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)

January to June, 1934

OTTAWA
J. O. PATENAUDE, I.S.O.
PRINTER TO THE KING'S MOST EXCELLENT MAJESTY
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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). Semi-annually.

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.

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MINES BRANCH INVESTIGATIONS IN

ORE DRESSING AND METALLURGY, JANUARY TO JUNE, 1934

I

REVIEW OF INVESTIGATIONS

W. B. Timm
Chief of Division

During the half year ending June 30, 1934, twenty-nine reports were published. Of these, twenty-two were on ores in which gold was the chief metal of value; one on a lead-zinc ore containing gold and silver from the Howard mine in the Nelson district, B.C.; one on the silver and silver-pitchblende ores from Contact lake, Great Bear Lake district, N.W.T.; one on the silver ore from Camsell River, N.W.T.; one on the lead-copper-zinc ore containing gold and silver from the Stirling mine, Cape Breton, N.S.; one on copper-nickel ore containing platinum group metals from the Timagami Forest Reserve, Ontario; one on the sintering of the flotation concentrates from Beattie mine, Duparquet township, Abitibi county, Quebec; and one on the concentration of corundum from Dungannon township, Renfrew county, Ontario.

Of the gold ores investigated, one was from British Columbia, one from Saskatchewan, eleven from Ontario, seven from Quebec, and two from Nova Scotia. As the result of these investigations, by the end of 1934 a 50-ton amalgamation plant was in operation at the Casey Summit mine in the Patricia district, Ontario; a 200-ton amalgamation-cyanide plant at Little Long Lac mine, Geraldton, Ontario; and a 100-ton cyanide plant at the Matachewan Consolidated property, Matachewan, Ontario. In addition a 125-ton amalgamation-cyanide plant is being built at the Pickle Crow mine in Patricia district, Ontario; and a 100-ton cyanide plant at the McKenzie-Red Lake property in the Red Lake area of the Patricia district, Ontario. A 75-ton amalgamation-flotation-cyanide plant was built at the Greene Stabell mine in Dubuisson township, Abitibi county, Quebec; a 75-ton amalgamation plant at the McWatters mine in Rouyn township, Quebec, to which a cyanide plant is to be added later; and a cyanide plant installed at the Beattie mine, Duparquet township, Abitibi county, Quebec, for the treatment of the flotation concentrates from their 1,200-ton concentrator; a 50-ton amalgamation-concentration plant was built at the Montague mine, Montague, Nova Scotia. A 150-ton cyanide plant is contemplated for the Arntfield mine, Arntfield, Quebec, and milling plants are being considered for the other properties, the ores of which have been investigated. The investigation of the lead-copper-zinc ore of the Stirling mine was undertaken in anticipation of higher prices for the base metals, when the concentration plant would be operated again after remodelling to conform with the results of the test work.

In addition to the reports of investigations which are published, thirty-five investigations of ores, non-metallic minerals, and metallurgical products were reported upon as follows:—

Gold ore from Wayside Consolidated Gold Mines, Limited, Bridge River district, B.C.

Gold ore from Ceepeecee, west coast of Vancouver island, B.C.

Antimonial-gold tailings from West Gore, Hants county, N.S.

Gold-bearing concentrates from Bussières Mining Company, Limited, Senneterre, Que.

Lead-zinc-gold ore from Canyon Creek, Bayonne district, B.C.

Gold-copper ore from Rosebery mine, Slocan City, B.C.

Arsenical gold ore from Thompson-Cadillac Mining Company, Limited, Cadillac township, Abitibi county, Que.

Gold ore from Mr. Algot Nelson, Kenora, Ontario.

Gold ore and mill products from Granada Gold Mines, Limited, Rouyn, Quebec.

Gold ore from McCarthy-Webb property in the Goudreau-Lochalsh area, Algoma district, Ontario.

Gold ore from the Mathews Gold Mines, Limited, Pascalis township, Abitibi county, Quebec.

Gold-silver-lead-zinc ore from the Goodenough mine, Ymir Gold Mines, Limited, Ymir, B.C.

Copper-gold ore from the Lee Gold Mines, Ltd., Greenlaw township, Sudbury district, Ontario.

Gold ore from Mining Claims 29114-5, Scadding township, Sudbury district, Ontario.

Gold ore from the Wanapitei Gold Syndicate, Scadding township, Sudbury district, Ontario.

Gold-bearing concentrates from Northern Empire Mines Co., Limited, Empire, Ontario.

Gold ore from Craig Gold Mines, Limited, Madoc, Ontario.

Gold ore from Mining Claim T B 10971, Little Long Lac area, Ontario.

Gold-silver ore from Favourable lake, Patricia district, Ontario.

Opal silica from Wanaki, Ontario.

Graphite from St. John, New Brunswick.

Quartz from Chicoutimi, Quebec, for sandblasting.

Garnet-bearing rock from Labelle county, Quebec, for sandblasting.

Quartz from Lac Rémi, Quebec, for sandblasting.

Silica sand from Keoma, Alberta, for sandblasting.

Anhydrite from Sydney, Nova Scotia.

Silica sand from St. Andrews East, Quebec, for sandblasting.

Garnet products from Labelle county, Quebec, for sandblasting.

Gypsum from Consolidated Mining and Smelting Company, Limited, Trail, B.C.

Endurance and other tests on two locomotive side rods for Canadian National Railways, Montreal, Quebec.

Grain size determinations of several samples of steel for Canadian Atlas Steels, Limited, Welland, Ontario.

Carburization and heat treatment of 72 small parts for camera mounts for Department of National Defence.

Deep etching tests and inspection of 64 recuperator adapter breech lugs from 18-pounder Q. F. Guns for Department of National Defence.

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

The microscopic examination and spectrographic analyses of ores and mill products were performed by M. H. Haycock.

The test work on metallic ores was performed by C. S. Parsons, R. J. Traill, A. K. Anderson, J. D. Johnston, W. R. McClelland, and W. S. Jenkins.

The test work on non-metallic minerals was performed by R. K. Carnochan and R. A. Rogers.

The metallurgical work on iron and steel products was performed by T. W. Hardy and H. H. Bleakney.

The chemical work was performed by a staff of chemists under the direction and supervision of H. C. Mabee, Chief Chemist.

II

REPORTS OF INVESTIGATIONS

Ore Dressing and Metallurgical Investigation No. 551

COPPER-GOLD ORE FROM THE MANITOBA AND EASTERN MINES, LIMITED, GOWARD, ONTARIO

Shipment. A shipment consisting of 24 bags containing 1,900 pounds of ore was received November 14, 1933, from the Manitoba and Eastern Mines, Limited, Horace F. Strong, Manager. The material was said to have been taken from mining leases W. S. 13 and 14, W. D. 458-465, Strathy township, Timagami, Ontario. The shipment consisted of three lots, Nos. 1, 2, and 3, of approximately equal weight.

Characteristics of the Ore. These lots were apparently from the surface and were somewhat oxidized.

Specimens of the three lots were examined in the mineragraphic laboratory. Polished sections showed that the gangue differs considerably in the three lots. In Lot No. 1 it is light grey to white quartz; in Lot No. 2, fine-textured, grey dioritic rock locally impregnated with grey quartz; and in Lot No. 3, dense, greenish grey rock penetrated by irregular veinlets of quartz.

The metallic minerals present in considerable amounts are pyrite, arsenopyrite, chalcopyrite, sphalerite, and limonite. Minor amounts of pyrrhotite, covellite, native bismuth, chalcocite, galena, and cosalite are present. Pyrite and arsenopyrite are the most abundant sulphides and are usually disseminated in the rock gangue. Both sulphides are rather coarse-textured and many grains show numerous fractures which have been filled with later gangue and chalcopyrite.

Experimental Tests

Each lot was sampled and assayed with the following results:—

Lot No.	Assay			
	Gold, oz/ton	Silver, oz/ton	Copper, per cent	Arsenic, per cent
1	3·88	6·98	2·08	0·10
2	0·60	3·70	1·36	1·46
3	0·39	1·44	0·11	12·55

For experimental purposes, equal weights of all three lots were taken and combined. This composite sample assayed 1·59 ounces gold, 3·69 ounces silver per ton; 1·41 per cent copper, 0·09 per cent bismuth, and a trace of tellurium.

As the ore contains much copper, straight cyanidation is not applicable. This metal must first be removed.

Numerous tests were made to note the effect of flotation, the chief of which are detailed below.

Test No. 1

A sample of ore was ground wet with 4 pounds lime and 0·14 pound Aerofloat No. 25 per ton until 85 per cent passed 200 mesh, 0·10 pound sodium xanthate and 0·07 pound pine oil were added to the cell, and a flotation concentrate removed.

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Copper, per cent	Gold, oz/ton	Silver, oz/ton	Copper	Gold	Silver
Feed	100·0	1·43	1·52	3·59	100·0	100·0	100·0
Flotation concentrate	38·3	3·28	3·62	7·22	87·9	91·1	77·0
Flotation tailing	61·7	0·28	0·22	1·34	12·1	8·9	23·0

No concentration of copper was obtained. Iron pyrites floated quite readily. This constitutes a bulk flotation with a low ratio of concentration, viz.: 2·6 : 1.

Test No. 2

In this test an attempt was made to separate the copper minerals from the iron pyrite.

A sample was ground wet with 20 pounds soda ash, 0·10 pound sodium cyanide and 0·14 pound Aerofloat No. 25 per ton, 0·10 pound sodium xanthate was added to the flotation cell and a concentrate removed after adding 0·08 pound pine oil per ton.

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Copper, per cent	Gold, oz/ton	Silver, oz/ton	Copper	Gold	Silver
Feed	100·0	1·27	1·27	3·43	100·0	100·0	100·0
Concentrate	8·6	12·02	11·38	24·22	81·3	77·0	60·6
Tailing	91·4	0·26	0·32	1·48	18·7	23·0	39·4

It is apparent that the larger amount of the gold is associated with the copper minerals. The concentrate assaying 12·02 per cent copper and 11·38 ounces gold per ton contained 77 per cent of the gold.

Test No. 3

In this test, conditions were such that with a normal gold ore most of the gold should be found in the tailing. The copper was removed by flotation and the tailing cyanided.

A sample of the ore was ground to pass 86 per cent through 200 mesh with 16 pounds lime per ton to depress gold and pyrite. Aerofloat No. 25, 0·14 pound, was added to the grinding mill; 0·10 pound sodium xanthate and 0·12 pound pine oil per ton were added to the flotation cell and a concentrate removed. The flotation tailing was then agitated, 1 : 3 dilution, with a 3·0 pound KCN solution for 48 hours; 7 pounds lime per ton of ore was added for protective alkalinity.

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Copper, per cent	Gold, oz/ton	Silver, oz/ton	Copper	Gold	Silver
Feed	100·0	1·32	1·61	3·40	100·0	100·0	100·0
Flotation concentrate	20·4	6·18	6·54	12·34	95·8	82·7	74·0
Flotation tailing	79·6	0·07	0·35	1·11	4·2	17·3	26·0

Here again, 82·7 per cent of the gold is found with the copper concentrate. This has the following analysis:—

Gold	6·54	oz./ton
Silver	12·34	"
Copper	6·18	per cent
Sulphur	23·9	"
Arsenic	7·47	"
Bismuth	0·82	"
Silica	19·2	"

Cyanidation of the flotation tailing gave a residue containing 0·065 ounce gold and 0·38 ounce silver per ton, with recoveries of 81·5 and 65·8 per cent respectively.

This combination, flotation followed by cyanidation, gives recoveries of 95·8 per cent of the copper, 96·0 per cent of the gold, and 88·8 per cent of the silver. The concentrate represents 20·4 per cent of the weight of ore milled.

SUMMARY AND CONCLUSIONS

The sample furnished was somewhat oxidized, therefore these tests are inconclusive; fresh ore, probably, will respond more readily. A separation of most of the copper from the bulk of the ore is necessary if cyanidation be desired, otherwise an excessive consumption of cyanide will result. The concentrate produced will require separate treatment at a smelter.

Ore Dressing and Metallurgical Investigation No. 552

GOLD ORE FROM B.C. CARIBOO GOLD FIELDS, NEAR MOYIE, BRITISH COLUMBIA

Shipment. A shipment consisting of five bags of ore weighing 560 pounds was received November 14, 1933, from C. S. Lord, Moyie, B.C., for the B.C. Cariboo Gold Fields, 919 Stock Exchange Building, Vancouver. The ore was said to have been taken from the company's property five miles from Moyie.

Characteristics of the Ore. Specimens of heavily mineralized ore were selected and examined in the mineragraphic laboratory. Polished sections showed the gangue to be chiefly white to greyish white vein quartz. The metallic minerals in their order of abundance are pyrite, arsenopyrite, sphalerite, galena, tetrahedrite, and chalcopyrite.

Pyrite and arsenopyrite are coarsely disseminated in the quartz and commonly form massive granular aggregates, which have been much fractured. The remaining sulphides are mutually associated in small irregular stringers in quartz or in small masses and fine veinlets in the massive pyrite and arsenopyrite. Sphalerite, galena, and tetrahedrite are present in appreciable amounts but chalcopyrite is rare. No native gold was seen.

EXPERIMENTAL TESTS

The shipment was crushed, ground, and sampled. Analysis showed it to contain 0·295 ounce gold, 1·18 ounces silver per ton; 0·10 per cent copper, and 0·73 per cent arsenic.

Experimental tests were made to determine if the gold and silver could be concentrated by flotation. Cyanidation was also tried.

The results of the investigation show that only 64 per cent of the gold is recovered by cyanidation and that 93 per cent of the gold and 95 per cent of the silver are recovered by flotation in a concentrate assaying 0·92 ounce gold, 3·6 ounces silver per ton, with a ratio of concentration of 3·2 : 1.

AMALGAMATION

Test No. 1

A sample of the ore ground to pass 48 mesh was amalgamated and a screen analysis made of the tailing.

Screen Analysis:

Mesh	Weight, per cent	Assay, gold, oz./ton
− 48+ 65	1·2	....
− 65+100	18·2	0·30
−100+150	19·6	0·31
−150+200	20·4	0·40
−200	40·6	0·34

These results show no free gold in the sample, and no segregation of gold in any of the sized parts of the sample.

Test No. 2

A second amalgamation test was made on material ground to pass 100 mesh. The tailing was found to contain 0·29 ounce gold per ton, the same as the original ore. This checks the previous test, no free gold is present.

CYANIDATION

Test No. 3

A series of tests was made to determine the recovery obtained by cyanidation.

Samples of the ore ground to pass 48, 100, 150, and 200 mesh were cyanided with a 1·0 pound KCN solution, 1 : 3 dilution, for 48 hours.

Results:

Mesh	Agitation, hours	Feed assay		Tailing assay		Per cent extraction	
		Gold, oz/ton	Silver, oz/ton	Gold, oz/ton	Silver, oz/ton	Gold	Silver
48	24	0·295	1·18	0·12	0·76	59·3	35·6
48	48	0·295	1·18	0·12	0·61	59·3	48·3
100	48	0·295	1·18	0·105	0·56	64·4	52·5
150	24	0·295	1·18	0·12	0·84	59·3	28·8
150	48	0·295	1·18	0·11	0·50	62·7	57·6
200	24	0·295	1·18	0·14	0·83	52·5	29·7
200	48	0·295	1·18	0·11	0·55	62·7	52·5

Reagent consumption:

KCN .... 3·6 lb./ton ore
CaO .... 8·0 "

Cyanidation of any size of material does not yield high recovery.

FLOTATION

Test No. 4

A sample of the ore was ground wet with 14 pounds soda ash per ton until 86 per cent passed 200 mesh and then floated with 0·20 pound sodium xanthate and 0·14 pound pine oil per ton.

Product	Weight, per cent	Assay		Distribution of precious metals, per cent	
		Gold, oz/ton	Silver, oz/ton	Gold	Silver
Feed	100·0	0·285	1·28	100·0	100·0
Concentrate	31·1	0·84	3·90	91·5	94·6
Tailing	68·9	0·035	0·10	8·5	5·4

Ratio of concentration, 3·2:1.

Test No. 5

A flotation test similar to the preceding one was made and the tailing run over a corduroy blanket to note if any further concentration took place.

Product	Weight, per cent	Assay		Distribution of precious metals, per cent	
		Gold, oz/ton	Silver, oz/ton	Gold	Silver
Feed (cal.)	100·0	0·31	1·21	100·0	100·0
Flotation concentrate	31·5	0·92	3·64	93·2	95·1
Flotation tailing	....	0·03	0·087	....	....
Blanket concentrate	2·2	0·21	0·30	1·5	0·5
Blanket tailing	66·3	0·025	0·08	5·3	4·4

Flotation recovers 93·2 per cent of the gold and 95·1 per cent of the silver in a concentrate weighing 31·5 per cent of the weight of ore milled. This concentrate has the following analysis: gold 0·92 ounce per ton, silver 3·64 ounces per ton, 0·20 per cent copper, 0·61 per cent lead, 2·2 per cent zinc, and 3·65 per cent arsenic.

Passing the flotation tailing over blankets does not produce a high-grade concentrate. The final tailing is slightly lower, but the concentrate is of lower grade than the flotation feed.

SUMMARY AND CONCLUSIONS

The results of the investigation show that flotation is the only method for the economic recovery of the precious metals. The gold apparently is associated with the pyrite and arsenopyrite and for its recovery these minerals must be concentrated. This results in a low ratio of concentration and a rather low-grade product for shipment to a smelter.

Cyanidation of the ore does not give high recovery, 64 per cent of the gold being recovered.

Ore Dressing and Metallurgical Investigation No. 553

GOLD ORE FROM THE EDWARDS MINING PROPERTY AT LOCHALSH, ONTARIO

Shipment. Two shipments of the ore were received on November 23 and December 8, 1933, respectively. The weight of the first shipment was approximately 250 pounds and that of the second shipment 200 pounds. The samples were submitted by H. C. Miller, 702 Atlas Building, Toronto, Ontario.

Characteristics of the Ore. Sample No. 1 and Sample No. 2 are so similar that their characteristics can be described together.

The gangue is fine-textured, light grey carbonate schist which is veined by white quartz. Some of this quartz contains veinlets and fine disseminated grains of carbonate.

The metallic minerals are, in order of their abundance, pyrite, ilmenite, chalcopyrite, and native gold. The quartz of the polished sections is barren, the rather abundant pyrite being disseminated as medium to fine irregular grains and imperfectly formed cubes wholly within the schist. In some places small hazily-bounded inclusions of schist occur within the quartz and contain disseminated pyrite, and to the naked eye this pyrite appears to be contained in impure quartz. Ilmenite occurs as numerous, small, irregular, corroded grains in the schist; locally the ilmenite has been altered to leucoxene.

Chalcopyrite is present as very small rounded and irregularly shaped grains within the pyrite.

All native gold seen occurs within pyrite. It forms tiny rounded or irregularly shaped grains and small, very irregular stringers, and is apparently more abundant in Sample No. 2 than in Sample No. 1. Grains of gold exceeding 200 mesh in size were not seen and most are well below 325 mesh.

Except that native gold, and possibly carbonate, is more abundant in Sample No. 2 than in Sample No. 1 no significant difference was noted. The evidence obtained from microscopic examination indicates that it might be possible to concentrate the gold with the pyrite.

Average assays of the two samples were as follows:

Sample No. 1	Au, 0·59 oz/ton
Sample No. 2	Au, 6·66 "

Experimental Tests

A series of small-scale tests was made on the first shipment of ore only, the second shipment being so rich that results obtained from test work on it would have no practical value. The work on the first shipment included tests by cyanidation, amalgamation, flotation, and hydraulic classification. By cyanidation the maximum extraction obtained was 97·5 per cent of the gold. In the hydraulic classifier 60·2 per cent of the gold settled out in a product amounting to 2·8 per cent of the weight of the ore used. By flotation approximately 95 per cent of the gold was recovered in a sulphide concentrate amounting to about 8 per cent of the weight of feed. About 80 per cent of the gold is recoverable by barrel amalgamation and this drops to about 65 per cent when amalgamation plates are used.

Details of the tests follow:

AMALGAMATION AND CYANIDATION

Tests Nos. 1 and 2

In these tests samples of the ore ground dry through 48- and 100-mesh screens were amalgamated with mercury for 30 minutes in a jar mill. The amalgamation tailings were sampled and assayed for gold and portions of each agitated in cyanide solution, 2·0 pounds KCN per ton, for 24 hours.

Results:

Test No.	Amalgamation tailing, Au, oz/ton	Cyanide tailing, Au, oz/ton	Extraction by amalgamation, per cent	Extraction by cyanidation, per cent	Total extraction, per cent	Reagents consumed, lb/ton	
						KCN	CaO
1	0·12	0·03	79·7	15·2	94·9	1·0	3·5
2	0·125	0·02	80·5	16·1	96·6	1·3	4·1

CYANIDATION

Tests Nos. 3 to 10

In this series of tests samples of the ore ground dry through 48-, 100-, 150-, and 200-mesh screens were agitated in cyanide solution, 2·0 pounds KCN per ton, for periods of 24 and 48 hours. The tailings were assayed for gold.

Results:

Test No.	Mesh	Period of agitation, hours	Tailing assay, Au, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
					KCN	CaO
3	− 48	24	0·035	94·1	1·3	4·1
4	−100	24	0·02	96·6	1·3	4·3
5	−150	24	0·015	97·5	1·3	4·3
6	−200	24	0·015	97·5	1·6	4·7
7	− 48	48	0·03	94·9	1·6	4·8
8	−100	48	0·02	96·6	1·9	5·8
9	−150	48	0·015	97·5	2·5	5·8
10	−200	48	0·015	97·5	2·7	8·5

HYDRAULIC CLASSIFICATION

Test No. 11

In this test a sample of the ore at −14 mesh was ground in a ball mill for 20 minutes and the pulp put through a hydraulic classifier where coarse gold and heavy minerals were allowed to settle against a slowly rising current of water. The oversize and the overflow were assayed for gold. A screen test was made on the overflow to determine the grinding.

Results:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Classifier oversize	2·8	12·60	60·2
Classifier overflow	97·2	0·24	39·8
Feed (cal.)	100·0	0·586	100·0

A screen test on 200 grammes of the classifier overflow showed the grinding to be as follows:

Mesh	Weight, per cent	Cumulative weight, per cent
−100+150	0·4	0·4
−150+200	5·0	5·4
−200	94·6	100·0
Total	100·0	....

FLOTATION

Test No. 12

In this test 2,000 grammes of the ore at minus 14 mesh was ground in a ball mill for 20 minutes and then floated.

Charge to ball mill:

Ore	2,000 grammes
Water	1,500 c.c.
Soda ash	5·0 lb/ton
Aerofloat No. 25	0·07 "

Reagents to cell:

Potassium amyl xanthate	0·10 lb/ton
Pine oil	0·05 "

Results:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Concentrate	7·9	6.54	94·9
Tailing	92·1	0·03	5·1
Feed (cal.)	100·0	0·54	100·0

Ratio of concentration, 12·6 : 1.

AMALGAMATION AND FLOTATION

Test No. 13

In this test a sample of the ore at minus 14 mesh was ground in a ball mill for 20 minutes and the pulp amalgamated with mercury for 30 minutes in a jar mill. The amalgamation tailing was then floated. The flotation concentrate and tailing were assayed for gold and the amalgamation tailing assay calculated from these.

Charge to ball mill:

Ore	2,000 grammes
Water	1,500 c.c.

Reagents to cell:

Soda ash	5·0 lb/ton
Potassium amyl xanthate	0·10 "
Pine oil	0·05 "

Summary:

Product	Weight, per cent	Assay, Au, oz/ton	Distribution of gold, per cent
Flotation concentrate	3·3	1·30	64·0
Flotation tailing	96·7	0·025	36·0
Amalgamation tailing (cal.)	100·0	0·067	100·0

Recovery by amalgamation, (0·59—0·067) ÷ 0·59	= 88·6 per cent
Net recovery in flotation concentrate, (100—88·6) × 64·0	= 7·3 "
Total recovery	95·9 "

PLATE AMALGAMATION

Test No. 15

In this test a sample of the ore at minus 14 mesh was ground in a ball mill for 20 minutes and then passed over a small amalgamation plate. The plate tailing was assayed for gold.

Results:

Feed sample	0·59 oz/ton
Amalgamation tailing	0·21 "
Recovery	64·4 per cent

CONCLUSIONS

This ore can be treated more efficiently by cyanidation than by any other method and for this reason as well as its greater freedom from trouble the process is to be recommended.

A hydraulic trap at the ball mill discharge to take care of any coarse gold would serve to lighten the load on the cyanide plant. The trap cleanings could be barrel-amalgamated and then passed on to the cyanide agitators for further treatment.

However, the ore concentrates very well by flotation (*see* Test No. 12) and although this treatment is somewhat less efficient than cyanidation the original outlay for plant would be considerably less.

Flotation with barrel amalgamation of the concentrate may, therefore, be worthy of consideration.

Ore Dressing and Metallurgical Investigation No. 554

GOLD ORE FROM THE PICKLE CROW MINE, PATRICIA DISTRICT, ONTARIO

Shipment. A shipment of one box of ore, net weight 116 pounds, was received December 13, 1933. The sample was submitted by A. G. Hattie, Manager, Pickle Crow Mine, on instructions from J. E. Hammell, President, Northern Aerial Canada Golds, Limited, 1406 Concourse Building, 100 Adelaide Street West, Toronto, Ontario.

Characteristics of the Ore. The gangue is chiefly translucent quartz of fine texture and light grey colour. Rare, tiny veinlets of carbonate occur in the quartz, and a small amount is associated with the sulphides.

The metallic minerals in their order of abundance are pyrite, pyrrhotite, chalcopyrite, and magnetite. No native gold was seen.

Pyrite and pyrrhotite are present together in rather coarse, irregular stringers in the quartz, and are associated with a small amount of carbonate. The pyrite contains numerous small inclusions of gangue. Chalcopyrite occurs as small irregular grains in the stringers referred to above and is usually more closely associated with the pyrrhotite than with the pyrite. Rare, small grains of magnetite occur as inclusions in the pyrite.

An average analysis of the ore was as follows:

Gold	1·21 oz/ton
Silver	0·11 "
Copper	Trace
Iron	11·78 per cent
Arsenic	0·02 "
Sulphur	3·24 "

Experimental Tests

A series of small-scale tests was made on this ore to determine the methods by which it might be treated. The work included tests by cyanidation, amalgamation, flotation, blanketing, and hydraulic classification.

By straight cyanidation as much as 99·2 per cent of the gold can be extracted. By barrel amalgamation 93·9 per cent of the gold can be extracted with ore ground wet. When amalgamation plates are used, however, extraction drops to 56·2 per cent. By straight flotation of the ore 94·7 per cent of the gold was recovered in a concentrate amounting to 16·0 per cent of the weight of feed. By blanket concentration of the ore as much as 87·1 per cent of the gold was recovered in a concentrate amounting to 5·4 per cent of the weight of feed. The blanket tailing assaying 0·175 ounce gold per ton was reduced to 0·01 ounce per ton in 24 hours by cyanidation.

In the hydraulic classifier 76·6 per cent of the gold settled out in a product amounting to 3·5 per cent of the weight of feed. The classifier overflow, assaying 0·265 ounce gold per ton, cyanided readily down to 0·005 ounce gold per ton in 24 hours.

Details of the tests follow:

CYANIDATION

Tests Nos. 1 to 8

This series of tests was made on four samples of the ore ground dry through the following screen sizes 48, 100, 150, and 200 mesh. Samples of each were agitated in cyanide solution, 2·0 pounds KCN per ton, for periods of 24 and 48 hours. The tailings were assayed for gold.

Test No.	Mesh	Period of agitation, hours	Tailing assay, Au, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
					KCN	CaO
1	− 48	24	0·065	94·6	3·36	9·75
2	−100	24	0·03	97·5	4·56	16·0
3	−150	24	0·035	97·1	5·16	18·25
4	−200	24	0·04	96·7	5·46	15·8
5	− 48	48	0·015	98·8	3·36	11·1
6	−100	48	0·01	99·2	5·76	19·0
7	−150	48	0·01	99·2	6·06	19·0
8	−200	48	0·01	99·2	6·06	18·1

AMALGAMATION AND CYANIDATION

Tests Nos. 9 and 10

These tests were made on samples of the ore ground dry through 48- and 100-mesh screens. The ore was amalgamated with mercury in a jar mill for 30 minutes. The amalgamation tailings were then sampled and assayed and portions of each agitated in cyanide solution, 2·0 pounds KCN per ton, for 24 hours. The final tailings were also assayed for gold.

Results:

Test No.	Amalgamation, tailing, Au, oz/ton	Cyanide tailing, Au, oz/ton	Extraction by amalgamation, per cent	Extraction by cyanidation, per cent	Total extraction, per cent	Reagents consumed, lb/ton	
						KCN	CaO
9	0·125	0·015	89·7	9·1	98·8	3·06	10·7
10	0·27	0·010	77·7	21·5	99·2	5·16	16·25

HYDRAULIC CLASSIFICATION

Test No. 11

In this test the ore at −14 mesh was ground for 20 minutes in a ball mill and then put through a hydraulic classifier where the gold and heavy minerals were allowed to settle against a slowly rising current of water. The classifier oversize and overflow were assayed for gold and a sample of the overflow was agitated in cyanide solution, 2·0 pounds KCN per ton for 24 hours. The cyanide tailing was also assayed for gold. This test is intended to give some idea of the results to be expected from the use of a hydraulic trap in practice.

Results:

Product	Weight, per cent	Assay, Au, oz/ton	Distribution of gold, per cent
Classifier oversize	3·5	23·86	76·6
Classifier overflow	96·5	0·265	23·4
Feed (cal.)	100·0	1·09	100·0

By cyanidation the gold in the classifier overflow was reduced to 0·005 ounce per ton with a reagent consumption of 2·2 pounds of KCN and 5·2 pounds of lime per ton of classifier overflow.

A screen test of the classifier overflow showed the grinding to be as follows:—

Mesh	Weight, per cent	Cumulative weight, per cent
− 65+100	0·3	0·3
−100+150	3·9	4·2
−150+200	13·7	17·9
−200	82·1	100·0
Total	100·0	....

FLOTATION

Test No. 12

The ore at −14 mesh was ground in a ball mill for 30 minutes and then floated.

Charge to ball mill:

Ore	2,000 grammes
Water	1,500 c.c.
Soda ash	5·0 lb/ton
Aerofloat No. 25	0·07 "

Reagents to cell:

Potassium amyl xanthate	0·10 lb/ton
Pine oil	0·05 "

Summary:

Product	Weight, per cent	Assay, Au, oz/ton	Distribution of gold, per cent
Concentrate	16·0	6·06	94·7
Tailing	84·0	0·065	5·3
Feed (cal.)	100·0	1·02	100·0

A screen test of the flotation tailing showed the grinding to be as follows:

Mesh	Weight, per cent	Cumulative weight, per cent
−100+150	1·4	1·4
−150+200	7·9	9·3
−200	90·7	100·0
Total	100·0	....

AMALGAMATION AND FLOTATION

Test No. 13

The ore at −14 mesh was ground in a ball mill for 20 minutes and then amalgamated with mercury in a jar mill for 30 minutes. The amalgamation tailing was then floated. The flotation concentrate and tailing were assayed for gold and the amalgamation tailing assay calculated from them.

Charge to ball mill:

Ore	2,000 grammes
Water	1,500 c.c.

Reagents to cell:

Soda ash	5·0 lb/ton
Potassium amyl xanthate	0·10 "
Pine oil	0·05 "

Summary:

Product	Weight, per cent	Assay, Au, oz/ton	Distribution of gold, per cent
Concentrate	2·5	1·58	53·7
Tailing	97·5	0·035	46·3
Amalgamation tailing (cal.)	100·0	0·074	100·0

Recovery by amalgamation	93·9 per cent
Recovery in flotation concentrate, 53·7 × 6·1	= 3·3 "

Screen test of the flotation tailing showed the grinding to be as follows:

Mesh	Weight, per cent	Cumulative weight, per cent
− 65+100	0·6	0·6
−100+150	3·2	3·8
−150+200	12·9	16·7
−200	83·3	100·0
Total	100·0	....

PLATE AMALGAMATION

Test No. 14

The ore at −14 mesh was ground in a ball mill for 20 minutes and passed over a small amalgamation plate. The tailing was assayed for gold.

Results:

Feed	1·21 oz/ton gold
Tailing	0·53 "
Recovery	56·2 per cent

BLANKETING

Tests Nos. 15 to 18

Four samples of the ore at −14 mesh were ground in ball mills for periods of 10, 15, 20, and 25 minutes respectively. In each case the pulp was passed over a small corduroy blanket set at a slope of 2·5 inches per foot. The concentrates and tailings were assayed for gold and a portion of each of the tailings was agitated in cyanide solution, 2·0 pounds KCN per ton, for 24 hours.

The cyanide tailings were also assayed for gold.

Results:

Test No.	Product	Weight, per cent	Assay, Au, oz/ton	Extraction, per cent total gold	Distribution of gold, per cent	Total recovery, per cent
15	Concentrate	3·9	23·88		81·2	
	Tailing	96·1	0·225		18·8	
	Feed (cal.)	100·0	1·15		100·0	
	Tailing cyanided	96·1	0·015	17·5		98·7
16	Concentrate	5·0	21·41		83·4	
	Tailing	95·0	0·225		16·6	
	Feed (cal.)	100·0	1·28		100·0	
	Tailing cyanided	95·0	0·01	15·9		99·3
17	Concentrate	5·7	17·56		82·5	
	Tailing	94·3	0·225		17·5	
	Feed (cal.)	100·0	1·21		100·0	
	Tailing cyanided	94·3	0·005	17·1		99·6
18	Concentrate	3·6	22·90		70·4	
	Tailing	96·4	0·36		29·6	
	Feed (cal.)	100·0	1·17		100·0	
	Tailing cyanided	96·4	0·01	28·3		98·7

Reagents consumed in the above tests are as follows in pounds per ton of blanket tailing:—

—	KCN	CaO
Test No. 15	0·7	3·7
" 16	1·0	4·0
" 17	1·3	4·0
" 18	1·3	4·0

Screen test of the blanket tailings produced in Tests Nos. 15 to 18 showed the grinding to be as follows:—

Test No.	Weight, per cent −48+65	Weight, per cent −65+100	Weight, per cent −100+150	Weight, per cent −150+200	Weight, per cent −200	Total
15	0·5	5·0	10·4	20·1	64·0	100·0
16	0·0	0·8	4·2	15·4	79·6	100·0
17	0·0	0·3	2·7	10·9	86·1	100·0
18	0·0	0·3	2·8	9·7	87·2	100·0

CYANIDATION WITH TABLING

Test No. 19

The ore at −14 mesh was ground in a ball mill for 20 minutes and agitated in cyanide solution, 4·0 pounds KCN per ton for 24 hours. A sample of the cyanide tailing was taken out for assay and the remainder passed over a small laboratory-size concentrating table. The table concentrate assayed 0·03 ounce per ton in gold and after treatment with cyanide solution the gold content was reduced to 0·02 ounce per ton. In this operation 14 pounds of potassium cyanide was used per ton of table concentrate. It is, therefore, evident that this step would be of no economic value.

CYCLE TEST

Tests Nos. 20 to 23

In this test the ore, ground dry all through 150-mesh screen, was agitated in cyanide solution, 2·0 pounds KCN per ton for 24 hours. The pregnant solution was then filtered, made up to proper strength and volume and used to treat another batch of fresh ore. The operation was repeated until four batches of ore had been treated with this one batch of solution. The tailing from each cycle was assayed to see if any falling-off in extraction occurred from repeated use of the same solution.

Results:

Test No.	Tailing assay, Au, oz/ton	Extraction, per cent
20	0·035	97·1
21	0·045	96·3
22	0·08	93·4
23	0·105	91·3

Chemical analyses of the pregnant solution showed that it contained the following ingredients:

Iron	0·0030 gramme per litre
Potassium thiocyanate	2·67 " "
Reducing power	1,843 c.c. $\frac{N}{10}$ $KMnO_4$ per litre

BLANKETING

Test No. 24

In this test 8,000 grammes of ore was ground 65 per cent through 200 mesh and passed over a corduroy blanket set at a slope of 2·5 inches per foot. Samples of the concentrate and tailing were assayed for gold. A sample of the concentrate was barrel-amalgamated with mercury for two hours. Two dozen steel balls were put in the amalgamation barrel to grind the concentrate. Two samples of the blanket tailing were cyanided, one without further grinding and one ground 79 per cent through 200 mesh. Two other samples of the blanket tailing were floated, one ground 79 per cent through 200 mesh and the second 90 per cent through 200 mesh.

The flotation reagents used in pounds per ton of blanket tailing were as follows:

Soda ash	5·0 lb./ton
Potassium amyl xanthate	0·20 "
Pine oil	0·10 "

A barrel-amalgamation test was made on a composite sample of the two flotation concentrates.

Results of Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz/ton	Extraction, per cent	Distribution of gold, per cent
Concentrate	5·4	20·72		87·1
Tailing	94·6	0·175		12·9
Head (cal.)	100·0	1·284		100·0
Concentrate amalgamated	5·4	0·19	99·1	

Net recovery by barrel amalgamation of blanket concentrate, 99·10 × 87·1=86·3 per cent total gold.

Results of Cyanidation of Blanket Tailing:

(a) Without further grinding—

Feed to cyanidation	0·175 oz/ton in gold
Cyanidation tailing	0·01 " "
Extraction	94·3 per cent
Net recovery	94·3 × 12·9 = 12·2 per cent total gold
Reagents consumed, lb/ton tailing	KCN 1·15
	CaO 5·10

(b) Blanket tailing ground 79 per cent through 200 mesh—

Feed to cyanidation	0·175 oz/ton in gold
Cyanidation tailing	0·005 " "
Extraction	97·1 per cent
Net recovery	97·1 × 12·9 = 12·5 per cent total gold
Reagents consumed, lb/ton tailing	KCN 1·50
	CaO 5·25

Results of Flotation of Blanket Tailing:

(c) Ground 79 per cent through 200 mesh—

Product	Weight, per cent	Assay, Au, oz/ton	Distribution of gold, per cent
Flotation concentrate	9·1	1·58	86·4
Flotation tailing	90·9	0·025	15·6
Blanket tailing (cal.)	100·0	0·167	100·0

Net recovery in flotation concentrate, 86·4 × 12·9=11·1 per cent total gold.

(d) Ground 90 per cent through 200 mesh—

Product	Weight, per cent	Assay, Au, oz/ton	Distribution of gold, per cent
Flotation concentrate	6·0	2·60	89·2
Flotation tailing	94·0	0·02	10·8
Blanket tailing (cal.)	100·0	0·175	100·0

Net recovery in flotation concentrate, 89·2 × 12·9 = 11·5 per cent total gold

BARREL AMALGAMATION OF FLOTATION CONCENTRATE

A composite sample of the two flotation concentrates was amalgamated with mercury for two hours in a jar mill. Two dozen steel balls were added to grind the concentrate.

Results:

Feed to amalgamation	1·98 oz/ton in gold
Amalgamation tailing	0·295 " "
Extraction	85·1 per cent
Net recovery	85·1 × 11·3* = 9·6 per cent total gold

*Average recovery in flotation concentrates Tests Nos. 24 (c) and 24 (d).

The mercury in this test was badly flocculated. The addition of some lime to the amalgamation barrel before the mercury is put in might prevent this.

CONCLUSIONS

The method of treatment most suitable to this ore is blanket concentration followed by cyanidation of the blanket tailing. The blanket concentrate should be barrel-amalgamated and the amalgamation tailing discarded because its introduction into the cyanide circuit would cause excessive reagent consumption as well as poor extraction on account of its fouling properties. A comparison of the reagent consumption in Tests Nos. 1 to 8, where all of the ore was treated by cyanidation, with that in the cyanidation tests on blanket tailing Tests Nos. 15 to 18, will show the extent of the saving in reagents to be so effected. Tests Nos. 20 to 23 show a rapid ascent in tailing assays when four lots of ore were treated successively with one batch of solution. This is probably caused by the pyrrhotite in the ore forming thiocyanates and otherwise causing a strongly reducing condition in the solution.

An alternative to cyanidation of the blanket tailing would be to regrind it and concentrate it by flotation. The flotation concentrate would then be barrel-amalgamated either with the blanket concentrate or by itself. In this way 95·9 per cent of the total gold in the ore can be recovered as bullion against 98·8 per cent when the blanket concentrate is barrel-amalgamated and the blanket tailing cyanided.

If flotation of the blanket tailing were to be considered, the problem of flocculated or sickened mercury caused by amalgamation of this concentrate would have to be dealt with. It is probable, however, that it could be overcome by washing and conditioning the concentrate before the mercury is added, as flotation reagents, such as xanthates, are at least part of the cause.

Ore Dressing and Metallurgical Investigation No. 555

GOLD ORE FROM CASEY SUMMIT GOLD MINES, LTD., SUMMIT LAKE DISTRICT, ONTARIO

Shipment. A shipment of 55 pounds of ore was received on January 4, 1934, from Casey Summit Gold Mines, Ltd., Summit Lake district, Ontario. The shipment was submitted by H. F. Fancy, superintendent of the mine.

Characteristics. The gangue consists of bluish grey, smoky quartz which contains small veinlets of carbonate. The metallic minerals are arsenopyrite, pyrrhotite, pyrite, chalcopyrite, and native gold. Arsenopyrite is the most abundant, but only very sparingly disseminated.

Native gold is present in the quartz, usually associated with the fine carbonate veinlets.

Except for the absence of galena, the occurrence of the gold in the quartz and its absence from the arsenopyrite, and the smoky character of the quartz, this sample is very similar to the one submitted previously from Ventures, Limited, Summit Lake property.

The ore was crushed to minus 14 mesh and sampled by standard methods.

The analysis of the ore was as follows:

Gold	1·795 oz/ton
Arsenic	0·64 per cent

Experimental Tests

The test work consisted of pan and barrel amalgamations on different sizes of ore, followed by blanket concentration of the amalgamation tailing. The blanket concentrates were combined and barrel-amalgamated.

A classification test was made in which the classifier overflow was run over blankets. The oversize from the classifier and the blanket concentrate were then barrel-amalgamated.

Results indicate that the gold is suitable for amalgamation treatment and good recoveries were obtained.

Three pan-amalgamation tests were carried out on 1,000-gramme samples of minus 14-mesh ore ground wet for different periods of time. An amalgamated copper gold pan was used in a similar manner to panning a gold gravel. The amalgamation-pan tailings were then run over corduroy blankets and the blanket concentrates of the three tests combined and barrel-amalgamated.

Test No. 1

A sample, 1,000 grammes, of ore ground wet for 10 minutes in a pebble jar was pan-amalgamated. The tailing assayed 0·51 ounce per ton, showing a gold recovery of 71·5 per cent.

Screen Test of Pan Tailing:

Mesh	Weight, per cent
+ 48	7·4
+ 65	10·7
+100	21·6
+150	15·2
+200	13·4
−200	31·7
	100·0

Blanket Test of Pan Tailing:

Product	Weight, per cent	Assay, Au, oz/ton	Distribution of gold, per cent	Ratio of concentration
Concentrate	6·39	5·49	68·8	15·6:1
Tailing	93·61	0·17	31·2	

Test No. 2

A sample, 1,000 grammes, was ground wet for 20 minutes and pan-amalgamated. The tailing assayed 0·275 ounce per ton, showing a gold recovery of 84·6 per cent.

Screen Test of Pan Tailing:

Mesh	Weight, per cent
+ 65	0
+100	6·8
+150	13·5
+200	21·8
−200	57·9
	100·0

Blanket Test of Pan Tailing:

Product	Weight, per cent	Assay, Au, oz/ton	Distribution of gold, per cent	Ratio of concentration
Concentrate	8·12	2·70	79·9	12·3 : 1
Tailing	91·88	0·06	20·1	

Test No. 3

A sample, 1,000 grammes, was ground wet for 30 minutes and pan-amalgamated. The tailing assayed 0·115 ounce per ton, showing a gold recovery of 93·5 per cent.

Screen Test of Pan Tailing:

Mesh	Weight, per cent
+ 65	0
+100	2·1
+150	9·5
+200	18·4
−200	70·0
	100·0

Blanket Test of Pan Tailing:

Product	Weight, per cent	Assay, Au, oz/ton	Distribution of gold, per cent	Ratio of concentration
Concentrate	9·09	0·815	64·4	11 : 1
Tailing	90·91	0·045	35·6	

The combined blanket concentrates from Tests Nos. 1, 2, and 3, weight 191·7 grammes, were barrel-amalgamated with 250 c.c. of water and 25 grammes of mercury for one-half hour. The calculated gold content was 2·70 ounces per ton.

Product	Assays		Recovery, per cent
	Au, oz/ton	Arsenic, per cent	
Tailing	1·03	3·80	61·8

The high tailing from the amalgamation of the blanket concentrates would indicate that the gold in these products is closely tied up with the aresnopyrite grains. This product represents only about 6 to 9 per cent of the original weight of the ore.

Capitulation of Results Tests Nos. 1, 2, and 3

Test No. 1: Ore 31·7 per cent minus 200 mesh

Gold recovered on amalgamation pan	71·5	per cent.
Possible gold recovery by blankets, 68·8 per cent of 28·5 per cent.	19·6	"
Possible overall recovery	91·1	"

Test No. 2: Ore 57·9 per cent minus 200 mesh

Gold recovered on amalgamation pan	84·6	per cent
Possible gold recovery by blankets, 79·9 per cent of 15·4 per cent	12·3	"
Possible overall recovery	96·9	"

Test No. 3: Ore 70 per cent minus 200 mesh

Gold recovered on amalgamation pan	93·5	per cent
Possible gold recovery by blankets, 64·4 per cent of 6·5 per cent	4·2	"
Possible overall recovery	97·7	"

The following two tests were barrel amalgamations on samples given different periods of grinding. The tailing in each case was run over corduroy blankets. The blanket concentrates were combined and barrel-amalgamated.

Test No. 4

A sample, 1,000 grammes, of minus 14-mesh ore was ground wet for 15 minutes and then barrel-amalgamated with 100 grammes of mercury for one-half hour. The tailing assayed 0·145 ounce per ton showing a gold recovery of 91·9 per cent. No sickening of the mercury was noted.

Screen Test of Amalgamation Tailing:

Mesh	Weight, per cent
+ 48	2·0
+ 65	4·5
+100	23·1
+150	18·7
+200	15·6
−200	36·1
	100·0

Blanket Test of Amalgamation Tailing:

Product	Weight, per cent	Assay Au, oz/ton	Distribution of gold, per cent	Ratio of concentration
Concentrate	8·13	0·823	46·2	12·3 : 1
Tailing	91·87	0·085	53·8	

Test No. 5

In this test the ore was ground for 30 minutes prior to amalgamation for one-half hour. The tailing assayed 0·16 ounce per ton showing a gold recovery of 91 per cent.

Screen Test of Amalgamation Tailing:

Mesh	Weight, per cent
+100	2·7
+150	8·7
+200	21·0
−200	67·6
	100·0

Blanket Test of Amalgamation Tailing:

Product	Weight, per cent	Assay, Au, oz/ton	Distribution of gold, per cent	Ratio of concentration
Concentrate	9·40	0·64	37·7	10·6 : 1
Tailing	90·60	0·11	62·3	

The combined blanket concentrates from Tests Nos. 4 and 5 (139·7 grammes) were barrel-amalgamated for a half hour. The calculated gold content was 0·724 ounce per ton.

Product	Assays		Recovery, per cent
	Au, oz/ton	Arsenic, per cent	
Tailing	0·41	3·15	43·3

Capitulation of Results of Tests Nos. 4 and 5:

Test No. 4: Ore 36·1 per cent minus 200 mesh.

Gold recovered by barrel amalgamation	91·9 per cent
Possible gold recovery by blankets, 46·2 per cent of 8·1 per cent	3·7 "
Possible overall recovery	95·6 "

Test No. 5: Ore 67·6 per cent minus 200 mesh.

Gold recovered by barrel amalgamation	91·0 per cent
Possible gold recovery by blankets, 37·7 per cent of 9·0 per cent	3·4 "
Possible overall recovery	94·4 "

In the barrel amalgamation very fine grinding does not appear to be essential.

Test No. 6

A sample, 1,000 grammes, of −14-mesh ore was ground wet in a pebble jar for 15 minutes. The pulp was then run through a hydraulic classifier. The oversize was collected and assayed. The overflow was run over corduroy blankets. Results show the possible gold retained in a trap from the ball mill discharge.

Assay of classifier oversize, gold	31·0 oz/ton
Per cent weight	4·78
Assay of blanket concentrate, gold	2·99 oz/ton
Assay of blanket tailing, gold	0·105 "

Screen Test of Blanket Tailing:

Mesh	Weight, per cent
+ 65	1·2
+100	17·0
+150	21·4
+200	18·7
−200	41·7
	100·0

The classifier oversize contains 84·3 per cent of the gold in the charge.

CONCLUSIONS

The results of the tests indicate that amalgamation will prove a satisfactory method of treatment for gold recovery. Tests show a recovery of over 91 per cent of the gold in barrel amalgamation with ore ground to have 36 per cent pass a 200-mesh screen. By blanketing the amalgamation tailing a further concentrate is obtained which, when barrel-amalgamated, shows a further 40 per cent (approx.) recovery of the remaining gold.

