRO-DEPOSITION OF ZINC

Electrolysis was carried out in 6 lead-lined cells arranged in cascade and connected electrically in parallel. Each cell carried 2 aluminium cathodes measuring roughly 20 by 14 inches and pure lead anodes of approximately similar dimensions.

A continuous run of 93 hours' duration was made at an average current of 246 amperes (C.D. 30 amps. per square foot) and 3·58 volts per cell, producing some 334 pounds of zinc metal, with a 90·7 per cent current efficiency. These figures show the actual power required for deposition of zinc to be 1·47 k.w.-hr. direct current per pound or 2,940 k.w.-hr. per ton of zinc.

The average deposition period was 35 hours and the spacing between electrodes approximately 3 inches. No glue or other colloid or addition agent was used. The head tank liquor analysed 10·6 per cent zinc and 2·0 per cent acid, and the outflow liquor analysed 3·6 per cent zinc and 12·5 per cent acid. The resulting cathodes were smooth and dense.

RECOVERY OF THE ZINC AND LEAD FROM LEACHED RESIDUES BY THE WAELZ PROCESS

The process developed and patented in Germany consists briefly in the treatment of a mixture of leached or other base metal residues with a reducing agent such as anthracite coal dust, coke breeze, or waste coal at temperatures from 1,000° to 1,200° C., under which conditions the lead and zinc content is reduced to the metallic form and immediately volatilizes. The volatile metals coming in contact with free air are again converted to the oxides and pass out of the kiln as fume through cleaners and ultimately to the collectors.

The operation is carried out in a long rotary kiln from 120 to 140 feet in length and from 6 to 9 feet in diameter.

The charge consisting of the ore intimately mixed with the carbonaceous material is charged through feeders into the upper or slightly elevated end of the kiln, the residue being carried forward by the rotary action, from one to two revolutions of the kiln per minute, and is finally discharged at the lower end in the form of sinter.

In the case of ores or material carrying gold, silver, copper, or other values such as, for example, in the ore under investigation the sinter produced by the process is in a state very suitable for the smelting and recovery of these values. Where no value is placed in the metallic content of the final sinter it may be dumped or can be worked up, as in the case in some parts of Germany, into road metal.

The kiln used for the volatilization experiments in the present case consisted of a rotary furnace, 22 feet long, with a uniform diameter of 15 inches and with a slight incline so that the upper or charging end was about 10 inches higher than the lower or discharge end. The necessary heat was obtained by means of city gas through air pressure burners; the air pressure being supplied by means of a rotary blower.

The upper or feed end of the furnace was inserted into a square chamber through which the charge was fed into the furnace by means of a worm screw.

The metallic fumes discharging into the chamber were conducted upward from the top through a 10-inch pipe and then downward, a total length of about 12 feet to a suction fan. The discharge from the fan being forced through another 10-inch pipe to a second chamber which was directly connected with the flue to the stack.

The object of these chambers was to check as far as possible a direct draught and to collect some of the fume for further examination.

It may be well to mention that the furnace, as well as the arrangement of the apparatus as described above, was employed to demonstrate the principle only of the Waelz process and is not intended to be descriptive of the Waelz process. None of the features of the process were embodied other than the passing of the material through a revolving furnace under the proper condition for the volatilization of the contained zinc and lead. Whereas considerable mechanical losses from dusting were experienced, with the regular Waelz plant there are practically no losses for the reason that provision is made for the gravity separation and settling-out of the heavy dust particles from the fumes and their return automatically to the furnace.

No definite arrangement was made other than the cooling-chambers as mentioned above for the condensation and collection of the metallic fume. It was assumed the process of cleaning, condensing, and collecting of metallic fumes by means of electrical precipitators, etc., to be of common knowledge and practice and therefore unessential in the present experiment.

ANALYSIS OF LEACHED RESIDUES

A sample of the leached residues was taken from each batch after filtering and analysed for the zinc-lead-copper content. The remaining portions being collected into one composite sample from which an average sample representing the complete run was selected for complete analysis as follows:—

Zinc	17·40 per cent	Lead	5·88 per cent
Copper	8·30 "	Gold	0·24 oz./ton
Iron	31·39 "	Silver	12·36 "
Moisture	21·0 "		

The residues after partial drying were intimately mixed with fine coke breeze in the proportion of about 36 per cent of breeze to 74 per cent of residues in the dry state.

Some 2,550 pounds of the mixture was fed through the furnace requiring 38 hours to complete the run.

