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, 1933

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

Investigations of Mineral Resources and the Mining Industry.

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

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

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

Other reports on Special Investigations are issued as completed.

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MINES BRANCH INVESTIGATIONS IN

ORE DRESSING AND METALLURGY, JANUARY to JUNE, 1933

The volume of experimental test work being conducted in the Ore Dressing and Metallurgical Laboratories of the Mines Branch, Department of Mines, has grown to such an extent as to make it desirable to issue the published reports thereon more frequently than annually as has been the practice hitherto. It is now proposed to publish a small edition of the more important individual reports as soon as possible after completion of the investigation and to accumulate these in semi-annual volumes.

Company engineers and consulting engineers engaged in the design of new milling plants and concentrators find the reports of the utmost value in deciding on the flow-sheet to be adopted. Moreover, in this connection they have the benefit of the experience and knowledge of the staff, gained in conducting the experimental work, and with whom they consult in most cases. The plants are, therefore, designed for the most economical treatment of the ores. The use of the laboratory facilities and co-operation of the staff are extended to and taken advantage of by consulting engineers and metallurgists from the operating companies. During the first six months of the year, a number of engineers spent considerable time in the laboratories investigating their own particular problems. The general expression of opinion from the letters received, is that the service being rendered has been and is of inestimable value to the progress being made by the mineral industry in Canada. New milling plants and concentrators are constantly being erected, based on the results of the investigations carried out in the Ore Dressing and Metallurgical Laboratories.

During the half year ending June 30, 1933, as shown by the foregoing list of investigations twenty-eight reports were issued. Of these, five were on ores from British Columbia, two from Manitoba, twelve from Ontario, five from Quebec, one from New Brunswick, and three from Nova Scotia. Twenty-two were on ores in which gold was the chief metal of value. It is interesting to note that three Nova Scotia gold ores were investigated indicating the revival of gold mining in that province.

Ore Dressing and Metallurgical Investigation No. 483

GOLD ORE FROM THE TAMARAC MINE, YMIR, BRITISH COLUMBIA

Shipment. A shipment consisting of one sack containing 85 pounds of ore was received by freight December 2, 1932. The sample was forwarded by Lieut-Col. A. T. Powell, Nelson, B.C., and was said to have come from the Tamarac mine, Ymir, B.C.

Characteristics of the Ore. The ore consisted of heavy sulphides of iron and arsenic associated with a small amount of siliceous gangue.

Experimental Tests

After crushing, grinding, and sampling by standard methods, a sample was obtained which on analysis showed that the shipment contained 0·38 ounce gold, 0·09 ounce silver per ton, and 10·36 per cent arsenic.

In the tests conducted cyanidation recovered 76 per cent of the gold when the ore was ground to minus 150 mesh. Owing to the high sulphide content of the ore, flotation does not give any appreciable ratio of concentration, 1·4 : 1 being obtained. The flotation tailing contained over 0·10 ounce gold per ton.

Amalgamation of the ore ground to minus 48 mesh with 19 per cent minus 200 mesh recovered no gold. Grinding 63 per cent minus 200 mesh and amalgamating recovered 10·5 per cent.

CYANIDATION

Samples of the ore ground to varying degrees of fineness were agitated for 40 hours, 1 : 3 dilution, with a cyanide solution equivalent to 3·0 pounds KCN per ton. Lime up to 22 pounds per ton was added to maintain protective alkalinity.

Mesh grind	Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Per cent extraction	Reagent consumption, lb./ton	
					KCN	CaO
− 48	24	0·38	0·12	68·4	2·4	16·6
− 48	40	0·38	0·12	68·4	3·3	19·5
−100	24	0·38	0·11	71·0	3·6	18·5
−100	40	0·38	0·11	71·0	4·2	19·5
−150	24	0·38	0·10	73·7	4·5	18·7
−150	40	0·38	0·09	76·3	5·4	20·7
−200	24	0·38	0·09	76·3	5·1	20·6
−200	40	0·38	0·085	77·6	6·0	22·0

FLOTATION

Test No. 1

A sample of the ore was ground in a rod mill, 66 per cent solids, together with 10 pounds soda ash and 0·12 pound coal-tar creosote per ton until 75 per cent passed 200 mesh. The pulp was then floated with 0·10 pound sodium ethyl xanthate and 0·10 pound pine oil per ton.