The gold in the blanket concentrates may be closely associated with the arsenopyrite, although the microscopic analysis does not indicate this.

A suggested flow-sheet is as follows:

Crushed ore to ball mill, mercury trap, amalgamation, and classifier. The oversize from the classifier returned to the ball mill while the classifier tailing passed over corduroy blanket tables. The blanket concentrate would be barrel-amalgamated. There should be an advantage in placing the amalgamators between the ball mill and the classifier.

Ore Dressing and Metallurgical Investigation No. 556

GOLD ORE FROM THE DUFFERIN MINE, NOVA SCOTIA

Shipment. A shipment consisting of two barrels containing 740 pounds of gold ore was received December 16, 1933. It was submitted by A. M. Nairn, Secretary, Saddle Reef Gold Mines, Limited, 3555 Shuter Street, Montreal. The material was said to have been taken from the Dufferin mine, Port Dufferin, Nova Scotia.

Characteristics of the Ore. The ore was examined in the mineragraphic laboratory and found to consist of a grey to white vein quartz gangue with local dark grey, slaty or quartzitic material. The metallic minerals are arsenopyrite, galena, and native gold. The arsenopyrite is disseminated in the quartz as coarse crystals and coarsely crystalline aggregates. Galena is rare and occurs as small irregular grains in the quartz and arsenopyrite. The gold noticed in the specimens examined was in small irregular grains in quartz and in contact with large crystals of arsenopyrite.

Experimental Tests

The shipment was crushed, sampled, and assayed, showing the ore to contain 5·94 ounces gold, 0·19 ounce silver per ton; and 0·88 per cent arsenic.

Tests were made by amalgamation and cyanidation, the results of which showed that 79 per cent of the gold can be amalgamated from ore ground −48 mesh. Straight cyanidation of the same coarsely-ground ore gives a recovery of 99·5 per cent of the gold.

AMALGAMATION

Test No. 1

A sample of the ore was ground to pass 48 mesh with 42 per cent through 200 mesh, and amalgamated.

Results:

Feed	5·94 oz./ton
Tailing	1·26 "
Recovery	78·8 per cent

Test No. 2

A sample ground to pass 100 mesh was treated as in Test No. 1. The amalgamation tailing was then cyanided for 48 hours with a 1·0 pound per ton cyanide solution, 1 : 3 dilution.

Results:

Feed	5·94 oz./ton
Amalgamation tailing	1·36 "
Recovery by amalgamation	77·1 per cent
Cyanide tailing	0·02 oz./ton
Recovery, amalgamation and cyanidation	99·7 per cent

CYANIDATION

Test No. 3

Samples of the ore were ground to pass 48-, 100-, 150-, and 200-mesh and agitated, 1 : 3 dilution, for 48 hours with a 1·0 pound per ton KCN solution. Sufficient lime was added to maintain protective alkalinity.

Results:

Mesh grind	Agitation, hours	Feed, Au, oz/ton	Tailing, Au, oz/ton	Extraction, per cent	Reagent consumption, lb/ton	
					KCN	CaO
48	24	5·94	0·03	99·5	0·9	5·0
48	48	5·94	0·03	99·5	0·9	5·2
100	24	5·94	0·04	99·3	0·6	5·7
100	48	5·94	0·04	99·3	1·5	6·2
150	24	5·94	0·03	99·5	2·6	5·7
150	48	5·94	0·045	99·2	2·4	6·7
200	24	5·94	0·03	99·5	2·4	5·6
200	48	5·94	0·04	99·3	2·1	6·6

These results show that the maximum extraction of 99·5 per cent is attained from 48-mesh material within 24 hours.

SUMMARY AND CONCLUSIONS

The sample furnished is of such extremely high grade that it can not be regarded as representative of the ore that would be mined.

This investigation indicates that the gold is readily soluble in cyanide solution and that 75 per cent is freed at minus 48-mesh grinding.

Cyanidation is the process indicated by these tests. A trap could be placed at the discharge end of the grinding mill to ensure that no coarse gold finds its way into the cyanide agitators. The clean-up from this trap should be barrel-amalgamated and the tailing added to the cyanide circuit for further treatment.

If it be desired to recover the arsenopyrite in the ore, the cyanide tailing after filtration could be repulped in water and passed over concentrating tables.

Ore Dressing and Metallurgical Investigation No. 557

SILVER AND SILVER-PITCHBLENDE ORE FROM BEAR EXPLORATION AND RADIUM, LIMITED, GREAT BEAR LAKE, N.W.T.

Shipment. A shipment of silver ore, net weight 77 pounds, was received on September 21, 1932. The sample was reported as coming from a property on Contact lake, Great Bear lake, N.W.T., and was submitted by Major B. Day of the Bear Exploration and Radium, Limited.

A second shipment consisting of 6 bags, weight 580 pounds, was received on October 7, 1933, from a different locality on the same property. The first shipment was designated Lot No. 1, the second, Lot No. 2.

Experimental work on Lot No. 2 consisted principally of preliminary tests on concentration of the pitchblende.

Lot No. 1

Characteristics. The sample consisted of high-grade silver ore in which the silver is most abundant in the native state. It occurs as irregular grains in the gangue, as tiny veinlets in earlier sulphides, and quite often assumes a pattern suggestive of dendritic growth. Tetrahedrite is present in minor quantities. Bornite and chalcocite are also present. Pitchblende is not present in this sample.

The gangue minerals are largely manganese and iron-bearing carbonates and minor amounts of quartz and a black silicate.

The chemical analysis of the ore is as follows:

Manganese	21·15 per cent
Iron	9·05 "
Copper	0·10 "
Arsenic	0·02 "
Sulphur	0·095 "
Soluble sulphate (SO_3)	0·008 "
Gold	None
Silver	1,565 oz/ton

Experimental Tests

A number of tests were made, including tabling, amalgamation, blanket concentration, cyanidation, and flotation.

TABLING

Test No. 1

A sample, 3,000 grammes, of the ore, 20 mesh in size, was fed to a small laboratory Wilfley table and the following products cut out and assayed for silver:

Cut No. 1	Silver,	8,684·62	oz/ton
" 2A	"	3,690·54	"
" 2B	"	877·60	"
" 2C	"	722·20	"
" 3	"	633·40	"
Sand tailing from 1,000 grammes	"	381·92	"
" " 2,000 "	"	451·07	"
Slime " 3,000 "	"	410·61	"

AMALGAMATION

Three amalgamation tests were made. The ore was ground with mercury in a small ball mill and the amalgam recovered in a hydraulic classifier.

Test No. 1

Charge to mill:

Ore	1,000 grammes
Mercury	1,000 "
Pulp ratio	2 : 1
Time of grinding	9 hours
Assay of tailing	Silver, 130·56 oz/ton
Silver recovery	91·6 per cent

The pulp was found to be too thick.

Test No. 2

Charge to mill:

Ore	1,000 grammes
Mercury	1,000 "
Sodium hydroxide	5 lb/ton
Pulp ratio	1 : 1
Time of grinding	6 hours
Assay of tailing	Silver, 132·24 oz/ton
Silver recovery	91·5 per cent

Amalgam cleaner than in Test No. 1.

Test No. 3

Charge to mill:

Ore	1,000 grammes
Mercury	1,000 "
KCN	5·0 lb/ton
CaO	5·0 "
Pulp ratio	1 : 1
Time of grinding	6 hours
Assay of tailing	Silver, 98·0 oz/ton
Silver recovery	93·7 per cent

Amalgam was very clean.

Silver Recovery from Amalgam

Mercury used in 3 tests	3,000	grammes
Mercury recovered by straining and retorting	2,902	"
Weight of residue from retort	193	"
Weight of silver ingot recovered	152·75	"

Barrel amalgamation affords a simple method of obtaining high silver recovery on high-grade ore. Test No. 3 gave the cleanest amalgam mixture and also a lower tailing than the other tests.

PRELIMINARY FLOTATION TESTS ON AMALGAMATION TAILING

The object of flotation is to concentrate the low-grade silver ore sufficiently to make it suitable for amalgamation. The preliminary tests run were on amalgamation tailing as no low-grade ore was available.

On Tests Nos. 1, 2, and 4 a frothing mixture of 15 per cent pine oil, 75 per cent coal-tar creosote, and 10 per cent coal tar was used.

Test No. 1

Charge, 400 grammes amalgamation tailing (silver 130·56 ounces per ton), 4 drops frothing mixture. Conditioned in cell.

Product	Weight, per cent	Silver, oz/ton	Distribution of silver, per cent
Concentrate	16·2	124·82	15·5
Tailing	83·8	131·96	84·5

No concentration was effected.

Test No. 2

Charge, same as Test No. 1. Potassium xanthate 1 pound per ton, and 2 drops frothing mixture added.

Product	Weight, per cent	Silver, oz/ton	Distribution of silver, per cent
Concentrate	10·7	188·32	15·6
Tailing	89·3	122·00	84·4

A little concentration was effected.

Test No. 3

In this test sodium sulphide, 1 pound per ton, and pine oil were used. No concentration was effected.

Test No. 4

A charge, 400 grammes, of amalgamation tailing (silver 132·24 ounces per ton) was ground in the mill for 15 minutes with sodium sulphide, 2 pounds per ton, potassium xanthate, 1 pound per ton, and a frothing mixture, 1 drop.

Product	Weight, per cent	Silver, oz/ton	Distribution of silver, per cent
Concentrate	6·7	529·4	27·2
Tailing	93·3	101·8	72·8

A concentration was effected, but the tailing still retained a large bulk of the silver and was high in silver.

TABLING

Test No. 2

A sample, 3,000 grammes of ore, minus 20 mesh, was tabled on a laboratory Wilfley table. The first cut was retained for amalgamation. The second cut and the sand tailing were reground for twenty minutes and again tabled. The first cut (1A) was retained, the tailing and slime were reserved for subsequent flotation.

Results:

Product	Weight, per cent	Silver, oz/ton	Distribution of silver, per cent
Cut 1	22·14	4,738·66	64·4 } 81·5
Cut 1A	3·80	7,627 54	17·1 }
Tailing and slime	65·81	388·80	15·7
Slime (loss)	8·25	390·31	2·8
	100·00		

FLOTATION OF TABLE TAILING

Test No. 5

The tailing and slime from Table Test No. 2 were ground for one hour with: sodium silicate, 1 pound per ton; potassium xanthate, 2 pounds per ton; and coal-tar creosote, 0·1 pound per ton. To the cell pine oil, 0·05 pound per ton, was added; and the pulp was conditioned for 15 minutes.

Results:

Product	Weight, per cent	Silver, oz/ton	Distribution of silver, per cent
Concentrate	5·6	4,291·6	62·1
Tailing	94·4	155·34	37·9
	100·0		

Screen Test on Flotation Tailing:

Mesh	Per cent
+ 48	0·30
+ 65	0·04
+100	0·10
+150	0·50
+200	0·90
−200	98·16
Total	100·0

The recoveries by tabling and flotation are as follows:

Silver recovery on tables	81·5 per cent
Silver recovery from tailing and slime by flotation, 15·7 × 0·621	9·7 "
Overall recovery	91·2 "

TABLING AND FLOTATION

Test No. 3

A sample, 3,000 grammes, of ore minus 20 mesh was tabled on a laboratory Wilfley table. The first cut was retained, and the second cut and the sand tailing were ground for 45 minutes and again tabled. Two cuts (1A and 1B) were retained, the tailing and slime being subsequently floated.

Results of Table Test No. 3:

Product	Weight, per cent	Silver, oz/ton	Distribution of silver, per cent
Cut 1	21·10	5,546·81	69·2
Cut 1A	2·13	8,348·45	11·4 } 86·3
Cut 1B	3·11	3,103·73	5·7
Tailing and slime	65·74	314·82	12·2
Slime loss	7·92	288·05	1·5
	100·00		

The table tailing and slime, just over 1,600 grammes, were ground for one hour with 30 drops of coal-tar creosote, 10 c.c. sodium silicate, and 1 gramme potassium xanthate, and floated.

The first concentrate was taken off without any reagents to cell. A drop of pine oil was added and a second concentrate floated.

Results of Flotation Test No. 6:

Product	Weight, per cent	Silver, oz/ton	Distribution of silver, per cent
Concentrate A	5·6	3,548·20	63·0 } 69·0
Concentrate B	1·9	1,000·28	6·0
Tailing	92·5	105·40	31·0

The recoveries by tabling and flotation are as follows:

Recovery by tabling	86·3 per cent
Recovery from tailing and slime by flotation, 12·2 × 0·69	8·4 "
Overall recovery	94·7 "

BLANKET CONCENTRATION AND FLOTATION

Test No. 1

A sample, 500 grammes of ore, minus 20 mesh, was ground for 30 minutes and the pulp then run over a special corduroy blanket having a slope of 2·5 inches in 16 inches.

Results of Blanket Concentration Test No. 1:

Product	Weight, per cent	Silver, oz/ton	Distribution of silver, per cent
Concentrate	20·4	6,476·63	79·9
Tailing	79·6	415·84	20·1

The tailing from the blankets was ground for one hour with 15 drops of coal-tar creosote, 5 c.c. sodium silicate, and 0·5 gramme potassium xanthate, and floated,

Results of Flotation Test No. 7:

Product	Weight, per cent	Silver, oz/ton	Distribution of silver, per cent
Concentrate	19·56	1,596·8	80·8
Tailing	80·44	91·8	19·2

The recoveries by blanket concentration and subsequent flotation of tailing are as follows:

Recovery in blanket concentrate	79·9 per cent
Recovery in flotation of blanket tailing, 20·1 × 0·808	16·2 "
Overall recovery	96·1 "

The combined table concentrate and flotation concentrate were barrel-amalgamated.

Weight of products	1,451·5 grammes
Calculated assay, silver	5,137·35 oz/ton

Charge for amalgamation:

Ore	1,451·5	grammes
Mercury	1,500	"
KCN	2	lb/ton
CaO	2	"
Water	750	c.c.
Grinding time	6	hours

The tailing assayed 296·0 ounces silver per ton, showing a recovery by amalgamation of 94·2 per cent.

CYANIDATION TESTS

A number of cyanidation tests were run under various conditions. Difficulties were encountered owing to the manganese content of the ore. These difficulties are discussed in detail after a description of the tests.

Test No. 1

A sample, 2,000 grammes, of ore was ground for 24 hours in a small ball mill. The ball mill was then opened and 75 pounds chloride of lime per ton added and the charge ground for a further period of 16 hours. The charge was then washed out and the metallic silver recovered on a 200-mesh screen. The 200-mesh product was filtered and washed to remove soluble chlorides and then treated with cyanide. This was found unsatisfactory owing to the difficulty of keeping the pulp alkaline. Over 250 pounds of lime per ton was added, the lime being consumed at a uniform rate of about 40 pounds per ton per hour.

Test No. 2

A sample, 500 grammes, of ore was ground for 9 hours and then agitated in a pail with cyanide solution:

Charge	Pulp ratio	Total CaO, grm.	Total NaCN, grm.	Reagents consumed, lb/ton		Time, hours	Assay tailing, oz/ton	Recovery, per cent
				KCN	CaO			
500.........	10·7 : 1	189	66·5	209	750	72	34·5	97·7

The high cyanide and lime consumption is no doubt due to the manganese carbonate present in the ore. Subsequent tests tend to confirm this and a more complete discussion follows later in this report.

The ore pulp without addition of reagents was found to have a pH value of 8·4.

Test No. 3

A sample, 500 grammes, of ore was ground in a small ball mill for 16 hours. The charge was then screened on a 200-mesh screen and the silver metallics retained. The 200-mesh product was agitated in a pail at a 10 : 1 dilution with sodium cyanide and ammonium carbonate. The initial charge was 20 grammes of cyanide and 50 grammes ammonium carbonate.

The ammonium carbonate was used in place of lime, because of the heavy lime consumption in Test No. 2.

Time of treatment was 72 hours during which the KCN consumption was 394·1 pounds per ton. There was no apparent consumption of ammonium carbonate.

The tailing assayed 107·55 ounces per ton and the metallics 116·97 milligrams of silver, giving an overall recovery of silver of 93·52 per cent.

Test No. 4

A sample, 500 grammes, of ore was roasted for half an hour at a temperature around 725° C. to break up the carbonates. The calcine, weight 480 grammes, was then ground in a ball mill for 16 hours. The charge was screened on a 200-mesh screen to recover the metallics and the 200-mesh product was treated with cyanide in a pail at a dilution of 10·4 : 1 for 72 hours. The KCN concentration was kept as nearly as possible to 10 pounds per ton. The KCN consumption was 97·5 pounds per ton and the lime 171·92 pounds per ton.

The metallics assayed 1,920 milligrams of silver and the tailing 419·20 ounces per ton, giving a total silver recovery of 79·45 per cent.

Calcination of the ore prior to cyanidation appears to lower the silver recovery.

Test No. 5

A sample, 500 grammes, of ore ground for 16 hours as in previous tests and then agitated for 96 hours in cyanide solution at a dilution of 10 : 1. Ammonium carbonate, 10 grammes, was added, and a total of 80 grammes of NaCN. After a time it became impossible to titrate for free cyanide owing to the murkiness of the solution on further dilution. The solution was tested and found to contain manganese and iron.

The tailing assayed 46·54 ounces per ton with 303·2 milligrams silver in the metallics. The recovery was 97·02 per cent.

Test No. 6

The same as Test No. 5, with 12·5 grammes CaO instead of ammonium carbonate. A total of 60 grammes of NaCN was added. The same difficulty was encountered in the solution so that a KCN consumption was not obtained. The tailing assayed 3·38 ounces per ton, with 173·4 milligrams silver in the metallics, giving a recovery of 99·74 per cent.

Test No. 7

This test was on tailing from Amalgamation Test No. 3 which assayed 98 ounces silver per ton. A sample, 200 grammes, was ground for 16 hours and agitated in a bottle at a dilution of 23·7 : 1 without any alkaline reagents for 72 hours. A total of 28 grammes of NaCN was added. The solution became fouled with manganese preventing any accurate titration of the cyanide. The tailing was only 0·50 ounce silver per ton, giving a recovery of 99·4 per cent.

DISCUSSION OF CYANIDATION TESTS

The results of the cyanidation, although making for a high silver recovery, show a very high cyanide consumption. It is apparent that the manganese carbonate acts as a cyanicide.

The very high lime consumption may be due to a reaction with the carbonate, or more likely it is due to the alkali precipitating the dissolved manganese and iron. The latter idea is borne out by the fact that the fouling of the solution does not appear to be so marked when large quantities of alkali are used as when no alkali or only small quantities are added. Roasting the ore does not overcome this difficulty and lowers the silver extraction as well.

Present results from these tests would indicate that cyanidation is not adaptable to the ore owing to the high consumption of cyanide and the resultant fouling of the solution with manganese and iron.

CONCLUSIONS

On high-grade ore such as Lot No. 1, barrel amalgamation offers a satisfactory form of treatment for silver recovery. Cyanidation is complicated by the action of the manganese and is not a suitable method. Tabling gives a high silver tailing. Flotation of the table tailing increased the silver recovery, giving an overall silver recovery by tables and flotation of 94·7 per cent. Blanket concentration followed by flotation of the tailing gives an overall silver recovery of 96·1 per cent.

Lot No. 2

The ore in Lot No. 2 had a lower silver content and contained pitchblende.

The shipment was sampled and assayed as follows:

Manganese	14·90 per cent
Iron	11·50 "
Sulphur	0·92 "
Barium	Nil
Carbon dioxide	20·10 "
Silica	12·50 "
Pitchblende (U_3O_8)	9·05 "
Silver	833·35 oz/ton

PRELIMINARY TESTS ON PITCHBLENDE CONCENTRATION

Screen Analysis of 500 Grammes of 48-mesh Ore:

Mesh	Weight, per cent	Pitchblende, per cent	Manganese, per cent	Iron, per cent	Lime, per cent	Silica, per cent
+ 65	3·04	10·49	14·07	10·20	2·60	12·87
+100	14·68	10·36	13·96	10·10	3·43	13·45
+150	16·75	10·24	14·03	10·10	2·56	13·29
+200	18·54	9·60	14·52	10·55	2·50	12·76
−200	47·00	9·47	15·50	11·42	2·92	10·07
	100·00					

The screen analysis indicates a close association of the constituent minerals in the ore.

TABLING

Test No. 1

A sample, 3,000 grammes of the −14-mesh ore, was ground to pass a 48-mesh screen and run over a laboratory Wilfley table.

Results of Table Test No. 1:

Product	Weight, per cent	Assays			Distribution of metals		
		Silver, oz/ton	Pitchblende, per cent	Manganese, per cent	Silver, per cent	Pitchblende, per cent	Manganese, per cent
Cut 1	11·68	2,925·97	35·00	5·77	43·7	43·2	4·5
Cut 2	19·40	876·80	8·50	17·55	21·7	17·3	23·1
Middling	39·69	448·40	7·00	15·35	22·7	29·3	41·3
Slime	29·23	313·70	3·25	15·60	11·9	10·2	31·1

Screen Tests:

Cut 1		Cut 2		Middling	
Mesh	Weight, per cent	Mesh	Weight, per cent	Mesh	Weight, per cent
+ 65	5·2	+ 65	2·5	+ 65	7·0
+100	21·2	+100	14·2	+100	27·3
+150	21·0	+150	21·9	+150	23·2
+200	18·8	+200	24·9	+200	15·1
−200	33·8	−200	36·5	−100	27·4
	100·0		100·0		100·0

The silver recovery is not so good as on the higher grade material. The concentration of pitchblende is fair, showing over 60 per cent recovered in the concentrate.

Test No. 2

The test was made on screen-sized products with the object of determining the concentration of pitchblende at the different sizes. A sample, 4,500 grammes, was crushed in rolls to pass 48-mesh and screened. The different sizes were tabled on a small, laboratory Wilfley table. The middling product was re-run over the tables three times.

Results of Table Test No. 2:

Size	Weight, per cent	Product	Weight, per cent	Pitchblende, per cent	Distribution of pitchblende, per cent	Total, per cent
−48+65	25·5	Concentrate	11·5	39·42	52·0	14·32
		Middling	2·1	18·40	4·4	
		Tailing	86·4	4·39	43·6	
−65+100	20·2	Concentrate	10·0	45·92	50·8	12·58
		Middling	3·9	26·60	11·5	
		Tailing	86·1	3·95	37·7	
−100+150	9·6	Concentrate	7·7	51·32	45·0	4·76
		Middling	1·4	28·76	4·6	
		Tailing	90·9	4·84	50·4	
−150+200	11·1	Concentrate	8·9	47·42	45·3	5·69
		Middling	1·8	31·05	6·0	
		Tailing	89·3	5·09	48·7	
−200	33·6	Concentrate	9·4	41·26	48·5	16·70
		Middling	0·4	23·80	1·2	
		Tailing	90·2	4·46	50·3	
Total						54·05

Total slime................215·1 grammes   Pitchblende..........4·65 per cent

The table concentrate with the small amount of middling constitutes a high-grade pitchblende product. The tailings average under 5 per cent pitchblende but they represent 45 per cent of the pitchblende in the ore.

Sized material makes possible the production of a higher grade product on the tables than a straight test on −48-mesh ore, although the actual recoveries of pitchblende were higher in Test No. 1.

Test No. 3

This test, on a full-size Wilfley table, was carried out with 225 pounds of ore previously ground to pass a 65-mesh screen.

The ore was fed wet to the table from an automatic feeder at the rate of 120 pounds per hour. During the course of the run, the middling product was collected and re-run over the table at frequent intervals. The concentrate from the clean-up of the table was run over a small laboratory Wilfley table and the products added to those already obtained on the large table.

Results of Table Test No. 3:

Product	Weight, per cent	Assays		Distribution, per cent	
		Silver, oz/ton	Pitch-blende, per cent	Silver	Pitchblende
Concentrate	9·36	3,198·0	40·85	36·8	53·9
Middling	0·62	1,142·1	10·05	0·8	0·8
Tailing and slime	90·02	450·6	3·56	62·4	45·3

Screen Tests:

Concentrate		Middling		Tailing	
Mesh	Weight, per cent	Mesh	Weight, per cent	Mesh	Weight, per cent
+ 65	0·6	+ 65	3·1	+ 65	0·1
+100	14·7	+100	34·6	+100	13·8
+150	29·2	+150	25·7	+150	33·9
+200	20·7	+200	14·4	+200	23·5
−200	34·8	−200	22·2	−200	28·7
	100·0		100·0		100·0

Three flotation tests were run on the tailing from the table test in order to concentrate further the silver contained therein.

Test No. 4

Additions to grinding jar:

Tailing	1,000 grammes
Water	750 c.c.
Coal-tar creosote	20 drops
Sodium ethyl xanthate	1·0 lb/ton
Sodium silicate	2·0 "
Grinding time:	30 minutes.

Product	Weight, per cent	Assays		Distribution, per cent		Ratio of concentration
		Silver, oz/ton	Pitch-blende, per cent	Silver	Pitchblende	
Concentrate	19·8	2,274·2	8·02	80·1	44·6	5·05 : 1
Tailing	80·2	139·0	2·46	19·9	55·4	

Screen Test Flotation Tailing:

Mesh	Weight, per cent
+150	0·7
+200	14·9
−200	84·4
	100·0

Recapitulation of Tests Nos. 3 and 4:

Silver recovery in table concentrate and middling .... 37·6 per cent
Silver recovery by flotation of table tailing, 80·1 per cent of 62·4 per cent .... 50·0 "

Possible overall recovery of silver .... 87·6 "

Test No. 5

Additions to grinding jar:

Tailing .... 1,000 grammes
Water .... 750 c.c.
Sodium silicate .... 1·0 lb/ton
Sodium sulphide .... 2·0 "
Sodium ethyl xanthate .... 1·0 "
Grinding time .... 15 minutes
Pine oil to cell .... 0·05 lb/ton

Product	Weight, per cent	Assay, silver, oz/ton	Distribution of silver, per cent	Ratio of concentration
Concentrate	7·9	3,584·0	54·1	12·6 : 1
Tailing	92·1	280·8	45·9	

Screen Test on Tailing:

Mesh	Weight, per cent
+100	1·4
+150	7·9
+200	29·7
−200	61·0
Total	100·0

Test No. 6

Additions to grinding jar:

Tailing .... 1,000 grammes
Water .... 750 c.c.
Coal-tar creosote .... 30 drops
Sodium silicate .... 1·0 lb/ton
Grinding time .... 15 minutes
No pine oil added

Product	Weight, per cent	Assay, silver, oz/ton	Distribution of silver, per cent	Ratio of concentration
Concentrate	8·4	3,441·6	54·7	11·9 : 1
Tailing	91·6	280·9	45·3	

Screen Test on Tailing:

Mesh	Weight, per cent
+100	0·9
+150	5·7
+200	25·7
−200	67·7
Total	100·0

Fine grinding is apparently an essential to obtain a high recovery in the concentrate. A small concentration of pitchblende is obtained by flotation.

Test No. 7

This flotation test was run to determine whether a separation of the carbonates and the pitchblende could be effected.

Tailing	1,000 grammes
Water	750 c.c.
Soda ash	0·5 lb/ton
Grinding time	30 minutes

Reagents to cell:

Oleic acid	3 drops
Pine oil	3 "

Product	Weight, per cent	Assays			Distribution, per cent			Ratio of concentration
		Silver, oz/ton	Pitchblende, per cent	Manganese, per cent	Silver	Pitchblende	Manganese	
Concentrate	39·36	742·0	3·82	17·15	55·2	39·3	40·9	2·5 : 1
Tailing	60·64	389·8	3·82	16·03	44·8	60·7	59·1	

Screen Test of Flotation Tailing:

Mesh	Weight, per cent
+150	1·0
+200	8·75
−200	90·25
	100·00

DISCUSSION OF PITCHBLENDE CONCENTRATION

The large table test gives slightly better results than obtained on the small table with sized material.

The percentage recovery on the small table (including the middling product) was 54·05 per cent, whereas the recovery on the large table was 54·70 per cent. The grade of concentrate on the large table (including the small middling product) had a pitchblende content of 39·09 per cent.

A lower tailing resulted in the large table test, showing a pitchblende content of 3·56 per cent against a 4 to 5 per cent tailing on the small table.

Flotation tests showed little if any concentration of pitchblende. With over 90 per cent of the material ground to pass a 200-mesh screen, no appreciable separation or concentration of the pitchblende from the manganese carbonate was effected. From the tests made, flotation offers little encouragement for concentration of the pitchblende.

GENERAL SUMMARY AND CONCLUSIONS

Silver. It is evident that by concentration methods about 90 per cent of the silver can be concentrated. In the case of the ore tested, a tailing carrying from 100 to 200 ounces per ton remained. This appeared to be the lowest tailing obtainable by table and flotation tests. From these results it is apparent that a chemical treatment will be necessary to recover the silver from such tailings.

Pitchblende. By table concentration it was possible to obtain a good grade of pitchblende concentrate representing a recovery of almost 60 per cent.

Flotation is, however, unsatisfactory for the pitchblende in the ore.

Ore Dressing and Metallurgical Investigation No. 558

GOLD ORE FROM LAC DES MILLE LACS PROPERTY, KASHABOWIE, THUNDER BAY DISTRICT, ONTARIO

Shipment. A shipment consisting of a sample of gold ore, weight 25 pounds, was received on January 3, 1934, at the Ore Dressing and Metallurgical Laboratories from the Lac des Mille Lacs property, Thunder Bay district, Ontario, and was submitted by Ventures, Limited.

Characteristics of the Ore. Polished sections of the ore contain various sulphides and native gold distributed sporadically in a white milky quartz gangue. Pyrite, chalcopyrite, sphalerite, and galena are the most abundant metallic minerals, but aikinite is common. Small amounts of tetrahedrite, native gold, native bismuth, and two undetermined minerals are present. Surface alteration has led to the presence of considerable "limonite" and covellite.

The ore was sampled and assayed as follows:

Gold	1·273 oz/ton
Silver	9·80 "
Copper	1·41 per cent

Experimental tests consisted of amalgamation, flotation and blanket concentration, and cyanidation.

Experimental Tests

Test No. 1

This was a straight amalgamation test on the raw ore; 1,000 grammes of −14-mesh ore was ground wet in a pebble jar for fifteen minutes and then amalgamated in a jar with 100 grammes of mercury for one hour. The amalgam, separated and collected in a hydraulic classifier, showed no sign of sickening.

Product	Assay, gold, oz/ton	Recovery, per cent
Amalgamation tailing	0·42	67·0

Mesh	Weight, per cent
+ 48	1·1
+ 65	1·5
+100	16·7
+150	22·1
+200	19·6
−200	39·0
Total	100·0

Test No. 2

This test consisted in grinding 1,000 grammes of the ore wet for 25 minutes and then barrel-amalgamating with 100 grammes of mercury for one hour. The amalgam was collected and the tailing floated.

Amalgamation Results:

Product	Assay, gold, oz/ton	Recovery, per cent
Amalgamation tailing	0·45	64·6

Reagents to flotation cell:

Soda ash	2 lb/ton
Sodium ethyl xanthate	0·4 "
Aerofloat No. 31	2 drops

Flotation Results:

Product	Weight, per cent	Assays, oz/ton		Distribution of metals, per cent		Ratio of concentration
		Gold	Silver	Gold	Silver	
Concentrate	10·71	3·10	74·30	72·6	76·7	9·3 : 1
Tailing	89·29	0·14	2·70	27·4	23·3	

Copper in flotation concentrate.................................. 12·58 per cent

Screen Test Flotation Tailing:

Mesh	Weight, per cent
+ 48	0
+ 65	0·1
+100	3·9
+150	15·1
+200	25·4
−200	55·5
Total	100·00

Gold recovered by amalgamation	64·6 per cent
Gold recoverable in flotation concentrate, 72·6 per cent of (100−64·6)	25·7 "
Possible overall gold recovery	90·3 "

Test No. 3

This test consisted of floating the raw ore and running the flotation tailing over corduroy blankets.

A sample, 1,000 grammes, of −14-mesh ore was ground wet for 25 minutes with the following reagents:

Soda ash	2 lb/ton
Sodium ethyl xanthate	0·4 "
Aerofloat No. 31	3 drops
Pine oil to cell	0·05 lb/ton

Product	Weight, per cent	Assay, oz/ton		Distribution of metals, per cent		Ratio of concentration
		Gold	Silver	Gold	Silver	
Concentrate	9·08	10·06	94·70	74·0	78·6	11 : 1
Tailing	90·92	0·352	2·57	26·0	21·4	

Copper in flotation concentrate ... 14·68 per cent

Blanket Test of Flotation Tailing:

Product	Weight, per cent	Assays, oz/ton		Distribution of metals, per cent		Ratio of concentration
		Gold	Silver	Gold	Silver	
Concentrate	6·53	4·26	9·32	78·7	23·7	15·3 : 1
Tailing	93·47	0·08	2·10	21·3	76·3	

Screen Test on Blanket Tailing:

Mesh	Weight, per cent
+100	0·7
+150	9·3
+200	22·5
−200	67·5
Total	100·00

Gold by flotation	74·00 per cent
Gold by blanketing of flotation tailing, 78·7 per cent of 26 per cent	20·46 "
Total gold in concentrates	94·46 "
Silver by flotation	78·6 "
Silver by blanketing of flotation tailing, 23·7 per cent of 21·4 per cent	5·1 "
Total silver in concentrates	83·7 "

Test No. 4

This test was the same as Test No. 3, followed by barrel amalgamation of the combined flotation and blanket concentrates.

Weight of blanket tailing	810·5 grammes
Gold assay of blanket tailing	0·05 oz/ton
Gold assay of combined flotation and blanket concentrates	6·45 "

Results of Amalgamation of Concentrates:

Product	Weight, grammes	Assay, gold, oz/ton	Recovery, per cent
Amalgamation tailing....................................	191·9	1·75	72·86

Total gold in flotation and blanket concentrates..................	96·8	per cent
Gold recovered from concentrates by amalgamation, 72·86 per cent of 96·8 per cent....................................	70·53	"

Test No. 5

A sample, 500 grammes, of ore was ground for 25 minutes in a pebble jar and then barrel-amalgamated with 50 grammes of mercury for one hour.

Assay of amalgamation tailing, gold.............................	0·33 oz/ton
Per cent recovery by amalgamation.............................	74·0 per cent

A sample, 200 grammes, of amalgamation tailing was agitated for 24 hours in a cyanide solution of 2 pounds KCN per ton and 5 pounds lime per ton at a pulp ratio of 3·42 : 1.

Product	Assay, gold, oz/ton	Final solution, lb/ton		Consumption, lb/ton		Recovery, per cent
		KCN	CaO	KCN	CaO	
Cyanide tailing......	0·03	1·5	0·8	11·97	8·31	90·9

Recovery of gold by amalgamation..............................	74·0	per cent
Recovery of gold by cyanidation of amalgamation tailing, 90·9 per cent of 26 per cent....................................	23·6	"
Overall gold recovery...............................	97·6	"

Despite the good recovery and low tailing obtained, the high cyanide consumption and consequent fouling of the solution with copper indicate the disadvantages of cyanide treatment.

Test No. 6

Cyanidation test on straight ore. The ore was ground for 25 minutes prior to agitation at a 3·44 : 1 pulp ratio in cyanide solution of 2 pounds KCN per ton and 5 pounds lime per ton for 24 hours.

The cyanide consumption was high. During the first three hours the consumption was 2·88 pounds per ton per hour.

Product	Assays, oz/ton		Final solution, lb/ton		Consumption, lb/ton		Recovery, per cent	
	Gold	Silver	KCN	CaO	KCN	CaO	Gold	Silver
Cyanide tailing............	0·035	2·88	1·1	0·55	13·42	9·20	97·2	70·6

Screen Test of Cyanide Tailing:

Mesh	Weight, per cent
+150	0·1
+200	2·0
−200	97·9
	100·0

CONCLUSIONS

Barrel amalgamation tests on this ore indicate that there is a maximum of 70 per cent of the gold existing in the free state that, under ideal conditions, could be recovered by barrel amalgamation.

Flotation, followed by blanket concentration of the flotation tailing, showed that over 94 per cent of the gold and 83·7 per cent of the silver were obtained in the combined concentrates.

Cyanidation does not offer a satisfactory method of treatment. The copper content of the ore will tend towards high cyanide consumption and consequent fouling of the solution, which will result in lowering the solvent action of the cyanide solution on the gold.

In regard to a flow-sheet for the treatment of this ore, it is suggested that grinding be carried out so that at least 70 per cent will be −200 mesh. The use of a unit flotation cell with a trap bottom, similar to the machine used at the McIntyre mill, is recommended; the cell to be placed at the ball mill discharge, between the ball mill and the classifier; the classifier overflow to be re-floated in a standard mechanical flotation cell and the flotation tailing then passed over blankets or concentrating tables.

The use of the unit cell will relieve the grinding circuit of its accumulation of free gold. The trap in the base of the cell catches the coarse nuggets.

Ore Dressing and Metallurgical Investigation No. 559

GOLD ORE FROM PASCALIS GOLD MINES, LIMITED, PASCALIS TOWNSHIP, ABITIBI COUNTY, QUEBEC

Shipment. A shipment of 31 bags of ore, net weight 2,542 pounds, was received December 18, 1933. The sample was submitted by J. M. C. Dunlop for Ventures, Limited, 100 Adelaide Street West, Toronto, Ontario.

Characteristics of the Ore. A polished section was prepared and examined microscopically to determine the metallic minerals.

The metallic minerals seen in the polished section are pyrite and "limonite." Pyrite is moderately abundant and is disseminated as coarse imperfectly-formed cubes up to one centimetre in diameter. It contains abundant small, irregular inclusions of gangue.

The limonite, which is abundant in the outer part of the large block of rock examined, is present also in the fresher parts as patches of finely-divided mineral, giving the rock a somewhat mottled appearance. Some coarse free gold was panned from a sample of ground ore. The gangue is essentially siliceous.

An average analysis of the head sample was as follows:

Gold	0·64	oz/ton
Silver	0·19	"
Iron	12·87	per cent
Sulphur	11·10	"
Insoluble	57·37	"

Experimental Tests

A series of small-scale tests was made on the ore to determine how it could be treated in practice. The work included tests by cyanidation, amalgamation, flotation, and hydraulic classification.

By cyanidation, 96·9 per cent of the gold can be extracted in 24 hours when all of the ore is ground dry through 200 mesh. By barrel amalgamation, 78·9 per cent of the gold can be extracted when all of the ore is ground dry through 100 mesh. This figure dropped to 67·2 per cent when the ore was wet-ground 93·5 per cent through 200 mesh. When the ore was ground wet 90·0 per cent through 200 mesh and passed over an amalgamation plate, 62·5 per cent of the gold was extracted.

By straight flotation of the ore (Test No. 18), 98·7 per cent of the gold was recovered in a flotation concentrate amounting to 27·2 per cent of the weight of feed used.

In the hydraulic classifier 70·9 per cent of the gold settled out in a product amounting to 8·1 per cent of the weight of feed used.

Details of the tests follow:

CYANIDATION

Tests Nos. 1 to 8

In this series of tests four lots of the ore were ground dry through 48-, 100-, 150-, and 200-mesh screens. Samples of each were agitated in cyanide solution, 2·0 pounds KCN per ton, for periods of 24 and 48 hours. Protective alkalinity was maintained by the addition of lime. The tailings were washed and assayed for gold.

Results:

Feed sample: gold, 0·64 oz/ton

Test No.	Mesh	Period of agitation, hours	Tailing assay, gold, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
					KCN	CaO
1	− 48	24	0·035	94·5	1·0	19·7
2	−100	24	0·025	96·1	1·0	21·0
3	−150	24	0·03	95·3	1·3	35·4
4	−200	24	0·02	96·9	2·2	50·0
5	− 48	48	0·04	93·8	1·0	20·4
6	−100	48	0·03	95·3	1·0	21·3
7	−150	48	0·02	96·9	2·2	35·5
8	−200	48	0·02	96·9	3·4	50·1

BARREL AMALGAMATION

Tests Nos. 9 and 10

Samples of the ore, ground dry through 48-, 100-, 150-, and 200-mesh screens, were amalgamated with mercury in jar mills for 30 minutes. The tailings were assayed for gold.

Results:

Feed sample	Gold,	0·64 oz/ton
− 48-mesh amalgamation tailing	Gold,	0·14 "
Recovery		78·1 per cent
−100-mesh amalgamation tailing	Gold,	0·135 oz/ton
Recovery		78·9 per cent

PLATE AMALGAMATION

Test No. 13

A sample of the ore was ground 90·0 per cent through 200 mesh in a ball mill and then passed over a small amalgamation plate. The tailing was assayed for gold. The grinding time was 25 minutes, with the ore all through 14 mesh to start with.

Results:

Feed sample	Gold,	0·64 oz/ton
Amalgamation tailing	Gold,	0·24 "
Recovery		62·5 per cent

BARREL AMALGAMATION AND FLOTATION

Test No. 12

The ore was ground 93·5 per cent through 200 mesh in a ball mill and then amalgamated with mercury in a jar mill for 30 minutes. The amalgamation tailing was filtered and washed and then floated. The flotation concentrate and tailing were assayed for gold and the amalgamation tailing assay was calculated from them. Samples of the concentrate were reground and cyanided 24 and 48 hours.

Charge to ball mill:

Ore	2,000 grms. at −14 mesh
Water	1,500 c.c.

Grinding time: 30 minutes.

Reagents to cell:

Soda ash	5·0 lb/ton
Potassium amyl xanthate	0·20 "
Pine oil	0·25 "

Summary:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Flotation concentrate	28·9	0·70	96·6
Flotation tailing	71·1	0·01	3·4
Amalgamation tailing (cal.)	100·0	0·21	100·0
Flotation concentrate cyanided	28·9	0·03	....

Ratio of concentration	3·46 : 1
Recovery by amalgamation	67·2 per cent total gold
Recovery in flotation concentrate (100·0−67·2) × 96·6	31·7 " "
Extraction by cyanidation of flotation concentrate, 31·7 × 95·7	30·3 " "
Net recovery, 67·2 + 30·3	97·5 " "
Average tailing from flotation and cyanidation	0·016 oz/ton in gold.

Reagents consumed in pounds per ton of concentrate:

KCN	2·50
CaO	18·80

FLOTATION

Test No. 18

The ore was ground 90·0 per cent through 200 mesh in a ball mill and floated. The concentrate and tailing were assayed for gold.

Charge to ball mill:

Ore	2,000 grms. at −14 mesh
Water	1,500 c.c.
Soda ash	5·0 lb/ton

Grinding time: 25 minutes.

Reagents to cell:

Potassium amyl xanthate	0·20 lb/ton
Pine oil	0·10 "

Summary:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Concentrate	27·2	2·06	98·7
Tailing	72·8	0·01	1·3
Feed (cal.)	100·0	0·568	100·0

Ratio of concentration........ 3·68 : 1

GRINDING TESTS

Samples of the ore at −14 mesh were ground for different periods of time and then screened to determine the grinding. The results are shown in the following table:

Period of grinding, minutes	Weight, per cent −65+100	Weight, per cent −100+150	Weight, per cent −150+200	Weight, per cent −200	Total
15	0·2	8·6	16·5	74·7	100·0
20	0·2	4·7	12·5	82·6	100·0
25	0·2	1·5	8·2	90·1	100·0
30	0·1	0·6	5·8	93·5	100·0

HYDRAULIC CLASSIFICATION

Test No. 16

The ore was ground 82·6 per cent through 200 mesh in a ball mill and then put through a hydraulic classifier where the coarse gold and heavy minerals were allowed to settle out against a slowly rising current of water. The classifier oversize and overflow were assayed for gold.

Summary:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Classifier oversize	8·1	6·21	70·9
Classifier overflow	91·9	0·225	29·1
Feed (cal.)	100·0	0·71	100·0

MILL RUNS

A series of large-scale mill runs was made to check up on the small-scale tests. For this purpose a unit of 100 pounds per hour capacity was used.

In the first run the pulp from the rod mill passed over an amalgamation plate into an Akins classifier. The oversize from the classifier was returned to the rod mill for regrinding and the classifier overflow went to flotation. Approximately 45 per cent of the gold was recovered on the amalgamation plate and an additional 53 per cent was recovered in the flotation concentrate, leaving a flotation tailing assaying 0·015 ounce gold per ton.

In the second large-scale test, a hydraulic trap replaced the amalgamation plate. Otherwise the flow-sheet was the same as was used for the first run.

About 10 per cent of the gold was caught in the trap in a product assaying more than 10 ounces gold per ton. The flotation tailing again assayed 0·015 ounce gold per ton, representing an overall recovery of approximately 98 per cent of the gold.

The third large-scale test was made without trap or amalgamation plate. The rod mill pulp went to an Akins classifier, the overflow from which went to flotation. The flotation tailing was blanketed, but little or nothing was recovered in this way. The flotation tailing assayed 0·025 ounce gold per ton both before and after blanketing.

For all three tests the reagents used were as follows:

Soda ash	5·0 lb/ton
Potassium amyl xanthate	0·25 "
Pine oil	0·10 "

CONCLUSIONS

The work done on the sample submitted points to cyanidation as the most satisfactory method of treatment for recovery of the gold. Amalgamation plates or a hydraulic trap should be used at the ball mill discharge to prevent coarse gold building up in the classifier and agitators. A hydraulic trap would be preferable in this case, as it would permit grinding in cyanide solution. The trap cleanings would be barrel-amalgamated and then re-united with the main body of the ore in the cyanide agitators. Good net recoveries were obtained by amalgamation and flotation with cyanidation of the flotation concentrate, but, owing to the high sulphide content of the ore, the ratio of concentration is so low that this step would be of doubtful economic value unless a ready market were available for the sulphide concentrate.

Lime consumption was very high, but this was, no doubt, due to the oxidized condition of the sample submitted. This condition is not likely to be found in fresh ore mined at depth.

Ore Dressing and Metallurgical Investigation No. 560

GOLD ORE FROM DIKDIK EXPLORATION COMPANY, LONGLAC, THUNDER BAY DISTRICT, ONTARIO

Shipment. The shipment consisting of gold ore, weight 25 pounds, was received on January 5, 1934, from the Dikdik Exploration Company, Atigogama lake, Thunder Bay district, Ontario, and was submitted by K. M. Fritsche, Longlac, Ontario.

Characteristics of the Ore. The gangue is light grey to white translucent quartz. The metallic minerals present in the polished sections are: pyrite, sphalerite, arsenopyrite, chalcopyrite, galena, tetrahedrite, and native gold. Of these pyrite is the only abundant mineral.

Native gold is abundant in the sections examined. A large percentage of the metal is within pyrite, which it veins.

The ore was crushed to −14 mesh and sampled by standard methods.

The head sample assayed as follows:

Gold	4·525	oz/ton
Silver	1·58	"
Arsenic	0·06	per cent
Copper	0·09	"

Experimental Tests

Test work carried out was as follows:

1. Barrel amalgamation on straight ore.
2. Barrel amalgamation followed by flotation of amalgamation tailing.
3. Flotation of raw ore followed by blanketing of flotation tailing.
4. As in No. 3 above, followed by amalgamation of combined flotation and blanket concentrates.

Test No. 1

A sample, 1,000 grammes, of −14-mesh ore was ground for 15 minutes in a pebble jar and then barrel-amalgamated with 100 grammes of mercury for one hour.

Product	Assays, oz/ton		Recovery, per cent	
	Gold	Silver	Gold	Silver
Amalgamation tailing	1·29	1·14	71·49	27·85

Screen Test on Tailing:

Mesh	Weight, per cent
+ 48	0·1
+ 65	0·1
+100	6·4
+150	15·5
+200	21·5
−200	56·4
	100·0

Test No. 2

A sample, 1,000 grammes, of −14 mesh was ground for 15 minutes and then barrel-amalgamated for one hour with 100 grammes of mercury. The amalgam was separated in a hydraulic classifier and the tailing floated.

Reagents:

Soda ash	2·0 lb/ton
Sodium ethyl xanthate	0·4 "
Aerofloat No. 31	3·0 drops

Amalgamation Results:

Product	Assays, oz/ton		Recovery, per cent	
	Gold	Silver	Gold	Silver
Amalgamation tailing	1·18	1·095	73·9	30·7

Flotation Results:

Product	Weight, per cent	Assays, oz/ton		Distribution of precious metals, per cent		Ratio of concentration
		Gold	Silver	Gold	Silver	
Concentrate	14·9	7·24	6·38	91·3	86·8	6·7 : 1
Tailing	85·1	0·12	0·17	8·7	13·2	

Screen Test Flotation Tailing:

Mesh	Weight, per cent
+ 48	0
+ 65	0·7
+100	7·3
+150	13·1
+200	22·6
−200	56·3
	100·0

Gold recovery by amalgamation	73·9 per cent
Gold in flotation concentrate, 91·3 per cent of 26·1 per cent	23·8 "
Possible overall gold recovery	97·7 "

Silver recovery by amalgamation	30·7 per cent
Silver in flotation concentrate, 86·8 per cent of 69·3 per cent	60·1 "
Possible overall silver recovery	90·8 "

Test No. 3

A sample, 1,000 grammes, of −14-mesh ore was ground wet for 15 minutes with the following reagents:

Soda ash	2·0 lb/ton
Sodium ethyl xanthate	0·4 "
Aerofloat No. 31	3·0 drops

Flotation Test:

Product	Weight, per cent	Assays, oz/ton		Distribution of precious metals, per cent		Ratio of concentration
		Gold	Silver	Gold	Silver	
Concentrate	12·3	34·81	13·54	81·6	90·4	8·1 : 1
Tailing	87·7	1·10	0·20	18·4	9·6	

Blanket Test:

Product	Weight, per cent	Assays, oz/ton		Distribution of precious metals, per cent		Ratio of concentration
		Gold	Silver	Gold	Silver	
Concentrate	7·1	13·41	0·90	86·49	31·5	14 : 1
Tailing	92·9	0·16	0·15	13·51	68·5	

Screen Test Blanket Tailing:

Mesh	Weight, per cent
+ 65	0·6
+100	6·7
+150	12·8
+200	22·8
−200	57·1
	100·0

Gold in flotation concentrate	81·6 per cent
Gold in blanket concentrate of flotation tailing, 86·49 per cent of 18·4 per cent	15·9 "
Total gold in concentrates	97·5 "

Silver in flotation concentrate	90·4 per cent
Silver in blanket concentrate of flotation tailing, 31·5 per cent of 9·6 per cent	3·0 "
Total silver in concentrates	93 4 "

Test No. 4

A sample, 1,000 grammes, of ore was ground, floated, and the flotation tailing was run over blankets.