Owing to the combined action of the air blast and the exhaust fan considerable difficulty was experienced in fine dust being carried out of the furnace. About 120 pounds of fines collected toward the end of the run were moistened and re-mixed with the charge.

From the total weight of residues treated, 950 pounds of sinter was produced which represents about 74 per cent of the original weight.

Samples of sinter were collected every half hour during the entire run and those collected towards the end of the run showed an average of 0·8 per cent zinc, and from a trace to 0·4 per cent lead.

These results represent a recovery, or successful volatilization, of 96·8 per cent of the zinc content in the leached residues or 99·48 per cent recovery of the total zinc in the original calcine and from 95 per cent to very nearly a total extraction of the lead.

A sample representing the last few hours of the run gave the following analysis:—

Zinc	0·61 per cent	Sulphur	4·18 per cent
Copper	11·76 "	Gold	0·34 oz./ton
Lead	trace	Silver	15·26 "
Iron	43·57 "		

These results show that the main object of the test was fully accomplished, viz., the elimination of the zinc and lead to a very satisfactory degree, and, further, the production of a sinter carrying the copper and precious metal values in a suitable form for smelting.

SUMMARY AND CONCLUSIONS

Although it is shown that successful results may be accomplished by adopting this method of treating mixed concentrates, it is doubtful if the procedure could be recommended in the commercial treatment of a bulk or mixed Treadwell-Yukon concentrates owing to the complex nature of this material and the insoluble compounds which are brought about in the roasting.

In the course of concentrating this ore it has been found necessary to produce a copper-lead concentrate followed by a zinc concentrate and it would seem advisable to leave the concentration process at this stage. With these two products the zinc could be roasted and leached separately, the resultant solutions electrolyzed for the zinc content and the residues passed through the Waelz process for the recovery of the undissolved zinc: or it may be found even more advantageous to add these residues to the copper-lead roasted concentrates, and the mixture of residues and roasted concentrates treated by the Waelz process direct, provided, however, that the addition of these residues does not bring about too great a dilution of the total copper content and thereby a final product uneconomic for smelting. By this means much overlapping would be avoided and better recoveries could be expected both in the milling and in subsequent operations.

V. PRELIMINARY TESTS ON THE USE OF COKING COALS IN METALLIZATION OF IRON ORES

W. R. McClelland

Ore and Coal. The ore used in the following tests was from a sample of gravity concentrate of low-grade banded iron ore from Kaministikwia, Ont., concentrated in the Ore Dressing Laboratory.

The coals used were samples of bituminous coking coals from Sydney, N.S., and Minto, N.B.

Purpose of Test Work. In the usual process for the making of sponge iron carried out with direct heat on the charge, the use of coking coals has been found to be quite unsatisfactory. A newly developed process, known as the Musso process, claims that coking coals can be used with the same ease as those of non-coking quality.

The Musso process is a direct carbon reduction one, as opposed to that of gas (CO) reduction used in most other processes.

Briefly a mixture of ore and coke is charged into a revolving air-tight cylinder. The carbon monoxide as formed is drawn off by means of tubes and burned in a furnace surrounding the cylinder, thus furnishing the required heat for the reaction.

The purpose of the following tests was to determine if a satisfactory sponge iron product could be obtained with the use of a coking coal.

Characteristics of Ore and Coal. The ore is a mixture of magnetite and hematite in the following proportions:—

Fe_3O_4	24·36 per cent
Fe_2O_3	19·12 "

The concentrate analysed approximately 50 per cent iron.

The coals have an approximate fixed carbon analysis as follows:—

Sydney	60 per cent
Minto	50 "

EXPERIMENTAL WORK

The ore and coal were crushed to pass a 50-mesh screen and mixed in suitable proportions for each charge.

As no small rotary furnace was available for these tests, a small vitreosil muffle was employed to contain the charge. To approximate conditions as close as was possible to the Musso furnace, the muffle was closed with an asbestos cover through which a silica tube extending outside the furnace was placed. This tube was connected to a vacuum pump and a slight vacuum maintained to draw off the gases formed during the reaction.

Test No. 1

Charge:

Ore	200 grammes
Coal (Sydney Mines)	150 "

The ore and coal were thoroughly mixed and charged to the vitreosil muffle and placed in the electric muffle furnace. A slight suction was maintained, the rate being gauged by a wash bottle in the air-line circuit.

The temperature ranged from 726° C. to 976° C. for 6 hours. The charge was allowed to cool in the furnace.

Tar and low-temperature oils collected in the tube leading from the muffle.