Product	Weight, per cent	Assay			Distribution of metals, per cent	
		Au, oz./ton	Ag, oz./ton	As, per cent	Au	As
Feed	100·0	0·38	0·09	10·56	100·0	100·0
Concentrate	71·4	0·50	0·10	11·96	92·2	80·9
Tailing	28·6	0·105	0·03	7·05	7·8	19·1

A test similar to the above was made and flotation continued for double the time. Practically all the sample was recovered as concentrate.

Test No. 2

A sample of the ore was floated as in Test No. 1. The tailing from flotation was then passed over a laboratory-size Wilfley table and a second concentrate removed.

Product	Weight, per cent	Assay		Distribution of metals, per cent	
		Au, oz./ton	As, per cent	Au	As
Feed	100·0	0·42	10·36	100·0	100·0
Flotation concentrate	40·9	0·70	6·63	68·5	26·2
Table concentrate	34·3	0·24	17·68	19·7	58·5
Table tailing	24·8	0·20	6·37	11·8	15·3

CONCLUSIONS

The results show that no separation of the sulphides with segregation of the metals is to be expected. Apparently a larger percentage of the gold is associated with the iron pyrite, as the table concentrate although higher in arsenic is lower in gold than the flotation concentrate.

Being practically solid sulphides to begin with, this ore does not lend itself to concentration. A slight increase in value can be obtained by flotation. Whether this increase in gold content, coupled with a loss in the tailing and an increase in the arsenic content of the shipping product would be economic depends solely on freight and smelter charges.

Ore Dressing and Metallurgical Investigation No. 484

GOLD ORE FROM CRANBERRY HEAD, YARMOUTH COUNTY, NOVA SCOTIA

Shipment. A shipment of 230 pounds of gold ore was received November 22, 1932. The shipment was submitted by Douglas M. Fraser, 63 Fellsway West, Medford, Mass., U.S.A.

Characteristics of the Ore. Quartz is the dominant gangue mineral, but some sericite was observed in fractures in the quartz.

Arsenopyrite, galena, and free gold have previously been reported from this district.[1] These minerals are visible to the unaided eye.

Relations of the Minerals. Native gold is common, and is usually rather coarse, being easily detected by the unaided eye. In addition, some very small grains of gold can be seen with the microscope.

Arsenopyrite, pyrrhotite, and pyrite are relatively abundant, and galena occurs in isolated irregular grains and rather prominent masses. Chalcopyrite, sphalerite, tetrahedrite, geocronite, and unknown mineral X occur intimately associated in galena, and in very small amounts. Unknown minerals Y and Z also occur in very small amounts in quartz, sometimes associated with pyrrhotite. Mineral Y is prismatic in habit, and is replaced by Z. Some of the above minerals have not been hitherto reported from Nova Scotia gold ores.

Purpose of Experimental Tests. The tests were made for the purpose of determining the most efficient method to apply for the recovery of the contained gold.

Sampling and Analysis. On crushing and sampling by standard methods the ore was found to contain 1·50 ounces gold per ton.

PAN AMALGAMATION

Test No. 1

A sample of minus 20-mesh ore was panned in an amalgamated copper pan. The tailing from amalgamation was screened and assayed.

[1]"Gold Fields of Nova Scotia," Geol. Surv., Canada, Mem. 156, p. 77 (1929).