The flotation and blanket concentrates were combined and barrel-amalgamated with 100 grammes of mercury for one hour.

Weight of blanket tailing..		82·7 per cent
Assay of blanket tailing..	Gold,	0·13 oz/ton
	Silver,	0·15 "
Calculated assay of combined concentrates................	Gold,	25·53 oz/ton
	Silver,	8·42 "

Amalgamation Results:

Product	Weight, grms.	Assays, oz/ton		Recovery, per cent	
		Gold	Silver	Gold	Silver
Amalgamation tailing...........	173·3	9·08	5·67	64·4	32·6

Gold in flotation and blanket concentrates........................	97·6 per cent
Gold recovery by amalgamation of concentrates, 64·4 per cent of 97·6 per cent..	62·85 "
Silver in flotation and blanket concentrates.......................	92·2 "
Silver recovery by amalgamation of concentrates, 32·6 per cent of 92·2 per cent..	30·06 "

Test No. 5

A sample, 500 grammes, of the ore was ground wet for 15 minutes and then barrel-amalgamated with 50 grammes of mercury for one hour. The mercury was separated in a hydraulic classifier and the amalgam tailing filtered. A sample of the tailing was cut out for a cyanidation test.

The tailing was agitated for 24 hours in a solution of 2 pounds KCN per ton and 5 pounds CaO per ton at a pulp ratio of 3·41 : 1.

Amalgamation Test:

Product	Assays, oz/ton		Recovery, per cent	
	Gold	Silver	Gold	Silver
Tailing....................................	1·01	1·11	77·6	30·3

Cyanidation Test:

Product	Assays, oz/ton		Final solution, lb/ton		Consumption, lb/ton		Recovery, per cent	
	Gold	Silver	KCN	CaO	KCN	CaO	Gold	Silver
Tailing.....	0·10	0·36	1·3	0·35	2·39	4·33	90·1	67·5

Screen Test of Cyanide Tailing:

Mesh	Weight, per cent
+100	0·2
+150	1·7
+200	8·6
−200	89·5
	100·0

Gold recovery by barrel amalgamation	77·6 per cent
Gold recovery by cyanidation of amalgamation tailing, 90·1 per cent of 22·4 per cent	20·18 "
Overall gold recovery	97·78 "
Silver recovery by barrel amalgamation	30·3 per cent
Silver recovery by cyanidation of amalgamation tailing, 67·5 per cent of 69·7 per cent	47·0 "
Overall silver recovery	77·3 "

Test No. 6

In this test the straight ore was cyanided for 24-hour and 48-hour periods in a cyanide solution of 2 pounds KCN per ton with 5 pounds CaO per ton at a pulp ratio of approximately 3 : 1.

Results:

Time of agitation, hours	Feed assays, per cent		Tailing assays, per cent		Final solution, lb/ton		Consumption, lb/ton		Recovery, per cent	
	Gold	Silver	Gold	Silver	KCN	CaO	KCN	CaO	Gold	Silver
24	4·525	1·58	0·115	0·36	1·6	0·40	1·20	3·80	97·4	77·2
48	4·525	1·58	0·125	0·39	1·3	0·35	2·10	3·95	97·2	75·3

Screen Test Tailing—24-hour Agitation:

Mesh	Weight, per cent
+ 65	0·1
+100	1·7
+150	6·5
+200	18·2
−200	73·5
Total	100·0

Results of cyanidation on the straight ore would indicate that prolonged contact of the ore with the solution causes a reprecipitation of the gold.

A lower tailing is obtained by first barrel-amalgamating the ore and then cyaniding the amalgamation tailing.

CONCLUSIONS

Tests by barrel amalgamation on the ore would indicate that at least 75 per cent of the gold is in the free state and under ideal conditions could be recovered by amalgamation.

Cyanidation does not appear to offer any serious difficulties. The consumptions of cyanide and lime are within reasonable limits.

Flotation tests on both amalgamation tailing and the raw ore show a high concentration of gold.

The cyanidation tests indicate a possible recovery of over 97 per cent of the gold and over 77 per cent of the silver. The tests would indicate that better extractions are possible at 24-hour agitation than at 48-hour.

Ore Dressing and Metallurgical Investigation No. 561

ORE FROM THE HOWARD MINE, NELSON, B.C.

Shipment. A shipment consisting of 9 bags of ore weighing 490 pounds was received on November 27, 1933. It was said to have been taken from the Howard mine, near Nelson, British Columbia, held by the B.C. Cariboo Gold Fields, Vancouver.

Characteristics of the Ore. Specimens of the ore were taken and a microscopic study of polished sections was made in the mineragraphic laboratory.

The gangue consists of greenish grey impure quartz with a minor amount of disseminated carbonate.

The metallic minerals are, in their order of abundance, pyrite, sphalerite, galena, pyrrhotite, chalcopyrite, tetrahedrite (or tennantite), and native gold.

Pyrite usually forms coarsely-granular masses or is disseminated as comparatively large irregular grains and is commonly contained in an assemblage composed of sphalerite and galena. It is veined by sphalerite, galena, and native gold, the sphalerite and galena being intimately associated. Native gold occurs in very tiny irregular grains in galena, sometimes against sphalerite or pyrite. It is present in pyrite as tiny veinlets.

EXPERIMENTAL TESTS

The shipment was crushed, ground, and sampled, and was found to contain 8·4 per cent lead, 9·1 per cent zinc, 0·10 per cent copper, 0·595 ounce gold, and 3·40 ounces silver per ton.

Flotation tests were made giving a lead concentrate, a zinc concentrate, and an iron pyrite concentrate. Cyanidation tests were conducted on the zinc flotation tailing to note what recovery of gold would result.

The results of the investigation show that 96 per cent of the lead, 82 per cent of the zinc, 60 per cent of the gold, and over 90 per cent of the silver can be recovered by flotation in marketable lead and zinc concentrates. Cyanidation of the flotation tailing reduces the gold content to 0·025 ounce, an extraction by flotation and cyanidation of 95·8 per cent.

Tests were made to determine the percentages of gold that would be found with the lead, zinc, and iron sulphides. To make a lead concentrate, it was necessary to depress the zinc and iron sulphides with sodium cyanide. It was found that this cyanide was dissolving approximately 10 per cent of the gold in the ore. In standard flotation practice this gold would be lost. In the following tests, for purposes of comparison, this loss of gold has been disregarded, recoveries being calculated from the sum of the gold in the products. This gives a feed assay approximately 0·09 ounce per ton of gold lower than the actual value of the shipment.

FLOTATION

Test No. 1

A sample of the ore was ground wet with 10 pounds soda ash, 0·30 pound sodium cyanide, and 1·0 pound zinc sulphide per ton until 90 per cent passed 200 mesh. A lead concentrate was then removed by flotation after the addition of 0·05 pound butyl xanthate and 0·24 pound cresylic acid per ton.

The pulp was then conditioned with 0·5 pound per ton copper sulphate and after adding 0·05 pound sodium xanthate and 0·12 pound pine oil per ton, a zinc concentrate was removed.

A further addition of 1·0 pound per ton copper sulphate was made, followed by 0·30 pound sodium xanthate and 0·12 pound pine oil per ton. An iron pyrite concentrate was then removed.

Results:

Product	Weight, per cent	Assay				Distribution of metals, per cent			
		Lead, per cent	Zinc, per cent	Gold, oz/ton	Silver, oz/ton	Lead	Zinc	Gold	Silver
Feed (cal.).......	100·0	8·31	9·10	0·50	3·56	100·0	100·0	100·0	100·0
Lead concentrate	14·1	57·40	5·25	2·40	23·10	97·4	8·1	67·2	91·3
Zinc concentrate.	14·2	0·60	52·10	0·12	0·98	1·0	81·2	3·4	3·9
Pyrite concentrate..........	25·2	0·30	2·47	0·22	0·40	0·9	6·9	11·0	2·8
Tailing..........	46·5	0·12	0·75	0·20	0·15	0·7	3·8	18·4	2·0

These results show that the bulk of the gold and silver is recovered with the lead. The gold content of the zinc concentrate and pyrite concentrate is lower than the feed sample. The flotation tailing carries 0·20 ounce gold per ton.

Test No. 2

In this test, the cyanide used to depress zinc and iron sulphides was reduced to 0·15 pound per ton. Other conditions were the same as in Test No. 1.

Results:

Product	Weight, per cent	Assay				Distribution of metals, per cent			
		Lead, per cent	Zinc, per cent	Gold, oz/ton	Silver, oz/ton	Lead	Zinc	Gold	Silver
Feed (cal.).......	100·0	8·37	9·08	0·49	3·68	100·0	100·0	100·0	100·0
Lead concentrate	17·5	46·70	7·37	2·14	20·00	97·7	14·2	76·6	95·1
Zinc concentrate.	14·5	0·40	51·80	0·10	0·50	0·7	82·7	3·0	2·0
Pyrite concentrate..........	28·0	0·25	0·40	0·14	0·22	0·8	1·2	8·1	1·6
Tailing..........	40·0	0·17	0·42	0·15	0·12	0·8	1·9	12·3	1·3

Decreasing the amount of cyanide used to depress zinc and pyrite results in a bulkier, lower grade lead concentrate. A lower grade tailing is made which can be accounted for by the larger weights of concentrates removed.

Test No. 3

In this test, lead and zinc concentrates were made and cleaned to note the grade of product produced. Zinc and iron sulphides were depressed with 0·20 pound cyanide per ton. Other conditions were similar to preceding tests.

Results:

Product	Weight, per cent	Assay				Distribution of metals, per cent			
		Lead, per cent	Zinc, per cent	Gold, oz/ton	Silver, oz/ton	Lead	Zinc	Gold	Silver
Feed (cal.).......	100·0	8·40	9·10	0·499	3·54	100·0	100·0	100·0	100·0
Lead concentrate	10·0	74·00	3·18	1·82	28·20	88·0	3·5	36·5	79·6
Lead middling...	4·3	15·60	14·79	2·82	8·86	8·0	7·0	24·3	10·8
Zinc concentrate.	13·3	0·85	55·25	0·35	1·02	1·3	80·7	9·3	3·8
Zinc middling...	2·4	1·19	13·52	1·125	1·92	0·4	3·6	0·5	1·3
Pyrite concentrate..........	31·7	0·35	1·36	0·10	0·20	1·4	4·7	6·4	1·8
Tailing..........	38·3	0·20	0·11	0·30	0·25	0·9	0·5	23·0	2·7

It is noted that the gold has a tendency to drop out of the concentrates on cleaning. This indicates advisability of treating the middling or cleaner tailing in separate circuits. If these products are returned to the main circuit increased tailing losses can be expected.

FLOTATION AND CYANIDATION

Test No. 4

The results of preceding tests indicate that there is no appreciable concentration of gold in an iron pyrite concentrate. In this test, no pyrite concentrate was made but the tailing from the zinc flotation circuit was cyanided for 48 hours with a 1·0 pound KCN solution, 1 : 3 dilution. Five pounds of lime was added to maintain protective alkalinity.

To establish the fact that the cyanide used to depress the zinc and iron sulphides in the lead flotation circuit was dissolving gold, the solutions from the two concentrates and flotation tailing were filtered off, evaporated, and the gold recovered in the assay furnace.

Product	Weight, per cent	Assay				Weight, per cent × assay, gold	Distribution of metals, per cent			
		Lead, per cent	Zinc, per cent	Gold, oz/ton	Silver, oz/ton		Lead	Zinc	Gold	Silver
Feed.............	100·0	8·40	9·10	0·595	3·40	59·50	100·0	100·0	100·0	100·0
Feed (cal.)........	100·0	8·45	9·09	0·559	3·61	55·90	......	......	......	......
Lead concentrate..	14·6	55·40	3·20	2·42	22·72	35·33	95·7	5·1	63·2	91·8
Zinc concentrate...	14·2	0·80	51·90	0·10	0·64	1·42	1·4	81·1	2·5	2·5
Tailing...........	71·2	0·35	1·76	0·185	0·29	13·27	2·9	13·8	23·6	5·7
Solutions.........	....	....	....	....	....	6·01	......	......	10·7	......

Cyanidation of the zinc flotation tailing reduced the gold content from 0·185 ounce to 0·025 ounce within 24 hours with a reagent consumption of 1·5 pounds KCN and 4·4 pounds lime per ton of ore. No increase in extraction was obtained by longer agitation.

These results show that a recovery of 95·8 per cent of the gold can be effected by flotation and cyanidation.

Loss of gold owing to the use of cyanide in the flotation circuits amounts to 0·06 ounce per ton. If flotation is conducted with a pulp density of 30 per cent solids, the solutions will contain approximately 0·026 ounce gold per ton.

SUMMARY AND CONCLUSIONS

Flotation should recover 95 per cent of the lead, over 90 per cent of the silver, and from 60 to 70 per cent of the gold, in a concentrate containing over 70 per cent lead.

Approximately 80 per cent of the zinc can be recovered in a product assaying over 50 per cent zinc and carrying from 0·10 ounce to 0·30 ounce gold per ton.

Most of the gold is found in the lead concentrate. The zinc and iron sulphides do not carry much gold nor is any noticeable concentration made in these concentrates.

The gold remaining in the flotation tailing can readily be reduced to 0·025 ounce per ton by cyanidation.

Apparently the gold occurs in such a form as to be quite soluble in cyanide solution. To make a separation by flotation of the lead and zinc minerals, it is necessary to use cyanide to depress zinc and iron sulphides. This has been shown to result in a loss of about 10 per cent of the gold in the ore.

To recover this gold, two methods may be employed. The flotation tailing as delivered from the circuit could be brought up to strength with cyanide and lime and agitated for extraction of the remaining gold. Filtration and precipitation of the solution will recover all gold. However, this solution after precipitation would have to be wasted, owing to accumulation of incoming fresh flotation tailing.

Provision should be made to destroy any cyanide in the solution before running to waste, otherwise stream pollution may occur.

In this arrangement, one set of filters would be sufficient for the cyanide plant.

The other plan, which would require two sets of filters, would be to filter all or part of the flotation tailing and precipitate the clear filtrate by running through charcoal. Owing to the lack of free cyanide, regular zinc precipitation probably would not prove satisfactory. The dewatered tailing would then be agitated with cyanide solution and the barren solution returned to the head of the cyanide plant.

Ore Dressing and Metallurgical Investigation No. 562

GOLD ORE FROM THE NORMONT GOLD MINE AT ROUYN, QUEBEC

Shipment. A shipment of ten sacks of ore, net weight 750 pounds, was received January 15, 1934. The sample was submitted by J. F. B. Davies, General Manager, Normont Gold Mines, Limited, Rouyn, Quebec.

Characteristics of the Ore. The gangue consists chiefly of greenish grey material, probably chloritic in nature. A considerable amount of finely disseminated carbonate is present, and in places small masses of white carbonate can be seen with the naked eye. Small disseminated grains of leucoxene (?), probably resulting from the alteration of iron oxide, are common in some sections.

The metallic minerals present are: pyrite, chalcopyrite, and native gold. Pyrite is disseminated in rather coarse irregular grains, and contains small inclusions of gangue, chalcopyrite, and gold. Chalcopyrite was seen only as rare, small irregular grains in pyrite. Native gold occurs almost wholly within pyrite, but a few tiny grains were seen in the gangue and against the gangue-pyrite boundaries. All of the gold seen is minus 325 mesh in size.

An average analysis of the sample submitted is as follows:

Gold	0·52 oz/ton
Silver	0·04 "
Arsenic	0·02 per cent
Copper	Trace "

Experimental Tests

A series of small-scale tests was made on the ore to determine the methods by which it might be treated in practice. The work consisted of tests by cyanidation, amalgamation, flotation, blanketing, and hydraulic classification. By straight cyanidation of the ore 95·2 per cent of the gold can be extracted in 24 hours. When the ore is barrel-amalgamated and then cyanided for 24 hours, extraction is increased to 96·2 per cent of the gold. By flotation 98·4 per cent of the gold was recovered in a concentrate assaying 3·52 ounces gold per ton, and a tailing assaying 0·01 ounce gold per ton was left. The ratio of concentration was 6·84 : 1.

By plate amalgamation 45·0 per cent of gold was recovered as amalgam and an additional 52·5 per cent of the gold was recovered by floating the amalgamation tailing. The flotation tailing assayed 0·015 ounce gold per ton. By barrel amalgamation 75·0 per cent of the gold was extracted when the ore was ground dry through 100 mesh. By blanket concentration 44·1 per cent of the gold was recovered in a product amounting to 4·3 per cent of the weight of feed used. By floating the blanket tailing an additional 53·6 per cent of the gold was recovered, bringing the total recovery up to 97·7 per cent. The combined ratio of concentration was 6·85 : 1. In the hydraulic classifier only about 8·0 per cent of the gold settled out in the oversize product, which amounted to about 2·5 per cent of the weight of the ore used.

Details of the tests follow:

GRINDING TESTS ON THE ORE

Samples of the ore at −14 mesh were ground in a ball mill for different periods of time and the dried pulp was screened to determine the grinding that had taken place. The results may be tabulated as follows:

Mesh	Period of grinding in minutes			
	15	20	25	30
− 48+ 65	0·3	....	....	....
− 65+100	4·9	0·6	0·5	0·1
−100+150	9·0	3·7	2·6	1·4
−150+200	17·1	13·9	9·7	7·3
−200	68·7	81·8	87·2	91·2
Total	100·0	100·0	100·0	100·0

CYANIDATION

Tests Nos. 1 to 8

Samples of the ore ground dry through 48-, 100-, 150-, and 200-mesh screens were agitated in cyanide solution, 2·0 pounds KCN per ton, for periods of 24 and 48 hours. The tailings were assayed for gold.

Summary:

Test No.	Mesh	Period of agitation, hours	Tailing assay, gold, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
					KCN	CaO
1	− 48	24	0·03	94·2	0·4	6·0
2	−100	24	0·025	95·2	0·4	7·0
3	−150	24	0·025	95·2	0·7	8·3
4	−200	24	0·025	95·2	0·7	11·8
5	− 48	48	0·03	94·2	0·7	6·0
6	−100	48	0·025	95·2	0·7	7·1
7	−150	48	0·025	95·2	0·7	8·5
8	−200	48	0·025	95·2	0·7	12·8

AMALGAMATION AND CYANIDATION

Tests Nos. 9 and 10

Samples of the ore crushed dry through 48- and 100-mesh screens were amalgamated with mercury in jar mills for 30 minutes. The amalgamation tailings were sampled and assayed and portions of each were agitated in cyanide solution, 2·0 pounds KCN per ton, for a period of 24 hours. The cyanide tailings were filtered, washed, and assayed for gold.

Summary:

Test No.	Mesh	Amalgamation tailing, assay, gold, oz/ton	Extraction by amalgamation, per cent	Cyanide tailing, assay, gold, oz/ton	Extraction by cyanidation, per cent	Reagents consumed, lb/ton	
						KCN	CaO
9..........	− 48	0·17	67·3	0·02	28·9	0·4	6·0
10..........	−100	0·13	75·0	0·02	21·2	0·4	7·0

AMALGAMATION AND FLOTATION

Test No. 11

A sample of the ore was ground 87·2 per cent through 200 mesh in a ball mill, then passed over a small amalgamation plate and the amalgamation tailings floated.

Charge to ball mill:

Ore.. 2,000 grms. at −14 mesh
Water.. 1,500 c.c.

Grinding time.. 25 minutes

Reagents to cell:

Soda ash.. 8·0 lb/ton
Potassium amyl xanthate.. 0·10 "
Pine oil.. 0·10 "

Summary:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Flotation concentrate..................................	13·4	2·04	95·5
Flotation tailing......................................	86·6	0·015	4·5
Plate tailing (cal.)...................................	100·0	0·286	100·0

Feed sample..................................Gold, 0·52 oz/ton

Recovery by amalgamation........................... 45·0 per cent of total gold
Recovery in flotation concentrate (100·0—45·0)×95·5.. 52·5 " "

Total recovery.......................... 97·5 " "

FLOTATION

Test No. 12

A sample of the ore was ground 91·2 per cent through 200-mesh in a ball mill and floated.

Charge to ball mill:

Ore.. 2,000 grms. at −14 mesh
Water.. 1,500 c.c.
Soda ash... 5·0 lb/ton

Grinding time.. 30 minutes

Reagents to cell:

Potassium amyl xanthate.. 0·10 lb/ton
Pine oil.. 0·05 "

Summary:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Flotation concentrate	14·6	3·52	98·4
Flotation tailing	85·4	0·01	1·6
Feed (cal.)	100·0	0·522	100·0

Ratio of concentration........ 6·84 : 1

BLANKETING AND FLOTATION

Test No. 13

A sample of the ore was ground 91·2 per cent through 200 mesh in a ball mill and passed over a corduroy blanket set at a slope of 2·5 inches per foot. The blanket tailing was then floated. The blanket concentrate, the flotation concentrate, and the flotation tailing were assayed for gold.

Charge to ball mill:

Ore........ 2,000 grms. at −14 mesh
Water........ 1,500 c.c.

Grinding time........ 30 minutes

Reagents to cell:

Soda ash........ 5·0 lb/ton
Potassium amyl xanthate........ 0·10 "
Pine oil........ 0·05 "

Summary:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Blanket concentrate	4·3	5·66	44·2
Flotation concentrate	10·3	2·86	53·5
Flotation tailing	85·4	0·015	2·3
Feed (cal.)	100·0	0·551	100·0

Combined ratio of concentration........ 6·84 : 1

HYDRAULIC CLASSIFICATION

Test No. 14

A sample of the ore was ground 91·2 per cent through 200 mesh and passed through a hydraulic classifier where coarse gold and heavy minerals were allowed to settle out against a slowly rising current of water. The classifier oversize and the overflow were assayed for gold.

Summary:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Classifier oversize	2·5	1·74	8·2
Classifier overflow	97·5	0·50	91·8
Feed (cal.)	100·0	0·53	100·0

FLOTATION IN LIME PULP

Test No. 15

This test was made to see if the pyrite could be floated in a lime pulp, leaving a tailing low enough in gold to be discarded, and thus simplify the problem of cyaniding the concentrates.

The ore was ground 65 per cent through 200 mesh in a ball mill and then floated. The products were assayed for gold, a screen analysis being made on the flotation tailing.

Charge to ball mill:

Ore	2,000 grms. at −14 mesh
Water	1,500 c.c.
Lime	2·0 lb/ton
Grinding time	15 minutes

Reagents to cell:

Potassium amyl xanthate	0·10 lb/ton
Pine oil	0·05 "

Summary:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Flotation concentrate	9·7	5·12	89·9
Flotation tailing	90·3	0·062	10·1
Feed (cal.)	100·0	0·553	100·0

Screen Analysis Flotation Tailing:

Mesh	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
+100	7·4	0·115	13·8
−100+150	10·9	0·125	22·1
−150+200	18·3	0·095	38·2
−200	63·4	0·035	35·9
Average tailing (cal.)	100·0	0·062	100·0

FLOTATION IN NEUTRAL PULP

Test No. 16

This test was made to see if the pyrite could be floated in neutral pulp to facilitate its subsequent cyanidation and leave a flotation tailing low enough in gold to be discarded. The ore was ground 69 per cent through 200 mesh and floated. The concentrate was reground and cyanided for 48 hours. The cyanide tailing and flotation tailing were assayed for gold.

Charge to ball mill:

Ore	2,000 grms. at −14 mesh
Water	1,500 c.c.
Barrett No. 4	0·10 lb/ton
Potassium amyl xanthate	0·10 "
Grinding time	15 minutes

Reagents to cell:

Pine oil.. 0·10 lb/ton

Summary:

Product	Weight, per cent	Assay, gold, oz/ton	Extraction, per cent
Flotation concentrate cyanided..........................	16·0	0·13	....
Flotation tailing..	84·0	0·07	....
Average tailing (cal.)...................................	100·0	0·08	84·6

Test No. 17

This test differs from Test No. 16 in that the ore was ground 55 per cent through 200 mesh and Tarol No. 1 was added to the reagent combination. The concentrate was not cyanided.

Charge to ball mill:

Ore... 2,000 grms. at −14 mesh
Water... 1,500 c.c.
Barrett No. 4.................................. 0·10 lb/ton
Potassium amyl xanthate....................... 0·10 "

Grinding time.................................. 10 minutes

Reagents to cell:

Pine oil.. 0·05 lb/ton
Tarol No. 1.. 0·05 "

Summary:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Flotation concentrate................................	11·2	4·74	93·0
Flotation tailing.......................................	88·8	0·045	7·0
Feed (cal.)...	100·0	0·57	100·0

CYANIDATION WITH TABLING

Test No. 18

The ore was ground 87 per cent through 200 mesh and agitated in cyanide solution, 2·0 pounds KCN per ton, for 24 hours. The cyanide tailing was sampled and assayed and the remainder passed over a small concentrating table. The table concentrate and tailing were also assayed for gold.

Summary:

Cyanidation:

Feed sample.. 0·52 oz/ton in gold
Cyanide tailing .... 0·025 " "
Extraction.. 95·2 per cent

Reagents consumed:

KCN.. 1·13 lb/ton ore
CaO.. 7·9 " "

Tabling:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Table concentrate	8·7	0·20	79·2
Table tailing	91·3	0·005	20·8
Cyanide tailing (cal.)	100·0	0·022	100·0

With 4·8 per cent of the gold going to the table, the net recovery in the table concentrate is $79 \cdot 2 \times 4 \cdot 8 = 3 \cdot 80$ per cent. If this concentrate were reground and cyanided in a separate circuit, the total extraction might be increased by 2 or 3 per cent.

SUMMARY

This ore is amenable to cyanidation and to flotation. Cyanidation, with table concentration and regrinding of the sulphides, is to be recommended as the most practical process because upwards of 95 per cent of the gold can be recovered on the property.

Most of the gold, however, is associated with the pyrite and can be concentrated by flotation. By this process, a tailing assaying 0·01 ounce per ton in gold can be produced with a ratio of concentration of 6·84 : 1. If a ready market is available for this concentrate it would, therefore, be good practice to ship it. This concentrate was produced in a soda ash pulp and could not be cyanided without washing and reconditioning with lime. All attempts to float the pyrite in neutral or lime pulp were unsuccessful, the tailings in such cases assaying too high in gold to be discarded.

Ore Dressing and Metallurgical Investigation No. 563

GOLD ORE FROM MONTAGUE GOLD MINES, LIMITED, MONTAGUE, N.S.

Shipment. A shipment of 1,150 pounds of ore was received February 13, 1934, from the Montague Gold Mines, Limited, Montague, Nova Scotia.

Characteristics of the Ore. The ore consists of grey slate and milky-white to greyish white quartz. Twelve polished sections were prepared and examined microscopically. The metallic minerals present in the polished sections are, in their order of abundance, arsenopyrite, pyrite, chalcopyrite, galena, pyrrhotite, sphalerite, magnetite or ilmenite (?), and native gold. Arsenopyrite and pyrite are quite abundant; chalcopyrite occurs in moderate amounts; galena, pyrrhotite, and sphalerite occur in very small amounts; and magnetite or ilmenite (?) is very rare.

Mode of Occurrence of the Metallic Minerals. Arsenopyrite occurs as coarse crystals which are usually not elongated; however, a small amount does occur as fine needles. Coarsely crystalline aggregates are common in both quartz and slate, and these commonly form stringers along the walls of the veins of quartz, or within either quartz or slate. Most of the arsenopyrite is somewhat shattered, and veinlets are occupied by chalcopyrite, galena, native gold, or later gangue.

Pyrite is present as irregular grains in arsenopyrite or gangue, and in several places forms narrow stringers in the latter. It is commonly closely associated with arsenopyrite.

Chalcopyrite occurs as irregular grains and fine veinlets in arsenopyrite, and as large granular masses in quartz; the latter mode of occurrence is not common. The large masses contain sparse, small irregular grains of sphalerite, which itself encloses numerous tiny dots of chalcopyrite.

Galena is present in the arsenopyrite as small irregular grains and veinlets, and in gangue as small grains.

Pyrrhotite is common as tiny irregular grains in arsenopyrite, but the amount present is very small.

Sphalerite was seen only in chalcopyrite, as noted under that mineral.

Magnetite, and possibly ilmenite (?) is present as small grains in gangue, and very rarely in arsenopyrite. Leucoxene (?) was seen in some sections, and is probably a product of the alteration of ilmenite (?).

Native gold was seen only within arsenopyrite. It occurs as tiny irregular grains and discontinuous veinlets from one to ten microns in width. Some portions of the sections that show numerous tiny grains of gold, show them in lines. These suggest that they are really segments of irregular veinlets that have been cut by the polished surface in such a manner as to appear as isolated grains. A minor amount of gold occurs in chalcopyrite that veins arsenopyrite.

Average assays of the ore for gold and arsenic were as follows:

Gold	0·98 oz/ton
Arsenic	0·98 per cent

Experimental Tests

The following experimental tests were made to determine methods of recovering the gold in the ore.

(1) Amalgamation.

(2) Amalgamation and table concentration of the tailing from the amalgamation plates.

(3) Cyanide tests on table concentrate.

(4) Barrel amalgamation tests on table concentrate.

(5) Flotation test on amalgamation tailing.

(6) Cyanide tests on amalgamation tailing.

(7) Straight cyanide tests on crude ore.

AMALGAMATION

Tests Nos. 1 and 2

Two lots of 1,000 grammes of ore were ground to the following screen sizes and amalgamated by barrel amalgamation in a jar.

Screen Analysis of Amalgamation Tailing:

Mesh	Weight, per cent	Assay, gold, oz/ton
+ 28	0·51	
− 28+ 35	1·69	1·855
− 35+ 48	6·74	
− 48+ 65	18·00	0·39
− 65+100	19·82	0·52
−100+150	11·84	0·335
−150+200	12·46	0·295
−200	29·21	0·20
Total	100·00	0·47

Note.—This screen analysis amounts to a higher assay than the actual tailing. This is due no doubt to the high content of metallic gold in the coarser sizes, which are very spotty in distribution.

Results of Amalgamation:

Test No.	Feed assay, gold, oz/ton	Tailing assay, gold, oz/ton	Recovery, per cent
1	0·98	0·32	67·3
2	0·98	0·22	77·5

Table Concentration of Amalgamation Tailing:

Test No.	Product	Weight, per cent	Assay, gold, oz/ton	Distribution, per cent
1	Table concentrate	10·4	2·04 (cal.)	66·4
	Table tailing	89·6	0·12	33·6
	Total	100·0	0·32	100·0
2	Table concentrate	15·6	1·03 (cal.)	73·2
	Table tailing	84·4	0·07	26·8
	Total	100·0	0·22	100·0

Note.—The assay of the table concentrate is calculated, and no attempt was made to make a clean concentrate.

Overall Recoveries:

Test No. 1—

Amalgamation	67·3 per cent
In table concentrate	21·7 "
Total	89·0 "

Test No. 2—

Amalgamation	77·5 per cent
In table concentrate	16·5 "
Total	94·0 "

Tests Nos. 3 and 4

Two lots of 1,000 grammes of ore were ground to the following screen sizes and amalgamated by barrel amalgamation in a jar.

Screen Test on Amalgamation Tailing:

Mesh	Weight, per cent
+ 65	0·1
− 65+100	6·4
−100+150	10·4
−150+200	20·6
−200	62·5
Total	100·0

Results of Amalgamation:

Test No.	Feed assay, gold, oz/ton	Tailing assay, gold, oz/ton	Recovery, per cent
3	0·98	0·145	85·2
4	0·98	0·100	88·7

Table Concentration of Amalgamation Tailing:

Test No.	Product	Weight, per cent	Assay, gold, oz/ton	Distribution, per cent
3	Table concentrate	6·4	1·53 (cal.)	67·7
	Table tailing	93·6	0·05	32·3
	Total	100·0	0·145	100·0
4	Table concentrate	13·4	0·41	64·6
	Table tailing	86·6	0·04	35·4
	Total	100·0	0·10	100·0

NOTE.—The above assays of the concentrates are calculated, and no attempt was made to make a clean concentrate on the small laboratory table used.

Overall Recoveries:

Test No. 3:

Amalgamation	85·2 per cent
In table concentrate	8·5 "
Total	93·7 "

Test No. 4:

Amalgamation	88·7 per cent
In table concentrate	7·3 "
Total	96·0 "

LARGE-SCALE CONTINUOUS AMALGAMATION AND TABLE CONCENTRATION

Flow-Sheet. The ore was fed to a small rod mill that discharged to an amalgamation plate. The plate tailing was pumped over a belt screen, a 35-mesh screen being used. The oversize returned to the rod mill and the undersize was concentrated on a standard-deck Wilfley table. A concentrate and a tailing were made on the table.

Test No. 1

About 600 pounds of ore was used in this test and the rate of feed was 100 pounds per hour.

Results:

Product	Assays		Distribution, per cent	
	Gold, oz/ton	Arsenic, per cent	Gold	Arsenic
Feed to mill	0·98	0·98	100·0	100·0
Retained on plate	....	....	79·0	0·0
Amalgamation tailing	0·205	0·98	21·0	100·0
Feed to table	0·205	0·98	21·0	100·0
Table concentrate	4·66	27·15	13·2	71·4
Table tailing	0·078	0·28	7·8	28·6

Screen Test Table Feed:

Mesh	Weight, per cent
+ 48	1·3
− 48+ 65	4·0
− 65+100	16·8
−100+150	15·9
−150+200	20·6
−200	41·4
Total	100·0

Summary. Ratio of concentration on the table was 36·2 : 1. The table recovered 63 per cent of the gold remaining in the amalgamation plate tailing. The recovery by amalgamation was 79 per cent. The recovery in the table concentrate was 13·2 per cent. The total recovery by amalgamation and table concentration was 92·2 per cent. The grade of the table concentrate was 4·66 ounces gold per ton and 27·15 per cent arsenic.

Test No. 2

The feed rate in this test was increased to 150 pounds per hour, and about 450 pounds of ore was used.

Results:

Product	Assays		Distribution, per cent	
	Gold, oz/ton	Arsenic, per cent	Gold	Arsenic
Feed to mill	0·98	0·98	100·0	100·0
Retained on plate	....	....	77·5	0·0
Amalgamation tailing	0·22	0·98	22·5	100·0
Feed to table	0·22	0·98	22·5	100·0
Table concentrate	4·3	20·80	17·35	....
Table tailing	0·075	....	5·15	....

Screen Test Table Feed:

Mesh	Weight, per cent
+ 48	4·4
− 48+ 65	8·95
− 65+100	16·70
−100+150	14·95
−150+200	16·00
−200	39·00
Total	100·00

Summary. Ratio of concentration on the table, 29·2 : 1. The table recovered 67·1 per cent of the gold remaining in the amalgamation plate tailing. The recovery by amalgamation was 77·5 per cent. The recovery in the table concentrate was 17·35 per cent. The total recovery by amalgamation and table concentration was 94·85 per cent. The grade of the table concentrate was 4·3 ounces gold per ton and 20·80 per cent arsenic.

Tests for the Recovery of the Gold in Table Concentrate

CYANIDATION

Two cyanide tests were run on the table concentrate obtained from continuous Run No. 2.

In Test No. 1 the concentrate was cyanided without regrinding, and in Test No. 2 the concentrate was reground in water to 87 per cent through 200 mesh and filtered before cyanidation.

Results:

Feed sample of concentrate, gold, 3·67 oz./ton.

Test No.	Grinding	Tailing assay	Extraction, per cent	Hours agitated	Reagents consumed		
					KCN	CaO	PbO
1	None..................	0·30	91·8	48	21·41	8·5	....
2	87 per cent −200 mesh..	0·11	97·0	48	5·61	10·75	1·0

Remarks. These results show that the table concentrate responds readily to treatment by cyanidation, good recoveries being obtained with a low reagent consumption.

Summary. Recapitulating this result with the results of the continuous Run No. 2, it is evident that by regrinding the table concentrate 94·3 per cent of the gold in the ore can be recovered, and without regrinding 93·5 per cent can be recovered by cyaniding the table concentrate.

Gold recovered by plate amalgamation Run No. 2.................	77·5 per cent
Gold recovered in table concentrate Run No. 2....................	17·35 "
Gold recovered by cyaniding raw table concentrate................	91·8 "
Gold recovered by cyaniding reground table concentrate...........	97·0 "

Therefore total gold recoverable by plate amalgamation followed by cyanidation of the raw table concentrate is $77\cdot5 + (91\cdot8 \times 17\cdot35 = 16\cdot0) = 93\cdot5$ per cent, and for reground concentrate $77\cdot5 + (97\cdot0 \times 17\cdot35 = 16\cdot8) = 94\cdot3$ per cent.

BARREL AMALGAMATION

A barrel amalgamation test was run on the table concentrate obtained from continuous Run No. 1.

Results: The concentrate was reground for one hour in a ball mill and mercury then added. The mercury floured rather badly.

Product	Assays, gold, oz/ton	Recovery, per cent
Feed (table concentrate)..	4·66	76·6
Amalgamation tailing...	1·09	

Remarks. Recapitulating this result with the results of continuous Run No. 1, it is evident that 89·1 per cent of the gold in the ore can be recovered by plate amalgamation and barrel amalgamation of the table concentrate.

Gold recovered by plate amalgamation Run No. 1.................	79·0 per cent
Gold recovered in table concentrate Run No. 1....................	13·2 "
Gold recovered by barrel amalgamation of table concentrate.......	76·6 "

Therefore total gold recoverable by plate amalgamation and barrel amalgamation of table concentrate is 79·0+(76·6 per cent of 13·2 per cent =10·1 per cent)=89·1 per cent.

FLOTATION TEST ON AMALGAMATION TAILING

A large sample was cut from the amalgamation tailing during continuous Run No. 2. Part of this sample was used for a flotation test.

A sample, 2,000 grammes, of this tailing was reground to approximately 65 per cent through 200 mesh in a ball mill in a 1 : 1 pulp.

The following reagents were used:

Added to ball mill: Soda ash	3·0	lb./ton
" " Xanthate	0·3	"
Added to flotation cell: Pine oil	0·05	"

Results:

Products	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Flotation concentrate	2·4	3·84	73·0
Flotation tailing	97·6	0·035	27·0

CYANIDE TESTS ON AMALGAMATION TAILING

Two cyanide tests were made on part of the sample cut from the amalgamation tailing during continuous Run No. 2. Test No. 1 was run on straight tailing and Test No. 2 on reground tailing. The strength of the cyanide solution used was 0·50 pound KCN per ton and the time of treatment 24 hours.

Results:

Test No.	Grinding	Tailing assay	Extraction, per cent	Agitation, hours	Reagent consumption	
					KCN	CaO
1	None	0·02	90·9	24	0·15	9·0
2	73·4 per cent —200 mesh	0·015	93·2	24	0·42	9·5

Remarks. It is evident from these results that the ore after amalgamation can be readily cyanided and high extraction of the gold obtained with a very low reagent consumption.

Recapitulating these results with the plate amalgamation results obtained in continuous Run No. 2:—

Gold recovered by plate amalgamation Run No. 2	77·5	per cent
Gold recovered by cyanidation plate tailing	90·9	"
Gold recovered by cyanidation reground tailing	93·2	"

Therefore total gold recoverable by amalgamation and by cyanidation of the amalgamation tailing is:—

$$77\cdot5+(90\cdot9\times22\cdot5=20\cdot4)=97\cdot9 \text{ per cent.}$$
$$77\cdot5+(93\cdot2\times22\cdot5=21\cdot0)=98\cdot5 \text{ per cent.}$$

STRAIGHT CYANIDATION

Four cyanide tests were made on the ore. The strength of the solution used was 0·5 pound per ton, and the solution was maintained at this strength by frequent additions during the tests.

Results:

Feed sample: gold, 0·98 oz. /ton.

Test No.	Grinding mesh	Agitation, hours	Assay of tailing	Extraction, per cent	Reagent consumption, lb./ton	
					KCN	CaO
1	– 48	24	0·07	92·8	1·06	11·88
2	– 48	48	0·125	87·2	....	....
3	–100	24	0·02	97·9	1·18	....
4	–100	48	0·025	97·4	1·02	15·30

Remarks. These tests show that the ore is amenable to straight cyanidation, but owing to the presence of such a high proportion of coarse free gold it would be advisable to use traps and blankets in the cyanide circuit to remove the coarse gold, otherwise losses would undoubtedly occur owing to metallic gold passing through the mill before it had time to be dissolved.

CONCLUSIONS AND RECOMMENDATIONS

Metallurgically, the ore is extremely simple to treat and responds equally well to a number of different flow-sheets.

Considering the treatment problem wholly from a metallurgical point of view, the best practice would be a flow-sheet in which the ore was ground in water, passed over plates and blankets, and the tailing cyanided.

However, amalgamation and table concentration of the plate tailing gives a high recovery (94 to 95 per cent), 77·5 per cent being recovered by amalgamation and an additional 17 per cent in a table concentrate containing 20 per cent arsenic. The milling machinery already on the property can best be adapted to this latter practice.

The question of treatment or recovery of the gold in the table concentrate should not be overlooked. It can be shipped to a smelter, but experience has shown that this should be done only as a last resource.

The test work has shown that a fair overall recovery can be obtained by barrel amalgamation of the table concentrate, and that an excellent overall recovery can be obtained by cyaniding this product.

The type of stamp battery in the mill at the property is not the best for amalgamation purposes, being a heavy-duty, 1,900-pound-per-stamp, Rand battery. The test work has shown that the ore should be crushed as fine as is practical, but it is doubtful whether it will be practical to use a 40-mesh screen on this type of battery.

In conclusion, attention is directed to the possibility of sorting the ore before milling. Judging from the sample received, it should be possible to hand-sort a considerable amount of barren slate from the ore on a picking-belt before crushing.

Ore Dressing and Metallurgical Investigation No. 565

LEAD-ZINC ORE FROM THE STIRLING MINE, STIRLING, N.S.

Shipment. A shipment of two carloads of ore was received November 6, 1933, from the British Metals Corporation (Canada), Limited. The shipment contained 80 tons of ore taken from the dumps at the Stirling mine, Stirling, Nova Scotia.

Character of Ore. Weeks[1] has described the gangue as consisting mostly of quartz and feldspar with small amounts of other minerals common to volcanic rocks. Although little work was carried out in this laboratory on the identification of the gangue minerals, the sections examined contain predominatingly light-coloured silicates (probably feldspar) with some white quartz. A small amount of white carbonate is present as narrow veinlets and disseminated grains.

The metallic minerals in the polished sections are, in their order of abundance, pyrite, sphalerite, galena and chalcopyrite, and tetrahedrite. Both gangue and ore minerals are very fine-grained and are everywhere intimately admixed. Pyrite commonly forms small irregular grains and imperfect cubes disseminated in the silicate gangue. It is much shattered, particularly in the larger sizes, and contains veinlets with sphalerite, chalcopyrite, galena, tetrahedrite, and carbonate.

Sphalerite occurs as small irregular grains and fine stringers, and is intimately associated with the other sulphides; in rare cases it is veined and somewhat replaced by chalcopyrite, galena, and tetrahedrite.

Chalcopyrite also occurs as small irregular grains and fine stringers, and is intimately associated with the other sulphides. In places it veins and corrodes sphalerite.

Galena forms irregular patches and small grains intimately associated with the other sulphides. It veins pyrite, sphalerite, and chalcopyrite.

Tetrahedrite is common and occurs as irregular patches and grains in galena and chalcopyrite, and sometimes in sphalerite. In some places this mineral veins sphalerite, chalcopyrite and galena, and is rarely present in the late carbonate veinlets.

The relative proportions of the metallic and gangue minerals differ considerably over short distances, imparting to the ore in some sections a distinct banded or streaky appearance. The gangue and the several metallic minerals are intimately mixed and the degree of combination of the sulphides and their grain sizes are brought out by the tables given later.

Paragenesis. The relationship of the minerals as shown in the polished sections allows some interpretation of their order of deposition. This interpretation is best represented graphically, and is shown in Table I

[1] Weeks, L. J.: Geol. Surv., Canada, Sum. Rept., 1924, pt. C; p. 212c.

TABLE I

Paragenesis of Ore at the Stirling Mine

Quartz	––––– – ? –
Pyrite	–––
Sphalerite	–––––
Chalcopyrite	–––––
Galena	–––––
Tetrahedrite	––––––––
Carbonate	– ? – – –––––––
"Limonite" (Supergene)	––––

Spectral Analyses of the Metallic Minerals. Analyses by means of the quartz spectrograph were carried out on small samples of the metallic minerals, obtained by drilling from the mineral grains in the polished sections under the microscope. The results of these analyses for tetrahedrite, galena, sphalerite, and chalcopyrite are shown in Table II. The strength of the spectra of the contained elements was estimated, and this is a rough indication of the relative amounts of these elements in the samples.

TABLE II

Spectral Analyses of the Metallic Minerals

Elements	Tetrahedrite (dissolved in HNO_3 and converted to chlorides after placing on the electrode)	Galena (Powder)	Sphalerite (Powder)	Chalcopyrite (Powder)
Copper	Essential, Strong	Nil	Trace	Essential, Strong
Iron	Trace	Trace	Strong Trace	Essential
Lead	Nil	Essential, Strong	Trace	Nil
Zinc	Nil	Nil	Essential, Strong	Nil
Tin	Nil	Trace	Faint Trace	Nil
Gold	Nil	Nil	Nil	Nil
Silver	Strong Trace	Strong Trace	Faint Trace	Nil
Arsenic	Essential, Strong	Nil	Nil	Nil
Antimony	Essential, Moderate	Nil	Nil	Nil
Bismuth	Nil	Strong Trace	Nil	Nil
Cadmium	Nil	Strong Trace	Moderate Trace	Nil

It will be seen from the above analysis of tetrahedrite, that arsenic is estimated as stronger than antimony; this would indicate that the mineral is not true tetrahedrite, but an intermediate member of the tetrahedrite-tennantite series.

Mode of Occurrence of the Precious Metals. The spectrographic analyses show that tetrahedrite and galena contain appreciable traces of silver, but chalcopyrite contains none; the silver reported in the sphalerite is undoubtedly due to the inclusion of a small amount of galena, and possibly tetrahedrite, in the sample analysed. The presence of gold in any of the metallic minerals was not detected, but exceedingly small traces might be present and the concentration still be below the limits at which the methods employed might be expected to detect this metal.

Grain Size of the Minerals in the Ore. Quantitative microscopic analyses of the minerals in eleven sections of the ore were carried out by making a large number of traverses across the surfaces. The final calculations of these analyses are shown in Tables III to VI (pages 82 and 83). For the purpose of providing a rough check on these results, the percentages of the metals and sulphur in the ore were calculated, using the results shown in Table IV as the basis; these percentages are shown in Table VII.