Weight of product	253·5 grammes

The product had a similar appearance to sponge made with non-coking coals. A portion formed a compact sinter which was easily broken up. Concentration was effected by a hand magnet.

Weight of magnetics	217·5 grammes
Weight of non-magnetics	33·0 "

Analysis of Magnetics

Total iron	43·37 per cent
Metallic iron	36·43 "

Metallization—84 per cent.

Test No. 2

Charge:

Ore	200 grammes
Coal (Sydney Mines)	100 "

Less coal was used than in Test No. 1, where there was a large excess of coal.

The front of the muffle was cemented up with alundum cement and a manometer was connected to the suction line from the muffle to determine the vacuum established in the hot muffle. A slight vacuum was recorded during the run.

The usual distillate of oil was found in the suction line. The temperature range for 6 hours was 746° C. to 976° C. The final product weighed 218 grammes. The sponge differed from Test No. 1 in that it was practically in powdered form, any sintered masses breaking up on handling. The appearance in every way resembled sponge made with non-coking coals.

Weight of magnetics	191 grammes
Weight of non-magnetics	1 "
Weight of lump sample (specimen)	26 "
	218

Analysis of magnetics

Total iron	45·61 per cent
Metallic iron	37·55 "

Metallization—82·3 per cent.

Test No. 3

Charge:

Ore	200 grammes
Coal (Minto Mines)	100 "

Conditions of operation were similar in all respects to that of Test No. 2. Greater care was exercised in cementing the cover on the muffle to ensure against leakage of air.

The test was carried out for the same length of time as previous tests, but a slightly lower temperature was maintained. The range of temperature was 636° C. to 946° C. over a 6-hour period. The usual distillates from the coal collected in the delivery tube.

No difficulty was experienced in controlling the vacuum in the muffle.

The furnace product weighed 215 grammes. It was black in colour and formed a compact easily friable sinter. In appearance it closely resembled normal sponge iron product. The product was broken up and concentrated by a hand magnet.

Weight of magnetics	201·5	grammes
Weight of non-magnetics	0·5	"
Weight of sample specimen	12·0	"
	214·0	"

Analysis of magnetics

Total iron	47·55 per cent
Metallic iron	42·45 "

Metallization—89·2 per cent.

CONCLUSIONS

From the above preliminary tests, the claims of the Musso process regarding the use of coking coals are verified.

The fineness of the charge probably accounts for the low non-magnetic tail found on concentration. A hand magnet also tends to draw up finely divided carbonaceous matter with the metallics.

The establishing of a vacuum in the hot charge is an interesting feature of the process and one that opens up many possibilities.

The drawing off of the volatile constituents of the coal as formed, together with the CO, should make a much purer iron sponge. There is the possibility of eliminating some of the sulphur in this way, but further experimentation would be necessary to verify this assumption.

The use of coking coals should offer no serious obstacle to the carrying out of this process in the rotary furnace on a large scale.

The distillates from the coal do not block up the tube in the hot part of the furnace. A suitable receiver and scrubber immediately at the outlet of the furnace would eliminate any tendency for condensation of oils in the gas lines.

The recovery of these distillates might form a by-product as the process resembles somewhat the low-temperature carbonization of coal.

The low metallization in Tests Nos. 1 and 2 was probably due to the muffle cover not forming an air-tight seal, thus allowing air to enter the muffle. In Test No. 3 the cover was satisfactorily sealed.

V

REPORT OF INVESTIGATIONS: LABORATORY OF FERROUS METALLURGY

Thos. W. Hardy

The construction and equipping of the new pyrometallurgical laboratory extended over most of the year and on this account it was not possible to carry out any comprehensive plan of investigation. By the end of the year all of the equipment listed in the last summary report as proposed for the pyrometallurgical laboratory with the exception of the one-ton Héroult arc furnace had been installed.

Unavoidable circumstances have delayed the erection of the building that was to provide laboratory space for metallographic, mechanical testing, and other equipment necessary for the carrying out of the work of the section. Until these facilities are available the scope and activities of this section will necessarily be very restricted.

Arrangements have been completed for the carrying out of a series of tests to demonstrate the technical and commercial possibilities of the Musso process for the semi-direct production of steel from iron ore. This work is being undertaken in co-operation with a group of Canadian financiers interested in the process, at whose expense a pilot plant especially designed for this demonstration, will be erected in the pyrometallurgical laboratory. It is expected that these tests will begin in March and that they will extend over several months.