TABLE III

Complete Grain Analysis (By Volume) of Ore from the Stirling Mine, Nova Scotia

Mesh	Sphalerite		Pyrite		Chalcopyrite		Galena		Tetrahedrite		Gangue	Totals
	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined		
+ 20	...	...	...	...	...	...	...	...	...	...	18·48	18·48
− 20+ 28	...	...	...	0·25	...	...	...	...	...	...	4·44	4·69
− 28+ 35	...	0·23	...	0·41	...	...	...	...	...	...	5·40	6·04
− 35+ 48	...	0·39	...	0·55	...	0·30	...	...	...	...	4·25	5·49
− 48+ 65	...	0·53	0·08	0·87	...	0·36	...	...	...	0·11	4·10	6·06
− 65+100	0·07	0·92	0·13	1·33	...	0·65	...	0·18	...	0·07	5·87	9·22
−100+150	0·12	1·47	0·23	1·35	0·05	0·18	...	0·22	...	0·12	4·50	8·24
−150+200	0·37	2·06	0·47	2·00	0·17	0·23	0·03	0·30	...	0·16	3·70	9·49
−200+325	0·69	2·70	0·84	2·88	0·25	0·56	0·08	0·48	0·05	0·13	4·35	13·01
−325	1·30	4·33	1·31	3·83	0·90	0·80	0·28	0·92	0·05	0·24	5·33	19·29
Totals	2·55	12·63	3·06	13·47	1·37	3·08	0·39	2·10	0·10	0·83	60·42	100·00%
	15·18		16·53		4·45		2·49		0·93		60·42	100·00%

TABLE IV

Complete Grain Analysis (By Weight) of Ore from the Stirling Mine, Nova Scotia

Specific gravities assumed: Sp.—4·0; Py—5·0; Cp—4·2; Ga—7·5; Tet—4·7; Gangue—2·6

Mesh	Sphalerite		Pyrite		Chalcopyrite		Galena		Tetrahedrite		Gangue	Totals
	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined		
+ 20	...	...	...	...	...	...	...	...	...	...	14·08	14·08
− 20+ 28	...	...	...	0·24	...	...	...	...	...	...	3·38	3·62
− 28+ 35	...	0·27	...	0·60	...	...	...	...	...	...	4·11	4·98
− 35+ 48	...	0·46	...	0·81	...	0·37	...	...	...	...	3·24	4·88
− 48+ 65	...	0·62	0·12	1·28	...	0·44	...	...	...	0·15	3·12	5·73
− 65+100	0·09	1·08	0·19	1·94	...	0·80	...	0·40	...	0·09	4·47	9·06
−100+150	0·15	1·73	0·34	1·97	0·06	0·22	...	0·49	...	0·17	3·43	8·56
−150+200	0·41	2·41	0·69	2·94	0·21	0·28	0·07	0·66	...	0·22	2·82	10·71
−200+325	0·81	3·16	1·23	4·22	0·31	0·69	0·18	1·06	0·06	0·18	3·32	15·22
−325	1·39	5·07	1·92	5·60	1·10	0·99	0·61	2·01	0·08	0·33	4·06	23·16
Totals	2·85	14·80	4·49	19·60	1·68	3·79	0·86	4·62	0·14	1·14	46·03	100·00
	17·65		24·09		5·47		5·48		1·28		46·03	100·00

TABLE V

Complete Grain Analysis of Sulphides Alone (By Volume) in Ore from the Stirling Mine, Nova Scotia

Mesh	Sphalerite		Pyrite		Chalcopyrite		Galena		Tetrahedrite		Totals
	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	
+ 20											
− 20+ 28				0·63							0·63
− 28+ 35		0·58		1·04							1·62
− 35+ 48		0·98		1·39		0·77					3·14
− 48+ 65		1·34	0·21	2·21		0·90				0·27	4·93
− 65+100	0·19	2·34	0·32	3·35		1·64		0·46		0·16	8·46
−100+150	0·31	3·72	0·68	3·40	0·13	0·46		0·57		0·31	9·48
−150+200	0·92	5·20	1·19	5·06	0·43	0·68	0·08	0·75		0·41	14·52
−200+325	1·74	6·81	2·13	7·28	0·63	1·42	0·21	1·21	0·12	0·33	21·88
−325	3·28	10·93	3·31	9·66	2·27	2·02	0·70	2·31	0·15	0·61	35·24
Totals	6·44	31·90	7·74	34·02	3·46	7·79	0·99	5·30	0·27	2·09	100·00
	38·34		41·76		11·25		6·39		2·36		100·00

TABLE VI

Complete Grain Analysis of Sulphides Alone (By Weight) in Ore from the Stirling Mine, Nova Scotia

Specific gravities assumed: Sp—4·0; Py—5·0; Cp—4·2; Ga—7·5; Tet—4·7

Mesh	Sphalerite		Pyrite		Chalcopyrite		Galena		Tetrahedrite		Totals
	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	
+ 20											
− 20+ 28				0·43							0·43
− 28+ 35		0·49		1·12							1·61
− 35+ 48		0·84		1·49		0·69					3·02
− 48+ 65		1·15	0·22	2·37		0·81				0·27	4·82
− 65+100	0·16	2·01	0·35	3·60		1·48		0·74		0·16	8·50
−100+150	0·27	3·20	0·62	3·65	0·12	0·42		0·91		0·31	9·50
−150+200	0·80	4·47	1·28	5·44	0·39	0·52	0·13	1·21		0·41	14·65
−200+325	1·49	5·85	2·29	7·82	0·57	1·28	0·33	1·96	0·12	0·34	22·05
−325	2·57	9·40	3·56	10·38	2·04	1·83	1·14	3·73	0·15	0·62	35·42
Totals	5·29	27·14	8·32	36·30	3·12	7·03	1·60	8·55	0·27	2·11	100·00
	32·70		44·62		10·15		10·15		2·38		100·00

A comparison of the results given in Table VII with chemical analyses of the feed samples reveals that the calculated percentages are too high. This indicates merely that the microscopic investigations were carried out with material of higher metallic mineral content than the ore as a whole.

TABLE VII

Distribution of the Metals and Sulphur in the Minerals in Ore from the Stirling Mine, Nova Scotia

(Calculated from Quantitative Grain Analysis shown in Table IV)

Minerals	Iron	Sulphur	Zinc	Copper	Lead	Antimony and arsenic
Pyrite	11·28	12·81	....	....	....	....
Sphalerite	....	5·82	11·83	....	....	....
Chalcopyrite	1·65	1·92	....	1·90	....	....
Galena	....	0·74	....	....	4·74	....
Tetrahedrite	....	0·29	....	0·67	....	0·32

Analysis. No feed sample was cut representing the entire shipment. An approximate analysis indicated from the daily sample of the classifier overflow during the experimental tests would be:—

Copper	0·9 per cent
Lead	2·0 "
Zinc	8·0 "
Gold	0·035 oz/ton
Silver	2·5 "

Purpose of Experimental Tests. The British Metal Corporation (Canada), Limited, through its mining engineer, R. D. Hearn, requested experimental work on the ore in order to improve if possible the flow-sheet and methods previously in use in the mill at Stirling. The existing flow-sheet is briefly as follows: Grinding is carried to 98 per cent through a 200-mesh screen. A talc product is then removed by flotation and the flotation of the lead, copper, and zinc minerals follows progressively to make a lead, a copper, and a zinc concentrate for shipment to the smelter.

Mr. Hearn was present during the experimental tests and assisted in the test work and in interpreting the results.

Three main objectives guided the experimental tests:—

(1) To determine whether a talc product could be successfully removed at a coarser size. The removal of a coarse talc product at some stage in the grinding circuit would reduce the tonnage to be reground and therefore increase the capacity of the ball mills.

(2) To determine if a bulk concentrate of the lead and copper of marketable grade could be produced and through increased recoveries of these metals greater profits be obtained.

(3) To experiment with a lead-copper separation from this bulk concentrate product, in expectation of obtaining both a higher grade product for shipment and better recoveries than obtained by the progressive method of separation.

Experimental Tests

A previous microscopic study of the ore had shown that the talc was practically free from association with either the copper, lead, or zinc minerals and therefore there was no apparent reason why it should not be separated from the ore at a relatively coarse size. The sulphides themselves, however, were found to be intimately associated with each other, which suggests that extremely fine grinding would be required to free them. A flotation unit for the removal of the talc could, therefore, be placed at any point in the grinding circuit where it was found convenient, but the ore would require very fine grinding.

A series of separate individual daily runs was made, each of about 8 hours' duration. Some characteristics of the ore observed during these runs are mentioned here so that references in the test results will be better understood. The galena was found to be much slower in floating than that found in an average ore, and within reasonable limits the flotability of the galena was greatly increased by any increase in the quantity of cyanide that was added for the purpose of depressing the pyrite. The chalcopyrite was found much more sensitive to cyanide than an average chalcopyrite. Both conditions may be at least partly due to the fact that the ore treated had been mined over three years and was exposed to the weather during that time.

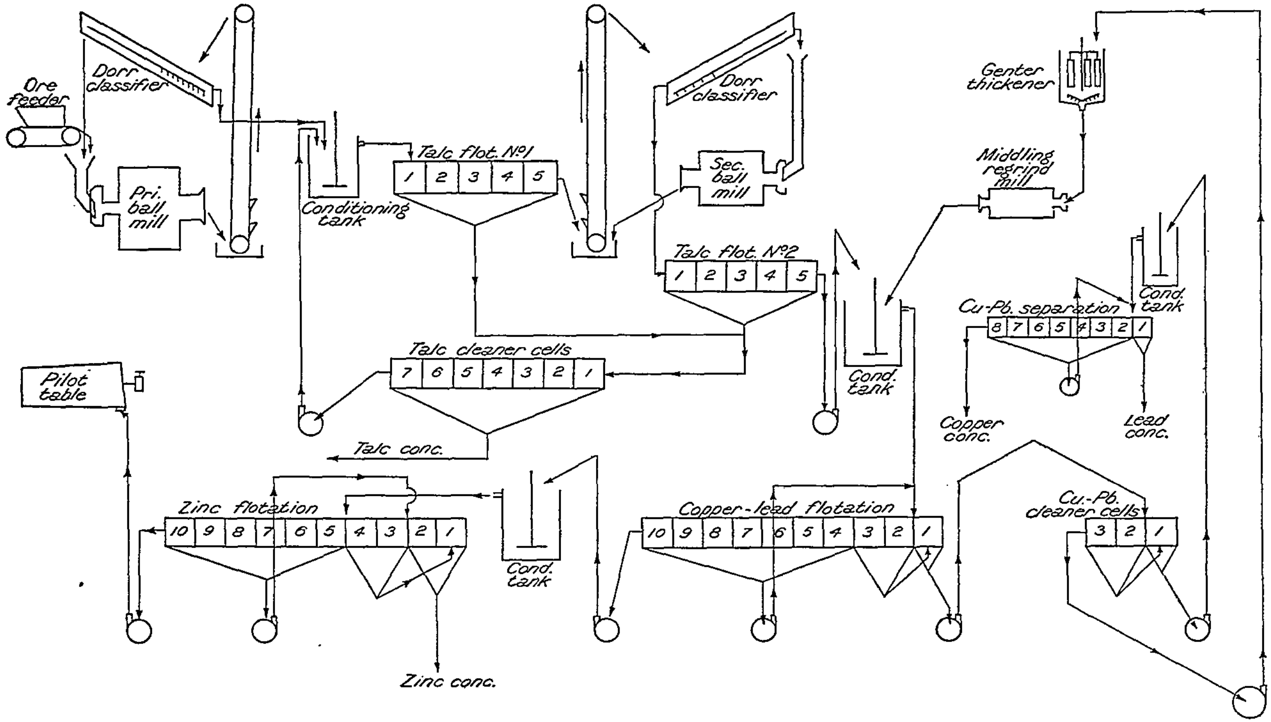
These two conditions made it necessary to keep the amount of cyanide within certain limits, and did not allow much latitude in controlling the depression of pyrite, for which purpose it is necessary to use the cyanide.

Twenty-four individual flotation tests or runs were made.

A few tests only are included in this report, selected to bring out characteristics that will be referred to.

Runs Nos. 1 to 5

The flow-sheet used for these runs was as follows: The ore at $\frac{3}{4}$ inch was fed to a 4-foot by 3-foot grate-discharge ball mill carrying a ball load of approximately 2,000 pounds of 3-inch balls. This ball mill was in closed circuit with a Dorr classifier. The classifier overflow went to a 5-cell mechanical type flotation unit where a talc float was taken off. The tailing from this flotation unit was pumped to a second Dorr classifier operating in closed circuit with a 30-inch by 3-foot ball mill. The overflow of the classifier then went to a second 5-cell flotation unit similar to the first where a second talc float was made. The talc concentrate was sampled and pumped to waste, first being passed over a Wilfley table, which was used as a pilot or guide to the operation of the talc cells. The tailing from the final talc float was sampled and pumped to a conditioning tank that fed the copper-lead flotation circuit. A bulk concentrate containing the copper and lead was floated in a 10-cell unit similar to the one used on the talc flotation. The reagents for the copper-lead float were introduced into the conditioning tank and the discharge of the conditioning tank was fed to Cell No. 2. Cells Nos. 2 and 3 made a concentrate that was recleaned in Cell No. 1. The froth from Cells Nos. 4 to 10 (inclusive) was returned as rougher concentrate with the feed into Cell No. 2. The tailing from the copper-lead circuit was sampled and pumped to a conditioning tank, where lime and copper sulphate were added, together with the other reagents



Flow-sheet used for continuous runs on Stirling ore.

required for the flotation of the zinc. The discharge of the zinc conditioning tank was fed to Cell No. 4 of a 10-cell unit similar to the other units. A rougher concentrate was taken off Cells Nos. 4 to 10 and returned for recleaning in Cells Nos. 2 and 3. The froth from Cells Nos. 2 and 3 was recleaned in Cell No. 1. The tailing from this circuit was pumped to waste.

Run No. 2

Feed rate: Approximately 950 pounds per hour.

Reagents added to primary ball mill:

Soda ash	1·0 lb/ton
Zinc sulphate	1·0 "
Cyanide	0·16 "
pH	8·6

Density:

Ball mill	69 per cent solids
Classifier overflow	44 " "

Reagents added to secondary ball mill:

Soda ash	0·25 lb/ton
Cyanide	0·04 "

Density:

Classifier overflow	25 per cent solids

Reagents to talc flotations:

Talc Float No. 1—
Cresylic acid........ 0·07 lb/ton

Talc Float No. 2—
Cresylic acid........ 0·03 lb/ton

Reagents to copper-lead float:

Potassium amyl xanthate	0·024 lb/ton
Aerofloat No. 25	0·03 "

Reagents to zinc float:

Lime	2·0 lb./ton
Copper sulphate	0·8 "
Aerofloat	0·06 "
Potassium ethyl xanthate	0·036 "

Screen Tests:

Primary Classifier Overflow		Secondary Classifier Overflow	
Mesh	Weight, per cent	Mesh	Weight, per cent
+ 48	1·4	+ 48	....
− 48+ 65	6·7	− 48+ 65	0·5
− 65+100	11·7	− 65+100	4·3
−100+150	7·8	−100+150	5·8
−150+200	12·1	−150+200	13·1
−200	60·3	−200	76·3
Total	100·0	Total	100·0

Assays of Products:

Product	Copper, per cent	Lead, per cent	Zinc, per cent	Gold, oz/ton	Silver, oz/ton
Feed sample	0·34	2·05	7·92	0·04	2·46
Talc concentrate	0·49	1·37	4·39	0·04	3·38
Feed to copper-lead flotation	0·89	2·20	8·02	0·035	2·46
Copper-lead concentrate	4·50	19·60	11·90	0·25	14·46
Copper-lead tailing and zinc feed	0·52	0·28	7·85	....	....
Zinc concentrate	1·70	0·73	56·61	....	....
Zinc tailing (final)	0·35	0·16	0·69	0·01	0·88

Remarks. From the behaviour of the chalcopyrite in this test and also in Test No. 1, it is evident that the cyanide is attacking and depressing this copper mineral. The total quantity of cyanide used was equivalent to 0·20 pound per ton of ore. The zinc flotation gave no trouble.

Run No. 3

The results of this run will not be given and reference is only made to it in order to show that the zinc flotation was successful. A concentrate was obtained running 54 per cent zinc with a tailing of 0·8 per cent. The loss of zinc in the final tailing was 7·1 per cent.

Run No. 4

Feed rate: Approximately 950 pounds per hour.

Reagents added to primary ball mill:

Soda ash, 9.00 a.m. to 3.30 p.m.	1·0 lb/ton
Soda ash, 3.30 p.m. to 4.15 p.m.	2·0 "
Cyanide	0·03 "
Zinc sulphate	1·0 "

Reagents added to secondary ball mill:

Soda ash	0·25 lb/ton
Cyanide	0·008 "
Zinc sulphate	0·4 "

Reagents to talc flotation:

Talc Float No. 1—
Cresylic acid........ 0·07 lb/ton

Talc Float No. 2—
Cresylic acid........ 0·03 lb/ton

Reagents to copper-lead float:

Butyl xanthate, Z-8	0·05 lb/ton
Barrett No. 4 oil	0·03 "

Reagents to zinc float:

Lime	2·0 lb/ton
Potassium ethyl xanthate	0·10 "
Copper sulphate	0·8 "
Pine oil	0·03 "

The grinding for the first talc flotation averaged 66·5 per cent through 200 mesh, and, for the final flotation, 79·7 per cent.

Assays of Products:

Product	Copper, per cent	Lead, per cent	Zinc, per cent	Gold, oz/ton	Silver, oz/ton
Feed sample	0·90	2·10	8·17	0·04	2·52
Talc product	0·58	1·46	5·37	....	....
Copper-lead concentrate	9·34	19·10	9·86	0·30	22·12
Copper-lead tailing } Zinc feed }	0·20	0·41	9·19	....	....
Zinc concentrate	0·46	1·44	57·74	....	....
Final tailing	0·25	0·17	1·20	....	....

Remarks. The test is given to show the effect of a change of reagents on the copper-lead float. Amyl xanthate, used in the previous tests, was replaced by secondary butyl (Z-8) with the result that there was a marked reduction in the amount of zinc in the copper-lead concentrate. The quantity of cyanide was reduced from 0·20 pound per ton used in Run No. 2 to 0·04 pound per ton in this run, with the result that the recovery of copper was improved.

Runs Nos. 6 to 9

(Recleaning Talc Product and Copper-Lead Concentrate)

The total number of cells available for these tests was thirty. These were all required when the zinc flotation was made. The first series of tests showed conclusively that there were no problems in connection with the flotation of the zinc and that both a high-grade concentrate and low tailing could be obtained. For this reason it was decided to leave out the zinc flotation and to use three of the ten cells made available, for recleaning the copper-lead concentrate, and the remainder (seven cells) for recleaning the talc product.

The flow-sheet for these runs was, therefore, similar to the previous ones, except that the copper-lead concentrate and talc product were re-cleaned.

The grade of the concentrate was raised without affecting the loss in the tailing, as can be seen from the assays of the products obtained in Run No. 9 given in the following table. A microscopic study was made of the middling returning from the cleaner cells to the feed, the result of which is given later in the report, where this unground middling is compared with a reground middling from Run No. 24.

Assays of Products:

Product	Copper, per cent	Lead, per cent	Zinc, per cent	Insol., per cent	Gold, oz/ton	Silver, oz/ton
Feed	0·90	1·93	7·34	....	0·035	2·45
Talc product	0·43	1·02	3·62	....	....	....
Copper-lead concentrate	10·84	22·80	8·67	2·40	0·37	29·10
Copper-lead middling	3·48	8·96	13·28	....	....	....
Copper-lead tailing	0·13	0·28	8·42	....	....	....

The grinding in these tests was rather coarse, namely 65 per cent through 200 mesh for the primary talc float and only 74 per cent through 200 mesh for the final grinding.

Runs Nos. 10 to 13

(Regrinding Concentrate)

In this series of runs the copper-lead concentrate from the first two cells of the 10-cell roughing unit was pumped to a Genter thickener and thickened for regrinding. The reground concentrate was then re-floated in a 5-cell unit, the concentrate from two cells being recleaned in the single cell, the greater part of the reagents ordinarily collecting in the concentrate was apparently washed away in the Genter solution during thickening and it was therefore necessary to add additional reagents to this circuit.

The addition of promoting reagents, such as xanthate, to this reground concentrate required very delicate control, and the small amount of material being handled increased the difficulty of arriving at the correct amount.

The best results obtained during the series of runs was with Run No. 13.

Run No. 13:

Feed rate: Approximately 950 pounds per hour.

Reagents added to primary ball mill:

Soda ash	1·0 lb/ton
Zinc sulphate	0·8 "
Cyanide	0·02 "

Reagents added to secondary ball mill:

Zinc sulphate	0·5 lb/ton

Reagents added for talc flotation:

Talc Product No. 1		Talc Product No. 2	
Cresylic acid	0·1 lb/ton	Cresylic acid	0·07 lb/ton

Reagents added for copper-lead float:

Butyl xanthate to conditioning tank	0·05 lb/ton
" " Cell No. 4	0·008 "
Cresylic acid to Cell No. 5	0·028 "

Reagents added for reground copper-lead concentrate circuit:

Zinc sulphate to regrind mill	0·16 lb/ton ore
Butyl xanthate to cells—average	0·04 "
Barrett No. 4 oil to mill	0·03 "

The regrinding for the first and second talc floats was respectively 66 per cent and 77 per cent through 200 mesh.

Assays of Products:

Product	Copper, per cent	Lead, per cent	Zinc, per cent	Insol., per cent	Gold, oz/ton	Silver, oz/ton
Feed	0·78	1·83	7·42	....	0·035	2·51
Talc product	0·29	1·10	3·62	....	....	....
Copper-lead concentrate (rougher)	7·80	16·40	12·20	5·02	0·21	18·56
Copper-lead reground and recleaned concentrate	11·44	24·22	7·23	3·23	0·33	30·08
Tailing	0·18	0·30	7·64	....	....	....

Runs Nos. 14 to 24

(Recleaning Middling)

In this series of runs the middling from the second and third recleaning of the copper-lead concentrate was reground, after first being thickened in a Genter thickener. The reground middling was returned to the head of the copper-lead flotation circuit.

The object was to improve the grade of the copper-lead concentrate and to increase the assay of both copper and lead in this product.

A few special reagents were used during some of these tests, but their use showed no marked improvement over the reagents that were adopted in this test work as standard for the ore, namely, soda ash, cyanide, zinc sulphate, butyl xanthate (Z-8), and cresylic acid. The control of the quantity and the point of addition of these reagents have been found important, and in an endeavour to obtain the best results the conditions of running were changed for each test.

Recoveries and Distribution of Metals. In the brief summary of a number of tests given, recoveries are not referred to because the calculated results of the recovery for individual runs were not satisfactory. Very small differences in the value of assays made tremendous differences in recoveries. The following tables have been worked out to show the recovery and distribution of the metals in actual tests. The tonnages, on a 100-ton basis, have been computed by using copper-zinc, lead-zinc, and copper-lead assay combinations, and also using each assay of copper, lead, and zinc separately. The results are shown in Table VIII. The figures in heavy type are abnormal, and a black X indicates an impossible calculation. In Table IX are shown the recoveries obtained using the average tonnage figures taken from Table VIII. In working these figures out it was found that the copper-lead combinations did not give very good results and that the copper-zinc and lead-zinc combinations were more useful. In Table IX the per cent recoveries are on the actual feed. The copper and zinc, as a rule, worked out well but the lead was always irregular.

Table X has been compiled to represent what could be expected from a prolonged run under the average conditions maintained during the entire test work on Runs Nos. 9, 12, 13, 14, 15, 16, 17, 18, 22, 23, 24—eleven tests in all. The following method of averaging was adopted after it became apparent than no one particular test could be picked out as illustrative of the best practice to be obtained, and this situation was probably caused by errors that could not be controlled—in the sampling or through very small errors of assay.

Feed Assays:

Averages—

Copper. Discarded all assays below 0·8 and over 1·00. Leaves 9 out of 11 assays for average. One high and one low discarded.

Lead. Discarded all below 1·75. Leaves 9 out of 11 assays for average. There are no abnormal highs recorded.

Zinc. No assays discarded. No abnormal highs or lows. Average of 11 taken.

Talc Concentrate Assays:

Averages—

Copper. One high out of 11 assays discarded.

Lead. One low and three highs discarded. Seven retained for average.

Zinc. Two highs discarded. No abnormal lows. Nine retained for average.

Lead-Copper Concentrate Assays:

Not averaged, as values entirely too diverse. Lead-copper concentrate regarded as the unknown and calculated afterwards from the average assays of other products and the tonnages.

Zinc Feed Assays:

Averages—

Copper. One low and two highs discarded out of 11.

Lead. Three highs discarded. (Three discarded were all above 0·4 per cent, i.e., 0·43, 0·45, 0·45).

Zinc. One high discarded. No abnormal lows. Ten retained.

Talc Concentrate Tonnage:

All results lower than 4·5 tons and higher than 25 tons discarded. Copper-zinc and lead-zinc assay calculations used, but not the copper-lead calculations, which are obviously unreliable. This meant discarding 2 lows and 6 highs out of 22 results, giving average of 11·1 tons.

Zinc Feed Tonnage:

Discarded all results below 80 and over 90. Out of 26 results, 11 lows and 2 highs were discarded. Out of the 11 lows discarded, 10 were below 70 and although the number discarded seems large, it is also fairly obvious that 70 is in itself a low figure.

This reconstruction of the aforesaid averages worked out surprisingly well, especially when it is considered that conditions of running were changed for every test, which would not apply to a practical operation. This Table X shows, therefore, just about what could be expected from a prolonged run on the ore under the better conditions of the test runs. It is possible that the copper-lead concentrate is shown a little too high in grade; but better values were obtained during individual runs, and in a prolonged run, with time for the intermediate circuit to find a level grade, the whole operation would likely level out and steady the concentrates at a slightly higher grade.

TABLE VIII

Tons Product from 100 Tons Feed

Test No.	Product	$F = C + M + T$			$T = F\left(\frac{c-f}{c-t}\right)$			Tonnage taken for calculation	Remarks
		Copper-zinc assay	Lead-zinc assay	Copper-lead assay	Copper assay	Lead assay	Zinc assay		
9	Talc concentrate	18·7	22·8	28·3	2·1	X	5·1	20·0	Obvious low assay on zinc feed
	Lead-copper concentrate	6·7	6·6	6·4				6·5	
	Zinc feed	74·6	70·6	65·3				73·5	
12	Talc concentrate	**0·9**	**0·9**	**83·5**	16·1	X	X	16·1	Copper assays only used as basis; copper recovery works out well, but lead and zinc very bad
	Lead-copper concentrate	4·5	6·9	4·6				4·6	
	Zinc feed	94·6	94·1	**11·9**				79·3	
13	Talc concentrate	4·9	4·8	**45·1**	22·2	12·0	16·1	5·0	
	Lead-copper concentrate	5·3	6·2	4·9				5·5	
	Zinc feed	89·8	89·0	**50·0**				89·5	
14	Talc concentrate	13·1	14·1	X	24·6	28·3	15·5	10·0	Disproportion of copper and lead in copper-lead concentrate probably throws calculation out
	Lead-copper concentrate	6·7	3·8	12·51				4·0	
	Zinc feed	80·2	82·1	X				86·0	
15	Talc concentrate	7·9	7·5	**30·8**	23·1	X	12·6	9·0	
	Lead-copper concentrate	3·8	4·5	3·8				4·0	
	Zinc feed	68·3	88·0	**65·4**				87·0	
16	Talc concentrate	29·6	29·3	16·3	18·9	X	30·1	29·0	Copper and zinc recoveries check on this tonnage, but lead is off 30 per cent
	Lead-copper concentrate	4·7	3·3	5·1				4·9	
	Zinc feed	65·7	67·4	X				66·1	

TABLE VIII—(Continued)

Tons Product from 100 Tons Feed——(Continued)

Test No.	Product	$F = C + M + T$			$T = F\left(\frac{c - f}{c - t}\right)$			Tonnage taken for calculation	Remarks
		Copper-zinc assay	Lead-zinc assay	Copper-lead assay	Copper assay	Lead assay	Zinc assay		
17	Talc concentrate	14·7	14·4	**61·6**	23·9	11·6	10·0	14·5	Lead assay in talc concentrate not possible. Changed in calculations to 1·4 per cent.
	Lead-copper concentrate	4·0	4·9	3·2				4·5	
	Zinc feed	81·3	80·7	**35·2**				81·0	
18	Talc concentrate	12·8	12·8	11·5	19·7	15·2	23·0	12·8	All metals check to 100 per cent at this tonnage on this test.
	Lead-copper concentrate	4·4	4·1	4·4				4·4	
	Zinc feed	82·8	83·1	84·1				82·8	
22	Talc concentrate	31·3	32·0	**X**	37·8	37·5	34·9	31·0	This test does not check well. Zinc recovery close to 100 per cent. Lead and copper badly off.
	Lead-copper concentrate	5·0	3·4	6·4				4·5	
	Zinc feed	63·7	64·6	**X**				64·5	
23	Talc concentrate	9·1	8·1	**57·1**	**X**	**X**	11·6	9·0	Zinc feed assay for zinc obviously impossible. Changed to 8·4 per cent.
	Lead-copper concentrate	5·3	6·8	4·3				5·3	
	Zinc feed	85·6	85·1	**38·6**				85·7	
24	Talc concentrate	26·1	26·1	226·1	7·0	20·0	34·2	26·1	Calculates very well with F = C + M + T. However tonnage of talc concentrate is probably very high.
	Lead-copper concentrate	5·2	5·2	5·2				5·2	
	Zinc feed	68·7	68·7	68·7				68·7	

NOTE. Figures in heavy type are abnormal, and black X indicates impossible calculation.

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TABLE IX

Assuming that, as shown by earlier tests, on the average grade of the tests, 56 per cent zinc concentrate and 0·6 per cent zinc tailing are obtainable:—

Recoveries of Metals in Products on Tonnages taken for Calculation from Table VIII

Test No.	Product	Copper assay	Distribution, per cent	Lead assay	Distribution, per cent	Zinc assay	Distribution, per cent	Remarks
9	Talc concentrate........	0·43	9·5	1·02	10·6	3·62	9·8	Obviously low assay for zinc on zinc feed is responsible for poor check on zinc recovery.
	Lead-copper concentrate	10·84	78·3	28·80	76·8	8·67	7·6	
	Zinc feed..............	0·13	10·6 to 98·4	0·28	10·6 to 98·0	7·54	75·5 to 92·9	
	Zinc concentrate.......	....	....	....	....	58·0	70·3	
	Final tailing..........	....	....	....	....	0·6	5·2	
12	Talc concentrate........	0·36	6·6	0·92	7·6	3·42	7·1	Zinc and lead recoveries low.
	Lead-copper concentrate	11·60	60·6	24·40	58·1	4·42	2·9	
	Zinc feed..............	0·37	33·3 to 100·5	0·30	12·3 to 78·0	7·94	81·3 to 91·3	
	Zinc concentrate.......	....	....	....	....	56·0	76·0	
	Final tailing..........	....	....	....	....	0·6	5·3	
13	Talc concentrate........	0·29	1·8	1·10	3·0	3·62	2·4	Lead recovery low.
	Lead-copper concentrate	11·44	80·6	24·22	72·8	7·23	5·3	
	Zinc feed..............	0·18	20·6 to 102·5	0·30	14·7 to 90·5	7·34	92·2 to 99·9	
	Zinc concentrate.......	....	....	....	....	56·0	85·9	
	Final tailing..........	....	....	....	....	0·6	6·3	
14	Talc concentrate........	0·42	4·6	1·07	6·4	4·62	5·7	Lead recovery low.
	Lead-copper concentrate	13·68	59·5	32·55	77·5	6·25	3·1	
	Zinc feed..............	0·42	39·2 to 103·3	0·15	7·7 to 91·6	8·54	90·2 to 99·0	
	Zinc concentrate.......	....	....	....	....	56·0	85·2	
	Final tailing..........	....	....	....	....	0·6	5·0	
15	Talc concentrate........	0·42	4·1	1·07	5·7	4·62	5·1	Lead recovery low.
	Lead-copper concentrate	13·68	59·5	32·55	77·5	6·23	3·1	
	Zinc feed..............	0·42	39·7 to 103·3	0·15	7·7 to 90·9	8·54	91·3 to 99·5	
	Zinc concentrate.......	....	....	....	....	....	....	
	Final tailing..........	....	....	....	....	....	....	

TABLE IX—(Continued)

Assuming that, as shown by earlier tests, on the average grade of the tests, 56 per cent zinc concentrate and 0·6 per cent zinc tailing are obtainable:—

Recoveries of Metals in Products on Tonnages taken for Calculation from Table VIII—(Continued)

Test No.	Product	Copper assay	Distribution, per cent	Lead assay	Distribution, per cent	Zinc assay	Distribution, per cent	Remarks
16	Talc concentrate	0·35	12·3	1·42	25·3	4·12	16·7	No good check on recoveries except zinc.
	Lead-copper concentrate	11·62	69·4	31·75	95·4	9·15	6·3	
	Zinc feed	0·26	20·9 to 102·6	0·23	9·3 to 130·0	8·34	77·3 to 100·3	
	Zinc concentrate	....	....	....	....	56·0	73·0	
	Final tailing	....	....	....	....	0·6	4·3	
17	Talc concentrate	0·46	8·2	1·40	10·5	4·12	7·7	Lead recovery low. Copper recovery high. Zinc checks.
	Lead-copper concentrate	13·90	77·2	28·90	67·4	7·04	4·1	
	Zinc feed	0·23	23·0 to 108·4	0·40	16·8 to 94·7	8·43	88·2 to 100·0	
	Zinc concentrate	....	....	....	....	56·0	83·0	
	Final tailing	....	....	....	....	0·6	5·2	
18	Talc concentrate	0·35	5·3	1·32	8·9	4·02	6·6	Good check on all recoveries.
	Lead-copper concentrate	13·82	72·4	31·15	72·9	5·58	3·2	
	Zinc feed	0·23	22·6 to 100·3	0·43	18·9 to 100·7	8·43	90·2 to 100·0	
	Zinc concentrate	....	....	....	....	56·0	84·7	
	Final tailing	....	....	....	....	0·6	5·5	
22	Talc concentrate	0·42	15·6	1·53	26·6	4·42	18·1	Very poor check on copper and lead.
	Lead-copper concentrate	11·04	59·8	31·15	78·7	6·94	4·3	
	Zinc feed	0·23	17·8 to 93·2	0·38	13·7 to 119·0	9·15	76·2 to 98·6	
	Zinc concentrate	....	....	....	....	56·0	....	
	Final tailing	....	....	....	....	0·6	....	
23	Talc concentrate	0·48	4·1	1·53	6·9	4·95	5·6	Probably large amount of lead tied up in middling.
	Lead-copper concentrate	16·46	83·1	21·50	57·5	6·36	4·2	
	Zinc feed	0·16	13·0 to 100·2	0·45	19·5 to 83·9	8·40	90·2 to 100·0	
	Zinc concentrate	....	....	....	....	56·0	84·8	
	Final tailing	....	....	....	....	0·6	5·4	
24	Talc concentrate	0·45	12·0	1·45	18·4	4·54	16·3	
	Lead-copper concentrate	12·98	70·0	26·10	66·2	7·48	5·3	
	Zinc feed	0·27	19·0 to 100·0	0·45	15·1 to 99·7	8·28	78·3 to 99·9	
	Zinc concentrate	....	....	....	....	56·0	73·5	
	Final tailing	....	....	....	....	0·6	4·8	

TABLE X

Reconstructed Assays and Recoveries by Averages

(Abnormal Highs and Lows Discarded)

Averages on Runs Calculated (i.e., 9, 12, 13, 14, 15, 16, 17, 18, 22, 23, 24)

Product	Tons	Copper, per cent	Recovery, per cent	Lead, per cent	Recovery, per cent	Zinc, per cent	Recovery, per cent
Average feed..........	100	0·87	100	1·90	100	7·58	100
Talc concentrate......	11·1	0·38	4·8	1·22	7·1	3·95	5·8
Lead-copper concentrate................	4·3	14·93	73·8	35·52	80·4	3·70	2·1
Zinc feed.............	84·6	0·22	21·4	0·28	12·5	8·25	92·1

From preliminary tests: zinc concentrate=56 per cent; zinc tailing=0·6 per cent.

$$\text{Therefore } T = F\frac{(c-f)}{(c-t)} = \frac{84\cdot6\ (56-8\cdot25)}{(56\ -\ 0\cdot6)} = 72\cdot91 \text{ tons}$$

Zinc feed	84·6	tons at	8·25	per cent zinc	Per cent recovery of feed
Zinc concentrate	11·69	tons at	56	per cent zinc	86·4
Zinc tailing	72·91	tons at	0·6	per cent zinc	5·7
	84·60				92·1

APPENDIX TO TABLE X

Lead-Copper Separation

	Tons	Copper, per cent	Recovery	Lead, per cent	Recovery
Lead-copper concentrate feed.....	4·3	14·93	73·8	35·52	80·4
Lead concentrate.............	2·4	3·8	10·5	60·8	76·9
Copper concentrate...........	1·9	28·9	63·3	3·5	3·5
			73·8		80·4

(Presumably in this separation 3·5 per cent of lead is lost economically and, depending on ore contract, probably 1·3 per cent of copper in the lead concentrate. The grade of these concentrates is high and owing to attachments it is possible that difficulty would appear in practice in obtaining them. According to contract terms it may not matter as to the quantity of copper in the lead concentrate, but probably any lead in the copper concentrate will be lost economically. Therefore, in practice, and according to terms of the ore contracts, it may be more practicable and easier to make a lower grade lead concentrate with higher copper percentage, and as long as the lead in the copper concentrate can be kept low, there is no increase in economic loss.)

Separation of Copper and Lead from the Bulk Concentrate. In Runs Nos. 21, 22, 23, and 24, the copper-lead bulk concentrate was pumped to a conditioning tank where approximately 12 to 15 minutes' conditioning was given to it with the equivalent of 0·1 pound of cyanide per ton of ore, or approximately 2·0 pounds per ton of copper-lead concentrate. No other reagent was added except a very small quantity of cresylic acid, sufficient only to maintain a proper amount of froth. These were the conditions arrived at in Runs Nos. 23 and 24.

Table XI gives the results of these tests, but special attention is directed to results of Runs Nos. 23 and 24.

TABLE XI

Results of Copper-Lead Separation

Test No.	Bulk concentrate					Lead concentrate					Copper concentrate				
	Copper, per cent	Lead, per cent	Gold, oz/ton	Silver, oz/ton	Per cent total metals	Copper, per cent	Lead, per cent	Gold, oz/ton	Silver, oz/ton	Per cent total lead	Copper, per cent	Lead, per cent	Gold, oz/ton	Silver, oz/ton	Per cent total copper
21	15·0	30·75	0·40	34·6	100·0	7·54	51·0	0·35	57·4	83·9	18·4	10·0	0·44	8·6	84·3
22	11·04	31·15	0·40	29·12	100·0	3·8	46·1	0·42	42·84	95·0	19·56	4·4	0·26	4·64	81·3
23	16·46	21·5	0·45	34·36	100·0	10·06	44·0	0·22	65·4	93·5	21·8	2·55	0·46	9·46	72·20
24	12·98	26·1	0·36	30·26	100·0	8·96	48·5	0·46	64·44	95·9	20·5	2·20	0·13	3·9	55·11

This table shows that under the conditions obtained in Runs Nos. 23 and 24, 5·3 per cent of the lead in the bulk concentrate would be lost in the copper concentrate. This recovery should be improved in practice. The recovery of the copper in the copper concentrate is not particularly good, but this may not matter as, depending on smelter contract terms, the copper in the lead concentrate will have a value, but any lead in the copper concentrate must be considered as lost. The economics of this separation rest entirely on smelter contracts and the cost of handling the separate products.

Recovery of Gold and Silver. Table XII has been compiled showing these results.

TABLE XII

Gold and Silver Recovery

Test No	Feed		Lead and copper concentrate	
	Gold, oz.	Silver, oz.	Gold, oz.	Silver, oz.
9	0·035	2·45	0·37	29·10
12	0·04	2·36	0·50	33·16
13	0·035	2·51	0·33	30·08
14	0·035	2·53	0·35	31·78
15	0·035	2·40	0·44	36·26
16	....	....	0·38	30·54
17	....	....	0·42	32·58
18	....	....	0·46	34·74
22	....	....	0·40	29·12
23	....	....	0·45	34·36
24	....	....	0·36	30·26
Average	0·038	2·46	....	....

1100 tons at.................... 0·038 oz. gold and 2·46 oz. silver per ton
Total....................41·8 oz. gold and 2706 oz. silver

Test No.	Tons of concentrate	Total, oz.	
		Gold	Silver
9	6·5	2·405	189·150
12	4·6	2·300	152·536
13	5·5	1·815	165·940
14	4·0	1·400	127·120
15	4·0	1·760	145·040
16	4·9	1·862	149·146
17	4·5	1·890	146·610
18	4·4	2·024	152·856
22	4·5	1·800	131·040
23	5·3	2·385	182·108
24	5·2	1·872	157·352
	....	21·513	1,698·898

Recovery	Gold	Silver
Average of 11 tests....................	51·5 per cent	62·8 per cent

NOTES:

1. Individual tests range as high as 5 per cent over these figures for gold and 15 per cent higher for silver.
2. The gold and silver in hypothetical lead-copper concentrate carrying 40 per cent lead and 13 per cent copper was given a recovery of 58·8 per cent gold and 72·4 per cent silver.
3. Practical operation of the Stirling mill should give higher recoveries than those indicated in the average of above tests.
4. Owing to the fact that the gold and silver are not stable with either the lead or copper content of the ore, it is difficult to assume values to correspond with the copper and lead content of concentrate in Table X, and hence calculate an assumed recovery.

GENERAL DISCUSSION ON TEST WORK

Results show that, although concentration is far from easy and that the control of the operation requires more than usual care and skill, there is no reason why the improved recoveries made in the test work will not be obtained in practice.

The work was carried out under conditions approaching those of large-scale mill operations. All operations were continuous, with the return middling circuits all closed so that there was no question where and how such middling would report. Small-scale batch testing on this ore is guess-work when it becomes necessary to decide in what product the middling will report or in what manner it will affect the grade of the products after a circuit has become built up. No machine was cleaned out and no cell dumped during the testing of the entire two carloads of ore, and the building up in any of the secondary circuits was, therefore, undisturbed.

The work has proved that the talc can be removed at any point in the grinding circuit found convenient to put flotation cells, and that fine grinding is required to free the sulphides from each other.

A microscopic examination of mill products selected from certain runs was made in order to throw more light on the degree of attachment of the sulphide minerals. The results of this study are tabulated in Tables XIII to XXII.

Tables XIII and XIV show the degree of combination in a concentrate of the lead and copper. The concentrate used in this study was taken from Run No. 4 and represents a product that was cleaned only once. The chemical analysis is given under Table XIV. The microscopic study shows that 54·4 per cent of the sulphides is free and 45·6 per cent combined; and from Table XIV it can be seen that 46·7 per cent of the galena is combined with other sulphides, 19·3 per cent being combined with the sphalerite and 7 per cent with the pyrite. The remaining 20·4 per cent is combined with the chalcopyrite and tetrahedrite, which fact is of no importance as these minerals are required in the product when a bulk concentrate is produced.

In Tables XV and XVI, and Tables XVII and XVIII, the results of a study of two middling products are given. The middling shown in Tables XVII and XVIII has been reground and represents the middling from Run No. 24, and in Table XXII a microscopic screen analysis also shows the degree of grinding. This reground middling is compared with a middling obtained under similar circumstances but not reground and is represented in Tables XV and XVI, and Table XXI gives the microscopic screen analysis. Comparing the screen analyses of the two products, the unground middling is 35 per cent plus 325 mesh, whereas the reground middling is 7·7 per cent plus 325 mesh, and the reduction in the finer grains is as marked.

The unground middling is shown to contain 48·5 per cent of the sulphides free and the remaining 51·5 per cent, still combined with each other. In the reground middling the amount of combined sulphides has been reduced to 22·4 per cent, that is, it contains 77·6 per cent free as compared with only 48·5 per cent. The most important combinations in this product from a practical point of view are those of the galena and copper with the sphalerite, pyrite, and gangue. The proportion combined

TABLE XIII

Microscopic Analysis of Copper-Lead Concentrate, Run No. 4, Stirling Ore

Percentage by Volume

Mesh	Pyrite		Chalcopyrite		Sphalerite		Galena		Tetrahedrite		Total		Total
	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	
+200	1·1	2·2	....	....	....	....	....	....	....	....	1·1	2·2	3·3
−200 +325	3·1	6·1	0·6	1·3	....	1·3	0·7	....	....	....	4·4	8·7	13·1
−325 +560	5·0	3·4	2·1	3·4	0·6	3·4	1·4	1·1	1·5	0·5	10·6	11·8	22·4
−560 +1100	4·0	1·2	2·5	3·9	0·7	3·9	1·2	1·2	1·2	0·3	9·6	10·5	20·1
−1100+1600	2·6	1·0	3·8	2·3	1·5	2·3	1·7	1·8	1·6	0·2	11·2	7·6	18·8
−1600+2300	1·4	0·3	3·3	1·2	2·1	1·1	2·1	0·9	0·7	0·1	9·6	3·5	13·1
−2300	0·9	0·1	3·7	0·3	1·0	0·3	1·1	0·6	1·2	....	7·9	1·3	9·2
Total	18·1	14·3	16·0	12·3	5·9	12·3	8·2	5·6	6·2	1·1	54·4	45·6	100·0
	32·4		28·3		18·2		13·8		7·3		100·0		

TABLE XIV

Degree of Combination of the Metallic Minerals in Copper-Lead Concentrate, Run No. 4, Stirling Ore

Minerals	Combined with pyrite	Combined with chalcopyrite	Combined with sphalerite	Combined with galena	Combined with tetrahedrite	Total		Total, per cent
						Combined	Free	
Pyrite	....	6·9	16·4	20·9	....	44·2	55·8	100·0
Chalcopyrite	1·0	....	26·3	18·2	2·5	48·0	52·0	100·0
Sphalerite	6·0	18·5	....	27·9	15·0	67·4	32·6	100·0
Galena	7·0	17·4	19·3	....	3·0	46·7	53·3	100·0
Tetrahedrite	....	0·1	7·2	7·8	....	15·1	84·9	100·0

Chemical Analysis

Copper, 9·34 per cent; Lead, 19·10 per cent; Zinc, 9·86 per cent; Insoluble, 4·08 per cent

TABLE XV

Microscopic Analysis of Lead-Copper Middling, Run No. 9, Stirling Ore

Percentage by Volume

Mesh	Pyrite		Chalcopyrite		Sphalerite		Galena		Tetrahedrite		Gangue		Total		Total
	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	
+150...	....	....	....	....	....	....	....	....	....	....	0·9	1·6	0·9	1·6	2·5
−150 +200...	0·5	0·5	....	0·5	....	1·4	....	....	....	....	1·8	1·7	2·3	4·1	6·4
−200 +325...	4·6	6·1	....	0·9	....	5·3	....	....	0·3	....	1·4	2·4	6·3	14·7	21·0
−325 +560...	7·1	3·4	0·9	1·6	1·1	4·8	0·9	0·9	0·2	0·4	1·4	1·2	11·6	12·3	23·9
−560 +1100..	6·3	1·6	1·5	1·4	3·2	4·6	1·1	1·9	0·3	....	1·3	0·3	13·7	9·8	23·5
−1100+1600..	2·3	0·6	0·7	0·6	1·8	1·5	0·4	0·7	....	....	0·5	0·2	5·7	3·6	9·3
−1600+2300..	1·5	0·1	1·6	0·9	1·7	1·4	0·4	1·2	....	0·1	0·1	0·2	5·3	3·9	9·2
−2300........	0·4	....	0·9	0·2	0·9	0·7	0·5	0·6	....	....	....	....	2·7	1·5	4·2
Total....	22·7	12·3	5·6	6·1	8·7	19·7	3·3	5·3	0·8	0·5	7·4	7·6	48·5	51·5	100·0
	35·0		11·7		28·4		8·6		1·3		15·0		100·0		

TABLE XVI

Degree of Combination of the Minerals in Lead-Copper Middling, Run No. 9, Stirling Ore

Minerals	Combined with pyrite	Combined with chalcopyrite	Combined with sphalerite	Combined with galena	Combined with tetrahedrite	Combined with gangue	Total		Total, per cent
							Combined	Free	
Pyrite........................	....	4·9	22·9	5·5	....	2·1	35·4	64·6	100·0
Chalcopyrite..................	2·5	....	32·8	7·1	0·6	8·6	51·6	48·4	100·0
Sphalerite....................	12·3	12·4	....	41·5	0·2	3·0	69·4	30·6	100·0
Galena........................	6·6	5·7	42·9	....	0·6	6·3	62·1	37·9	100·0
Tetrahedrite..................	....	13·3	3·3	20·1	....	....	36·7	63·3	100·0
Gangue........................	16·9	11·4	10·9	11·4	....	....	50·6	49·4	100·0

TABLE XVII

Microscopic Analysis of Lead-Copper Middling, Run No. 24, Stirling Ore, Reground

Percentage by Volume

Mesh	Pyrite		Chalcopyrite		Sphalerite		Galena		Tetrahedrite		Gangue		Total		Total
	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	
+325...	1·4	0·7	0·5	....	....	....	....	....	0·5	....	1·8	0·5	4·2	1·2	5·4
−325 +560...	3·7	1·6	0·8	0·4	0·5	1·0	0·8	....	0·8	....	1·3	1·2	7·9	4·2	12·1
−560 +1100..	10·0	2·2	2·0	1·2	6·5	4·6	4·2	1·0	....	0·4	1·3	1·1	24·0	10·5	34·5
−1100+1600..	4·9	0·6	2·5	0·2	4·3	1·4	1·2	0·3	0·3	....	0·7	....	13·9	2·5	16·4
−1600+2300..	5·1	0·2	4·2	0·5	5·8	1·2	2·0	0·4	0·5	....	....	0·1	17·6	2·4	20·0
−2300........	2·1	....	2·2	0·4	3·4	0·4	1·5	0·8	0·1	....	0·7	....	10·0	1·6	11·6
Total....	27·2	5·3	12·2	2·7	20·5	8·6	9·7	2·5	2·2	0·4	5·8	2·9	77·6	22·4	100·0
	32·5		14·9		29·1		12·2		2·6		8·7		100·0		

TABLE XVIII

Degree of Combination of the Minerals in Lead-Copper Middling, Run No. 24, Stirling Ore, Reground

Minerals	Combined with pyrite	Combined with chalcopyrite	Combined with sphalerite	Combined with galena	Combined with tetrahedrite	Combined with gangue	Total		Total, per cent
							Combined	Free	
Pyrite..........................	....	1·9	7·7	5·5	....	1·2	16·3	83·7	100·0
Chalcopyrite....................	2·6	....	9·0	5·5	....	1·3	18·4	81·6	100·0
Sphalerite......................	2·7	5·8	....	17·9	1·7	1·3	29·4	70·6	100·0
Galena..........................	0·8	2·4	16·2	....	....	0·8	20·2	79·8	100·0
Tetrahedrite....................	....	....	16·7	....	....	....	16·7	83·3	100·0
Gangue..........................	4·4	2·2	24·3	2·8	....	....	33·7	66·3	100·0

TABLE XIX

Microscopic Analysis of Tailing, Run No. 24, Stirling Ore

Percentage by Volume

Mesh	Pyrite		Chalcopyrite		Sphalerite		Galena		Tetrahedrite		Gangue		Total		Total
	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	Free	Combined	
+100	....	....	....	....	....	....	....	....	....	....	1·6	....	1·6	....	1·6
−100 +150	....	....	....	....	....	....	....	....	....	....	2·8	2·7	2·8	2·7	5·5
−150 +200	....	....	....	....	....	....	....	....	....	....	14·2	5·0	14·2	5·0	19·2
−200 +325	1·6	0·5	....	....	....	2·0	....	....	....	....	12·9	6·6	14·5	9·1	23·6
−325 +560	2·6	0·6	....	....	0·5	1·8	....	....	1·0	....	11·5	2·5	15·6	4·9	20·5
−560 +1100	2·5	0·5	....	0·2	1·0	2·5	....	0·4	....	....	4·7	0·9	8·2	4·5	12·7
−1100+1600	1·2	0·1	0·1	0·5	1·5	1·5	....	0·3	....	....	2·7	0·3	5·5	2·7	8·2
−1600+2300	0·9	0·1	0·3	0·3	1·4	1·0	....	0·2	....	....	1·9	0·2	3·7	1·8	5·5
−2300	0·6	Trace	0·2	0·2	1·3	0·5	Trace	0·1	....	....	0·2	0·1	2·3	0·9	3·2
Total	9·4	1·8	0·6	1·2	5·7	9·3	Trace	1·0	1·0	....	51·7	18·3	68·4	31·6	100·0
	11·2		1·8		15·0		1·0		1·0		70·0		100·0		

TABLE XX

Degree of Combination of the Minerals in Tailing, Run No. 24, Stirling Ore

Minerals	Combined with pyrite	Combined with chalcopyrite	Combined with sphalerite	Combined with galena	Combined with tetrahedrite	Combined with gangue	Total		Total, per cent
							Combined	Free	
Pyrite	....	2·0	9·2	....	....	5·2	16·4	83·6	100·0
Chalcopyrite	2·8	....	47·2	....	....	19·4	69·4	30·6	100·0
Sphalerite	4·0	10·4	....	15·9	....	31·8	62·1	37·9	100·0
Galena	....	....	45·8	....	....	50·0	95·8	4·2	100·0
Tetrahedrite	....	....	....	....	....	....	....	100·0	100·0
Gangue	3·3	3·8	13·3	5·8	....	....	26·2	73·8	100·0

with the gangue is not serious, but in the unground middling 42·9 per cent of the galena is combined with the sphalerite, as shown in Table XVI. This has been reduced to 16·2 per cent by regrinding the middling, shown in Table XVIII. The amount of chalcopyrite combined in the unground middling is 51·6 per cent, which is reduced to 18·4 per cent by regrinding. The greater part of this chalcopyrite is combined with the sphalerite.

From this analysis it becomes evident that the chief disturbing element will be zinc, and if the zinc be forced out of the product it will carry with it both chalcopyrite and galena and cause high losses owing to attachments. Fine grinding is therefore essential in order to free the chalcopyrite and galena from the sphalerite, otherwise it will be impossible to obtain a high-grade copper-lead concentrate without high loss of both minerals.

Tables XIX and XX show the results of a similar study made on the tailing from the copper-lead flotation obtained from Run No. 24. This examination showed that 95·8 per cent of the galena in this tailing was attached, only 4·2 per cent being free. About 50 per cent was combined with gangue and 45·8 per cent with sphalerite. In the case of the chalcopyrite, 30·6 per cent was free, and of the 69·4 per cent that was combined, 47·2 per cent was combined with the sphalerite.

This microscopic study of the mill products has brought out clearly that it is necessary to grind the sulphides very fine.

The grinding during the test work was slightly coarser than 80 per cent through 200 mesh. This was about the best that could be done and still maintain a density that would allow efficient flotation; a bowl classifier was not available. It might have been possible to grind the sulphides finer and still maintain a suitable density in a bowl overflow had one been used. Decidedly the best practice, however, would be to regrind the middling, or possibly a concentrate from the copper-lead flotation, rather than to endeavour to grind too fine in the main grinding circuit.

Whether a middling or a concentrate should be reground is still an open question. In the test runs the regrinding of the middling was favoured, because on a small laboratory scale it was found easier to control than the operation of regrinding and re-floating a concentrate. This feature is referred to under the description of Runs Nos. 10 to 13, which were made on reground concentrate, and under Runs Nos. 14 to 24, made on reground middling.

Air Conditioning. Owing to the sensitivity of the chalcopyrite to cyanide and to the tendency of a proportion of the pyrite to be difficult to depress, air was used in the conditioning tank preceding the flotation of the copper-lead concentrate. In the last three tests a Pachuca tank was used. The action of this extra air conditioning was beneficial to the copper flotation by increasing the flotability of the chalcopyrite.

The use of air conditioning may not be essential on freshly mined ore, but there is no doubt that it would prove beneficial. The cost under normal conditions should not exceed one cent per ton.

TABLE XXI

Microscopic Analysis of Grain Size of Lead-Copper Middling, Run No. 9, Stirling Ore

Mesh	Per cent
− 65+ 100	2·1
− 100+ 150	3·7
− 150+ 200	7·2
− 200+ 325	22·0
− 325+ 560	20·2
− 560+1100	23·9
−1100+1600	7·6
−1600+2300	7·3
−2300	6·0
Total	100·0

TABLE XXII

Microscopic Analysis of Grain Size of Lead-Copper Middling (Reground), Run No. 24, Stirling Ore

Mesh	Per cent
− 150+ 200	1·3
− 200+ 325	6·4
− 325+ 560	17·0
− 560+1100	26·2
−1100+1600	13·2
−1600+2300	16·2
−2300	18·7
Total	100·0

Ore Dressing and Metallurgical Investigation No. 566

COPPER-NICKEL ORE FROM THE CUNIPTAU MINES, GOWARD, ONTARIO

Shipment. A shipment of 30,028 pounds of ore was received on January 10, 1934. It was submitted by the Cuniptau Mines Development Company, Limited, 465 Bay Street, Toronto, and the experimental tests were arranged for by E. P. Muntz, Vice-President.

Characteristics of the Ore. This shipment was submitted as representing a disseminated type of ore. The metallic sulphides are disseminated throughout a medium- to fine-grained, greenish grey to black serpentine rock. Numerous fibrous needles of the silicate penetrate the small sulphide masses.

The metallic minerals identified in the present sample are magnetite, pyrrhotite, chalcopyrite, and pentlandite. All are intimately associated, and occur in mutual intergrowths forming small masses disseminated rather evenly through the gangue. Pentlandite is present in considerable amount and occurs chiefly in pyrrhotite and chalcopyrite, with which minerals it is intimately intergrown. Minor amounts of magnetite, pyrrhotite, and chalcopyrite occur as small irregular or elongated grains and stringers in the gangue.

Sampling and Analysis. The whole shipment was sampled, and the sample was found to contain the following:—

Copper	1·12 per cent.	Gold	0·01 oz/ton
Arsenic	Nil	Silver	0·18 "
Sulphur	5·62 per cent	Platinum and palladium	0·13 "
Nickel	1·02 "		
Iron	16·74 "		

Experimental Tests

A series of large-scale continuous flotation tests was made, the object being to concentrate the ore by flotation to obtain a product containing a higher percentage of copper and nickel. A previous shipment of ore from this property had concentrated without any particular difficulty. It consisted of a sample of high-grade massive sulphide ore, differing from this later shipment in that it contained very little or none of the serpentine rock.

A serious difficulty was encountered immediately on attempting to concentrate the present sample by flotation. On crushing the ore, the serpentine gangue yielded a product of a "talcy" nature that floated with the sulphides and prevented a high-grade concentrate being made.

A separate set of flotation machines was used in the flow-sheet in order to float away this talcy material before the flotation of the copper and nickel was attempted. This has been accomplished successfully on a number of ores containing a talcy gangue, but in this case practically the whole of the gangue when ground in the ball mill was converted into material of a talcy nature that tended to float.

After numerous attempts to remove this troublesome material, it was decided to try the use of a colloid, such as glue, which would disperse the talcy gangue material and tend to keep it from floating with the copper and nickel. The glue was found to be very effective, but it was still necessary to float off a part of the talcy material before floating the copper and nickel. When this was first done the addition of a solution of glue to the flotation cells prevented the rest of the talcy material from floating with the bulk concentrate of copper and nickel. The amount of glue used was about one-half pound per ton of ore.

Flow-Sheet. During the first four tests, the ore was fed at the rate of 500 pounds per hour, and during the remaining four tests at 300 pounds per hour. The ore was fed to a small 30 inch by 36 inch ball mill in closed circuit with a small Dorr classifier. The overflow of the Dorr went to a conditioning tank where pine oil was added for the flotation of the talc product. The talc was floated in a 10-cell mechanical flotation machine. The tailing from the talc flotation cells was pumped to a conditioning tank and the glue solution and the flotation reagents for the copper-nickel float were added. (The reagents used for the copper-nickel float were amyl xanthate and pine oil.) Then into a second 10-cell flotation machine, where the copper-nickel concentrate was floated. The feed entered the second cell, Nos. 2 and 3 cells making a concentrate, which was recleaned in Cell No. 1. Cells Nos. 4 to 10, inclusive, made a rougher concentrate, which was returned to Cell No. 2 with the feed from the conditioning tank.

Results. It is unnecessary to give in detail the results of the eight individual runs made. Runs Nos. 3 and 4 only will be given, as they represent the best results obtained.

Run No. 3

Feed rate—500 pounds per hour.

Total amount of feed—approximately 3,500 pounds.

Reagents Used:

Talc Float:

Pine oil added to conditioner and to Cells Nos. 3, 5, 7, and 9. Total amount equivalent to 0·10 pound per ton.

Copper-Nickel Float:

Glue	0·5	lb/ton of original ore	
Amyl xanthate	0·25	"	"
Copper sulphate	0·4	"	"
Pine oil	None		

Screen Test of Feed to Flotation:

Mesh	Weight, per cent
+ 65	0·7
− 65+100	5·0
−100+150	8·5
−150+200	17·0
−200	68·8
Total	100·0

Results of Test:

Product	Proportional weight	Analysis, per cent		Distribution, per cent	
		Copper	Nickel	Copper	Nickel
Feed	100·0	1·15	0·91	100·0	100·0
Talc float	35·50	0·64	0·41	19·75	16·0
Copper-nickel concentrate	15·76	5·14	4·05	70·50	70·1
Flotation tailing	48·74	0·23	0·26	9·75	13·9

Total ratio of concentration: 1 : 6·45.

Explanation of Table. These tables of results show that the ore was ground to 68·8 per cent through a 200-mesh screen; and that out of 100 tons of ore 35·5 tons of talcy material would be floated, 15·76 tons of copper-nickel concentrate leaving 48·74 tons as flotation tailing. The talc concentrate must also be considered a tailing, giving a loss of 19·75 per cent of the copper and 16 per cent of the nickel in the talc product, plus 9·75 per cent of the copper and 13·9 per cent of the nickel in the flotation tailing. This makes a total loss of 29·5 per cent of the copper and 29·9 per cent of the nickel, leaving a recovery of 70·50 per cent of the copper and 70·1 per cent of the nickel in the copper-nickel concentrate.

Run No. 4

The feed rate, grinding, and reagents were the same as for Run. No. 3.

Results:

Product	Proportional weight	Analysis, per cent		Distribution, per cent	
		Copper	Nickel	Copper	Nickel
Feed	100·0	0·96	0·88	100·0	100·0
Talc float	45·6	0·41	0·39	19·5	20·3
Copper-nickel concentrate	12·8	5·26	4·42	70·1	64·5
Flotation tailing	41·6	0·24	0·32	10·4	15·2

Total ratio of concentration: 1 : 7·82.

Analysis of Copper-Nickel Concentrate:

Copper	5·26 per cent	Gold	0·04 oz/ton
Iron	29·45 "	Platinum and palladium	0·36 "
Silica	16·81 "		
Nickel	4·42 "		
Sulphur	18·56 "		

CONCLUSIONS

This disseminated ore is difficult to concentrate; it is too fine-grained to use gravity concentration, and the only remaining method is flotation. The results shown in this test work would undoubtedly be improved in actual milling operations as experience is gained.

Ore Dressing and Metallurgical Investigation No. 567

GOLD ORE FROM THE ARNTFIELD GOLD MINES, LIMITED, ARNTFIELD, QUEBEC

Shipment. A carload shipment of 19,547 pounds of ore was received January 12, 1934. It was made by the Arntfield Gold Mines, Limited, 159 Bay Street, Toronto, from its property at Arntfield, Quebec.

Sampling and Analysis. The entire shipment was crushed to $\frac{1}{2}$ inch and a tenth cut for a sample by means of a Vezin automatic sampler. This tenth was then reduced by stage crushing and dividing in a Jones sampler until the final sample for the assay laboratory was obtained. This sample was found to assay:—

Gold	0·075 oz/ton

Characteristics of Ore:

Twelve polished sections were made from selected specimens of the ore and examined microscopically to determine the metallic minerals and their mode of occurrence.

The *gangue minerals* were not identified, but in general the gangue consists of fine-textured, dense silicates which show considerable range in colour, imparting a mottled appearance.

The *metallic minerals* in the sections examined are: pyrite, magnetite, chalcopyrite, and pyrrhotite. Pyrite and magnetite are comparatively abundant; chalcopyrite and pyrrhotite are present in extremely small amounts.

Pyrite occurs as disseminated poorly-formed cubes and irregular grains which in some places are present in sufficient quantity to form granular masses of the mineral. It contains small inclusions of gangue, chalcopyrite, and pyrrhotite.

Magnetite occurs as small disseminated grains; locally it is abundant as very fine disseminations which give to the ore a dull black appearance.

Chalcopyrite occurs as tiny irregular grains in pyrite, and more rarely in the gangue.

Pyrrhotite is exceedingly rare as small grains within pyrite.

Grain Size of the Pyrite. A quantitative microscopic analysis was carried out on the twelve sections for the purpose of determining the grain size of the pyrite. The result is shown on page 112:—

Grain Size of the Pyrite:

Mesh	Per cent	Accumulative per cent
+ 10	3·6	3·6
− 10+ 14	7·8	11·4
− 14+ 20	8·0	19·4
− 20+ 28	9·9	29·3
− 28+ 35	16·2	45·5
− 35+ 48	12·4	57·9
− 48+ 65	8·3	66·2
− 65+100	7·8	74·0
−100+150	8·0	82·0
−150+200	6·4	88·4
−200+325	5·7	94·1
−325	5·9	100·0
Total	100·0	

Purpose of Experimental Tests. The following metallurgical tests were made to obtain metallurgical data on the proper method of milling the ore for the recovery of the gold.

Experimental Tests

The experimental work included the following methods of treatment:

1. Amalgamation, and flotation of the amalgamation tailing.
2. Trapping of the coarse gold, and flotation of the trap tailing.
3. Straight cyanidation of the ore.
4. Cyanidation of the flotation concentrate obtained from methods 1 and 2.

AMALGAMATION AND FLOTATION

Test No. 1

The ore, crushed to ½ inch, was fed to a 32-inch (diameter) by 36-inch ball mill, containing a charge of 2,000 pounds of assorted ball sizes, at the rate of 650 pounds per hour. The ball mill discharge passed over an amalgamation plate, which discharged into a Dorr classifier. The classifier sands returned to the ball mill and the overflow went to a 10-cell flotation machine, which produced concentrate and tailing.

Results:

Run No.	Product	Assays, gold, oz/ton	Distribution of gold, per cent of total	
1	Feed	0·075	100·0	
	Amalgamation tailing	0·045	40·0	
	Feed to flotation	0·045	60·0	
	Flotation concentrate	0·90	47·2	87·2
	" tailing	0·01	12·8	

Ratio of concentration........25·5 : 1

Results:

Run No.	Product	Assays, gold, oz/ton	Distribution of gold, per cent of total	
2	Feed	0·075	100·0	
	Amalgamation tailing	0·04	46·7	
	Feed to flotation	0·04	53·3	
	Flotation concentrate	0·79	40·5	87·2
	" tailing	0·01	12·8	

Ratio of concentration .... 26 : 1

Run No.	Product	Assays, gold, oz/ton	Distribution of gold, per cent of total	
3	Feed	0·075	100·0	
	Amalgamation tailing	0·04	46·7	
	Feed to flotation	0·04	53·3	
	Flotation concentrate	0·91	40·9	87·6
	" tailing	0·01	12·4	

Ratio of concentration .... 30 : 1

Screen Tests on Flotation Tailing:

Mesh	Run No. 1, weight, per cent	Run No. 2, weight, per cent	Run No. 3, weight, per cent
+ 65	?	?	?
− 65+100	5·0	7·3	5·75
−100+150	11·0	12·2	13·90
−150+200	13·4	14·5	15·50
−200	70·6	66·0	64·85
Total	100·0	100·0	100·0

From the results of three runs made in this test, it will be observed that about 46 per cent of the gold can be recovered by amalgamation when using this flow-sheet and that 76 per cent of the remaining gold can be recovered in a flotation concentrate assaying about 0·90 ounce per ton, which gives a ratio of concentration of one ton of concentrate for approximately every 26 to 30 tons of crude ore. The total recovery by amalgamation and by flotation is 87·3 per cent.

The flotation concentrate is not a suitable product to ship to a smelter, so it is necessary to determine whether the gold in this product can be extracted and recovered by cyanidation, before concentration by flotation can be considered as a possible method of treatment.

Test No. 2

The ore, crushed to ½ inch, was fed to the ball mill as in the previous run, but instead of the ball mill discharge passing over an amalgamation plate a hydraulic trap was placed at the discharge end of the mill. The overflow of the trap then went to the Dorr classifier, which operated as before in closed circuit with the ball mill.

Results:

Run No.	Product	Assays, gold, oz/ton	Distribution of gold, per cent of total
1	Feed	0·075	100·0
	Trap tailing	0·045	40·0
	Feed to flotation	0·045	60·0
	Flotation concentrate	1·06	47·1 87·1
	" tailing	0·01	12·9

Ratio of concentration.......... 30 : 1

Run No.	Product	Assays, gold, oz/ton	Distribution of gold, per cent of total
2	Feed	0·075	100·0
	Trap tailing	0·045	40·0
	Feed to flotation	0·045	60·0
	Flotation concentrate	1·08	47·1 87·1
	" tailing	0·01	12·9

Ratio of concentration.......... 30·6 : 1

Screen Tests on Flotation Tailing:

Mesh	Run No. 1, weight, per cent	Run No. 2, weight, per cent
−100	6·3	4·1
−100+150	11·0	7·8
−150+200	13·6	13·4
−200	69·1	74·7
Total	100·0	100·0

The results with the trap in the grinding circuit are not conclusive, for the reason that the weight of the material taken from the trap clean-up at the end of the test was only 26 pounds and it assayed only 1·75 ounces per ton in gold. During the test there was approximately 7,600 pounds of ore fed to the mill, assaying 0·075 ounce to the ton. The classifier overflow going to flotation assayed 0·045, leaving 0·030 ounce per ton in the grinding circuit. The units of gold remaining in the grinding circuit were, therefore, 7,600 multiplied by 0·03, which is 228 units. The units of gold retained in the trap were: 26 multiplied by 1·73, which is 54·98 units. This means that the trap contained only 41·5 per cent of the gold that did not overflow the classifier. It is evident that the classifier and ball mill acted as a better trap for the gold than did the trap itself, and therefore it is possible that when the grinding circuit becomes saturated gold would overflow the classifier in such a condition that it would not be recovered in the flotation circuit.

STRAIGHT CYANIDATION

Eight tests were made on samples of the ore crushed to various sizes and treated for periods of 24 and 48 hours.

Results:

Feed sample: gold, 0·075 oz/ton

Test No.	Grinding mesh	Period of agitation, hours	Tailing assay, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
					KCN	CaO
1	− 48	24	0·015	80·0	0·4	6·1
2	−100	24	0·01	86·7	0·7	6·7
3	−150	24	0·01	86·7	0·7	6·85
4	−200	24	0·01	86·7	0·85	7·00
5	− 48	48	0·01	86·7	0·4	6·35
6	−100	48	0·01	86·7	0·7	6·95
7	−150	48	0·01	86·7	0·85	7·10
8	−200	48	0·01	86·7	1·15	7·25

These tests indicate that the ore cyanided readily and that there is no advantage in grinding any finer than 48 mesh. The fact that a tailing lower than 0·01 ounce per ton was not obtained by either flotation or cyanidation, regardless of the fineness to which the ore was crushed, is worthy of attention. The microscopic study of the ore indicates the probable reason for this uniform tailing. It shows the occurrence of tiny grains of chalcopyrite disseminated throughout both the pyrite and gangue. These grains of chalcopyrite probably carry sufficient gold to account for the 0·01 tailing.

CYANIDATION OF FLOTATION CONCENTRATE

A series of cyanidation tests was made on samples of a mixture of the flotation concentrates produced in Tests Nos. 1 and 2. Samples of this mixture, with and without grinding, were agitated in cyanide solution, 1·0 pound KCN per ton, for periods of 24 and 48 hours.

Results:

Feed sample: gold, 1·07 oz/ton

Test No.	Grinding, per cent −200 mesh	Period of agitation, hours	Tailing assay, gold, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
					KCN	CaO
1	61·5	24	0·23	78·5	3·1	5·0
2	61·5	48	0·235	78·0	4·0	5·4
3	90·2	24	0·185	82·7	3·4	6·1
4	90·2	48	0·155	85·5	4·8	7·1

In Tests Nos. 3 and 4 the concentrate was reground for 15 minutes in a ball mill. Much frothing occurred during agitation of the reground concentrate.

CONCLUSIONS

The results of the experimental work on this sample of ore show clearly that cyanidation is the only method of treatment.

The flotation tests show that a recovery of 87 per cent can be obtained, but that when the flotation concentrate is cyanided after regrinding to 90 per cent through 200 mesh only 85 per cent of the gold in the concentrate is extracted. This indicates only 74 per cent total recovery of the gold by flotation and cyanidation of the flotation concentrate.

Ore Dressing and Metallurgical Investigation No. 568

GOLD ORE FROM LITTLE LONG LAC, GERALDTON, ONTARIO

Shipment. A shipment consisting of 24 bags of gold ore weighing 1,000 pounds was received February 7, 1934, from the Little Long Lac Gold Mines, Limited, Geraldton, Ontario.

Characteristics of the Ore. Polished sections of the ore were examined in the mineragraphic laboratory and studied microscopically to determine the metallic minerals and the mode of their occurrence. The gangue was found to consist of medium- to fine-grained "greenstone" with a minor amount of quartz and a small amount of carbonate. The metallic minerals in the order of their abundance in the sections examined are pyrite, arsenopyrite, pyrrhotite, chalcopyrite, and native gold. Pyrite, arsenopyrite, pyrrhotite, and chalcopyrite occur mostly in the "greenstone", while native gold occurs in both quartz and "greenstone". No coarse native gold was seen, that present in the sections being finer than 200 mesh.

EXPERIMENTAL TESTS

The entire shipment was crushed, sampled, and assayed. This showed it to contain 1·58 ounces gold, 0·17 ounce silver per ton; 0·21 per cent arsenic, 0·02 per cent copper, 3·65 per cent iron, and 0·72 per cent sulphur.

The investigation included tests by amalgamation, concentration, and cyanidation. The results obtained showed that 67·7 per cent of the gold could be amalgamated from material ground minus 48 mesh. Straight cyanidation of minus 200-mesh ore gave 94 per cent extraction. Blanket concentration recovered from 36 per cent to 70 per cent of the gold when the ratio of concentration was varied from 15·2 : 1 to 25 : 1. Amalgamation of the concentrates recovered from 95 per cent to 67 per cent of the gold contained, depending on the grade of product so treated. When the blanket concentrate after amalgamation and the blanket tailing are reground and cyanided, an overall recovery by amalgamation and cyanidation of 95·9 per cent is obtained.

AMALGAMATION

Test No. 1

A sample of the ore was ground to pass 48 mesh and amalgamated. The amalgamation tailing was then cyanided with a sodium cyanide solution equivalent in strength to 1·0 pound of potassium cyanide per ton. Five pounds of lime per ton of ore was added to maintain protective alkalinity.

Results:

Feed	1·58 oz./ton
Amalgamation tailing	0·51 "
Recovery	67·7 per cent

After 24 hours' agitation, the cyanide tailing contained 0·125 ounce gold per ton, a recovery by amalgamation and cyanidation of 92·1 per cent.

Screen Analysis of Amalgamation Tailing:

Mesh	Weight, per cent	Assay, gold, oz./ton
− 48+100	14·2	0·60
−100+150	11·2	0·585
−150+200	17·4	0·535
−200	57·2	0·46

Test No. 2

A test similar to the above was made on a sample of the ore ground to pass 100 mesh with 79 per cent minus 200 mesh.

Results:

Feed	1·58 oz./ton
Amalgamation tailing	0·79 "
Recovery	50·0 per cent
24-hour cyanide tailing	0·15 oz/ton
Total recovery	90·5 per cent

CYANIDATION

Test No. 3

A series of tests was made by cyanidation. Samples of the ore ground to different degrees were agitated 1 : 3 dilution with a cyanide solution equivalent in strength to 1·0 pound of potassium cyanide per ton ore. Six pounds of lime per ton was added to each test.

Results:

Mesh grind	Agitation, hours	Feed, gold, oz/ton	Tailing, gold, oz/ton	Extraction, per cent	Reagent consumption, lb/ton	
					KCN	CaO
−48	24	1·58	0·14	91·1	0·6	3·8
−48	48	1·58	0·13	91·8	1·2	4·8
−100	24	1·58	0·14	91·1	0·6	4·4
−100	48	1·58	0·115	92·7	1·2	5·0
−150	24	1·58	0·13	91·8	0·6	4·2
−200	24	1·58	0·13	91·8	0·6	4·2
−200	48	1·58	0·09	94·3	1·2	6·4

Fine grinding is indicated. The highest recovery obtained was from the minus 200-mesh size.

A series of tests was undertaken to note the effect of fine grinding with lime prior to cyanidation.

Test No. 4

A sample of the ore was ground for two hours, 65 per cent solids together with 6 pounds of lime per ton. The pulp was then diluted to a ratio of 1 : 3, and cyanide added to make a solution equivalent to 1·0 pound of potassium cyanide per ton. Two pounds additional lime was added during the 24-hour agitation period.

Results:

Feed	1·58 oz./ton
Cyanide tailing	0·225 "
Extraction	85·8 per cent
Reagent consumption: KCN	0·6 lb/ton
CaO	7·6 "

Test No. 5

A test similar to the preceding one was made with the exception that after grinding, the pulp was filtered and then cyanided with a fresh solution.

Results:

Feed	1·58 oz./ton
Cyanide tailing	0·11 "
Extraction	93·1 per cent
Reagent consumption: KCN	1·9 lb./ton
CaO	9·3 "

Test No. 6

In this test no lime was added during the 2-hour grinding period. After grinding, the pulp was filtered and repulped with a cyanide solution equivalent in strength to 1·0 pound potassium cyanide per ton. Six pounds lime per ton was added during the 24-hour agitation period.

Results:

Feed	1·58 oz./ton
Cyanide tailing	0·10 "
Extraction	93·7 per cent
Reagent consumption: KCN	0·9 lb/ton
CaO	5·1 "

Test No. 7

A sample of the ore was ground in water as in Test No. 6 and then diluted to a ratio of 1 : 3. Cyanide was added to make a solution strength equivalent to 1·0 pound of potassium cyanide per ton. Lime equal to 6 pounds per ton of ore was added during the 24-hour agitation period.

Results:

Feed	1·58 oz./ton
Cyanide tailing	0·10 "
Extraction	93·7 per cent
Reagent consumption: KCN	1·2 lb/ton
CaO	9·1 "

Conditioning with lime before agitation does not increase the amount of extraction. This series does not show increased recoveries over those of Test No. 3 in which the ore was ground minus 200 mesh. A slight saving of cyanide is noted when the ore is filtered between grinding and cyanidation.

BLANKET CONCENTRATION

Test No. 8

To note the effect of removing a concentrate prior to cyanidation, a sample of the ore was ground to pass 75 per cent through 200 mesh and passed over a corduroy blanket. The tailing was then filtered and cyanided, 1 : 3 dilution, with a 1·0 pound per ton cyanide solution. Five pounds of lime was added during the agitation period.

Results:

Product	Weight, per cent	Assay		Distribution of metals, per cent	
		Gold	Arsenic		
		Oz. /ton	Per cent	Gold	Arsenic
Feed (cal.)	100·00	1·49	0·29	100·0	100·0
Blanket concentrate	4·06	25·86	3·08	70·4	42·0
Blanket tailing	95·94	0·46	0·18	29·6	58·0

Ratio of concentration .... 24·6 : 1.

The blanket tailing containing 0·46 ounce gold per ton was reduced to 0·06 ounce in 24 hours and to 0·055 ounce in 48 hours by cyanidation. This represents an extraction of 88·0 per cent of the gold in the blanket tailing.

Test No. 9

To observe the effect of amalgamating the blanket concentrate prior to cyanidation, a sample was ground as in Test No. 8 and a blanket concentrate removed. The concentrate was then amalgamated and the amalgamation tailing added to the blanket tailing. This mixture was then cyanided.

Results:

Feed	1·58 oz./ton
Amalgamation and blanket tailing combined	1·05 "
Recovery by amalgamation	33·5 per cent
24-hour cyanide tailing	0·23 oz./ton
Recovery by amalgamation and cyanidation	85·5 per cent
48-hour cyanide tailing	0·10 oz./ton
Total recovery	93·7 per cent
Reagent consumption: KCN	2·7 lb/ton
CaO	8·6 "

Test No. 10

The recovery by amalgamation in Test No. 9 was lower than that obtained in previous tests. To observe the effects of amalgamation of the blanket concentrate made from a coarse grind, a sample was ground wet to pass 48 mesh and a blanket concentrate removed. This coarse concentrate was amalgamated and the tailing from this operation was added to the blanket tailing. This mixture was then reground to pass 97 per cent through 200 mesh and cyanided.

Results:

Feed	1·58	oz./ton
Combined amalgamation and blanket tailing	0·51	"
Recovery by amalgamation	67·7	per cent
24-hour cyanide tailing	0·085	oz./ton
Total recovery, amalgamation and cyanidation	94·6	per cent
48-hour cyanide tailing	0·065	oz./ton
Total recovery	95·9	per cent
Reagent consumption: KCN	0·6	lb/ton
CaO	6·2	"

The remaining ore was used to make two mill runs with a feed rate of 100 pounds per hour, grinding in a 12 inch by 24 inch rod mill.

Mill Run No. 1

In this test, the ore was ground in the rod mill and discharged over an amalgamating plate. The grind was coarse, 2·5 per cent remaining on 48 mesh and 44·7 per cent passing 200 mesh. The amalgamation tailing was concentrated on a Wilfley table. The table concentrate was reground to pass 200 mesh and barrel-amalgamated. The tailing from this operation was then cyanided. The Wilfley table tailing also was reground and cyanided.

Results:

Assays:

Mill discharge	1·585	oz./ton
Plate tailing	0·82	"
Table concentrate	12·74	"
Table tailing	0·355	"
Recovery by amalgamation	48·3	per cent
Recovery by table concentration	29·3	"
	77·6	"

Amalgamation of reground table concentrate:

Feed	12·74	oz./ton
Tailing	2·935	"
Recovery	77·0	per cent

Cyanidation of amalgamated table concentrate:

Feed	2·935	oz./ton
24-hour cyanide tailing	0·86	"
Extraction	70·7	per cent
48-hour cyanide tailing	0·815	oz./ton
Extraction	72·2	per cent
Reagent consumption: KCN	2·4	lb./ton
CaO	22·3	"

Cyanidation of reground table tailing:

Feed	0·355	oz./ton
48-hour cyanide tailing	0·05	"
Extraction	86·0	per cent
Reagent consumption: KCN	1·2	lb./ton
CaO	29·5	"

Recoveries:

Plate amalgamation	48·3 per cent
Amalgamation of reground table concentrate	22·6 "
Cyanidation of amalgamated table concentrate	4·8 "
Cyanidation of reground table tailing	19·3 "
Total recovery	95·0 "
Ratio of table concentration, 26·6 : 1	
Average mill tailing	0·078 oz./ton

Mill Run No. 2

In this test, the mill discharge passed over corduroy blankets that were changed every two hours. Grinding was the same as in the preceding run.

The blanket concentrate was barrel-amalgamated after regrinding to minus 200 mesh and the tailing from this operation cyanided. The blanket tailing was also reground and cyanided.

Results:

Assays:

Mill discharge	1·69 oz./ton
Blanket concentrate	87·30 "
Blanket tailing	1·125 "

Blanket concentration:

Recovery in concentrate	36·2 per cent
Ratio of concentration	152·5 : 1

Amalgamation of reground blanket concentrate:

Feed	87·30 oz./ton
Amalgamation tailing	4·22 "
Recovery	95·2 per cent

Cyanidation of reground amalgamated concentrate:

24-hour cyanide tailing	0·445 oz./ton
Extraction	89·5 per cent
Reagent consumption: KCN	0·75 lb./ton
CaO	17·5 "

(48-hour agitation did not reduce the tailing below that obtained in 24 hours.)

Cyanidation of reground blanket tailing:

Feed	1·125 oz./ton
24-hour cyanide tailing	0·20 "
Extraction	82·2 per cent
48-hour cyanide tailing	0·07 oz./ton
Extraction	93·8 per cent
Reagent consumption: KCN	1·65 lb./ton
CaO	16·0 "

Amalgamation of the unground blanket tailing reduced the gold content from 1·125 ounces to 0·61 ounce per ton, indicating that the blankets should be changed more frequently than once every two hours.

Recoveries:

Amalgamation of blanket concentrate	34·5 per cent
Cyanidation of reground amalgamated concentrate	1·5 "
Cyanidation of reground blanket tailing	59·8 "
Total recovery	95·8 "
Average mill tailing	0·073 oz./ton

SUMMARY AND CONCLUSIONS

1. Amalgamation tests show that 67 per cent of the gold is freed at 48-mesh grinding. Finer grinding tends to increase tailing losses.

2. Cyanidation at 48-mesh grind gives an extraction of 91 per cent. When the fineness of grinding is increased to minus 200 mesh, the extraction is raised to 94 per cent.

3. Pre-conditioning with lime is of no apparent benefit.

4. Blanket concentration at a ratio of 24·6 : 1, as shown in Test No. 8, recovers 70 per cent of the gold; and 36 per cent when the ratio of concentration is 152·5 : 1 as in Mill Run No. 2. This latter recovery is low, owing to the long period between blanket changes; 45·8 per cent of the gold passing the blankets in this test (Mill Run No. 2) can be recovered by amalgamation.

5. Amalgamation recovers 95 per cent of the gold in the blanket concentrate. Cyanidation recovers 89·5 per cent of the gold in the residue, and 93 per cent of the gold in the blanket tailing.

6. A total recovery of 95·5 per cent can be expected by a combination of concentration, amalgamation, and cyanidation from a mill feed of the grade tested.

7. Mill Run No. 2 and Test No. 10 do not indicate that any benefit will be derived by cyaniding the amalgamated blanket concentrate and the blanket tailing in separate circuits.

The flow-sheet indicated by this investigation is one including two-stage grinding. The first mill, grinding to about 48 mesh, should discharge into a hydraulic trap to remove coarse gold. The overflow from this should pass over blankets to a thickener. The underflow from this thickener should be reground to minus 200 mesh before passing to agitators with approximately 48 hours' contact. All grinding should be done in cyanide solution.

The blanket should be changed at frequent intervals and the concentrate reground and barrel-amalgamated together with the trap cleaning. The residue from this amalgamation should be cyanided. To avoid surges in the cyanide tailing assay, this product should be handled in a separate circuit. This might also prevent a slight fouling of solution owing to the formation of soluble cyanide compounds.

Ore Dressing and Metallurgical Investigation No. 569

SILVER ORE FROM WHITE EAGLE SILVER MINES, LIMITED, CAMSELL RIVER, NORTHWEST TERRITORIES

Shipment. A shipment of silver ore consisting of 16 bags, having a net weight of 1,365 pounds, was received at the Ore Dressing and Metallurgical Laboratories on February 13, 1934, from the property of the White Eagle Silver Mines, Limited, Camsell River, Great Bear Lake district, N.W.T., and was submitted by Col. C. D. H. MacAlpine, President.

Characteristics of the Ore. Fourteen polished sections were prepared and examined microscopically. Three types of gangue material are present in the specimens, quartz, country rock, and carbonate.

Quartz predominates and varies from greyish white, fine-grained sugary quartz to comb quartz.

Carbonate, probably chiefly calcite, is next in importance.

Country rock is present as brecciated fragments contained in the quartz-carbonate assemblage. It is medium- to fine-grained, and dark grey in colour.

The ore minerals occurring are galena, gersdorffite, niccolite, rammelsbergite-safflorite, native silver, löllingite, sphalerite, tetrahedrite, chalcopyrite, argentite (?), hematite, and two unknown minerals.

Galena occurs as large massive aggregates and has a coarse crystalline structure.

Native silver shows three typical modes of occurrence as follows:—

1. Small irregular grains interstitial to the grains in fine-textured quartz.

2. Irregular grains partially replacing rammelsbergite-safflorite which occurs in fine-textured quartz.

3. Irregular grains and masses intimately associated with and invading niccolite which occurs in quartz.

It is noteworthy that no native silver was seen to occur (a) in galena, (b) in gersdorffite, or (c) associated with carbonate.

Test Work. Test work on the ore embraced table concentration, blanket concentration, barrel amalgamation of concentrates, flotation, other small-scale tests and a large-scale mill run (crushing, tabling, and flotation).

The results of the test work indicate that over 98 per cent of the silver can be concentrated by tables and flotation, and over 96 per cent of the silver in the concentrates can be recovered by barrel amalgamation.

Crushing and Sampling. The ore was crushed and sampled by standard methods and an analysis of the feed sample was as follows:—

Silver	988·03 oz/ton
Gold	0·01 "
Lead	5·27 per cent
Cobalt	0·55 "
Nickel	1·10 "
Iron	3·55 "
Copper	0·04 "
Manganese	0·63 "
Arsenic	2·05 "
Antimony	0·08 "
Sulphur	1·85 "
Silica	44·90 "
Calcium oxide	8·90 "
Magnesium oxide	5·00 "
Carbon dioxide	12·50 "

A sample of the galena was crushed and panned free from gangue, and assayed for silver, giving a result of:—

Silver	10·05 oz/ton

A screen analysis on 1,000 grammes of the feed sample was as follows:—

Mesh	Weight, per cent	Assays, silver, oz/ton	Distribution of silver, per cent
− 14+ 20	13·90	1,364·90	19·8
− 20+ 35	30·43	1,175·90	37·4
− 35+ 48	11·63	838·26	10·2
− 48+ 65	8·77	1,009 34	9·2
− 65+100	8·97	937·70	8·8
−100	26·30	526·40	14·6
	100·00	956·11*	100·0

*Calculated feed assay.

Experimental Tests

TABLE CONCENTRATION

The feed sample, 101 pounds, was screened and sized into six products.

Each of the six products was run separately over a laboratory Wilfley table.

The results of these tests follow:—

Test No. 1

Feed: 3,000 grammes −14+20 mesh.

The concentrate was re-run and a second middling product obtained. This size is too coarse for satisfactory tabling.

Product	Weight, per cent	Assays		Distribution of metals, per cent	
		Silver, oz/ton	Lead, per cent	Silver	Lead
Concentrate	14·8	5,644·19	32·18	59·9	88·6
1st middling	33·5	524·73	0·44	12·6	2·7
2nd middling	23·4	1,209·93	1·51	20·3	6·5
Tailing	28·3	351·70	0·37	7·2	2·2

Test No. 2

Feed: 3,000 grammes −20+35 mesh.

The middling was re-run.

Product	Weight, per cent	Assays		Distribution of metals, per cent	
		Silver, oz/ton	Lead, per cent	Silver	Lead
Concentrate	12·3	4,249·26	39·06	47·5	92·4
Middling	2·8	3,100·70	3·54	7·8	1·7
Tailing	84·9	577·70	0·35	44·7	5·9

Test No. 3

Feed: 3,000 grammes −35+48 mesh.

The middling was re-run.

Product	Weight, per cent	Assays		Distribution of metals, per cent	
		Silver, oz/ton	Lead, per cent	Silver	Lead
Concentrate	10·9	5,868·35	44·52	53·2	88·7
Middling	2·5	6,043·97	6·37	12·5	2·9
Tailing	86·6	475·80	0·53	34·3	8·4

Test No. 4

Feed: 3,000 grammes −48+65 mesh.

The middling was re-run.

Product	Weight, per cent	Assays		Distribution of metals, per cent	
		Silver, oz/ton	Lead, per cent	Silver	Lead
Concentrate	12·4	6,524·38	58·85	62·2	93·5
Middling	2·3	5,312·53	4·65	9·4	1·4
Tailing	85·3	432·80	0·46	28·4	5·1

Test No. 5

Feed: 2,000 grammes −65+100 mesh.

The middling was re-run.

Product	Weight, per cent	Assays		Distribution of metals, per cent	
		Silver, oz/ton	Lead, per cent	Silver	Lead
Concentrate	18·6	4,491·67	32·18	83·3	94·0
Middling	2·9	1,198·50	3·53	3·4	1·6
Tailing	78·5	167·70	0·35	13·3	5·4

Test No. 6

Feed: 3,000 grammes −100 mesh.
The middling was not re-run.

Product	Weight, per cent	Assays		Distribution of metals, per cent	
		Silver, oz/ton	Lead, per cent	Silver	Lead
Concentrate	9·3	3,289·50	38·45	55·5	50·6
Middling	9·1	1,196·30	16·44	19·7	20·6
Tailing	81·6	167·00	2·43	24·8	28·8

Slime loss, 221·1 grammes=7·3 per cent.

The table concentration tests indicate that sizes between 20 and 65 mesh are most suitable. Coarser sizing is not satisfactory and finer grinding increases the tendency for high slime. The concentrate carries the bulk of the galena and, although the middling product is low in lead, it is sufficiently high in silver to be re-run over the tables, thus yielding a high silver-lead concentrate carrying over 5,000 ounces silver per ton and a tailing carrying from 200 to 300 ounces silver per ton.

CONCENTRATION ON BLANKETS

A series of tests was made on sized ore by running over a corduroy blanket at a slope of 2·8 inches to the foot.

The results seem to indicate that blanket concentration is not satisfactory.

Test No. 1

Charge of −14+20 mesh showed this product to be too coarse for blanket concentration.

Test No. 2

Feed: 2,000 grammes −20+35 mesh.
This size also was too coarse.

Product	Weight, per cent	Assays		Distribution of metals, per cent		Ratio of concentration
		Silver, oz/ton	Lead, per cent	Silver	Lead	
Concentrate	14·5	2,132·47	17·91	28·8	54·0	6·9 : 1
Tailing	85·5	898·30	2·58	71·2	46·0	

Test No. 3

Feed: 1,000 grammes −35+48 mesh.

Product	Weight, per cent	Assays		Distribution of metals, per cent		Ratio of concentration
		Silver, oz/ton	Lead, per cent	Silver	Lead	
Concentrate	26·1	3,195·15	22·26	64·8	88·6	3·83 : 1
Tailing	73·9	612·40	1·01	35·2	11·4	

Test No. 4

Feed: 750 grammes −48+65 mesh.

Product	Weight, per cent	Assays		Distribution of metals, per cent		Ratio of concentration
		Silver, oz/ton	Lead, per cent	Silver	Lead	
Concentrate..........	34·7	2,072·17	13·76	65·0	86·3	2·88 : 1
Tailing...............	65·3	594·10	1·16	35·0	13·7	

Test No. 5

Feed: 1,000 grammes −65+100 mesh.

Product	Weight, per cent	Assays		Distribution of metals, per cent		Ratio of concentration
		Silver, oz/ton	Lead, per cent	Silver	Lead	
Concentrate..........	28·6	1,694·72	12·70	53·0	72·0	3·5 : 1
Tailing...............	71·4	600·80	1·97	47·0	28·0	

Test No. 6

Feed: 2,000 grammes −100 mesh.

Product	Weight, per cent	Assays		Distribution of metals, per cent		Ratio of concentration
		Silver, oz/ton	Lead, per cent	Silver	Lead	
Concentrate..........	16·9	1,182·40	15·78	37·9	45·8	5·9 : 1
Tailing...............	8·31	394·50	3·80	62·1	54·2	

Charges of 1,000 grammes of table tailing were run over corduroy blankets. The results were unsatisfactory, showing no concentration of silver.

Results of three of these tests are as follows:—

Test No. 7

Feed: 1,000 grammes −20+35 mesh.

Silver = 577·70 oz/ton.

Product	Weight, per cent	Assays, silver, oz/ton	Distribution of silver, per cent	Ratio of concentration
Concentrate..............................	17·5	669·60	19·8	5·72 : 1
Tailing....................................	82·5	576·20	80·2	

Test No. 9

Feed: 1,000 grammes −48+65 mesh.
Silver = 432·80 oz/ton.

Product	Weight, per cent	Assays, silver, oz/ton	Distribution of silver, per cent	Ratio of concentration
Concentrate	25·6	569·80	31·8	3·9 : 1
Tailing	74·4	418·40	68·2	

Test No. 11

Feed: 1,000 grammes −100 mesh.
Silver = 167·0 oz/ton.

Product	Weight, per cent	Assays, silver, oz/ton	Distribution of silver, per cent	Ratio of concentration
Concentrate	24·4	239·00	35·4	4·1 : 1
Tailing	75·6	140·60	64·6	

BARREL AMALGAMATION

Test No. 1

A composite charge of 1,000 grammes was made of the concentrates from the table tests. The silver content of this concentrate was 5,016·84 ounces per ton.

The charge was ground wet for 20 minutes in a pebble jar and then barrel-amalgamated with 1,000 grammes of mercury for 6 hours. On separating the amalgam from the pulp it was found that insufficient mercury had been added. An additional 700 grammes was added and agitation continued for a short period.

No sickening of the mercury was apparent. There was a tendency, however, for the fine galena particles to adhere closely to the amalgam during separation. No additions of cyanide or soda ash were made during the amalgamation.

Weight of charge	1,000 grammes
Weight of amalgamation tailing	834 "
Silver assay on amalgamation tailing	234·30 oz/ton
Silver recovery	95·3 per cent

Barrel amalgamation does not appear to offer any serious difficulty.

Screen Test of Amalgamation Tailing:

Mesh	Weight, per cent
+100	0·8
+150	3·4
+200	13·4
−200	82·4
Total	100·0

Test No. 2

This test was carried out on raw ore. A 1,000-gramme charge was ground wet for 20 minutes and then transferred to an Abbé jar and agitated for 6 hours with 1,000 grammes of mercury.

The separation of the mercury amalgam from the ore was carried out in a hydraulic classifier. There was no indication of sickening of the mercury.

Assay of feed sample	Silver,	988·03 oz/ton
Assay of amalgamation tailing	Silver,	89·10 "
Recovery of silver		91·0 per cent

Screen Test of Amalgamation Tailing:

Mesh	Weight, per cent
+ 65	0·25
+100	4·05
+150	10·50
+200	22·90
−200	62·30
Total	100·00

JIG TEST

A 2,000-gramme charge of −14+20-mesh ore was run through a Richards pulsating jig. There was a clean galena product obtained high in silver. The overflow, although low in lead, contained much silver.

The test indicates the unsuitability of jigging for this ore at the above size.

FLOTATION TESTS

Test No. 1

This test was carried out on table tailing from Table Test No. 6 (−100 mesh).

Silver	167·0 oz/ton
Lead	2·43 per cent

Additions to cell:

(Conditioned in cell for 15 minutes)

Table tailing	750 grammes
Coal-tar creosote	20 drops
Sodium silicate	2·66 lb/ton
Sodium ethyl xanthate	0·53 "
Pine oil	0·13 lb/ton

Product	Weight, per cent	Assays		Distribution of metals, per cent		Ratio of concentration
		Silver, oz/ton	Lead, per cent	Silver	Lead	
Concentrate	15·22	964·20	14·16	90·5	88·6	6·57 : 1
Tailing	84·78	18·10		9·5	11·4	
	100·00				100·0	

Summary of Silver Recovery by Tables and Flotation:

Recovery of silver on table	75·2 per cent
Recovery of silver by flotation of table tailing, 90·5 per cent of 24·8 per cent	22·4 "
Overall recovery of silver	97·6 "

Test No. 2

This test was carried out on tailing from Blanket Test No. 6 (−100 mesh).