Involving as they do, as the first operation, the production of sponge iron, the proposed tests on the Musso process fit well into the plans for the investigation in general of sponge iron processes and their application to Canadian ores and conditions. As is now generally appreciated, Canada has no known large deposits of primary iron ore suitable for blast furnace use. However, deposits of considerable magnitude of siderites, magnetites and magnetite-hematite mixtures do exist, many of which can, as has already been demonstrated by tests carried out by the Ore Dressing Laboratory, be made suitable for blast furnaces by some form of beneficiation. That these deposits are not now being utilized is due to the fact that primary ores of foreign origin can at present be imported and laid down at Canadian furnaces more cheaply.

In considering the possible application of low-temperature reduction methods to Canadian low-grade ores, it must of course be kept in mind that in addition to reduction, the question of beneficiation is necessarily involved, since it is even less desirable to charge material high in gangue materials into a steel melting furnace than into the blast furnace. That certain ores may be more cheaply beneficiated after low-temperature reduction is one reason for the interest evinced in sponge iron. That some ores cannot be adequately beneficiated either before or after low-temperature reduction is apparently not fully appreciated by many of those interested in the commercial development of sponge iron. In general, it may be stated that the more intimately the gangue materials of low-grade ores are associated with the iron, the less likely they are to prove suitable for the production of a useful sponge iron. Small-scale experiments on the low-temperature reduction of one Canadian low-grade ore have been carried out already, and the results indicate that, in this case, the failure of the reduced material to adequately respond to magnetic concentration will make it just as unsuitable for the production of commercial sponge iron as it is for the production of pig iron by the means now generally used.

REPORT OF THE WORK AND INVESTIGATIONS OF THE CHEMICAL LABORATORIES OF THE DIVISION

H. C. Mabee

Chief Chemist

The activities of the staff of the chemical laboratory were as in previous years concerned with and, for the most part, guided by the activities and amount of research investigations conducted in the testing and research laboratories of the Division.

During the year 1929 the number of samples for chemical analyses from the various investigational divisions exceeded those of any previous year by over 50 per cent.

With insufficient housing provided for the accommodation of additional chemists or adequate laboratory facilities the present staff was so taxed in meeting the extra demand connected with the regular test work on ore treatment that very little opportunity was afforded for special research investigations.

The Mines Branch following the co-operative arrangement made with the Cassel Cyanide Company, in January, 1928, continued to provide accommodation and facilities for D. S. Halford, research fellow, in carrying out research on the cyanide selective flotation of base metal sulphide ores.

Active co-operation by members of the staff with the Canadian Base Metals Extraction Company was also continued throughout the year in the furnishing of analyses and assays connected with their experimental tests in the hydrometallurgical treatment of Canadian base metal sulphide ores.

The writer in addition to supervising the work and investigations of the chemical laboratory also assisted in supervising the installation of roasting, calcining, and other pyrometallurgical equipment, and in the carrying out of zinc-lead volatilization experiments on leached residues from base metal ores.

During the year extensive investigations, both physical and chemical, were conducted on Canadian gypsum, graphite, uraninite, and other non-metallic ores for members of the staff of the Division of Mineral Resources.

In the following table is given a list of the ores and products upon which reports were issued during 1929. This list indicates the class of ore and the number of samples in each particular case.

Carbonaceous material	3	Identification and valuation	24
China clay	11	Iron ore	160
Copper-gold ore	454	Lead ore	19
Copper-lead-zinc ore	178	Molybdenite	65
Copper-zinc-barite ore	73	Nickel-copper ore	21
Copper-zinc ore	215	Rock samples	20
Graphite	17	Silica sands	142
Gold ore	530	Silver ore	58
Gold-silver ore	132	Iron ore	4
Gypsum	43		

The investigations and work of the chemical laboratory were carried out under the supervision of H. C. Mabee, Chief Chemist of the Division of Ore Dressing and Metallurgy.

The investigations in connexion with the metallic ores were performed by B.P. Coyne and H. L. Beer; the non-metallic investigation by R. A. Rogers; and the fire assaying by L. Lutes with the assistance of N. A. Parker, laboratory helper.

At the close of the year it was found necessary to provide extra laboratory accommodation and equipment for the purpose of carrying on the analysis of iron and steel in connexion with the investigations in progress on the direct reduction of iron oxide and the final production of steel.

Temporary quarters for this work have been arranged in a part of the present hydrometallurgical laboratory, the work being assigned to J. D. Johnston with the assistance of A. E. LaRochelle, senior laboratory assistant.

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