Silver	394·5 oz/ton
Lead	3·80 per cent

Additions to cell:

(Conditioned in cell for 15 minutes)

Blanket tailing	1,000 grammes
Coal-tar creosote	20 drops
Sodium silicate	1 lb/ton
Sodium ethyl xanthate	0·4 "
Pine oil	0·05 lb/ton

Product	Weight, per cent	Assays		Distribution of silver, per cent	Ratio of concentration
		Silver, oz/ton	Lead, per cent		
Concentrate	15·9	2,391·20	25·4	95·5	6·28 : 1
Tailing	84·1	21·10	....	4·5	
Total	100·0	....	....	100·0	....

Test No. 3

This was a test on the tailing from the barrel amalgamation test of the table concentrate and was run to float off the lead and leave a product containing the bulk of the nickel and cobalt.

Additions to cell:

Tailing	500 grammes
Sodium cyanide	0·2 lb/ton
Sodium ethyl xanthate	0·1 "
Soda ash	2·0 "
Pine oil	0·2 "

Product	Weight, per cent	Lead, per cent	Nickel, per cent	Cobalt, per cent	Distribution of metals, per cent			Ratio of concentration
					Lead	Nickel	Cobalt	
Concentrate	32·3	64·98	2·60	1·00	52·4	17·4	11·1	3·09 : 1
Tailing	67·7	28·08	5·91	3·82	47·6	82·6	88·9	

Results show a very low-grade concentrate of nickel and cobalt.

SILVER RECOVERY FROM TABLE CONCENTRATES BY DIRECT SMELTING AND CUPELLATION

This test was run to determine the possibility of smelting the concentrate into a lead-silver bullion and by subsequent cupellation obtaining a silver bullion. The lead in the concentrate was utilized in the production of the crude silver-lead bullion.

A 200-gramme charge of the concentrate was first roasted at a dull red heat for 20 minutes to reduce the sulphur content. The calcine was then melted, using a flux made up as follows:—

Soda carbonate	390 grammes
Borax	50 "
Iron nails	(5)
Argol (for reduction)	35 grammes

The bullion obtained from the melt was then scorified to reduce the lead further and finally cupelled.

The concentrate assayed as follows:—

Silver	5,037·44 oz/ton
Lead	42·21 per cent

Silver in calcine	33·68 grammes
Lead in calcine	84·42 "
Silver recovered as bullion	28·865 "
Recovery	85·7 per cent

The recovery was low and this was probably due to insufficient roasting. Sulphur resulting in the formation of a matte during the melt would account for a loss of silver in the slag.

The test indicates, however, the possibility of developing a smelting method whereby the lead available in the concentrate could be utilized for the recovery of silver bullion.

Mill Run

Test No. MR-1

The ore, crushed to −4 mesh, was fed to a 12-inch by 24-inch rod mill. The mill discharge was pumped to a Callow screen (10 mesh), the oversize being returned to the mill, the undersize passing to a Wilfley concentrating table.

The sand tailing and slime were pumped into a cone tank. The sand tailing was drawn off into a settling-tank, collected, and dried. The slime remaining in the cone tank was run through a filter press and the solids recovered.

The grinding circuit used approximates closely the results obtainable from a stamp battery.

The table middling was re-run over the tables, and the mill, pump, and elevator clean-ups were also tabled.

Data of Test:

Mill feed: Silver = 988·03 oz/ton
Lead = 5·34 per cent
Table feed: Silver = 755·61 oz/ton
Rate of feed to rod mill = 160·5 lb/hour
Length of table stroke = 0·5 inch

The results of the run are condensed and tabulated in the following table:—

Products	Weight, per cent	Assays		Distribution of metals, per cent		Ratio of concentration
		Silver, oz/ton	Lead, per cent	Silver	Lead	
Concentrate............	15·41	6,488·17	19·90	83·61	57·4	6·49 : 1
Tailing................	84·59	231·68	2·69	16·39	42·6	
	100·00			100·00	100·0	

Concentration on the table appears to be satisfactory. A minimum of fines during grinding is essential for the best recovery.

The tailing was dried to a moisture content of from 5 to 8 per cent and reground and floated.

Screen Tests:

Table Concentrate		Table Tailing	
Mesh	Weight, per cent	Mesh	Weight, per cent
+ 14....................	7·7	+ 14....................	2·2
+ 20....................	10·6	+ 20....................	6·6
+ 28....................	9·7	+ 28....................	8·8
+ 35....................	12·7	+ 35....................	10·9
+ 48....................	11·4	+ 48....................	11·9
+ 65....................	9·5	+ 65....................	10·2
+100....................	9·4	+100....................	12·6
+150....................	7·1	+150....................	8·2
+200....................	8·8	+200....................	10·8
−200....................	13·1	−200....................	17·8
	100·0		100·0

FLOTATION OF TABLE TAILING

Test No. MR-2

The tailing was fed to the rod mill, containing 300 pounds of rods, at a feed rate of 105 pounds per hour. The mill discharged into an Akins classifier. The classifier overflow was pumped into a conditioner tank from which it was fed to the flotation cells. The cell tailing was run over a Butchart table.

The reagents added were as follows:—

To the rod mill:

Sodium silicate.. 1·0 lb/ton
Coal-tar creosote...................................... 1·25 "

To the conditioner tank:

Sodium ethyl xanthate.................................. 0·4 lb/ton

To the cells:

Pine oil... 0·05 lb/ton

It was found that the coal-tar creosote produced too heavy a froth, resulting in a rather low-grade concentrate and the particles were heavily oiled.

The rest of the tailing was run the following day, omitting the coal-tar creosote.

Results:

Products	Assays	
	Silver, oz/ton	Lead, per cent
Classifier overflow	172·40	....
Conditioner overflow	146·30	3·79
Flotation concentrate	1,949·0	27·78
Flotation tailing	10·75	Nil
Table concentrate	19·40	....
Table tailing	10·48	....

Screen Test Classifier Overflow:

Mesh	Weight, per cent
+ 65	0·6
+100	4·5
+150	9·9
+200	23·5
−200	61·5
	100·0

Calculations:

$$\text{Ratio of concentration} = \frac{1949 - 10{\cdot}75}{146{\cdot}3 - 10{\cdot}75} = 14{\cdot}3 : 1$$

$$\text{Recovery of silver} = \frac{100 \times 1949\ (146{\cdot}3 - 10{\cdot}75)}{146{\cdot}3\ (1949 - 10{\cdot}75)} = 93{\cdot}1 \text{ per cent}$$

Test No. MR-3

The reagents added were as follows:

To the mill:

Sodium silicate .... 1·0 lb/ton

To the conditioner tank:

Sodium ethyl xanthate .... 0·4 "

To the cells:

Pine oil .... 0·3 "

Results:

Products	Assays	
	Silver, oz/ton	Lead, per cent
Classifier overflow	202·8	....
Conditioner overflow	191·5	2·96
Flotation concentrate	2,021·0	28·67
Flotation tailing	13·68	0·15
Table concentrate	19·40	....
Table tailing	12·90	....

Screen Test Classifier Overflow:

Mesh	Weight, per cent
+ 65	0·1
+100	3·0
+150	7·8
+200	22·8
−200	66·3
	100·0

Calculations:

$$\text{Ratio of concentration} = \frac{2021 - 13 \cdot 68}{191 \cdot 5 - 13 \cdot 68} = 11 \cdot 28 : 1$$

$$\text{Recovery of silver} = \frac{100 \times 2021\ (191 \cdot 5 - 13 \cdot 68)}{191 \cdot 5\ (2021 - 13 \cdot 68)} = 93 \cdot 4 \text{ per cent}$$

Calculations on Silver Recovery by Table Concentration and Flotation

Silver recovery on tables	83·61 per cent
Silver recovery by flotation of table tailing = 93·4 per cent of 16·39 per cent	15·31 "
Overall recovery	98·92 "

The flotation tests indicate that the silver minerals float satisfactorily, yielding a fair grade concentrate.

AMALGAMATION OF SILVER CONCENTRATES

Test No. 1

A sample, 500 grammes, of the table concentrate from Test No. MR-1 was ground in a pebble jar for 20 minutes. The ground concentrate was then barrel-amalgamated with 1,000 grammes of mercury in water for 6 hours. The amalgam was separated in a hydraulic classifier. No sickening of the mercury occurred.

Screen Test Amalgamation Tailing:

Mesh	Weight, per cent
+150	1·0
+200	7·1
−200	91·9
	100·0

Assay of feed: Silver = 6,808·30 oz/ton
Assay of tailing: Silver = 242·30 "
Recovery = 96·4 per cent

A microscopic examination of the tailing showed a small amount of native silver as tiny grains included within gangue or metallic minerals. A very minor amount occurs as tiny, rare, free grains. No minerals that could be silver minerals were seen.

Test No. 2

This test was on 1,000 grammes of flotation concentrate from Test No. MR-3. The charge was barrel-amalgamated with 1,000 grammes of mercury and 10 pounds soda ash per ton and water for six hours. Results were not satisfactory.

Feed assay:	Silver	=	1,783·0 oz/ton
Amalgamation tailing:	Silver	=	1,115·1 "
Recovery		=	37·4 per cent

Test No. 3

This test was on 1,000 grammes of flotation concentrate from Test No. MR-2 and was carried out similarly to amalgamation Test No. 2.

Feed assay:	Silver	=	1,552·40 oz/ton
Amalgamation tailing:	Silver	=	1,304·60 "
Recovery		=	15·9 per cent

The results of barrel amalgamation on the flotation concentrate from the table tailing show a very low silver recovery. This may be due to several causes. The oiling of the silver particles and the action of xanthate on the silver may have a retarding influence on the action of the mercury on the silver. This is supported by the results obtained on the two concentrates. In Test No. MR-2 coal-tar creosote was used as a flotation reagent, whereas in Test No. MR-3 it was discontinued. Sufficient coal-tar creosote, however, remained in the circuit to influence amalgamation.

A microscopic examination of this tailing showed native silver rather abundant as irregular grains, which are free and appear to have been somewhat corroded. A moderate amount of native silver is enclosed as tiny grains in gangue and metallic minerals. No minerals that could be silver minerals were seen. Much galena is present and from a previous assay is known to carry a small amount of silver.

As a result of the microscopic examination, a further amalgamation test was made in which the concentrate was ground in a pebble jar with soda ash prior to barrel amalgamation.

The results of this test showed little improvement.

In order to effect a further concentration of silver in amalgamation tailing of the table concentrate another test, duplicating barrel amalgamation Test No. 1, was run and the tailing was floated. The results are as follows:

Test No. 4

A sample, 1,000 grammes, of table concentrate was ground wet for 20 minutes and then barrel-amalgamated with 2,000 grammes of mercury.

Silver in feed	6,808.30 oz/ton
Silver in tailing	212·30 "
Recovery by amalgamation	96·8 per cent

Reagents to flotation of tailing:

Soda ash	2·0 lb/ton
Sodium ethyl xanthate	0·4 "
Pine oil	0·1 "

Product	Weight, per cent	Assays	Distribution of silver, per cent	Ratio of con- centration
		Silver, oz/ton		
Concentrate	50·1	384·60	89·3	2 : 1
Tailing	49·9	46·10	10·7	
	100·0		100·0	

Silver recovery by amalgamation	96·8 per cent
Silver recovery by flotation of amalgamation tailing, 89·3 per cent of 3·2 per cent	2·8 "
Overall recovery	99·6 "

Test No. 5

A 1,000-gramme charge of flotation concentrate from Test No. MR-3 was ground wet with 10 pounds soda ash per ton for 15 minutes. The pulp was then barrel-amalgamated with 1,000 grammes of mercury, 4 pounds soda ash per ton, and 0·2 pound sodium cyanide, for 6 hours.

Silver in feed	1,783 oz/ton
Silver in tailing	1,080·80 "
Recovery	39·3 per cent

Screen Test on Amalgamation Tailing:

Mesh	Weight, per cent
+150	0·4
+200	3·5
−200	96·1
	100·0

CONCLUSIONS

The results of the test work on the White Eagle silver ore indicate that the silver is almost entirely present as native silver.

Crushing in stamp batteries followed by table concentration will yield a high-grade silver concentrate with a high silver recovery. By flotation of the table tailing, 93·4 per cent of the silver in this tailing is recovered. The overall recovery by tabling and flotation was 98·92 per cent.

Barrel amalgamation of the table concentrate offers no difficulty, 96·8 per cent of the silver being recovered as amalgam. By flotation of the amalgamation tailing a further 2·8 per cent of silver was obtained in a lead concentrate carrying 384·6 ounces silver per ton.

Amalgamation of the flotation concentrate appears to present some difficulty, as has already been pointed out. It might be considered advisable to retain all the flotation concentrate and treat it by a smelting process or ship it direct to a smelter.

Ore Dressing and Metallurgical Investigation No. 570

GOLD ORE FROM AMISK (BEAVER) LAKE, SASKATCHEWAN

Shipment. A shipment of four sacks of ore was received October 20, 1933. Two sacks contained Samples Nos. 1 and 2 mixed, the net weight being 111 pounds. Sample No. 3 was contained in one sack and weighed 77 pounds. Sample No. 4 was contained in one sack and weighed 56 pounds. The samples were submitted by Dr. J. F. Wright, Geological Survey, Victoria Museum, Ottawa.

Character of the Ore. Samples Nos. 1 and 2 are of a rusty, partly decomposed, fine-grained material derived from the weathering of quartz-feldspar porphyry, which is highly jointed, altered, and mineralized with finely-disseminated sulphides and stringers of vein quartz and carbonates. No. 3 is of unweathered andesitic lava highly altered and carrying disseminated sulphides, quartz, and carbonates. No. 4 is of rusty decomposed material derived from the weathering of material similar to No. 3. Samples Nos. 1 and 2 are from the Sonora prospect and Nos. 3 and 4 from the Amisk Gold Syndicate prospect.

The average assays of the three samples are as follows:—

Samples Nos. 1 and 2	0·0625 oz/ton
Sample No. 3	0·435 "
Sample No. 4	0·52 "

The metallic minerals in the polished sections are arsenopyrite, pyrite, pyrrhotite, and chalcopyrite named in their order of abundance, and, in addition, a grey mineral occurring in needles, possibly ilmenite, and native gold. The arsenopyrite occurs in irregular grains and well defined crystals, the pyrite in irregular grains, and the pyrrhotite in small grains in both the arsenopyrite and pyrite. Three grains of native gold were seen in the specimens studied, one in a gangue mineral, another in pyrite, and the third in arsenopyrite. The gold is exceedingly fine, that in the gangue being 3·0 microns, that in pyrite 1·5 microns, and that in arsenopyrite 1·0 micron. The fine character of the gold and the apparent association of most of it with sulphides makes this ore rather complex to treat, and only a very small recovery can be expected by washing the partly decomposed rusty capping material, as in Samples Nos. 1 and 2, over mercury plates.

Experimental Tests

The work on Samples Nos. 1, 2, and 4 was limited to barrel amalgamation tests, as any method by which this material could be economically treated would depend on the gold being finally collected in mercury. From Samples Nos. 1 and 2, 52·0 per cent of the gold was recovered as amalgam, and from Sample No. 4 only 32·7 per cent of the gold was recovered in this way.

The fresh rock sample was treated by cyanidation, amalgamation, and flotation. Maximum extraction obtained by cyanidation was 93·1 per cent of the gold when the ore was ground dry all through 200 mesh. By barrel amalgamation 54·0 per cent of the gold was extracted when the ore was ground dry all through 100 mesh. By flotation of the ore 93·1 per cent of the gold was recovered in a concentrate amounting to 29·6 per cent of the weight of feed used and assaying 1·16 ounces per ton in gold. By amalgamating the ore and floating the amalgamation tailing, a total recovery of 92·6 per cent of the gold was obtained. Of this, 61·4 per cent was recovered by amalgamation and 31·2 per cent in a flotation concentrate amounting to 28·2 per cent of the weight of feed used and assaying 0·48 ounce per ton in gold.

Details of the tests follow:—

Samples Nos. 1 and 2

Test No. 1—Amalgamation

The ore, dry crushed through 14 mesh, was amalgamated with mercury in a jar mill for 30 minutes. The tailing was assayed for gold. The ore was not crushed any finer because it was felt that this could not be done economically in practice.

Summary:

Feed sample	0·0625 oz/ton
Amalgamation tailing	0·03 "
Recovery	52·0 per cent

Sample No. 4

Test No. 1—Amalgamation

This test was carried out in exactly the same way as Test No. 1 on Samples Nos. 1 and 2.

Summary:

Feed sample	0·52 oz/ton
Amalgamation tailing	0·35 "
Recovery	32·7 per cent

Sample No. 3

Tests Nos. 1 to 8—Cyanidation

Four lots of the ore were crushed dry to pass through 48-, 100-, 150-, and 200-mesh screens respectively. A sample of each lot was agitated in cyanide solution, 2·0 pounds KCN per ton, for periods of 24 and 48 hours. The tailings were assayed for gold.

Summary:

Feed sample: 0·435 oz/ton

Test No.	Mesh	Period of agitation, hours	Tailing assay, gold, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
					KCN	CaO
1	− 48	24	0·06	86·2	0·70	13·0
2	−100	24	0·045	89·7	1·00	15·8
3	−150	24	0·035	92·0	1·00	17·3
4	−200	24	0·03	93·1	1·00	21·3
5	− 48	48	0·06	86·2	1·3	13·8
6	−100	48	0·04	90·8	1·3	17·3
7	−150	48	0·04	90·8	1·3	18·5
8	−200	48	0·045	89·7	1·9	23·8

Tests Nos. 9 and 10—Amalgamation and Cyanidation

Samples of the ore, crushed dry to pass through 48- and 100-mesh screens, were amalgamated with mercury in jar mills for 30 minutes. The tailings were assayed for gold.

Summary Tests Nos. 9 and 10: Amalgamation:

Feed sample: 0·435 oz/ton

Test No.	Mesh	Tailing assay, gold, oz/ton	Recovery, per cent
9	− 48	0·24	44·8
10	−100	0·20	54·0

Samples of the amalgamation tailing were agitated in cyanide solution, 2·0 pounds KCN, per ton for 24 hours. The cyanide tailings were assayed for gold.

Summary Tests Nos. 9 and 10—Cyanidation:

Test No.	Amalgamation tailing assay, gold, oz/ton	Cyanide tailing assay, gold, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
				KCN	CaO
9	0·24	0·045	44·8	1·00	14·6
10	0·20	0·05	34·5	1·10	16·6

Test No. 11—Flotation

The ore, at −14 mesh, was ground in a ball mill for 30 minutes and then floated.

Charge to ball mill:

Ore	2,000	grammes
Water	1,500	c.c.
Soda ash	7·0	lb/ton

Reagents to cell:

Potassium amyl xanthate	0·20 lb/ton
Pine oil	0·10 "
Copper sulphate	1·0 "

Summary Test No. 11:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Flotation concentrate	29·6	1·16	93·10
Average flotation tailing	70·4	0·036	6·90
Feed (cal.)	100·0	0·37	100·0

Screen Analyses of Flotation Tailing:

Product	Weight, per cent	Assay, gold, oz/ton
+200 mesh	7·2	0·11
−200 mesh	92·8	0·03
Average tailing (cal.)	100·0	0·036

Test No. 12—Amalgamation and Flotation

The ore was ground in a ball mill for 30 minutes and the pulp amalgamated with mercury in a jar mill for 30 minutes. The amalgamation tailing was then floated.

Charge to ball mill:

Ore	2,000 grms. at −14 mesh
Water	1,500 c.c.

Reagents to cell:

Soda ash	5·0 lb/ton
Potassium amyl xanthate	0·20 "
Pine oil	0·10 "
Copper sulphate	1·0 "

Summary Test No. 12:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Flotation concentrate	28·2	0·48	80·7
Flotation tailing	71·8	0·045	19·3
Amalgamation tailing (cal.)	100·0	0·168	100·0

Recovery by amalgamation	61·4 per cent total gold
Recovery in flotation concentrate	31·2 " "
Total recovery	92·6 " "

SUMMARY

Samples Nos. 1 and 2. These samples were collected from the Sonora deposit under the direction of E. B. Webster, and represent a large tonnage of oxidized material that has been determined as pre-Glacial in age. The deposit is not considered extensive enough for large-scale mining with mechanical shovels, but it has been suggested that if some simple method requiring an inexpensive plant could be found that would recover sufficient gold, prospectors and unemployed men might be encouraged to work this oxidized material.

The test work described was to determine if the gold could be recovered by some method such as by sluice boxes equipped with either riffles or blankets or by amalgamation. The microscopic examination and the test work performed show that the gold is present in a very fine state, and that blankets, riffles, or traps would not recover it. However, it was found that over half the gold could be recovered by amalgamation, provided the material were crushed to 14 or 20 mesh before passing it over the amalgamation plates. Some type of impact amalgamator, such as the Gibson, would be more efficient on this class of ore than stationary plates.

The possibility of prospectors or unemployed making wages by working this material is not very encouraging. Taking gold at $35.00 per ounce, the recoverable value per ton would be only $1.10.

Sample No. 3. This material represents the fresh rock underlying the oxidized material on the Amisk deposit. The grade, as shown by assay, is 0·43 ounce gold per ton. The principal sulphide in the ore is arsenopyrite, but the experimental tests show that it does not make the ore refractory.

Cyanidation gave recoveries of over 90 per cent.

Sample No. 4. This sample was collected from the Amisk deposit by George Bottoms, mining engineer for the Amisk Syndicate, London.

The grade of the material is much higher than that of the ore from the Sonora deposit, but the experimental tests show a much lower percentage recoverable by amalgamation. The grade of the material, according to the assay of the sample, is 0·52 ounce per ton and the amount recoverable by amalgamation was only 32·7 per cent—which is equivalent to about $6.00 per ton, when the ore is ground to 14 mesh.

It might be possible, therefore, to work this higher grade material profitably by using some type of simple grinding mill followed by amalgamation.

Ore Dressing and Metallurgical Investigation No. 571

CONCENTRATION AND CYANIDATION OF GREENE STABELL ORE

Shipment. Shipments consisting of two lots of ore and one of flotation concentrate were received. The first lot of 30 pounds was received from H. A. Kee, April 3, 1933, shipped from the Dorr Company of New York. The second lot of ore weighing 76 pounds and one lot of flotation concentrate which arrived February 15, 1934, was received from B. H. Budgeon, Manager of the Greene Stabell mine.

Experimental Tests

The experimental tests were to determine what results would be obtained by cyanidation. The later shipment of ore was for checking the results obtained on previous shipments. The concentrate was examined to note if any gold contained in it could be recovered by amalgamation.

The report is divided into two parts, "A" and "B". The first part deals with the shipment of H. A. Kee, and the second part with that of B. H. Budgeon.

Part A

The sample was crushed and sampled and found to contain 0·61 ounce of gold per ton and 0·50 per cent copper.

CYANIDATION OF RAW ORE

Test No. 1

A sample of the ore was ground to minus 100 mesh and cyanided for 24 hours, 1 : 3 dilution with cyanide solution maintained at approximately 1 pound KCN per ton. Lime was added at intervals so as to maintain an alkalinity of about 0·3 pound lime per ton of solution. After completion the test was filtered and water-washed with 10 per cent the volume of solution.

This solution was added to a fresh sample of ore, brought to 1 pound KCN per ton and sufficient lime added. This was repeated for 7 cycles. After the fourth cycle, the solution was precipitated with zinc dust.

Cycle No.	Feed, gold, oz/ton	Tailing, gold, oz/ton	Extraction, per cent	Reagent consumption	
				KCN	CaO
1	0·61	0·05	91·8	6·0	7·0
2	0·61	0·05	91·8	6·9	6·0
3	0·61	0·09	85·2	5·7	6·1
4	0·61	0·085	86·1	5·7	6·0
5	0·61	0·08	86·9	5·4	5·4
6	0·61	0·065	89·3	4·6	5·4
7	0·61	0·065	89·3	4·95	5·5

At the end of the run, the solution was analysed and found to contain 2 pounds copper per ton.

It will be noted that the copper sulphides in this ore are strong cyanicides consuming from 5 to 7 pounds KCN per ton.

From each 100 tons of ore treated approximately 28 pounds metallic copper would be precipitated in the zinc presses.

FLOTATION AND CYANIDATION

Test No. 2

In this test, the ore was ground minus 65 mesh with 64 per cent minus 200 mesh. Three pounds lime per ton was added to the mill. After grinding, 0·10 pound sodium xanthate and 0·06 pound pine oil were added and a rougher concentrate removed. This was cleaned once and the cleaner tailing added to the flotation tailing which was then cyanided after filtering with a 1 pound KCN per ton solution, 1 : 3 dilution. Lime was added to supply protective alkalinity.

Flotation:

Product	Weight, per cent	Assay		Distribution of metals, per cent	
		Gold, oz/ton	Copper, per cent	Gold	Copper
Feed	100·00	0·61	0·50	100·00	100·00
" (cal.)		0·54	0·51		
Concentrate	1·39	14·03	27·34	36·1	74·8
Tailing	98·61	0·35	0·13	63·9	25·2

Ratio of concentration, 72 : 1

Cyanidation:

Agitation, hours	Feed, gold, oz/ton	Tailing, gold, oz/ton	Extraction, per cent	Total extraction, flotation and cyanidation	Reagent consumption, lb./ton	
					KCN	CaO
24	0·35	0·09	74·3	85·2	2·4	4·1
48	0·35	0·045	87·1	92·6	2·7	4·1

The results show that after floating off a concentrate, which assays 14 ounces gold and 27 per cent copper, with a ratio of concentration of 72 : 1, the residue containing 0·35 ounce gold can be cyanided to 0·045 ounce, a total recovery of 92·6 per cent of the gold. The cyanide consumed is 2·7 pounds KCN per ton as against 5 to 7 pounds on the raw ore. A total of 7 pounds lime per ton for the combined process is noted.

AMALGAMATION OF FLOTATION CONCENTRATE

Test No. 3

In order to determine the amount of gold that could be recovered by barrel amalgamation of the flotation concentrate a large sample of the ore was ground with 2 pounds lime per ton to pass 65 mesh with 72 per cent minus 200 mesh. A rougher concentrate was then removed by adding 0·10 pound sodium xanthate and pine oil. This was not cleaned but ground 98·7 per cent minus 325 mesh and amalgamated.

Product	Weight, per cent	Assay		Distribution of metals	
		Gold, oz/ton	Copper, per cent	Gold	Copper
Feed	....	0·61	0·50	....	....
Flotation concentrate (cal.)	....	9·96	....	....	....
Amalgamation	....	....	....	31·2	....
Flotation concentrate amalgamation	4·3	5·53	10·40	39·0	94·0
Flotation tailing	95·7	0·19	0·03	29·8	6·0

The results indicate that 70·2 per cent of the gold was recovered in the rougher flotation concentrate. By amalgamation 31·2 per cent of the gold in the feed was recovered from this, leaving a product assaying 5·53 ounces gold and 10·4 per cent copper.

The investigation shows that straight cyanidation of the ore is not practical, as from 5 to 7 pounds of cyanide and 5½ to 7 pounds lime per ton will be consumed. In addition a large amount of copper will go into solution, resulting in fouled solutions and very low-grade bullion.

It therefore becomes necessary to remove most of the copper before attempting cyanidation. A high-grade copper concentrate can be made as shown in Test No. 2. Approximately 40 per cent of the gold can be recovered by barrel amalgamation as shown in Test No. 3. The residue from amalgamation will have to be shipped to a refinery.

Whether barrel amalgamation should be carried out can best be determined by a study of the difference between treatment charges on a high-grade concentrate and those on one from which 40 per cent of the gold has been removed, plus cost of barrel amalgamation, and the advantages of ready cash from bullion recovered.

Flotation followed by cyanidation of the flotation tailing will result in an overall recovery of over 90 per cent of the gold in the ore and cyanide consumption should not be in excess of 2 to 3 pounds per ton. Seven pounds of lime per ton should be sufficient for flotation and cyanidation.

Grinding should be done in water with $1\frac{1}{2}$ to 3 pounds lime per ton. The classifier overflow should enter a conditioning tank where the flotation reagents are added. If mechanical cells are used, frothing oil should be added to the cells.

The flotation tailing should be thickened and filtered, repulped with cyanide solution containing approximately 1 pound KCN per ton, and lime added to maintain protective alkalinity. The pulp should be agitated for at least 48 hours, thickened, filtered, repulped and refiltered before discarding.

The precipitating apparatus should include a clarifying tank, Merrill Crowe vacuum tank, zinc dust feeder, and precipitate press.

The flotation concentrate will require a thickener, preferably a Genter, and a small filter followed by a drier before bagging the product for shipment.

Adequate apparatus for flotation control should be provided, such as specific gravity cans, balance, measuring cylinders, flotation test machine, and sample filters.

Part B

Mill Concentrate. The 20-pound lot of concentrate was sampled, assayed, and found to contain 27·3 per cent copper, 27·7 per cent iron, 28·6 per cent sulphur, and 9·81 ounces gold per ton.

Amalgamation failed to recover any appreciable quantity of gold from this product, either as received or after regrinding.

The sample apparently had been amalgamated prior to its arrival.

Mill Feed. The sample submitted was sampled and assayed and found to contain 0·84 per cent copper, 0·52 ounce gold per ton.

FLOTATION AND CYANIDATION

Test No. 1

A sample of the ore was ground wet to pass 68 per cent through 200 mesh. Three pounds of lime per ton was added to the mill during the grinding period.

The pulp was transferred to a flotation machine, 0·10 pound sodium xanthate and 0·07 pound pine oil were added and a concentrate removed. This concentrate was not cleaned.

The flotation tailing was then filtered and cyanided for 48 hours, 1 : 3 dilution, with a 1·5 pound per ton KCN solution and 6 pounds of lime per ton ore.

Product	Weight, per cent	Assay		Distribution, per cent	
		Gold, oz/ton	Copper, per cent	Gold	Copper
Feed (cal.)	100·00	0·51	0·85	100·0	100·0
Concentrate	3·56	7·32	22·60	51·0	94·3
Tailing	96·44	0·26	0·05	49·0	5·7

Cyanidation of the flotation tailing for 48 hours reduced the gold content to 0·055 ounce per ton with a consumption of 2·1 pounds sodium cyanide and 5·2 pounds lime per ton ore.

Test No. 2

An additional test was made to note the effects of finer grinding. A sample was ground to pass 91 per cent through 200 mesh. Other conditions were the same as in Test No. 1, except that 0·05 pound sodium xanthate was used in place of 0·10 pound.

Product	Weight	Assay		Distribution, per cent	
		Gold, oz/ton	Copper, per cent	Gold	Copper
Feed (cal.)	100·00	0·54	0·89	100·0	100·0
Concentrate	4·06	8·76	21·15	65·0	95·7
Tailing	95·94	0·20	0·04	35·0	4·3

Cyanidation for 48 hours reduced the flotation tailing to 0·03 ounce gold per ton, a recovery of 94·7 per cent by flotation and cyanidation, 1·9 pounds sodium cyanide and 6·0 pounds lime per ton were consumed.

SUMMARY AND CONCLUSIONS

Fine grinding is necessary to obtain a low tailing for cyanidation.

The sample responds to flotation and cyanidation in the same manner as all previous lots tested. The ratio of copper to gold is somewhat higher than in previous shipments. It is fast floating and no trouble should be encountered in making a tailing low in copper. The percentage of this metal in the feed to the cyanide plant has a direct bearing on the cyanide consumption.

Part of the gold can be dropped from the rougher flotation concentrate by cleaning. The cleaner tailing may be added to the flotation tailing if low in copper without materially increasing reagent consumption.

Amalgamation, as indicated in former test work, will recover part of the gold in the concentrate. This investigation, therefore, adds nothing to former reports on the ore from this mine. Previous test work and recommendations are herein reiterated[1].

[1] Mines Branch, Dept. of Mines, Canada, Invest. Ore Dressing and Metallurgy, 1929, Rept. 720, p. 116; Rept. 736 (1932), p. 123.

Ore Dressing and Metallurgical Investigation No. 572

SINTERING TESTS ON FLOTATION CONCENTRATE FROM THE BEATTIE MINE, DUPARQUET TOWNSHIP, ABITIBI COUNTY, QUEBEC

Shipment. A sample shipment of flotation concentrate from the Beattie mine, Duparquet township, Abitibi county, Quebec, weighing 300 pounds, was submitted for sintering tests to determine how much of the arsenic could be removed and the character of the sinter produced.

Experimental test work conducted to date on Beattie ore has shown that the ore is not amenable to cyanidation, only about 65 per cent of the gold being recoverable. Cyanidation of the flotation concentrate gives about the same recovery of the gold in the concentrate. The gold is very closely associated, for the most part, with the pyrite and arsenopyrite minerals, which are very finely disseminated through the gangue rock, requiring very fine grinding to free them. Concentration by flotation of the finely ground ore, with the production of a sulphide concentrate, is the first step in its treatment. At present the concentrate is being shipped to a smelter.

Two methods of treatment are suggested for the concentrate. Firstly, roasting in a suitable roasting furnace and cyanidation of the roasted concentrate, and, secondly, roasting with or without copper concentrate, smelting in a reverberatory furnace with copper ore or concentrate to matte and converting to blister copper; or sintering with copper concentrate, smelting in a blast furnace and converting to blister copper. The sintering tests were conducted to obtain data on the latter method of treatment.

Experimental Sintering Tests

The concentrate used in the preliminary tests was obtained from some on hand from the flotation of the ore in the Ore Dressing and Metallurgical Laboratories and assayed 14·54 per cent sulphur and 2·50 per cent arsenic. A screen test showed 84·4 per cent to pass a 200-mesh screen.

The concentrate used in the later tests was from recent mine production and assayed 39·71 per cent sulphur and 5·85 per cent arsenic.

The copper concentrate used for the mix was some on hand at the Ore Dressing and Metallurgical Laboratories, and assayed 11·87 per cent copper and 31·26 per cent sulphur. A screen test showed 64·4 per cent to pass a 200-mesh screen.

Series A

UNMIXED BEATTIE CONCENTRATE (14·54 PER CENT SULPHUR GRADE)

This series of tests was made in order to work out some of the practical features of the problem before using the more recent 300-pound shipment. In view of the success that some have apparently met with in the sintering of properly prepared finely divided sulphide material without the use of any "returns" or other relatively coarse material, it was decided to carry out tests along these lines. To achieve success with such finely divided material, it must be mixed in such a manner as to "pelletize" it, i.e. convert it from a fine powder into the form of small balls. A mixer capable of producing commercially such a result is said to have been developed by Stelhi, of the Dwight & Lloyd Company.

In this work the mixing was done in a small tumbling-barrel, which with careful supervision achieved a satisfactory pelletization of the concentrate.

The sintering was carried out in a 12-inch by 48-inch Dwight & Lloyd continuous sintering machine. The results obtained were irregular. In all, six runs were made. With thick beds, 3½ or 4 inches, the arsenic sulphide distilled from the upper layer condensed in the mix just above the grate, thereby preventing the air from being drawn through, and rendering sintering impossible. With beds about 2 inches thick results were better, the material being well roasted but not always fused well enough to be classed as a satisfactory sinter. However, since this pelletized material, composed entirely of Beattie concentrate, did in some cases produce a satisfactory product when sintered in thin beds of 2 to 2½ inches, it may be assumed that it could be done regularly by an experienced operator.

The results obtained in those runs in which conditions were satisfactory are summarized below.

Run No.	Charge (lb.)			Depth of bed, inches	Analysis of sinter		Remarks
	Concentrate	Returns	Water		Sulphur, per cent	Arsenic, per cent	
3	210	Nil	31·5	2½	2·64	0·34	Not well sintered
B 6	100	Nil	13·0	2	10·72	0·03	Fair sinter

Series B

BEATTIE CONCENTRATE (14·54 PER CENT SULPHUR GRADE) PLUS "RETURNS"

In most, if not all, commercial sintering operations there is always some fine sinter and semi-sintered material in the product that have to be screened out and returned to the sintering plant for re-treatment. The mixing of such "returns" with the finely divided concentrate gives a mix with greatly increased porosity and makes possible the production of a much better sinter.

Series B consists of tests made on mixes made up of raw concentrate plus "return" material from Series A. As in Series A, care was taken, in the mixing of the charge, to "pelletize" it as completely as possible.

The results of the tests are summarized below. In general, it can be said that with the addition of "returns" in the proportion of one of return to three of concentrate a satisfactory sinter, chemically and physically, is obtained.

Run No.	Charge (lb.)			Depth of bed, inches	Analysis of sinter		Remarks
	Concentrate	Returns	Water		Sulphur, per cent	Arsenic, per cent	
B 8............	75	25	13	2	9·00	0·03	Good sinter
B 9............	75	25	11·5	3	9·67	0·02	Fair sinter

Series C

MIX OF BEATTIE CONCENTRATE (14·54 PER CENT SULPHUR GRADE), COPPER CONCENTRATE, AND "RETURNS"

As in the tests in Series A and B, the mixing was carried out in such a manner as to "pelletize" the mix as completely as possible. The results are summarized below, and it will be seen that good sinters, chemically and physically, were obtained.

Run No.	Charge (lb.)				Depth of bed, inches	Analysis of sinter		Remarks
	Concentrate	Copper concentrate	Returns	Water		Sulphur, per cent	Arsenic, per cent	
B 10..........	66	34	33	15	2½	9·51	0·01	Good sinter
B 11..........	66	34	25	14	2½	10·43	0·01	Good sinter

Series D

MIX OF BEATTIE CONCENTRATE (39·71 PER CENT SULPHUR GRADE), COPPER CONCENTRATE, AND "RETURNS"

In this series of tests the high-grade concentrate now being made at the Beattie property was used. In view of the results obtained in the preceding series of tests and of the probability that the proposed smelter would operate on a mixture of copper concentrate and Beattie concentrate, a mix composed of these two materials together with a commercial proportion of returns was decided upon.

The results are given below, and show that a physically good sinter was obtained, and that, although the arsenic content is higher than that obtained with the low-grade Beattie concentrate, it is low enough for practical purposes.

Run No.	Charge (lb.)				Depth of bed, inches	Analysis of sinter			Remarks
	Concentrate	Copper concentrate	Returns	Water		Sulphur, per cent	Arsenic, per cent	Gold, oz/ton	
B 12.......	100	50	38	16·0	$2\frac{1}{2}$	10·32	0·15	2·22	Good sinter
B 13.......	104	52	24	17·5	$2\frac{1}{2}$	10·10	0·12	2·08	Good sinter

CONCLUSION

Tests show that the sintering of flotation concentrate from the Beattie mine, without the admixture of "returns" or other coarse material is possible though difficult.

They also show that under the more probable practical conditions that obtain in commercial work, in which a certain amount of "returns" together with a proportion of copper concentrate would be included in the mix to be sintered, a satisfactory sinter, chemically as well as physically, can readily be obtained.

Ore Dressing and Metallurgical Investigation No. 573

GOLD ORE FROM THE McWATTERS GOLD MINE AT ROUYN, QUE.

Shipment. A shipment of seven bags of ore, net weight 375 pounds, was received February 17, 1934. Another shipment of seven bags of ore, net weight 550 pounds, was received March 13, 1934. The samples were submitted by W. J. Hosking, Manager, McWatters Gold Mines, Limited, Rouyn, Quebec.

Characteristics of the Ore. Six polished sections were prepared from specimens of rock taken from the first shipment and examined microscopically.

The gangue of the first shipment consists chiefly of fine-textured, grey, banded and somewhat schistose siliceous country rock, which contains numerous veins of smoky grey to glassy quartz. Locally, fine tourmaline needles are abundant in both the quartz and the country rock adjacent.

The metallic minerals determined in the polished sections from the first shipment are, in their order of abundance, pyrite, arsenopyrite, pyrrhotite, magnetite, chalcopyrite, and ilmenite.

Pyrite occurs most commonly as narrow irregular veinlets in quartz, and more rarely in country rock. It is also disseminated in the country rock in small amounts.

Arsenopyrite forms large crystals or masses in both quartz and country rock. Where it is associated with pyrite it is veined by this mineral, and in rare cases is included as small well formed crystals in pyrite.

Pyrrhotite is disseminated in irregular grains in the country rock. It is minor in quantity.

Magnetite is disseminated in small irregular grains in country rock. It contains a few fine lamellæ of ilmenite.

Chalcopyrite is rather common in the country rock, where it occurs as finely-disseminated grains, generally elongated parallel to the schistosity. A minor amount also occurs as fine veinlets in country rock and as tiny inclusions in pyrite.

No native gold was seen in the polished sections. An examination of specimens under the binocular microscope, however, revealed native gold in irregular grains and fine discontinuous veinlets in quartz, as irregular grains at the boundaries of quartz and dark silicates that occur within quartz. The occurrence of the gold is sporadic, which accounts for its absence from the polished surfaces.

No polished sections were prepared from the second shipment but specimens were examined under the binocular microscope. This sample appears to be similar in character with that of the first shipment, except that two points of difference were observed as follows: (1) There is more country rock and less quartz in the second shipment; (2) Native gold was not seen in specimens from the second shipment.

Average assays of samples from the two shipments were as follows:—

First shipment....................................	4·18 oz./ton in gold
Second shipment....................................	0·315 " "

Experimental Tests

A series of small-scale tests was made on a mixture of the two shipments of ore to determine in particular whether a worthwhile recovery could be made by blanket concentration of the ore and barrel amalgamation of the blanket concentrate. Amalgamation and cyanidation tests were also made on the ore, ground dry to different sizes. The mixed sample assayed 1·91 ounces per ton in gold and 98·7 per cent was extracted in 48 hours by cyanidation when the ore was all crushed through 150 mesh. In this case the tailing assayed 0·025 ounce per ton in gold. When the ore was all crushed dry through 200 mesh, however, extraction fell off considerably for a reason as yet undetermined. By barrel amalgamation of the ore crushed dry all through 48 mesh 64·4 per cent of the gold was recovered. By blanket concentration of the ore, ground in a ball mill 88·3 per cent through 200 mesh, 72·7 per cent of the gold was recovered in a blanket concentrate amounting to 4·2 per cent of the weight of feed used. The blanket tailing assayed 0·48 ounce per ton in gold. By barrel amalgamation 94·4 per cent of the gold in the blanket concentrate was recovered as amalgam. This represents a net recovery of 68·6 per cent of the total gold in the ore.

Details of the tests follow:—

CYANIDATION

Tests Nos. 1 to 8

Samples of the ore ground dry to pass through 48-, 100-, 150-, and 200-mesh screens were agitated in cyanide solution, 2·0 pounds KCN per ton, for periods of 24 and 48 hours. The cyanide tailings were assayed for gold.

Summary:

Feed sample: 1·91 oz/ton

Test No.	Mesh	Period of agitation, hours	Tailing assay, gold, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
					KCN	CaO
1	− 48....................	24	0·105	94·5	1·6	12·0
2	−100....................	24	0·05	97·4	3·1	14·1
3	−150....................	24	0·04	97·9	3·4	14·1
4	−200....................	24	0·17	91·1	3·7	14·5
5	− 48....................	48	0·035	92·2	1·7	13·1
6	−100....................	48	0·03	98·4	4·2	15·1
7	−150....................	48	0·025	98·7	4·4	15·2
8	−200....................	48	0·105	94·5	8·1	31·5

No explanation can be given for the high cyanide tailing from the 200-mesh sample. An amalgamation test on a sample of cyanide tailing from Test No. 8 failed to extract any of the gold, and this is good evidence that the high assay is not due to the presence of coarse gold.

GRINDING TESTS ON THE ORE

In order to determine the grindability of the ore, samples of it were ground in ball mills for different periods of time and after being filtered and dried were passed through a series of screens. The fraction caught on each screen was weighed and calculated to terms of percentages of total weight.

The results may be tabulated as follows:—

Period of grinding, minutes	Weight, per cent +28 mesh	Weight, per cent −28+35 mesh	Weight, per cent −35+48 mesh	Weight, per cent −48+65 mesh	Weight, per cent −65+100 mesh	Weight, per cent −100+150 mesh	Weight, per cent −150+200 mesh	Weight, per cent −200 mesh	Total
5	1·4	4·5	7·0	12·1	12·1	10·6	10·0	42·3	100·0
10	....	0·1	1·1	4·3	9·8	12·5	15·4	56·8	100·0
15	....	....	....	0·5	3·8	7·2	15·7	72·8	100·0
20	....	....	....	0·1	1·7	4·8	13·1	80·3	100·0
25	....	....	....	....	0·5	2·3	8·9	88·3	100·0
30	....	....	....	....	0·3	1·8	8·9	89·0	100·0

AMALGAMATION AND CYANIDATION

Tests Nos. 9 and 10

Samples of the ore ground dry to pass through 48- and 100-mesh screens were barrel-amalgamated with mercury for 30 minutes. The amalgamation tailings were sampled and assayed for gold and portions of each agitated in cyanide solution, 2·0 pounds KCN per ton, for 24 hours. The cyanide tailings were also assayed for gold.

Summary:

Feed sample: 1·91 oz./ton

Test No.	Amalgamation tailing assay, gold, oz/ton	Cyanide tailing assay, gold, oz/ton	Extraction by amalgamation, per cent	Extraction by cyanidation, per cent	Reagents consumed, lb/ton	
					KCN	CaO
9....................	0·68	0·13	64·4	28·8	1·3	13·1
10....................	0·71	0·015	62·8	36·4	2·1	15·3

BLANKET CONCENTRATION

Tests Nos. 11 to 18

A series of small-scale tests was carried out to find what recovery could be obtained by blanket concentration. The ore, in 2,000-gramme lots, was ground in ball mills for different periods of time and each lot passed over a corduroy blanket set at a slope of 2·5 inches per foot. The products were weighed and assayed for gold, and recovery in the blanket concentrate was calculated for each lot treated. The grinding most suitable for this operation was thus determined and a confirmatory test was carried out using 10,000 grammes of ore. A barrel-amalgamation test was also made on this blanket concentrate after it had been reground.

Summary:

Test No.	Period of grinding, minutes	Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
11	5	Concentrate	5·9	23·94	66·4
		Tailing	94·1	0·76	33·6
		Feed (cal.)	100·0	2·13	100·0
12	10	Concentrate	5·2	23·42	62·8
		Tailing	94·8	0·76	37·2
		Feed (cal.)	100·0	1·94	100·0
13	15	Concentrate	5·3	20·37	69·1
		Tailing	94·7	0·51	30·9
		Feed (cal.)	100·0	1·56	100·0
14	20	Concentrate	4·2	29·97	70·3
		Tailing	95·8	0·45	29·7
		Feed (cal.)	100·0	1·79	100·0
15	25	Concentrate	4·6	26·14	74·6
		Tailing	95·4	0·43	25·4
		Feed (cal.)	100·0	1·61	100·0
16	30	Concentrate	4·3	24·80	74·3
		Tailing	95·7	0·385	25·7
		Feed (cal.)	100·0	1·43	100·0
17	35	Concentrate	5·5	17·84	69·3
		Tailing	94·5	0·46	30·7
		Feed (cal.)	100·0	1·42	100·0
18	25	Concentrate	4·2	29·09	72·7
		Tailing	95·8	0·48	27·3
		Feed (cal.)	100·0	1·68	100·0
		Concentrate amalgamated	4·2	1·63	....

Net recovery as amalgam, 94·4 x 72·7...............68·6 per cent total gold

CONCLUSIONS

For efficient operation the ore should be treated by cyanidation.

In order to keep coarse gold out of the agitators a hydraulic trap, blankets, or amalgamation plates should be used after the grinding circuit.

Approximately 70·0 per cent of the gold can be recovered by a blanket concentrate, but the blanket tailing is much too high in gold to be discarded.

However, a blanket plant could be operated temporarily and the blanket tailing impounded for further treatment later on, possibly by cyanidation. The blanket concentrate would be barrel-amalgamated and the amalgamation tailing re-united with the blanket tailing for further treatment. This would give a fair return from the ore with a small initial outlay.

Ore Dressing and Metallurgical Investigation No. 574

GOLD ORE FROM MATACHEWAN, ONTARIO

Shipment. Three shipments of ore were received, No. 1 weighing 84 pounds and No. 2 weighing 102 pounds, on March 7, 1934; No. 3, weighing 50 pounds, on March 21, 1934.

The shipments were submitted by C. A. Floyd for the Arbade Gold Mines, Limited, Matachewan, Ontario.

Characteristics of the Ore. The gangue consists chiefly of fine-textured, pink siliceous rock containing a small amount of calcite in small disseminated grains and irregular stringers; this is crossed by veins of white quartz. A very small amount of leucoxene (?) is present as small grains in the siliceous gangue.

The metallic minerals in the polished sections are pyrite, "limonite", and native gold. Pyrite occurs mostly as small disseminated cubes and irregular grains in the pink siliceous gangue, but a less amount is present as coarse grains in the quartz. It is much fractured, and the fine fractures have been filled with later gangue, and locally with "limonite".

Native gold was seen only in the coarse pyrite associated with the vein quartz. Here small grains of gold occur either against pyrite grains or associated with gangue in the fine veinlets cutting the pyrite. One small rounded grain was seen within pyrite.

The small number of grains of gold does not allow of a grain analysis. It is estimated, however, that the grain size of the gold ranges between 200 mesh and −1,600 mesh, and that the average size may be somewhat below 200 mesh. Owing to the mode of occurrence of the gold seen, it is probable that grinding will be very efficient in liberating most of the metal, as it is quite likely that breaking will follow the fractures in the pyrite.

Sampling and Analysis. The ore was crushed and sampled by standard methods. It assayed as follows:—

Shipment No. 1	Gold	0·225 oz/ton
Shipment No. 2	Gold	0·12 "
Shipment No. 3	Gold	0·185 "

Experimental Tests

The experimental tests were carried out on Shipment No. 1 and Shipment No. 3 only.

Shipment No. 1

The following tests were made on Shipment No. 1:—

1. Amalgamation tests.
2. Straight cyanidation tests.
3. Cyanidation followed by tabling.
4. Blanketing followed by flotation.
5. Cyanidation of blanket concentrate.
6. Straight flotation tests.
7. Cyanidation of flotation concentrate.
8. Plate amalgamation followed by blanketing.

Summary of Tests

1. Amalgamation showed 75·5 per cent recovery at −48 mesh, and 78 per cent recovery at −65 mesh.

2. Straight cyanidation showed recoveries as follows: For 24-hour period: −48 mesh, 87 per cent; −65 mesh, 91 per cent; −100 mesh, 95·5 per cent; −150 mesh, 98 per cent. For the 48-hour period, the recoveries were 89 per cent, 93 per cent, 95 per cent, and 95 per cent respectively for the same sizes as above.

3. Cyanidation extracted 92 per cent; the tabling, 62 per cent of the remainder; making a total of 97 per cent recovery.

4. Approximately 80 per cent of the gold was saved on the blankets and 81 per cent of the rest recovered by flotation, an overall recovery of 96 per cent.

5. Cyanidation of blanket concentrate. Concentrate assayed 2·48 ounces gold per ton, tailing 0·04 ounce gold per ton, recovery 98 per cent. Reagents consumed: KCN at the rate of 1·2 pounds per ton, and lime at the rate of 8 pounds per ton.

6. In the straight flotation tests the concentrate assayed 5·34 ounces per ton, with recovery of 94 per cent. The ratio of concentration was 33 : 1. Finer grinding gave 97 per cent recovery; assay of concentrate, 5·62 ounces per ton; ratio of concentration, 32 : 1.

7. Cyanidation of flotation concentrate. Concentrate assayed 4·66 ounces gold per ton, cyanide tailing 0·17 ounce gold per ton, recovery 96 per cent. Reagents consumed: KCN at the rate of 6·75 pounds per ton, and lime at the rate of 28 pounds per ton.

8. Plate amalgamation gave 50 per cent recovery; blanket concentration, 42 per cent of the rest; overall recovery, 71 per cent.

AMALGAMATION

Test No. 1

Representative samples of the −14-mesh ore were crushed dry to pass 48- and 65-mesh screens.

The ore was amalgamated with mercury by barrel amalgamation in an Abbé jar mill at a pulp dilution of 1 : 1.

Screen tests were made on the tailings:—

Mesh No.	−48 weight, per cent	−65 weight, per cent
− 48+ 65	5·75	....
− 65+100	23·80	13·60
−100+150	14·95	18·50
−150+200	14·80	18·60
−200	40·70	49·30

The tailings were sampled and assayed for gold.

Recovery:

Mesh No.	Assay, gold, oz/ton		Extraction, per cent
	Feed	Tailing	
−48	0·225	0·055	75·56
−65	0·225	0·050	77·78

CYANIDATION

Test No. 2

Representative samples of the ore were crushed dry to pass 48-, 65-, 100-, and 150-mesh screens.

The samples were agitated in sodium cyanide solution equivalent in strength to 1·0 pound potassium cyanide per ton. The ratio of dilution was one part ore to three parts of solution. Lime was added to the pulp at the rate of 4·0 pounds per ton of ore. It was found necessary to add more lime during the test, using approximately 7·0 pounds per ton. One series of samples was agitated for a period of 24 hours, a second series was treated similarly for a period of 48 hours.

A summary showing the extraction of gold and the amount of reagents consumed in each case is given in the following table:—

Period of agitation, hours	Mesh No.	Assay, gold, oz/ton		Extraction, per cent	Reagents consumed, lb/ton of ore	
		Feed	Tailing		KCN	CaO
24	− 48	0·225	0·03	86·67	0·6	5·8
24	− 65	0·225	0·02	91·11	0·6	5·7
24	−100	0·225	0·01	95·56	....	6·2
24	−150	0·225	0·005	97·78	....	6·8
48	− 48	0·225	0·025	88·89	0·27	5·8
48	− 65	0·225	0·015	93·33	0·42	6·1
48	−100	0·225	0·01	95·56	0·42	6·5
48	−150	0·225	0·01	95·56	0·42	7·0

A screen test on the −100-mesh ore shows the following grind:—

Mesh	Weight, per cent
−100+150	10·8
−150+200	23·9
−200	65·3

CYANIDATION AND TABLING

Test No. 3

A representative sample of the −14-mesh ore was ground in a jar mill until approximately 60 per cent −200 mesh.

The sample was agitated in sodium cyanide solution equivalent in strength to 1·0 pound KCN per ton for 24 hours. The cyanide tailing was fed to a laboratory Wilfley table.

The Wilfley table products were concentrate, middling, and tailing.

The assay of the products shows that the table feed assayed 0·018 ounce gold per ton.

The extraction by cyanidation was 92·0 per cent. The recovery of gold from the table concentrate was 62·2 per cent of the gold in the cyanide tailing; referred to the original feed the recovery was 62·22 × 8·0=4·98 per cent. The overall recovery was, therefore, 96·98 per cent.

Summary:

Cyanidation

Period of agitation, hours	Assay, gold, oz/ton		Extraction, per cent	Reagents consumed, lb/ton of ore	
	Feed	Tailing		KCN	CaO
24	0·225	0·018	92·0	0·28	6·1

Tabling Test

Products	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent	Ratio of concentration	Recovery, per cent	
Cyanide tailing / Table feed	100·00	0·018	100·00			
Table concentrate	4·29	0·26	62·22	23·3 : 1	Cyanidation	92·0
Table middling	40·03	0·01	22·22		Table concentration	5·0
Table tailing	55·68	0·005	15·56		Total	97·0

Test No. 4

Four representative samples of the −14-mesh ore were ground in a jar mill for different periods of time.

The ground pulp was passed over an experimental blanket table. The table was set at a slope of $2\frac{1}{4}$ inches to the foot.

The blanket tailings were filtered and charged to a Denver Sub-A Flotation Cell and conditioned with the following reagents:—

Soda ash	2·0 lb/ton
Barrett No. 4	0·17 "
Sodium ethyl xanthate	0·1 "

The pulp was floated by the addition of pine oil at the rate of 0·05 to 0·1 pound per ton.

In calculating the recovery in this test, a calculated value for the feed assay was used. In a blanket test there is the obvious difficulty of obtaining all the valuable mineral from the blanket.

It was noted that with finer grinding the value of the blanket concentrate became generally lower, and of the flotation concentrate higher. The overall recovery from each test was fairly constant in three tests.

Summary:

Screen Test

Mesh	No. 1, weight, per cent	No. 2, weight, per cent	No. 3, weight, per cent	No. 4, weight, per cent
− 48+ 65	0·55	0·10	0·05	....
− 65+100	6·15	2·75	1·85	0·10
−100+150	12·05	8·45	6·60	1·70
−150+200	22·25	23·30	22·50	10·25
−200	59·00	65·40	69·00	87·95

Blanket Test No. 1 Plus Flotation

Products	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent	Overall distribution, per cent	Total distribution, per cent
Feed	100·0	0·22	100·00	100·00	100·00
Blanket concentrate	5·55	3·20	80·69	80·69	
Blanket tailing	94·45	0·045	19·31		
Both concentrates combined give					96·72
Flotation feed	100·0	0·057	100·00		
Flotation concentrate	3·04	1·56	83·01	83·01×19·31=16·03	
Flotation tailing	96·96	0·01	16·99	16·99×19·31= 3·28	3·28

Blanket Test No. 2 Plus Flotation

Feed	100·00	0·27	100·00	100·00	100·00
Blanket concentrate	8·77	2·70	86·64	86·64	
Blanket tailing	91·23	0·04	13·36		
Both concentrates combined give					97·45
Flotation feed	100·00	0·05	100·00		
Flotation concentrate	2·12	1·96	80·93	80·93×13·36=10·81	
Flotation tailing	97·88	0·01	19·07	19·07×13·36= 2·55	2·55

Blanket Test No. 3 Plus Flotation

Products	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent	Overall distribution. per cent	Total distribution, per cent
Feed....................	100·00	0·17	100·00	100·00	100·00
Blanket concentrate......	6·39	1·95	72·70	72·70	
Blanket tailing...........	93·61	0·05	27·30		
Both concentrates combined give..........	....	....	....	....	92·02
Flotation feed...........	100·00	0·067	100·00		
Flotation concentrate....	2·46	1·92	70·76	70·76×27·3=19·32	
Flotation tailing.........	97·54	0·02	29·24	29·24×27·3= 7·98	7·98

Blanket Test No. 4 Plus Flotation

Products	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent	Overall distribution. per cent	Total distribution, per cent
Feed....................	100·00	0·18	100·00	100·00	100·00
Blanket concentrate......	6·53	2·12	76·68	76·68	
Blanket tailing...........	93·47	0·045	23·32		
Both concentrates combined give..........	....	....	....	....	97·40
Flotation feed...........	100·00	0·044	100·00		
Flotation concentrate....	2·26	1·73	88·86	88·86×23·32=20·72	
Flotation tailing.........	97·74	0·005	11·14	11·14×23·32= 2·6	2·60

In the table the recoveries are percentages of the feed in each test. The blanket concentrate and flotation concentrate are shown separately and combined. The flotation tailing is the final tailing in each test. It will be seen that the blanket tailing is the feed for the flotation test in each case. The value of flotation feed assays is determined from the calculation of the values obtained from the products. For this reason they do not coincide exactly with the assay value of the blanket tailing.

It will be noted that fine grinding, that is, an increase from 59 per cent −200 mesh to 88 per cent −200 mesh, did not give much greater recovery.

CYANIDATION OF BLANKET CONCENTRATE

Test No. 5

In this test, the blanket concentrate from Test No. 4 was treated by cyanidation. The strength of the sodium cyanide solution was made equivalent to 2·0 pounds KCN per ton. Lime was added at the rate of 10 pounds per ton of concentrate. The period of agitation was 72 hours.

The feed assay was calculated from the assays of the four lots of concentrate.

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

Period of agitation, hours	Assay, gold, oz/ton		Extraction, per cent	Reagents consumed, lb/ton of ore	
	Feed	Tailing		KCN	CaO
72....................................	2·48	0·04	98·39	1·17	8·05

CYANIDATION OF FLOTATION CONCENTRATES

Test No. 7

A representative sample of −14-mesh ore was ground in a jar mill to give approximately 80 per cent −200 mesh.

The reagents added to the grinding mill were as follows:—

Soda ash..	2·0 lb/ton
Barrett No. 4..	0·16 "

The reagents added to the flotation cell were as follows:—

Sodium ethyl xanthate........................	0·1 lb/ton,	as a conditioner
Pine oil..................................	0·05 to 0·10 "	as a frother

All the concentrate made in this test was treated by cyanidation. The solution was equivalent to 2·0 pounds KCN per ton. Lime was added at the rate of 10 pounds per ton of concentrate. The period of agitation was 72 hours. Frequent tests of the solution were made, and the reagents were added as required to maintain the strength of the solution.

The following tables show the results of flotation and cyanidation:

Results:

Flotation

Products	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent	Ratio of concentration
Feed..	100·00	0·18	100·00	
Flotation concentrate........................	3·38	4·66	86·73	29·6 : 1
Flotation tailing............................	96·62	0·025	13·27	

Cyanidation

Period of agitation, hours	Assay, gold, oz/ton		Extraction, per cent	Reagents consumed, lb/ton of ore	
	Feed	Tailing		KCN	CaO
72..	4·66	0·17	96·35	6·75	27·9

Overall recovery of gold: 96·35×86·73=83·56 per cent

STRAIGHT FLOTATION

Test No. 6

Two representative samples of −14-mesh ore were ground in a jar mill to give 64 per cent −200 mesh and 80 per cent −200 mesh.

The reagents to the grinding mill were as follows:—

Soda ash..	2·0 lb/ton
Barrett No. 4..	0·16 "

The reagents to the flotation cell were as follows:—

Sodium ethyl xanthate........................	0·1 lb/ton	as a conditioner
Pine oil..................................	0·05 to 0·10 "	as a frother

The results are shown in the following table:—

Test No. 6, No. 1

Products	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·17	100·00	
Flotation concentrate	3·05	5·34	94·34	33 : 1
Flotation tailing	96·95	0·01	5·66	

Test No. 6, No. 2

Products	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·18	100·00	
Flotation concentrate	3·10	5·62	97·32	32 : 1
Flotation tailing	96·90	0·005	2·68	

PLATE AMALGAMATION AND BLANKETING

Test No. 8

A representative sample of −14-mesh ore was ground in a jar mill to approximately 60 per cent −200 mesh.

The ground pulp was passed over an amalgamation plate. The amalgamation tailing was sampled and passed over a blanket table.

The following table shows the results obtained:—

—	Weight, per cent	Assay, gold, oz/ton	Distribution of gold (in each stage), per cent	Distribution of gold in whole operation	
				Per cent	Per cent
Feed	100·00	0·15*	....	100·00	100·00
Amalgam	....	....	50·00	50·00	
Amalgamation tailing	....	0·075	....	....	
Blanket feed	100·00	0·066	100·00	....	
Blanket concentrate	4·15	0·66	41·70	20·85	
Blanket tailing	95·85	0·04	58·30	29·15	29·15

Recovery by amalgamation.................................... 50·00 per cent
Recovery by blanketing..............................41·7×50·0=20·85 "

Overall recovery by amalgamation and blanketing...... 70·85 "

*The ore remaining was mixed and sampled to give a feed sample for the above test. The representative sample was cut out of the freshly mixed ore and used for the test.

This test shows that 29·15 per cent of the gold is lost in the final tailing.

Shipment No. 3

The following tests were made on Shipment No. 3:—

1. Cyanidation. Tests No. 1 and No. 2 : 24 and 48 hours.

2. Plate amalgamation followed by blanket concentration. Test No. 3.

3. Flotation. Tests No. 4 and No. 5.

4. Cyanidation. Tests No. 6 and No. 7 : 48 hours.

5. Blanket concentration and amalgamation of concentrate. Test No. 8.

Characteristics of the Ore. The ore of Shipment No. 3 appeared to be from the surface and was more oxidized than the previous shipments.

SUMMARY OF TESTS

1. Cyanidation for 24 hours gave 89 per cent recovery; for 48 hours, 76 per cent recovery.

2. Plate amalgamation gave 49 per cent recovery; blankets, 49 per cent of the remainder; overall recovery, 74 per cent.

3. Straight flotation tests gave a concentrate assaying 9·74 ounces gold per ton; recovery, 93 per cent; ratio of concentration, 53 : 1. With finer grinding the concentrate assayed 8·15 ounces gold per ton; recovery, 95 per cent; ratio of concentration, 43·5 : 1.

4. Two cyanidation tests repeated on ore crushed 70 per cent −200 mesh and 80 per cent −200 mesh. The period of agitation was 48 hours. The recovery was 92 per cent.

5. Blanket concentration gave 53 per cent recovery. Amalgamation of concentrate recovered 82 per cent of the gold, an overall recovery of 44 per cent.

CYANIDATION

Tests Nos. 1 and 2

Two representative samples of −14-mesh ore were ground in jar mills to give approximately a 75 per cent −200-mesh product.

The pulp was agitated in cyanide solution at a dilution of 2 parts of ore to 5 parts of solution. Sodium cyanide solution equivalent in strength to 1·0 pound KCN per ton was used. The periods of agitation were 24 and 48 hours, respectively. The alkalinity of the pulp was maintained by using lime at the rate of 8·0 pounds per ton of ore, and additions as required.

The following table shows the results of the tests.

Tests Nos. 1 and 2

Test No.	Period of agitation, hours	Assay, gold, oz/ton		Extraction, per cent	Reagents consumed, lb/ton of ore	
		Feed	Tailing		KCN	CaO
1	24	0·185	0·02	89·2	0·50	7·42
2	48	0·185	0·045	75·7	0·98	8·25

These tests show high tailings and are not satisfactory.

PLATE AMALGAMATION AND BLANKET CONCENTRATION

Test No. 3

A representative sample of the −14-mesh ore was ground in a jar mill to give approximately a 60 per cent −200-mesh product.

The ground pulp was treated similarly to Test No. 8 (page 163), that is, passed over the same amalgamation plate, and the amalgamation tailing was sampled. The tailing was passed over a blanket.

Results:

Products	Weight, per cent	Assay, gold, oz./ton	Distribution of gold (in each stage), per cent	Distribution of gold in whole operation, per cent
Feed	100·00	0·185	100·00	100·00
Amalgamation	....	....	48·65	48·65
Amalgamation tailing	....	0·095	51·35	....
Overall recovery of gold	....	....	....	73·80
Blanket feed	100·00	0·098	100·00	....
" concentrate	8·9	0·54	48·98	25·15
" tailing	91·1	0·055	51·02	26·20

Blanket concentrate, $48·98 \times 51·35 = 25·15$ per cent
" tailing ..... $51·02 \times 51·35 = 26·20$ per cent

In this test, 48·65 per cent of gold is recovered by amalgamation. By blanket concentration 49 per cent of the gold in the amalgamation tailing is recovered, giving a total overall recovery of 73·80 per cent.

FLOTATION

Tests Nos. 4 and 5

The object of the tests was to ascertain the fineness of grinding necessary to give complete recovery of the gold.

Two representative samples of the ore were ground in a jar mill with the following reagents:—

Soda ash	2·0 lb/ton
Barrett No. 4	0·17 "

The pulp was conditioned in the cell with sodium xanthate, 0·1 pound per ton, and floated with pine oil, 0·05 to 0·1 pound per ton.

After floating, the pulp was reconditioned with the same amounts of reagents and frother to attempt to raise a middling product. No appreciable amount of sulphides was observed in the froth.

The small amount of concentrate gives a high ratio of concentration.

The grind in Test No. 4 was approximately 70 per cent −200 mesh, and that of Test No. 5, 80 per cent −200 mesh.

Test No.	Products	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent	Ratio conce tration
4	Feed	100·00	0·199	100·00	52·63 : 1
	Concentrate	1·90	9·74	92·64	
	Tailing	98·10	0·015	7·36	
5	Feed	100·00	0·197	100·00	43·5 : 1
	Concentrate	2·3	8·15	95·08	
	Tailing	97·7	0·010	4·92	

CYANIDATION

Tests Nos. 6 and 7

Owing to the unsatisfactory results shown in the cyanidation Tests Nos. 1 and 2 of Shipment No. 3, the cyanidation test was repeated.

Representative samples of the ore were crushed in jar mills to 70 per cent −200 mesh and 80 per cent −200 mesh. The pulp was agitated with cyanide solution at a strength of 1·0 pound KCN per ton, for 48 hours. The pulp was kept alkaline with lime.

Results:

Test No.	Agitation period, hours	Assay, gold, oz/ton		Extraction, per cent	Reagents consumed, lb/ton of ore	
		Feed	Tailing		KCN	CaO
6	48	0·185	0·015	91·89	0·75	11·13
7	48	0·185	0·015	91·89	1·00	11·25

In this test it was observed that considerably more lime was required, due no doubt to the acid-forming minerals in the ore. The sample showed more oxidation than Shipment No. 1.

BLANKET CONCENTRATION FOLLOWED BY BARREL AMALGAMATION OF THE CONCENTRATE

Test No. 8

A representative sample of the ore was ground in a jar mill to 60 per cent −200 mesh. The pulp was passed over a blanket table.

The blanket concentrate was amalgamated with mercury by barrel amalgamation.

The recovery of gold by blanket concentration amounted to 53·1 per cent. The recovery of the gold from the concentrate by amalgamation amounted to 82·3 per cent. The overall recovery was 43·7 per cent.

The results are shown in the tables following:

Results:

Products	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Feed	100·0	0·185	100·0
Blanket concentrate	8·3	1·38	53·1
Blanket tailing	91·7	0·11	46·9

Amalgamation

Assay, gold, oz/ton		Extraction, per cent	Overall extraction
Feed	Tailing		
1·38	0·245	82·25	82·25 × 53·1 = 43·7 per cent

Ore Dressing and Metallurgical Investigation No. 575

GOLD ORE FROM MATACHEWAN CONSOLIDATED MINES, LIMITED, AT MATACHEWAN, ONTARIO

Shipment. A shipment of two lots of ore marked "E" and "W" was received January 23, 1934. Sample "E" consisted of 19 bags of ore weighing 1,287 pounds, and sample "W" consisted of 13 bags of ore weighing 896 pounds. The samples were submitted by Thomas L. Wells, Matachewan Consolidated Mines, Limited, Matachewan, via Elk Lake, Ontario.

Characteristics of the Ore. The gangue material of sample "E" consists chiefly of brownish to greenish grey, dense, fine-textured, siliceous rock, which has a distinct but irregular banding and is locally somewhat schistose; this is penetrated by stringers of milky quartz containing white calcite.

The gangue material of sample "W" is very similar to that of sample "E", except that the specimens examined show a greater proportion of schist.

The metallic minerals in the ore, in their order of abundance, are: pyrite, magnetite, ilmenite, chalcopyrite, and native gold. Samples "E" and "W" are very similar and will be described as one.

Pyrite is the most abundant metallic mineral, and occurs in coarsely crystalline masses in quartz and as disseminated grains in both quartz and schist. It contains inclusions of gangue, magnetite, ilmenite, and chalcopyrite, and is veined by gangue, chalcopyrite, and native gold.

Magnetite and ilmenite are moderately abundant as irregular grains in country rock. Some of the ilmenite has been altered to leucoxene (?).

A small amount of chalcopyrite occurs as small irregular grains in both gangue and pyrite, and commonly forms veinlets in the latter.

An average assay of the samples was as follows:—

	Sample "E"	Sample "W"
Gold	0·18 oz/ton	0·305 oz/ton
Iron	8·30 per cent	9·25 per cent
Sulphur	3·13 "	3·78 "

Experimental Tests

A series of small-scale tests was made on the individual samples of ore, as well as large-scale mill runs on a mixture of the two samples.

The small-scale work included tests by amalgamation, flotation, and cyanidation. Amalgamation recovered 54·4 per cent of the gold from sample "E" when the ore was crushed 90·0 per cent through 200 mesh. An additional 30·4 per cent of the gold was recovered in a concentrate by floating the amalgamation tailing. By straight cyanidation of the ore, 94·4 per cent of the gold was extracted from sample "E" in 24 hours, leaving a tailing assaying 0·01 ounce per ton in gold.

From sample "W", 72·1 per cent of the gold was recovered by amalgamation when the ore was crushed 90·0 per cent through 200 mesh. An additional 18·9 per cent of the gold was recovered in a concentrate when the amalgamation tailing was floated. By straight cyanidation of the ore, 96·7 per cent of the gold was extracted from sample "W" when the ore was crushed 90·0 per cent through 200 mesh.

In the large-scale mill runs, using a mixture of the two lots of ore the flotation tailings produced from amalgamation or blanket tailings assayed approximately the same as those produced in the small-scale runs.

Details of the tests follow:—

Sample "E"

GRINDING TESTS

Samples of the ore were ground in a ball mill for different periods of time. In each case 1,000 grammes of the ore was ground with 750 c.c. of water, the pulp dried and screened to determine the grinding.

Results:

Mesh	Period of grinding (minutes)				
	10	15	20	25	30
Weight, per cent +48	1·3				
" −48+65	4·4	0·3	0·1		
" −65+100	11·1	2·6	1·2	0·3	0·2
" −100+150	12·4	5·6	3·8	1·5	1·2
" −150+200	15·3	15·8	12·1	7·2	6·6
" −200	55·5	75·7	82·8	91·0	92·0
Total	100·0	100·0	100·0	100·0	100·0

AMALGAMATION AND FLOTATION

Tests Nos. 1, 2, and 3

In this series of tests the ore was ground in a ball mill for periods of 8, 12, and 25 minutes, or to approximately 50, 70, and 90 per cent through 200 mesh. The pulp was barrel-amalgamated and then floated with the following reagents:—

Soda ash	2·0 lb/ton
Potassium amyl xanthate	0·20 "
Pine oil	0·10 "

Summary:

Feed sample: 0·18 oz/ton

Test	Product	Grinding time, minutes	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
1	Flotation concentrate	8	9·6	0·92	48·9
	Flotation tailing		90·4	0·03	15·0
	Amalgamation tailing (cal.)		100·0	0·115	63·9
2	Flotation concentrate	12	8·4	0·92	42·9
	Flotation tailing		91·6	0·025	12·7
	Amalgamation tailing (cal.)		100·0	0·100	55·6
3	Flotation concentrate	25	9·1	0·60	30·4
	Flotation tailing		90·9	0·03	15·2
	Amalgamation tailing (cal.)		100·0	0·082	45·6

CYANIDATION

Tests Nos. 4 and 5

Two samples of the ore were ground in a ball mill approximately 70·0 and 90·0 per cent through 200 mesh. The ground ore was agitated in cyanide solution, 1·0 pound KCN per ton, for 24 hours. The tailings were assayed for gold.

Summary:

Feed sample: 0·18 oz/ton

Test No.	Grinding, per cent—200 mesh	Tailing assay, gold, oz/ton	Extraction, per cent	Reagents consumed, lb/ton.	
				KCN	CaO
4	70·0	0·015	91·7	0·50	4·75
5	90·0	0·010	94·4	0·60	5·10

AMALGAMATION AND CYANIDATION

Test No. 6

A sample of the ore was ground in a ball mill approximately 70·0 per cent through 200 mesh. The pulp was then barrel-amalgamated for 30 minutes and the amalgamation tailing agitated in cyanide solution, 1·0 pound KCN per ton, for 24 hours. Samples of the amalgamation tailing and of the cyanide tailing were assayed for gold.

Summary:

Feed sample: 0·18 oz/ton

Product	Assay, gold, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
			KCN	CaO
Amalgamation tailing	0·095	47·2	....	....
Cyanide tailing	0·015	44·5	0·50	5·6

AGITATION WITH LIME, AND CYANIDATION

Test No. 7

In this test a sample of the ore was ground approximately 70·0 per cent through 200 mesh in a ball mill and the pulp agitated with lime for 4 hours. Cyanide was then added and agitation continued for another 24 hours. The tailing was assayed for gold.

Results:

Feed sample	0·18 oz/ton in gold
Cyanide tailing	0·01 " "
Extraction	94·4 per cent

Reagents consumed:

KCN	0·45 lb/ton ore
CaO	5·10 "

Sample "W"

GRINDING TESTS

A series of small-scale tests was carried out on sample "W", similar to those made on sample "E". The results are given in the following tables:—

Results:

Mesh		Period of grinding (minutes)				
		10	15	20	25	30
Weight, per cent	+ 48	0·9	0·1	....	....	....
"	− 48+ 65	4·0	0·8	....	....	....
"	− 65+100	11·4	5·7	1·8	1·0	0·2
"	−100+150	12·5	9·7	4·6	3·2	1·6
"	−150+200	16·2	16·6	12·7	11·6	7·3
"	−200	55·0	67·1	80·9	84·2	90·9
Total		100·0	100·0	100·0	100·0	100·0

AMALGAMATION AND FLOTATION

Tests Nos. 1, 2, and 3

Samples of the ore were ground approximately 50, 70, and 90 per cent through 200 mesh in a ball mill. The periods of grinding were, respectively, 13, 22, and 35 minutes. The pulps were barrel-amalgamated and the amalgamation tailings floated with the following reagents:—

Soda ash	2·0 lb./ton
Potassium amyl xanthate	0·20 "
Pine oil	0·10 "

Summary:

Feed sample: 0·305 oz/ton

Test No.	——	Grinding time, minutes	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
1	Flotation concentrate	13	10·3	0·99	33·4
	Flotation tailing	....	89·7	0·04	11·8
	Amalgamation tailing (cal.)	....	100·0	0·138	45·2
2	Flotation concentrate	22	9·1	1·20	35·7
	Flotation tailing	....	90·9	0·04	11·8
	Amalgamation tailing (cal.)	....	100·0	0·145	47·5
3	Flotation concentrate	35	8·4	0·69	18·9
	Flotation tailing	....	91·6	0·03	9·0
	Amalgamation tailing (cal.)	....	100·0	0·085	27·9

The recovery by amalgamation may be obtained by subtracting from 100 the recovery in the amalgamation tailing.

CYANIDATION

Tests Nos. 4 and 5

Two samples of the ore were ground approximately 70·0 and 90·0 per cent through 200 mesh and agitated in cyanide solution, 1·0 pound KCN per ton, for 24 hours. The tailings were assayed for gold.

Summary:

Feed sample: 0·305 oz/ton

Test No.	Grinding, per cent—200 mesh	Tailing assay, gold, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
				KCN	CaO
4	70·0	0·015	95·1	0·60	4·75
5	90·0	0·010	96·7	0·60	5·10

AMALGAMATION AND CYANIDATION

Test No. 6

A sample of the ore was ground 70·0 per cent through 200 mesh and barrel-amalgamated. The amalgamation tailing was agitated in cyanide solution, 1·0 pound KCN per ton for 24 hours. Samples of the amalgamation and cyanide tailings were assayed for gold.

Summary:

Feed sample: 0·305 oz/ton

Product	Assay, gold, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
			KCN	CaO
Amalgamation tailing	0·10	67·2	....	....
Cyanide tailing	0·01	29·5	0·40	5·50

AGITATION WITH LIME AND CYANIDATION

Test No. 7

A sample of the ore was ground 70·0 per cent through 200 mesh in a ball mill and then agitated with lime for four hours. Cyanide was then added and agitation continued for another 24 hours.

Summary:

Feed sample	0·305 oz/ton in gold
Cyanide tailing	0·01 " "
Extraction	96·7 per cent

Reagents consumed:

KCN	0·60 lb/ton ore
CaO	5·50 "

Large-Scale Mill Runs on Samples "E" and "W" Mixed

Run No. 1

In this test a unit having a capacity of 100 pounds per hour was used. The ore at −14 mesh was fed into a rod mill at the rate of 110 pounds per hour. The rod mill discharged into a hydraulic trap, the overflow from which went to an Akins classifier. The oversize was returned to the rod mill for regrinding and the overflow was passed over corduroy blankets. The blanket tailing was floated in a battery of five cells. The concentrate was not recleaned. The flotation tailing was passed over a concentration table in order to check up on the action of the cells.

The reagents used in this run were:—

Soda ash	2·0	lb/ton
Potassium amyl xanthate	0·10	"
Pine oil	0·075	"

After running a short time, cresylic acid, 0·30 pound per ton, was substituted for the pine oil.

Assays:

	Oz/ton
Mill feed	0·24
Rod mill discharge	0·23
Trap overflow	0·365
Classifier overflow	0·195
Blanket tailing	0·16
Flotation concentrate	1·76
Flotation tailing	0·03
Blanket concentrate	3·32
Trap cleanings	2·22
Trap cleanings and blanket concentrate, combined	2·73

In this run, which lasted three hours, 2·75 pounds of blanket concentrate, 6·25 pounds of trap cleanings, and 23 pounds of flotation concentrate were produced. Therefore, by difference, the weight of flotation tailing is $(110 \times 3)-(23+6{\cdot}25+2{\cdot}75)=298$ pounds actual, or 90·3 per cent of the weight of feed used. The loss in the flotation tailing in terms of total gold is therefore $90{\cdot}3\times0{\cdot}03\div100\times0{\cdot}24=11{\cdot}3$ per cent.

Recoveries in the flotation concentrate and in the combined blanket concentrate and trap cleanings respectively were 51·3 per cent and 30·8 per cent. The remaining gold, not accounted for in these three products, is locked up in the mill and classifier. The classifier overflow was 65·0 per cent through 200 mesh.

Cyanidation tests were made on the flotation concentrate for varying periods of time with and without grinding. The results of these tests may be summarized as follows:—

Feed sample: 1·76 oz/ton

Period of agitation, (hours)	Grinding, per cent −200 mesh	Tailing assay, gold, oz/ton	Extraction, per cent	Extraction, per cent total gold	Reagents consumed, lb/ton	
					KCN	CaO
24	82·8	0·15	91·5	46·9	3·9	9·4
24	95·0	0·085	95·2	48·8	5·7	9·8
48	82·8	0·09	94·9	48·7	5·0	12·0
48	95·0	0·06	96·6	49·6	9·0	13·1

Barrel-amalgamation tests were made on the combined trap cleanings and blanket concentrate with and without regrinding. The results may be tabulated as follows:—

Feed sample: 2·73 oz/ton

Grinding, per cent −200 mesh	Tailing assay, gold, oz/ton	Recovery, per cent	Recovery, per cent total gold
62·2	0·76	72·2	22·2
93·8	0·39	85·7	26·4

Run No. 2

In this run, amalgamation plates replaced the hydraulic trap and the blankets. The rod mill discharged onto a short plate, the tailing from which went to an Akins classifier. The classifier overflow was passed over another amalgamation plate, the tailing from which went to flotation. The reagents used were the same as for Run No. 1.

Assays:

	Oz/ton
Mill feed	0·255
Rod mill discharge	0·30
First plate discharge	0·22
Classifier overflow	0·14
Second plate discharge	0·105
Flotation concentrate	1·48
Flotation tailing	0·025

The classifier overflow was 85 per cent through 200 mesh, and 17 pounds of flotation concentrate was produced in a three-hour run. The feed rate was maintained at 110 pounds per hour.

Recovery by amalgamation on first plate, (0·30 −0·22)÷0·255=31·4 per cent.
Recovery by amalgamation on second plate, (0·14 −0·105)÷0·255=13·7 per cent.
Total recovery by amalgamation, 45·1 per cent.
Recovery in flotation concentrate, calculated by two-product formula, 42·5 per cent.
Loss in flotation tailing, (100·0 −87·6)=12·4 per cent.

Cyanidation tests were made on the flotation concentrate for periods of 24 and 48 hours with and without regrinding. The results of these tests may be summarized as follows:—

Feed sample: 1·48 oz/ton

Period of agitation (hours)	Grinding, per cent, −200 mesh	Tailing assay, gold, oz/ton	Extraction, per cent	Extraction, per cent total gold	Reagents consumed, lb/ton	
					KCN	CaO
24	84·2	0·155	89·5	38·0	5·10	9·5
24	96·0	0·12	91·9	39·1	7·8	10·0
48	84·2	0·15	89·9	38·2	8·3	11·8
48	96·0	0·085	94·3	40·1	9·8	13·3

At the end of Run No. 2, the mill and classifier were cleaned out and the cleanings assayed together. The assay of this material was 0·75 ounce per ton in gold. The conditioning tank was also cleaned out and its contents assayed 0·125 ounce per ton.

CONCLUSIONS

The results of the tests carried out on this ore show that cyanidation is the best method for recovering the gold. An 0·015-ounce tailing can be obtained by straight cyanidation, and if means of preventing the coarse gold from reaching the agitators are taken, an 0·01-ounce tailing is assured for ore of the grade tested. The ore should be ground to 70 per cent through 200 mesh to produce this tailing.

The lowest tailing obtained by flotation was 0·025 ounce when grinding to 85 per cent through 200 mesh, and in order to produce this grade of tailing it was necessary to amalgamate prior to flotation.

Ore Dressing and Metallurgical Investigation No. 576

GOLD ORE FROM ROCHESTER MINE, GILLIES LAKE-PORCUPINE GOLD MINES, LIMITED, TIMMINS, ONTARIO

Shipment. A shipment of gold-bearing ore having a weight of 1,367 pounds was received at the Ore Dressing and Metallurgical Laboratories on April 3, 1934. The shipment was from the Rochester mine of the Gillies Lake-Porcupine Gold Mines, Limited, and was submitted by G. S. Scott, P.O. Box 1844, Timmins, Ontario.

Characteristics of the Ore. The predominating gangue material is a dark grey to greenish grey schist, containing much of finely-divided carbonate. Irregular stringers of light grey to white translucent quartz penetrate this.

The metallic minerals determined from a microscopic examination of the polished sections are: pyrite, chalcopyrite, ilmenite (?), pyrrhotite, and an unidentified mineral that may be galena. The metallic minerals form only a very small part of the aggregate.

Pyrite is by far the most abundant and occurs as sparsely disseminated cubes. Chalcopyrite occurs in very small amount.

No native gold was seen in the polished sections, but much free gold in the mill discharge indicates that much of the gold in the ore is native.

The ore was crushed and sampled by standard methods, and the feed sample, assayed for gold, gave the following result:

Gold........2·91 oz/ton.

EXPERIMENTAL TESTS

Test work comprised the following:

1. Amalgamation on plates.
2. Cyanidation of amalgamation plate tailing.
3. Flotation of amalgamation plate tailing.
4. Barrel amalgamation of flotation concentrate.
5. Cyanidation of flotation concentrate.

Small-Scale Tests

A 4,000-gramme charge of ore was ground to pass a 48-mesh screen.

Screen Test on −48-mesh Ore:

Mesh	Weight, per cent
+ 65	10·1
+100	16·6
+150	13·5
+200	14·7
−200	45·1
	100·0

Test No. 1

A 1,000-gramme charge of the −48-mesh ore was ground wet for 10 minutes in an Abbé pebble mill and the pulp run over a small amalgamation plate.

Gold in feed	2·91 oz/ton
Gold in tailing	0·565 "
Recovery on plate	80·58 per cent

Screen Test of Plate Tailing:

Mesh	Weight, per cent
+ 65	0·2
+100	3·0
+150	11·2
+200	22·2
−200	63·4
	100·0

Test No. 2

A 2,000-gramme charge of −48-mesh ore was ground wet for 10 minutes and the pulp run over an amalgamation plate. The tailing was floated and the flotation concentrate barrel-amalgamated.

Reagents to flotation cell:

Soda ash	2·0 lb/ton
Sodium ethyl xanthate	0·4 "
Pine oil	0·1 "

Results:

Product	Weight, per cent	Assays, gold, oz/ton	Distribution of gold, per cent	Ratio of concentration
Concentrate	5·77	8·97	91·6	
Tailing	94·23	0·05	8·4	17·3 : 1
	100·00			

Barrel Amalgamation of Flotation Concentrate:

Weight of concentrate	115	grammes
Mercury added	66·7	"
Time of agitation	1	hour
Barrel amalgamation tailing ... Gold =	4·235	oz. /ton
Recovery	53·0	per cent
Recovery of gold on plates	80·58	"
Gold recovered by flotation	17·89	"
Recovery by barrel amalgamation = 53·0 per cent of 17·89 per cent =	9·48	"
Overall gold recovery, 80·58 per cent + 9·48 per cent =	90·06	"

The recovery was only fair, but such a small amount of metallic minerals in the ore renders a satisfactory test by small-scale experiments impossible. It was decided to carry out a large-scale mill run on the remainder of the ore and carry out small-scale cyanidation and barrel-amalgamation tests on the products of the mill run.

Large-Scale Tests

Mill Run No. 1

The ore crushed to ¼-inch size was reduced to −14 mesh in rolls and fed to a 12-inch by 14-inch rod mill containing 200 pounds of rods. The discharge from the mill passed over an amalgamation plate at a slope of 2·5 inches to the foot. The plate overflow was pumped to a conditioning tank, the overflow from which was fed to a Denver 10-unit laboratory flotation machine.

Early in the run it was found that pine oil gave too great a froth, so cresylic acid was substituted.

Much free gold was noted on the plate and also in the plate overflow, a fact that indicated the advisability of placing a trap between the mill discharge and the amalgam plate.

The results of this test are as follows:

Total feed to mill—629 pounds
Average rate of feed—104·8 pounds per hour

Reagents added:

To mill—Soda ash, 2 lb/ton
To conditioner—Sodium ethyl xanthate, 0·4 lb/ton.
To cells—Cresylic acid, 0·3 lb/ton

Products	Assays, gold, oz/ton
Mill feed	2·96
Plate overflow	1·75
Conditioner overflow	0·16
Flotation concentrate	5·26
" tailing	0·055

Ratio of concentration $\dfrac{5\cdot26-0\cdot055}{0\cdot16-0\cdot055} = 48\cdot6 : 1$

Recovery by plate amalgamation and metallics retained in mill.. 40·87 per cent

Gold trapped in pump and conditioner tank=90·8 per cent of (100−40·87 per cent) = 53·69 "

Recovery of gold by flotation $= \dfrac{100 \times 5\cdot26\ (0\cdot16-0\cdot055)}{0\cdot16\ (5\cdot26-0\cdot055)} = 66\cdot3$ "

Percentage of total gold in flotation concentrate = 66·3 per cent of 5·44 per cent 3·61 "

Overall recovery of gold = 40·87 + 53·69 + 3·61 = 98·17 "

The grinding on this run was found to be a little coarse. At a feed rate of 90 pounds per hour, the mill discharge was only 43 per cent −200 mesh; and at 78·75 pounds per hour, 45·2 per cent −200 mesh.

Screen Test on Mill Discharge (taken over run):

Mesh	Weight, per cent
+ 35	0·3
+ 48	1·5
+ 65	9·4
+100	10·9
+150	14·3
+200	19·7
−200	43·9
	100·0

Mill Run No. 2

A second mill run was made in which certain modifications were carried out.

A trap was placed between the mill discharge and the amalgamation plate. The weight of rods in the mill was increased to 250 pounds and the rate of feed reduced.

Total feed to rod mill	486·0 pounds
Average rate of feed	74·7 " per hour

Screen Test on Trap Overflow:

Mesh	Weight, per cent, 60 lb/hr. feed	Weight, per cent, 82 lb/hr. feed	Weight, per cent, taken over run
+ 65	....	1·3	1·8
+100	0·4	6·8	6·3
+150	3·0	10·1	10·2
+200	14·7	18·5	21·8
−200	81·9	63·3	59·9
	100·0	100·0	100·0

Reagents added:

To mill, soda ash	1·2 lb./ton
To conditioner, sodium ethyl xanthate	0·28 "
To flotation cells, cresylic acid	0·2 "

Products	Assays, gold, oz/ton
Mill feed	2·98
Trap overflow	0·67
Plate overflow	0·505
Conditioner overflow	0·295
Flotation concentrate	13·06
Flotation tailing	0·025

Mill clean-up	Gold	17·02 oz/ton
Trap clean-up	Gold	86·68 "
Conditioner clean-up	Gold	2·12 "
Weight of trap clean-up		5·75 pounds
Weight of mill clean-up		34·0 "
Weight of conditioner tank clean-up		30·0 "

Screen Test on Flotation Tailing:

Mesh	Weight, per cent
+ 65	3·6
+100	5·4
+150	8·8
+200	20·5
−200	61·7
	100·0

Ratio of concentration, $\frac{13 \cdot 06 - 0 \cdot 025}{0 \cdot 295 - 0 \cdot 025} = 48 \cdot 2 : 1$

Gold retained in mill and trap	77·5	per cent
Gold recovery on amalgamation plate	24·6	"
Percentage of total gold recovered on plate, 24·6 per cent of 100−77·5 per cent	5·53	"
Gold in pump and conditioner tank, 41·58 per cent of (100−83·03 per cent)	7·06	"
Recovery of gold by flotation, $\frac{100 \times 13 \cdot 06\ (0 \cdot 295 - 0 \cdot 025)}{0 \cdot 295\ (13 \cdot 06 - 0 \cdot 025)} =$	91·7	"
Percentage of total gold in flotation concentrate, 91·7 per cent of 9·91 per cent	9·09	"
Overall recovery of gold, 77·5+5·53+7·06+9·09	99·18	"

The finer grinding and the use of a trap in the circuit made for better recoveries and better grade products.

CYANIDATION AND BARREL AMALGAMATION TESTS ON PRODUCTS OF LARGE-SCALE RUNS

Test No. 3

A charge of 914 grammes (dry weight) of the plate overflow from Mill Run No. 1 was ground wet for 20 minutes and then cyanided in a Denver Super-Agitator for 24 hours in a pulp ratio of 3 : 1, the solution having a KCN content of 1 pound per ton, and lime, at the start, of 1 pound per ton.

Product	Assay, gold, oz/ton	Extraction of gold, per cent	Final solution, lb/ton		Consumption, lb/ton	
			KCN	CaO	KCN	CaO
Feed	1·75	....	....	....	....	....
Cyanide tailing	0·015	99·14	1·5	0·15	0·65	7·55

Screen Test of Cyanide Tailing:

Mesh	Weight, per cent
+100	0·6
+150	4·1
+200	18·4
−200	76·9
	100·0

Test No. 4

A 2,000-gramme charge of flotation concentrate from Mill Run No. 1 was ground wet in a pebble jar for 15 minutes.

The charge was then cyanided for 48 hours in a Denver Super-Agitator. The pulp ratio was 3 : 1 and the cyanide strength 1 pound KCN per ton and CaO 5 pounds per ton. Both cyanide and lime consumption were fairly high.

Product	Assay, gold, oz/ton	Extraction of gold, per cent	Final solution, lb/ton		Consumption, lb/ton	
			KCN	CaO	KCN	CaO
Feed	5·26	....	....	....	....	....
Cyanide tailing	0·125	97·6	0·8	0·35	8·152	28·95

Screen Test of Cyanide Tailing:

Mesh	Weight, per cent
+100	0·05
+150	0·15
+200	2·20
−200	97·60
	100·00

Test No. 5

A charge of 923 grammes (dry weight) of flotation concentrate from Mill Run No. 1 was barrel-amalgamated after a 15-minute grind; 100 grammes of mercury, 1,000 c.c. of water, and 2 grammes of lime were used.

Gold in feed	5·26 oz/ton
Gold in tailings	1·86 "
Recovery	65·2 per cent

Test No. 6

This was a cyanidation test on the amalgamation plate overflow from Mill Run No. 2; 1,100 grammes of the filtered pulp was agitated in a Denver Super-Agitator, without regrinding, for 24 hours. The pulp ratio was 3 : 1, and the cyanide solution 1 pound KCN per ton, and lime (CaO) 5 pounds per ton.

Product	Assay, gold, oz/ton	Extraction of gold, per cent	Final solution, lb/ton		Consumption, lb/ton	
			KCN	CaO	KCN	CaO
Feed	0·505	....	....	....	....	....
Tailing	0·015	97·03	0·8	0·5	1·35	15·5

Test No. 7

The trap clean-up, 2,302 grammes, was barrel-amalgamated with 400 grammes of mercury. The clean-up product assayed in gold, 86·68 ounces per ton, and after amalgamation assayed 6·935 ounces per ton, showing a recovery of 92·0 per cent of the gold.

Screen Test on Amalgamation Tailing:

Mesh	Weight, per cent
+ 35	2·1
+ 48	2·2
+ 65	4·5
+100	13·7
+150	16·3
+200	26·2
−200	35·0
	100·0

CONCLUSIONS

The experimental tests indicate that the gold is largely present in the free state.

Grinding to have 60 per cent pass a 200-mesh screen gave satisfactory results.

Recoveries by three methods of treatment are shown for purposes of comparison:

1. The use of traps, amalgamation plate, flotation of amalgamation tailing, and cyanidation of flotation concentrate.

Gold recovery in trap and amalgamation	74·83	per cent
Gold recovery by cyanidation of flotation concentrate	17·62	"
Overall recovery	92·45	"

2. Similar to No. 1, but barrel amalgamation of flotation concentrate.

Gold recovery in trap and amalgamation	74·83	per cent
Gold recovery by barrel amalgamation of flotation concentrate	11·77	"
Overall recovery	86·60	"

3. The use of traps, amalgamation plate, and cyanidation of amalgamation tailing.

Gold recovery in trap and amalgamation	74·83	per cent
Gold recovery by cyanidation of amalgamation tailing	17·51	"
Overall recovery	92·34	"

Owing to the presence of free gold in the ore, there is a danger of this free gold passing through the flotation machine without being caught.

From the metallurgical point of view, the use of traps and blankets and cyanidation of blanket tailing is recommended.

A suggested flow-sheet would be to grind in cyanide solution with ball mill discharge to traps, the classifier overflow passing over blankets. The trap clean-up and blanket concentrate to be barrel-amalgamated.

Increased recovery could be expected in plant operation on account of cyanidation of all amalgamation tailings.

Ore Dressing and Metallurgical Investigation No. 577

CORUNDUM FROM DUNGANNON TOWNSHIP, ONTARIO

Shipment. A shipment consisting of six bags of corundum-bearing rock, gross weight 401 pounds, was received from lot 12, concession 12, Dungannon township, Renfrew county, Ont., on September 6, 1933. The shipper was W. J. McCoy, 45 Richmond Street West, Toronto, Ont.

Characteristics of the Rock. The shipment consisted of slightly bluish corundum, ranging in size from very small grains to pieces ¼ inch in diameter, in a gangue of white feldspar and biotite mica.

Sampling and Analysis. A head sample cut out after crushing to 10 mesh analysed as follows:—

Corundum	9·52 per cent
Lime	3·95 "
Magnesia	0·09 "

Experimental Tests

Test No. 1

After removing some specimens, the shipment was crushed to −10 mesh, and sized into the following by a Hummer screen:—

−10 +14	86 lb.	14 oz.
−14 +20	72 "	
−20 +26	33 "	1 "
−26 +35	31 "	7 "
−35 +42	22 "	
−42 +60	30 "	6 "
−60	83 "	5 "
	359 lb.	1 oz.

Each of the above sizes was tabled separately on a large Wilfley table, making a corundum concentrate and a tailing. Some galena and magnetite were present in the concentrates, especially those from the first two sizes. The table concentrates were dried and run over a high-power Ulhrich magnetic separator, which removed the magnetite and some mica. The Ullrich non-magnetic products were re-tabled, making a lead product, a corundum concentrate, and a tailing. The weights of the corundum concentrates are given below, with their analyses:—

Size	Weight	Per cent
−10 +14	8 lb. 7 oz.	70·42 corundum
−14 +20	10 "	66·46 "
−20 +26	3 " 14 "	68·40 "
−26 +35	3 " 2 "	75·32 "
−35 +42	2 "	79·50 "
−42 +60	3 " 10 "	75·76 "
−60	5 " 15 "	85·20 "
	37 lb.	

The concentrates are not high in grade, especially the first three sizes.

Test No. 2

In an effort to get a higher grade concentrate a test was made in which the feed was all crushed to pass 26 mesh. For this test, the products of Test No. 1 were mixed and crushed to pass 26 mesh and screened into the following sizes:—

−26 +35	119 lb.	12 oz.
−35 +42	43 "	7 "
−42 +60	43 "	4 "
−60	116 "	8 "
	322 lb.	15 oz.

The above sizes were tabled on a large Wilfley table, and the concentrates run over the Ullrich to remove magnetite and mica. The non-magnetics were re-tabled, making a lead product, a corundum concentrate and a tailing. The concentrates were:—

Size	Weight	Per cent
−26 +35	13 lb. 4 oz.	72·22 corundum
−35 +42	4 "	76·56 "
−42 +60	4 " 2 "	83·32 "
−60	10 " 8 "	90·10 "
	31 lb. 14 oz.	

In grinding samples high in corundum for analysis, the mortar and pestle used become worn away and the percentage of all constituents of the sample are reduced with the exception of silica which is increased. In Test No. 2, the mortar and pestle and sample were weighed before and after each sample was ground, and a corrected analysis worked out. The following table gives the results of Test No. 2:

Product	Weight in pounds	Corrected corundum, per cent	Pounds of corundum
−26+35 concentrate	13·25	75·29	9·98
−35+42 "	4·00	77·75	3·11
−42+60 "	4·12	83·73	3·45
−60 "	10·50	90·80	9·53
Total	31·87	81·80	26·07

The feed to Test No. 2 was 322 pounds 15 ounces. This at 9·52 per cent corundum would contain 30·74 pounds of corundum. The recovery in Test No. 2 would be $\frac{26\cdot07\times100}{30\cdot74}=84\cdot81$ per cent.

The corundum concentrates were analysed for total alumina. The results and the corrected results were:—

Size	Alumina, per cent	Corrected percentage of alumina
−26 +35	87·10	90·81
−35 +42	93·86	95·32
−42 +60	96·13	96·60
−60	93·34	94·07

The recovery in this test is good and the products, both in corundum and alumina content, are also good.

The difference between the corundum assay and the total alumina assay of the concentrates is due to the presence of other minerals containing alumina.

FLOTATION

Several flotation tests were made on the corundum rock. It was found that, although the corundum floated, much of the gangue also floated, which resulted in a concentrate containing about 25 per cent corundum.

CONCLUSIONS

The corundum can be recovered from the rock by tabling, magnetic separation, and re-tabling the non-magnetics, but it is doubtful whether a price can be obtained for the finished concentrate to meet the cost of production.

Ore Dressing and Metallurgical Investigation No. 578

GOLD ORE FROM McKENZIE-RED LAKE GOLD MINES, LIMITED, AT RED LAKE, ONTARIO

Shipment. A shipment of 20 sacks of ore, net weight 2,460 pounds, was received April 16, 1934, from John W. Shaw, New Liskeard, Ontario.

Characteristics of the Ore. Six polished sections were prepared and examined microscopically.

The gangue consists of green chloritic (?) schist and milky white vein quartz, both of which contain small irregular patches and grains of white dolomitic carbonate. Leucoxene occurs as disseminated grains, and has probably resulted from the alteration of ilmenite.

Pyrite is the most abundant ore mineral; it occurs as sparsely disseminated cubes and irregular grains in both schist and quartz, and as narrow stringers and veinlets in the schist. Rare small irregular grains of chalcopyrite are present in the schist and more rarely in the quartz. Much native gold is present in the quartz as irregular grains, often closely associated with pyrite.

A few grains of an unidentified bright white mineral occur in the schist. This closely resembles arsenopyrite in appearance and behaviour when etched; a micro-chemical analysis, however, gave strong tests for iron and sulphur but negative tests for arsenic.

A quantitative microscopic analysis of the gold in the polished sections was made, and the results, shown in the following table, give a rough indication of the grain size:

Grain Size of the Gold:

Mesh	Gold, per cent
+100	25·0
−100+200	32·0
−200+325	34·0
−325	9·0
	100·0

An average analysis of the ore was as follows:—

Gold	0·795 oz/ton
Silver	0·17 "

Experimental Tests

A series of small-scale tests was made on the ore to determine probable treatment for the recovery of the gold. Three large-scale mill runs were made to confirm results obtained.

The small-scale work consisted of tests by cyanidation, amalgamation, blanket concentration, and flotation. Cyanidation extracted 97·5 per cent of the gold in 48 hours when the ore was dry-crushed all through 150 mesh. If lime be added to the agitators about 27 pounds per ton of ore will be consumed, and yet the ore is sufficiently alkaline to be cyanided without the addition of any lime. By plate amalgamation 62·3 per cent of the gold can be extracted when the ore is ground 84 per cent through 200 mesh. At this same grinding 69·3 per cent of the gold can be recovered in a blanket concentrate amounting to 4·0 per cent of the weight of feed, the blankets being set at a slope of 2·5 inches per foot. In a small-scale flotation test 86·6 per cent of the gold was recovered in a concentrate amounting to 9·0 per cent of the weight of the feed used. By passing the flotation tailing over a corduroy blanket an additional 8·2 per cent of the gold was recovered in a blanket concentrate amounting to 1·5 per cent of the weight of the original ore fed to the flotation cell. The blanket tailing assayed 0·07 ounce per ton in gold.

The large-scale mill runs were made in a unit of 100 pounds per hour capacity. The ore was ground in a rod mill, the discharge from which went to a flotation cell. A concentrate was taken off here and the tailing went to a classifier, the overflow from which went to a battery of cells where another concentrate was taken off. The classifier oversize was returned to the mill for regrinding. The flotation tailing was passed over blankets. An average of 93·0 per cent of the gold was recovered in the two concentrates, the average assay of which would be about 9 ounces per ton in gold. The flotation tailings assayed 0·06 ounce per ton in gold, and little or nothing was taken out of them by blanketing.

Details of the tests follow:

GRINDING

Samples of the ore at −14 mesh were ground in ball mills for different periods of time and the dried pulp passed through a series of screens to determine the amount of grinding that had been done. The results of these tests are laid down in the following table:

Mesh	Weight, per cent			
	Ground for 15 minutes	Ground for 20 minutes	Ground for 25 minutes	Ground for 30 minutes
+ 48	....	....	....	....
− 48+ 65	0·8	0·1	....	....
− 65+100	6·4	3·1	0·7	0·5
−100+150	10·4	7·0	3·2	2·4
−150+200	17·0	15·8	12·0	11·1
−200	65·4	74·0	84·1	86·0
Total	100·0	100·0	100·0	100·0

CYANIDATION

Tests Nos. 1 to 8

Samples of the ore, ground dry to various sizes, were agitated in cyanide solution, 1·0 pound KCN per ton, for periods of 24 and 48 hours. The tailings were assayed for gold.

Summary:

Feed sample: 0·795 oz/ton

Test No.	Mesh	Period of agitation, hours	Tailing assay, gold, oz/ton	Extraction, per cent	Reagents consumed, lb/ton	
					KCN	CaO
1	−48	24	0·05	93·7	0·30	26·0
2	−100	24	0·025	96·9	0·75	28·2
3	−150	24	0·03	96·2	0·45	27·8
4	−200	24	0·10	87·4	0·60	28·8
5	− 48	48	0·045	94·3	0·60	26·4
6	−100	48	0·025	96·9	0·90	28·4
7	−150	48	0·02	97·5	0·90	27·6
8	−200	48	0·10	87·4	1·20	28·7

NOTE.—No explanation can be given for the high tailing assays when the ore was ground all through 200 mesh. A sample of the tailing was re-agitated for a short time in cyanide solution and then barrel-amalgamated, but still assayed the same.

AMALGAMATION AND CYANIDATION

Tests Nos. 9 and 10

Samples of the ore, ground dry through 48- and 100-mesh screens, were amalgamated with mercury in a jar mill for 30 minutes. The amalgamation tailings were filtered, sampled and assayed, and portions of each agitated in cyanide solution, 1·0 pound KCN per ton, for periods of 24 hours. The cyanide tailings were also assayed for gold.

Summary:

Test No.	Mesh	Assay amalgamation tailing, gold, oz/ton	Extraction, per cent	Assay cyanide tailing, gold, oz/ton	Extraction, per cent	Total extraction, per cent	Reagents consumed, lb/ton	
							KCN	CaO
9	− 48	0·27	66·0	0·045	28·3	94·3	0·60	25·50
10	−100	0·38	52·2	0·02	45·3	97·5	0·90	27·60

It will be observed that the washing the pulp was subjected to during amalgamation and subsequent separation of amalgam from pulp did not reduce materially the consumption of lime during the agitation in cyanide solution that followed.

BLANKET CONCENTRATION

Tests Nos. 11 to 14

Samples of the ore, ground in ball mills approximately 65, 74, 84, and 86 per cent through 200 mesh, were passed over corduroy blankets set at a slope of 2·5 inches per foot. The concentrates and tailings were assayed for gold.

Summary:

Test No.	Product	Grinding, per cent −200 mesh	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
11	Concentrate	65·4	9·7	6·90	66·1
	Tailing	....	90·3	0·38	33·9
	Feed (cal.)	....	100·0	1·01	100·0
12	Concentrate	74·0	3·7	14·50	65·8
	Tailing	....	96·3	0·29	34·2
	Feed (cal.)	....	100·0	0·82	100·0
13	Concentrate	84·0	3·9	15·0	69·3
	Tailing	....	96·1	0·27	30·7
	Feed (cal.)	....	100·0	0·84	100·0
14	Concentrate	86·0	3·8	12·86	64·9
	Tailing	....	96·2	0·275	35·1
	Feed (cal.)	....	100·0	0·75	100·0

HYDRAULIC CLASSIFICATION

Test No. 15

A sample of the ore, ground 74·0 per cent through 200 mesh, was put through a hydraulic classifier where coarse gold and heavy minerals were allowed to settle out against a slowly rising current of water. The classifier oversize and overflow were assayed for gold.

Summary:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Classifier oversize	1·5	18·97	34·6
Classifier overflow	98·5	0·545	65·4
Feed (cal.)	100·0	0·82	100·0

PLATE AMALGAMATION

Test No. 16

A sample of the ore, ground 84·0 per cent through 200 mesh, was passed over a small amalgamation plate. The plate tailing was assayed for gold.

Results:

Feed sample	0·795 oz/ton gold
Plate tailing	0·30 "
Extraction	62·3 per cent

CYANIDATION WITHOUT LIME

Tests Nos. 18 to 21

As it was thought that carbonates in the ore were responsible for the high lime consumption, a test was made to find out whether the ore was acid or alkaline. To do this, a sample of the ore was ground with distilled water and filtered and the pH value of the filtrate determined. This was found to be 8·2 to 8·4, which is slightly alkaline. A cyanidation test was then made, using soda ash, 2·0 pounds per ton ore, for protective alkalinity, and little or none of this was consumed. Consumption of cyanide was reduced to the low figure of 0·20 pound per ton of ore. This test was made solely as confirmatory of the contention that the lime was being destroyed as active lime by the formation of insoluble calcium carbonate.

Three other cyanidation tests were made without addition of lime or soda ash. Good extractions were obtained, but cyanide consumption was higher, reaching as much as 1·13 pounds per ton of ore. In these four tests the grinding was 84 per cent through 200 mesh.

Summary:

Test No.	Protective alkali	Tailing assay, gold, oz/ton	Extraction, per cent	Consumption KCN lb/ton ore
18	Na_2CO_3	0·05	93·7	0·20
19	None	0·03	96·5	1·10
20	None	0·03	96·5	1·13
21	None	0·035	95·6	1·13

SETTLING TESTS

A number of settling tests were made to find how the ore would act when agitated without lime, particularly with respect to cyanide consumption, extraction, and thickening.

A batch of ore was ground in a ball mill and agitated in cyanide solution for 24 hours without lime. The pulp was then transferred to a settling tube 4 feet high and 2·1 inches inside diameter. The pulp level was observed and recorded at three-minute intervals for a period of time. Then lime was added, the pulp shaken up and allowed to settle again, and another set of readings was taken. This was repeated several times, more lime being added each time and in some cases the dilution was increased.

The results of the settling tests carried out on the ore can be laid down in the following table:

Results:

Time	Pulp Levels and Conditions				
	Dilution, 2 : 1	Dilution, 2 : 1	Dilution, 2 : 1	Dilution, 2·5 : 1	Dilution, 4 : 1
	KCN—1·25 lb/ton solution	KCN—1·25 lb/ton solution	KCN—1·25 lb/ton solution	KCN—1·00 lb/ton solution	KCN—0·625 lb/ton solution
	CaO added, none	CaO added, 2·0 lb/ton	CaO added, 4·0 lb/ton	CaO added, 6·0 lb/ton	CaO added, 8·0 lb/ton
	feet	feet	feet	feet	feet
Start	3·140	3·140	3·140	3·705	3·750
3 minutes	3·125	3·125	3·115	3·675	3·650
6 "	3·105	3·105	3·090	3·645	3·540
9 "	3·090	3·090	3·060	3·615	3·430
12 "	3·075	3·075	3·035	3·580	3·320
15 "	3·055	3·060	3·015	3·550	3·210
18 "	3·040	3·045	2·990	3·520	3·100
21 "	3·025	3·030	2·965	3·495	2·990
24 "	3·010	3·015	2·945	3·465	2·880
27 "	2·990	3·005	2·925	3·435	2·770
30 "	2·975	2·990	2·900	3·405	....
33 "	2·960	2·975	2·880	3·375	....
36 "	2·945	2·960	2·860	3·350	....
39 "	2·930	2·950	2·835	3·320	....
42 "	2·910	2·935	2·815	3·290	....
45 "	2·895	2·920	2·795	3·260	....
48 "	2·880	2·905	2·775	3·235	....
51 "	2·865	2·890	2·755	3·205	....
54 "	2·845	2·880	2·730	3·180	....
57 "	2·830	2·865	2·710	3·150	....
60 "	2·815	2·850	2·690	3·125	....

The sand was allowed to settle from a sample of pulp in which the dilution was four of solution to one of solids. The overflow from this, with about 5 per cent solids held in suspension, was transferred to the settling tube and shaken up with lime at the rate of 2·0 pounds per ton of original ore. The solids settled out at the rate of 4·5 to 5·0 feet per hour, leaving a clear overflow.

ANALYSES OF SOLUTIONS

Further analyses were made on the feed sample, and on solutions with which it was treated, to see what could be found about the nature of the ore. The following determinations were made:

Feed Sample:

	Per cent
Total iron	5·50
Ferrous iron	3·66
Acid soluble $CaCO_3$	11·7
Acid soluble $MgCO_3$	7·7
Total sulphur	0·98
Acid soluble SO_3	0·09
CO_2	9·82

The carbon dioxide is only enough to combine with the lime and magnesia, so that the analysis does not indicate the presence of iron carbonate.

A sample of the ore was ground with distilled water, filtered, and the filtrate analysed as follows:

pH value	8·2 to 8·4
Calcium carbonate	0·057 grm/litre
Magnesium carbonate	0·046 "
Sulphur trioxide	0·048 "
Carbon dioxide	0·182 "
Total solids	0·365 "
Iron	0·006 "
Reducing power	Nil

A sample of the ore was ground and agitated in cyanide solution for 24 hours without addition of lime. The solution was filtered off and the following determinations made on it:

Calcium carbonate	0·020 grm./litre
Magnesium carbonate	Nil
Sulphur trioxide	0·063 grm./litre
Iron	0·022 "
Reducing power	18·3 c.c. $\frac{N}{10}$ $KMnO_4$/litre

Another sample of ore was agitated in cyanide solution with an excess of lime. The solution was filtered off and analysed as follows:

Magnesium carbonate	Trace
Sulphur trioxide	0·030 grm./litre
Iron	0·0002 "
Reducing power	14·6 c.c. $\frac{N}{10}$ $KMnO_4$/litre

CYANIDATION WITH LIME ADDED NEAR THE END OF AGITATION PERIOD

Test No. 28

A sample of the ore was ground 84 per cent through 200 mesh in a ball mill and agitated in cyanide solution, 1·0 pound KCN per ton, for 20 hours. Lime at the rate of 4 pounds per ton ore was then added, and agitation continued for another four hours. The tailing was assayed for gold.

Results:

Feed sample	0·795 oz/ton
Cyanide tailing	0·025 "
Extraction	96·9 per cent
KCN consumed	1·10 lb/ton ore
CaO consumed	3·40 "

AERATION FOLLOWED BY CYANIDATION

Test No. 29

A sample of the ore was ground 84 per cent through 200 mesh in a ball mill and agitated with water in a Denver Super-Agitator for 16 hours. Lime and cyanide were then added and agitation continued for another 24 hours.

Results:

Feed sample	0·795 oz/ton
Cyanide tailing	0·06 "
Extraction	92·5 per cent
KCN consumed*	1·85 lb/ton ore
CaO consumed	5·85 "

Test No. 30

A sample of the ore was ground 84 per cent through 200 mesh and agitated in water in a Denver Super-Agitator for five hours. The pulp was then transferred to a bottle and agitated with lime and cyanide for 24 hours.

Results:

Feed sample	0·795 oz/ton
Cyanide tailing	0·015 "
Extraction	98·1 per cent
KCN consumed	0·35 lb/ton ore
CaO consumed	6·0 "

FLOTATION AND BLANKETING

Test No. 22

A sample of the ore was ground 86 per cent through 200 mesh in a ball mill and then floated. The flotation tailing was passed over a corduroy blanket set at a slope of 2·5 inches per foot. The products were all assayed for gold.

Charge to ball mill:

Ore	6,000 grammes at −14 mesh
Water	4,500 c.c.
Sodium carbonate	1·0 lb/ton
Grinding time	30 minutes

(*Note.—According to experience in the Ore Dressing and Metallurgical Laboratories, tests run in the Denver Super-Agitator always show more than normal cyanide consumption. This is probably due to the excessive aeration produced in this apparatus. This result should be compared with results of Test No. 30, when a bottle was used.)

Reagents to cell:

Sodium ethyl xanthate	0·10 lb/ton
Cresylic acid	0·09 "

Summary:

Product	Weight, per cent	Assay, gold, oz/ton	Distribution of gold, per cent
Flotation concentrate	9·0	11·60	86·6
Blanket concentrate	1·5	6·58	8·2
Blanket tailing	89·5	0·07	5·2
Feed (cal.)	100·0	1·20	100·0

Three large-scale mill runs were made on the ore, using a unit of 100 pounds per hour capacity. The ore at −14 mesh was fed to a rod mill the discharge from which went to a single flotation cell. A concentrate was taken off here and the tailing went to an Akins classifier. The classifier oversize was returned to the rod mill for regrinding, and the overflow went through a conditioning tank to a battery of flotation cells, where another concentrate was taken off. The flotation tailing was passed over a blanket set at a slope of 2·5 inches per foot. Samples of products at various stages in the flow-sheet were taken at regular intervals and assayed.

Mill Run No. 1

In the first run, the feed rate was kept at 140 pounds per hour. The mill discharge was 39 per cent through 200 mesh, and the classifier overflow was 57 per cent through 200 mesh.

The reagents used were as follows:

To rod mill:

Barrett No. 4	0·18 lb/ton

To cells:

Potassium amyl xanthate	0·16 lb/ton
Cresylic acid	0·03 "

The potassium amyl xanthate was divided, one-quarter going to unit cell and three-quarters to conditioning tank.

Assays:

	Gold, oz/ton
Mill feed	0·75
Rod mill discharge	0·41
Unit cell concentrate	14·95
Unit cell tailing	0·24
Classifier overflow	0·15
Flotation concentrate	2·86
Flotation tailing	0·06
Blanket tailing	0·06
Blanket concentrate	0·41

In the two concentrates 92·6 per cent of the total gold is recovered, with a ratio of concentration of 14 : 1, approximately. Average assay value of the two concentrates is 9·7 ounces per ton in gold.

Mill Run No. 2

The second mill run was similar to the first except that soda ash, 1·0 pound per ton, was added to the rod mill.

Assays:

	Gold, oz/ton
Mill feed	0·76
Rod mill discharge	0·52
Unit cell concentrate	23·12
Unit cell tailing	0·22
Classifier overflow	0·14
Flotation concentrate	1·23
Flotation tailing	0·06
Blanket tailing	0·05
Blanket concentrate	0·21

In this run the two concentrates account for 92·9 per cent of the gold, with a ratio of concentration of 10·7 : 1. The average assay value of the two concentrates is approximately 7·5 ounces per ton in gold.

Mill Run No. 3

In Mill Run No. 3 the feed rate was reduced to 100 pounds per hour, giving a rod mill discharge 46 per cent through 200 mesh and a classifier overflow 68·5 per cent through 200 mesh. The reagents were kept the same as in Mill Run No. 2.

Assays:

	Gold, oz/ton
Mill feed	0·74
Rod mill discharge	0·42
Unit cell concentrate	15·52
Unit cell tailing	0·22
Classifier overflow	0·14
Flotation concentrate	2·36
Flotation tailing	0·055
Blanket tailing	0·04
Blanket concentrate	0·26

In this run, 93·1 per cent of the gold is recovered in the two flotation concentrates, with a ratio of concentration of 13·4 : 1. The average assay value of the two concentrates is about 9·3 ounces per ton in gold.

CYANIDATION OF CONCENTRATES

Tests Nos. 23 to 26

Samples of the unit cell concentrate and of the flotation concentrate produced in Mill Run No. 3 were agitated in cyanide solution for periods of 24 hours with and without grinding. The rich unit cell concentrate was agitated in solution containing 5·0 pounds KCN per ton, and the flotation concentrate in solution containing 2·0 pounds KCN per ton; 98·1 per cent of the gold was extracted from the reground unit cell concentrate, and 78·8 per cent of the gold was extracted from the reground flotation concentrate, the cyanide tailings being 0·30 and 0·50 ounce per ton in gold respectively.

Summary:

Test No.	Feed assay, gold, oz/ton	Cyanide tailing assay, gold, oz/ton	Reground cyanide tailing assay, gold, oz/ton	Extraction, per cent	Reagents consumed, lb/ton ore	
					KCN	CaO
23	15·52	0·40	....	97·4	4·16	4·5
24	15·52	....	0·30	98·1	7·16	4·6
25	2·36	0·585	....	75·2	1·78	6·6
26	2·36	....	0·50	78·8	2·53	6·6

CONCLUSIONS

The test work has indicated three possible flow-sheets for the treatment of this ore:

1. Flotation followed by cyanidation of the flotation concentrate;
2. Grinding in cyanide solution and cyaniding without lime;
3. Grinding in water, with blanket concentration in the grinding circuit between the mills and classifiers, and pre-aeration of the pulp before the cyanide solution is brought into contact with the ore.

By the first method, flotation and cyanidation of the flotation concentrate, it will be difficult to exceed a 90 per cent recovery of the gold. The best result obtained in the test work was about 88 per cent. The flow-sheet used contained a unit cell at the ball mill discharge where the bulk of the gold was recovered.

The second method, cyaniding without lime and grinding in cyanide solution, should, according to the test results, recover 95 per cent of the gold, with a lime consumption of about 5 pounds per ton and a cyanide consumption of 1·13 pounds per ton. A brief outline of the flow-sheet is as follows: Grind in cyanide solution without lime, the classifier overflow going to a thickener. If the solution from this thickener is rich enough to go to precipitation it can go to a mixer where lime can be added to coagulate any suspended slimes before clarification. The thickened pulp could go direct to the agitators, still without lime being added. After agitation, lime can be added to the pulp going to the thickener for settling prior to filtering.

The third method calls for grinding in water and pre-aerating the pulp before cyanidation, and the use of blankets to treat the circulating load in the grinding circuit in order to prevent an excessive accumulation of gold in the grinding circuit. This method, from the results of the test work and by comparison with the other two methods, should recover 98 per cent of the gold, with a cyanide consumption of 0·35 pound and a lime consumption of 6 pounds per ton of ore. A brief outline of the flow-sheet would be as follows: Grinding in water, the circulating load of the grinding circuit to be passed over corduroy blankets; the overflow of the classifiers to be thickened in a thickener, where a small amount of lime can be added for settling purposes. The thickener underflow would then go to Pachuca tanks, where the pulp would be aerated for about four hours, after which the cyanide solution can be added. By this method, the cyanide consumption can be reduced from 1·13 pounds to 0·35 pound per ton, but to this latter figure must be added an additional loss from the excess solution which after precipitation must be run to waste. This loss would probably be between 0·25 and 0·5 pound per ton of ore. Using the maximum figure of 0·5 pound, the total cyanide consumption would be 0·85 pound per ton in comparison with 1·13 pounds from flow-sheet No. 2.

The concentrate obtained on the blanket would be barrel-amalgamated. The test work shows that over 65 per cent of the gold should be recovered at this stage of the treatment. This would have the advantage of reducing the grade of the feed going to cyanidation.

In reference to this third flow-sheet, particular attention is directed to Tests Nos. 11 to 14 and to Tests Nos. 28 to 30. The former have to do with blanket concentration and the latter with cyanidation after aeration in water.

Ore Dressing and Metallurgical Investigation No. 579

CYANIDATION AND FLOTATION OF BEATTIE CONCENTRATE

Shipment. A shipment consisting of 40 bags of moist flotation concentrate, weighing approximately 4,000 pounds, was received May 16, 1934, from the Beattie Gold Mines, Limited, Noranda, Quebec.

Purpose of Experimental Tests. The shipment was made to determine what extraction of gold by cyanidation could be obtained from the concentrate when ground to pass 98 per cent through 325 mesh. The cyanide tailing was then required to be concentrated by flotation to note the grade and quantity of concentrate that could be produced.

Experimental Tests

Three small-scale tests were made on a small shipment of concentrate previously received for similar test work. This lot had an assay value of 1·125 ounces gold per ton, the larger shipment assaying 1·23 ounces per ton.

Test No. 1

A sample, 1,000 grammes of concentrate, 750 c.c of water, and 5 grammes of lime were ground for two hours in a porcelain mill containing iron balls. This gave a product 98 per cent minus 325 mesh. The balls were then removed and the pulp agitated in a Denver Super-Agitator (Wallace type) for four hours at a dilution of 1 of ore to 3 of water.

The pulp was then filtered, made up to 1 : 3 dilution with a sodium cyanide solution equivalent in strength to 1·5 pounds KCN per ton, and agitated for 40 hours; 11·0 grammes of lime were added to maintain protective alkalinity.

Cyanidation Results:

Feed	1·125 oz./ton
40-hour cyanide tailing	0·25 "
Extraction	77·8 per cent

Titrations after 40 hours' agitation: potassium cyanide—0·5 lb.; CaO—0·15 lb.

Reagent consumption:	Sodium cyanide	2·37 lb./ton ore
	Lime	10·5 "

Flotation:

The cyanide tailing was filtered and washed twice on the filter. It was then conditioned with 3·3 pounds soda ash per ton for 10 minutes, after which 2·2 pounds copper sulphate, 0·55 pound sodium ethyl xanthate, and 0·22 pound pine oil were added and a flotation concentrate taken off for 20 minutes. Flotation was very sluggish.

Results:

Product	Weight, grammes	Weight, per cent	Assay, gold, oz/ton	Distribution, gold, per cent
Feed=Cyanide tailing	897·2	100·0	0·25	100·0
Concentrate	235·2	26·2	0·53	55·6
Tailing	662·0	73·8	0·15	44·4

Total recovery, cyanidation and flotation........................ 90·1 per cent

Test No. 2

This test is similar to Test No. 1. Cyanidation practice was identical, but the quantities of flotation reagents were changed as indicated below.

Cyanidation Results:

Feed	1·125	oz./ton
40-hour cyanide tailing	0·26	"
Extraction	76·9	per cent

Titrations after 40 hours' agitation: potassium cyanide—0·6 lb.; CaO—0·15 lb.

Reagent consumption: Sodium cyanide	2·1	lb./ton
Lime	10·5	"

Flotation:

A cyanide tailing was filtered, re-pulped with water, re-filtered, and washed.

Reagents:	Lb/ton
Soda ash	5·9
Copper sulphate	1·2
Sodium ethyl xanthate	0·59
Pine oil	0·24
Flotation time	15 minutes

Results:

Product	Weight, grammes	Weight, per cent	Assay, gold, oz/ton	Distribution, gold, per cent
Feed=Cyanide tailing	845·0	100·0	0·26	100·0
Concentrate	345·0	40·8	0·50	76·6
Tailing	500·0	59·2	0·105	23·4

The froth in this test was much more lively than that of Test No. 1, owing to the increase in soda ash.

Total recovery, cyanidation and amalgamation.................. 94·3 per cent

Test No. 3

This test is similar to Test No. 2. The cyanide agitation period was reduced from 40 hours to 24 hours.

Cyanidation Results:

Feed	1·125	oz./ton
Cyanide tailing	0·275	"
Extraction	75·6	"

Titrations after 24 hours' agitation: potassium cyanide—0·8 lb.; CaO—0·075 lb.

Reagent consumption: Sodium cyanide	1·65	lb./ton
Lime	7·8	"

Flotation:

Reagents:	Lb/ton
Soda ash	5·8
Copper sulphate	1·2
Sodium xanthate	0·59
Pine oil	0·24
Flotation time	15 minutes

After the flotation concentrate was removed, the pulp was acidified with sulphuric acid and a second concentrate removed.

Results:

Product	Weight, grammes	Weight, per cent	Assay, gold, oz/ton	Distribution, gold, per cent
Feed = Cyanide tailing	854·2	100·0	0·275	100·0
Concentrate No. 1	285·0	33·4	0·56	70·1
Concentrate No. 2	47·2	5·5	0·34	7·0
Tailing	522·0	61·1	0·10	22·9

Tests Nos. 4 to 9

A series of tests was undertaken to note the effect of various grinds on the recovery by cyanidation. This was made on the shipment of concentrate received May 16, and the grinding was done in the same jar mill as were Tests Nos. 1, 2, and 3. Tests Nos. 8 and 9 were cyanided in a Denver Super-Agitator, omitting the 4-hour pre-liming agitation.

Results:

Test No.	Hours of grinding	Feed, gold, oz/ton	Tailing, gold, oz/ton	Extraction, per cent	Reagent consumption, lb/ton	
					KCN	CaO
4	$\frac{3}{4}$	1·23	0·375	69·5	1·25	6·8
5	1	1·23	0·355	71·1	1·25	6·7
6	$1\frac{1}{4}$	1·23	0·35	71·5	1·25	7·2
7	$1\frac{1}{2}$	1·23	0·325	73·6	1·5	7·4
8	1	1·23	0·37	69·9	3·0	7·8
9	1	1·23	0·37	69·9	3·3	7·8

These results, together with those of Tests Nos. 1, 2, and 3, show that the fineness of grind has a direct bearing on the extraction by cyanide.

Omitting the 4-hour pre-liming period and agitating in Denver Super-Agitator lowers the extraction and increases cyanide consumption.

MILL RUNS

Two tests were made on a larger scale than the above on the larger shipment.

Mill Run No. 1

The concentrate was mixed with lime equal to 10 pounds per ton and fed at the rate of 50 pounds per hour to a rod mill in closed circuit with a classifier. The classifier overflow at 25 per cent solids with a screen analysis of 94 per cent minus 325 mesh was pumped to a Pachuca agitator.

When 150 pounds of concentrate had passed to the agitator, milling was discontinued and the pulp agitated for four hours. The charge was then filtered and the cake returned to the agitator where it was made up to 25 per cent solids with cyanide solution equivalent in strength to 1·5 pounds potassium cyanide per ton. Lime was added from time to time to maintain the protective alkalinity at about 0·2 pound lime per ton. Agitation was concluded after 36 hours. Samples were taken for assay and flotation tests at 4-, 8-, 12-, 16-, and 24-hour periods.

Cyanidation Results:

Period of agitation, hours	Feed, gold, oz/ton	Tailing, gold, oz/ton	Extraction, per cent
4	1·23	0·385	68·7
8	1·23	0·38	69·1
12	1·23	0·375	69·5
16	1·23	0·375	69·5
24	1·23	0·375	69·5
30	1·23	0·36	70·7
36	1·23	0·365	70·3

Reagent consumption after 36 hours:

Sodium cyanide	0·6 lb/ton
Lime	10·4 "

Flotation and Cyanide Tailing. Samples were taken from the agitator at 4-, 8-, 12-, 16-, and 24-hour periods, filtered, re-pulped, re-filtered, and washed. The samples were then conditioned with soda ash for 10 minutes, after which copper sulphate, xanthate, and pine oil were added and flotation carried on for 16 minutes. The flotation of the 4-hour sample was continued an additional 7 minutes to remove a middling product.

4-hour sample:

Reagents:

	Lb/ton	
Soda ash	4·0	
Copper sulphate	0·8	
Sodium xanthate	0·4	1st flotation
Pine oil	0·2	
Amyl xanthate	0·16	2nd flotation
Pine oil	0·08	

Results:

Product	Weight, grammes	Weight, per cent	Assay, gold, oz/ton	Distribution, gold, per cent
Feed=Cyanide tailing	1,240	100·0	0·385	100·0
Concentrate No. 1	518	41·8	0·76	80·1
Concentrate No. 2	136	11·0	0·33	9·2
Tailing	586	47·2	0·09	10·7

8- to 24-hour samples:

Reagents:

	Lb/ton
Soda ash	4·0
Copper sulphate	0·8
Amyl xanthate	3·0
Pine oil	0·2

Results:

Product	Weight, grammes	Weight, per cent	Assay, gold, oz/ton	Distribution, gold, per cent
8-hour sample:				
Feed=Cyanide tailing	1,275·4	100·0	0·38	100·0
Concentrate	599·0	47·0	0·66	83·0
Tailing	676·4	53·0	0·12	17·0
12-hour sample:				
Feed=Cyanide tailing	1,271·2	100·0	0·375	100·0
Concentrate	643·7	50·6	0·66	88·3
Tailing	627·5	49·4	0·09	11·7
16-hour sample:				
Feed=Cyanide tailing	1,299·0	100·0	0·375	100·0
Concentrate	608·0	46·8	0·63	84·1
Tailing	691·0	53·2	0·105	15·9
24-hour sample:				
Feed=Cyanide tailing	1,308·7	100·0	0·375	100·0
Concentrate	536·2	41·0	0·71	77·9
Tailing	772·5	59·0	0·14	22·1

Total Recoveries—Cyanidation and Flotation:

4 hours	93·8 per cent
8 "	94·7 "
12 "	96·4 "
16 "	95·1 "
24 "	93·3 "

Mill Run No. 2

In this run, an attempt was made to produce a finer grind. The slope of the classifier was reduced, giving a classifier overflow of 98 per cent minus 325 mesh. The procedure of the test was the same as in the preceding run. Samples were taken after 4, 8, and 12 hours, when cyanidation was concluded.

Cyanidation Results:

Period of agitation, hours	Classifier overflow, gold, oz/ton	Tailing, gold, oz/ton	Per cent extraction
4	1·20	0·375	68·7
8	1·20	0·365	69·6
12	1·20	0·37	69·2

Solution after 12 hours' agitation:

Potassium cyanide	1·35 lb./ton
Lime	0·2 "

Reagent consumption:

Sodium cyanide	0·35 lb/ton
Lime	5·8 "

Flotation of Cyanide Tailing. Samples taken at 4, 8, and 12 hours were floated as in the preceding run. Owing to larger samples having been taken, the reagent additions were less than in Mill Run No. 1.

Reagents:

	Lb/ton
Soda ash	2·8
Copper sulphate	0·6
Amyl xanthate	0·23
Pine oil	0·14

Results:

Product	Weight, grammes	Weight, per cent	Assay, gold, oz/ton	Distribution, gold, per cent
4-hour sample				
Feed=Cyanide tailing	1,753·5	100·0	0·375	100·0
Concentrate	763·5	43·5	0·72	83·5
Tailing	990·0	56·5	0·11	16·5
8-hour sample:				
Feed=Cyanide tailing	1,773·5	100·0	0·365	100·0
Concentrate	597·5	33·7	0·75	67·9
Tailing	1,176·0	66·3	0·18	32·1
12-hour sample:				
Feed=Cyanide tailing	1,779·0	100·0	0·37	100·0
Concentrate	621·0	34·9	0·74	70·0
Tailing	1,158·0	65·1	0·17	30·0

Here again is shown the effect of increased time of cyanidation on the flotability of the sulphides. The 4-hour sample was much faster floating than the others. However, the rate of flotation can be governed by reagent conditions.

CYANIDATION OF ROASTED FLOTATION CONCENTRATE

The flotation concentrate obtained from a large sample of cyanide tailing after 12 hours' agitation was dried and roasted until free from arsenic and sulphur fumes. Toward the end of the roast, the temperature was raised to approximately 950°C. The product from the roast had the following analysis: gold, 0·93 oz/ton; arsenic, 0·39 per cent; sulphur trioxide, 2·68 per cent; sulphur, 1·09 per cent; lime, 1·19 per cent; iron, 42·2 per cent. Roasting resulted in a loss in weight of 30·6 per cent.

After roasting, the product was ground for 30 minutes in water, filtered, and washed. Portions were then cyanided for 24, 48, and 72 hours, 1 : 3 dilution, with a cyanide solution equivalent in strength to 2·0 pounds potassium cyanide per ton. Seven to nine pounds of lime was added to supply protective alkalinity.

Results:

24-hour agitation:

Feed		0·93	oz/ton
Tailing		0·53	"
Extraction		43·0	per cent
Reagent consumption:	Potassium cyanide	1·2	lb/ton
	Lime	6·0	"

48-hour agitation:

Tailing		0·525	oz/ton,
Extraction		43·6	per cent
Reagent consumption:	Potassium cyanide	1·2	lb/ton
	Lime	6·3	"

72-hour agitation:

Tailing		0·535	oz/ton,
Extraction		42·2	per cent
Reagent consumption:	Potassium cyanide	1·05	lb/ton
	Lime	7·8	"

CYANIDATION OF ROASTED BEATTIE MILL CONCENTRATE

A sample of the concentrate as received was given an oxidizing roast, finishing at about 900°C. The roasted product had the following analysis:

Gold	1·53	oz/ton
Arsenic	0·26	per cent
Sulphur	1·83	"
Sulphur trioxide	4·33	"
Lime	3·56	"

Test No. R-1

A sample of the roasted concentrate was ground for one hour in water, filtered, and washed. It was then agitated for 48 hours, 1 : 3 dilution, with a cyanide solution equivalent in strength to 1·5 pounds potassium cyanide per ton. Seven pounds of lime per ton was added.

Results:

24-hour agitation:

Feed		1·53	oz/ton,
Tailing		0·17	"
Extraction		88·9	per cent
Reagent consumption:	Potassium cyanide	0·75	lb/ton
	Lime	5·4	"

48-hour agitation:

Tailing		0·185	oz/ton,
Extraction		87·9	per cent
Reagent consumption:	Potassium cyanide	1·5	lb/ton
	Lime	6·5	"

Test No. R-2

A sample was ground as above together with 10 pounds of lime per ton, filtered, and washed. It was then cyanided as in Test No. R-1.

Results:

24-hour agitation:

Feed		1·53	oz/ton,
Tailing		0·185	"
Extraction		87·9	per cent
Reagent consumption:	Potassium cyanide	0·9	lb/ton
	Lime	5·3	"

48-hour agitation:

Tailing		0·145	oz/ton,
Extraction		90·5	per cent
Reagent consumption:	Potassium cyanide	1·2	lb/ton
	Lime	6·3	"

Test No. R-3

A sample was ground in water with 10 pounds of lime per ton. It was then agitated for 48 hours, 1 : 2½ dilution, cyanide being added to bring the strength of the solution to 1·5 pounds potassium cyanide per ton.

Results:

Feed		1·53	oz/ton,
Tailing		0·15	"
Extraction		90·1	per cent
Reagent consumption:	Potassium cyanide	1·0	lb/ton
	Lime	10·5	"

Test No. R-4

A sample was ground in water without lime and the pulp was then diluted to 1 : 2½, cyanide added to make a 1·5 pound potassium cyanide per ton solution, and 10 pounds of lime per ton added. Agitation was carried on for 48 hours.

Results:

Feed		1·53	oz/ton,
Tailing		0·15	"
Extraction		90·1	per cent
Reagent consumption:	Potassium cyanide	2·7	lb/ton
	Lime	10·5	"

Test No. R-5

A sample was ground for one hour in water with 10 pounds of lime per ton and then aerated for 4 hours, filtered, and washed. It was then cyanided for 48 hours, 1 : 2½ dilution, with a 2·0 pound potassium cyanide per ton solution. Lime was added to maintain about 0·2 pound lime in the solution.

Results:

Feed		1·53	oz/ton,
Tailing		0·15	"
Extraction		90·1	per cent
Reagent consumption:	Potassium cyanide	1·4	lb/ton
	Lime	19·6	"

These results indicate that to obtain highest recoveries with low reagent consumption, the roasted product should be ground with lime. Filtering of the pulp prior to cyanidation apparently is not necessary.

CONCLUSIONS

The recovery by cyanidation, as indicated in Tests Nos. 1, 2, and 3, will only be obtained in practice by extremely fine grinding. This operation carried out in a ball or pebble mill should give a product more suitable for cyanidation than that of Mill Runs Nos. 1 and 2, in which grinding was done in a rod mill.

Thorough washing of the cyanide tailing prior to flotation is essential. The character of the froth can be controlled by the amount of soda ash added to the conditioner.

Amyl xanthate appears to be somewhat more suitable than sodium xanthate.

These tests indicate that extractions ranging from 77·8 per cent to 68·7 per cent may be obtained by cyanidation, and that from 55 per cent to 88 per cent of the gold remaining in the cyanide tailing can be recovered in a concentrate containing from 0·50 ounce gold to 0·75 ounce gold per ton. The ratio of concentration will be approximately 2 : 1 to obtain the lowest flotation tailing.

The investigation shows that an 8-hour cyanide treatment will be sufficient to obtain maximum extraction.

Ore Dressing and Metallurgical Investigation No. 580

GOLD ORE FROM THE BUSSIÈRES MINING COMPANY, LIMITED, SENNETERRE, QUE.

Shipment. A sample shipment of 3,600 pounds of gold ore was received June 11, 1934, from the Bussières Mining Company, Limited, Senneterre, Quebec.

Analysis:

Gold	0·26 oz/ton
Copper	0·16 per cent

Characteristics of the Ore. Six polished sections of specimens of the ore were prepared and examined microscopically.

The purpose of the microscopic examination was to determine the mode of occurrence of the chalcopyrite with a view to determining roughly the grinding and degree necessary in order to float the chalcopyrite from the pyrite.

The gangue is complex; some is white quartz through which occurs numerous needles of tourmaline; some is mottled quartz and black to light-green silicates; patches of white carbonate are conspicuous, and some contain streaks of light-green stain, possibly due to copper. Corroded grains of leucoxene (?) are common in the silicates.

The only metallic minerals noted in the polished sections are pyrite and chalcopyrite. Pyrite occurs as irregular grains and imperfect cubes disseminated in the silicates, and, more rarely, as large coarsely crystalline masses. Chalcopyrite occurs as very irregular grains mostly in the gangue, but a small quantity is contained as small grains in the pyrite.

Quantitative Analysis. A quantitative microscopic analysis was carried out on the six polished surfaces, to determine the grain size of the chalcopyrite and the degree to which it is combined with pyrite.

TABLE I

Grain Size of the Chalcopyrite in the Ore

Mesh	Per cent	Cumulative per cent
+ 28	12·5	12·5
− 28+ 35	12·6	25·1
− 35+ 48	15·4	40·5
− 48+ 65	18·6	59·1
− 65+100	17·3	76·4
−100+150	6·6	83·0
−150+200	5·8	88·8
−200+325	4·8	93·6
−325	6·4	100·0
Total	100·0	

Of the total chalcopyrite in the ore, 13·4 per cent is combined with pyrite. Of this, 10·5 per cent is comparatively coarse (+100 mesh) and occurs against pyrite, and should be readily freed; while 2·9 per cent occurs as tiny grains (−200 mesh) enclosed by pyrite and would be exceedingly difficult to free.

The figures given above, although only approximate, indicate that a fairly high degree of freedom of the chalcopyrite from the pyrite should be attainable with comparatively coarse grinding.

Purpose of Experimental Tests. The present flow-sheet used in the mill is briefly as follows: The ore is ground in ball mills and concentrated on blankets. The blanket tailing goes to flotation machines where a flotation concentrate and final tailing are produced. The blanket concentrate and flotation concentrate are amalgamated in a clean-up barrel. There is still much gold in the concentrate after amalgamation, and cyanidation of these products is proposed.

Tests conducted separately on the blanket concentrate after amalgamation and on the flotation concentrate as produced showed excellent extractions. The cyanide consumption on the flotation concentrate was between 11 and 16 pounds per ton, and on the amalgamated blanket concentrate only 2 to 3 pounds. The blanket concentrate contained 0·3 per cent copper, whereas the flotation concentrate ran over 2 per cent. The cyanide solutions from the treatment of the flotation concentrate showed a tendency to build up in soluble copper salts.

The purpose of the following experimental tests was to determine if the copper could be selectively separated from this flotation concentrate.

EXPERIMENTAL TESTS

The test work was performed in a continuous unit at a feed rate varying between 100 and 125 pounds of ore per hour.

The flow-sheet used was as follows: The ore crushed to 14 mesh was fed to a small rod mill, grinding in closed circuit with a classifier. The classifier overflow passed over a corduroy blanket to a surge tank where the reagent for the flotation of the copper was added. The copper was floated in six cells of a flotation machine. The feed entered No.

3 cell, the concentrate from which was recleaned in No. 2 cell and the concentrate from No. 2 cell was again recleaned in No. 1 cell. This cell produced the final copper concentrate. Double cleaning was, therefore, practised. The three remaining cells, namely Nos. 4, 5, and 6, were used as rougher cells, the concentrate from which was returned with the feed to No. 3 cell.

The tailing from the copper cells was passed to four additional cells, where a pyrite concentrate was made.

Reagents Used. The reagents used for this separation were as follows: Cresylic acid was added to the surge tank for the flotation of the copper. No other reagent was used, the flotation being carried out in a neutral pulp. The quantity of cresylic acid added to the test work was 0·05 pound per ton. The pyrite was floated by the addition of 0·15 pound of amyl xanthate per ton.

Results:

Run No.	Products	Assays	
		Copper	Gold
		per cent	oz/ton
1	Feed to mill	0·16	0·26
	Blanket tailing or feed to flotation	0·07	0·05
	Copper concentrate	23·06	4·41
	Copper tailing	0·02	0·02
	Pyrite concentrate	0·18	0·32
	Tailing	nil	0·01
2	Feed to mill	0·16	0·26
	Blanket tailing or feed to flotation	0·14	0·06
	Copper concentrate	18·94	4·41
	Copper tailing	0·01	0·02
	Pyrite concentrate	0·16	0·34
	Tailing	0·01	0·01
3	Feed to mill	0·16	0·26
	Blanket tailing or feed to flotation	0·19	0·05
	Copper concentrate	20·90	3·98
	Copper tailing	0·02	0·015
	Pyrite concentrate	0·17	0·39
	Tailing	0·01	0·0075
4	Feed to mill	0·16	0·26
	Blanket tailing or feed to flotation	0·14	0·045
	Copper concentrate	15·96	3·94
	Copper tailing	0·01	0·015
	Pyrite concentrate	0·16	0·39
	Tailing	0·005	0·01

Screen Tests on Feed to Flotation:

Run No.	Percentages						
	+48	+65	+100	+150	+200	−200	Totals
1	2·0	5·0	14·8	14·7	17·0	46·5	100·0
2	2·0	2·4	18·3	13·6	16·7	47·0	100·0
3			4·2	12·2	22·2	61·5	100·0
4			3·7	9·7	22·3	64·3	100·0

SUMMARY OF RESULTS

Using an approximate average of the copper assays for the four runs, and using the two product formulæ $C = F \frac{(f-t)}{(c-t)}$, the results obtained show the following tonnages:—

Feed to mill	100·0 tons
Blanket concentrate	1·0 "
Copper concentrate	0·7 "
Pyrite concentrate	4·4 "
Tailing	93·9 "

The blanket concentrate for the entire four runs assays: copper, 0·36 per cent; and gold, 4·26 ounces per ton.

Using the average gold assay for the four runs, it can be seen that about 60 per cent of the gold was retained in the ball mill and classifier circuit, and this accounts for the low assay of the pyrite concentrate.

There is a possibility that the copper concentrate will run higher in gold after the grinding circuit becomes saturated.

CONCLUSIONS

A high-grade copper concentrate can be produced, as can also a pyrite concentrate that will contain less copper than the blanket concentrate.

Six flotation cells will be required to produce the copper concentrate.

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