The results were as follows:

Screen	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
+28	10·07	1·24	18·07
−28+35	19·64	0·93	26·43
−35+48	14·98	0·975	21·14
−48+65	11·12	0·68	10·94
−65+100	9·94	0·56	8·06
−100	34·25	0·31	15·36

Feed....................Au 1·50 oz./ton
Tailing....................Au 0·59 oz./ton
Recovery....................60·67 per cent

Test No. 2

A sample of minus 20-mesh ore was reground in a Denver rod mill for 10 minutes, dilution 2 : 1, and amalgamated as in Test No. 1. A screen analysis was made on the tailing.

The results were as follows:

Screen	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
+35	8·18	0·65	9·15
−35+48	18·25	1·655	51·95
−48+65	17·02	0·525	15·38
−65+100	11·76	0·42	8·50
−100	44·79	0·195	15·02

Feed....................Au 1·50 oz./ton
Tailing....................Au 0·58 oz./ton
Recovery....................61·33 per cent

Metallics.................... Au 0·029 oz./ton

Test No. 3

A sample of minus 20-mesh ore was reground 15 minutes in a Denver rod mill, dilution 2 : 1, and amalgamated as in Test No. 1. A screen analysis was made on the tailing.

The results were as follows:

Screen	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
+48	18·54	1·815	56·16
−48+65	18·91	0·505	15·94
−65+100	13·56	0·42	9·51
−100	48·99	0·225	18·39

Feed....................Au 1·50 oz./ton
Tailing....................Au 0·50 oz./ton
Recovery....................66·67 per cent

Test No. 4

A sample of minus 20-mesh ore was reground in a Denver rod mill for 20 minutes and amalgamated as in Test No. 1. A screen analysis was made on the tailing.

The results were as follows:

Screen	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
+65	29·68	1·067	58·88
−65+100	15·69	0·47	13·70
−100	54·63	0·27	27·42

Feed....................Au 1·50 oz./ton
Tailing....................Au 0·57 oz./ton
Recovery....................61·67 per cent

Metallics in tailing sample....................Au 0·030 oz./ton
Metallics in +65-mesh tailing....................Au 0·067 oz./ton

AMALGAMATION AND FLOTATION

Test No. 5

A sample of minus 48-mesh ore was amalgamated. The tailing from amalgamation was treated by flotation with the following reagents:

	Lb./ton
Soda ash	2·0
Sodium ethyl xanthate	0·3
Risor pine oil	0·05
Steam-distilled pine oil	0·025

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	1·50		
Amalgamation tailing		0·14		
Recovery			90·67	
Flotation feed		0·14		
Flotation concentrate	2·50	3·07	5·28	40 : 1
Flotation tailing	97·50	0·06		

	Per cent
Recovery by amalgamation	90·67
Recovery by flotation	5·28
Total recovery	95·95

A screen test on the flotation tailing shows the ore to be ground as follows: Analyses +100, −100 mesh.

Mesh	Weight, per cent	Products	Assay, Au, oz./ton	Calculated assay of tailing
+65	18·58	+100	0·085	0·063 oz./ton
−65+100	21·08			
−100+150	14·59	−100	0·04	
−150+200	7·69			
−200	38·06			

Test No. 6

A sample of minus 65-mesh ore was amalgamated. The tailing from amalgamation was treated by flotation with the following reagents:

	Lb./ton
Soda ash	2·0
Sodium ethyl xanthate	0·2
Coal-tar creosote	0·20
Steam-distilled pine oil	0·10

Results:

Products	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	1·50		
Amalgamation tailing		0·13		
Recovery			91·33	
Flotation feed		0·13		
Flotation concentrate	3·28	2·16	4·91	30·49 : 1
Flotation tailing	96·72	0·06		

	Per cent
Recovery by amalgamation	91·33
Recovery by flotation	4·91
Total recovery	96·24

A screen test on the flotation tailing shows the ore to be ground as follows: Analyses +100, −100 mesh.

Mesh	Weight, per cent	Products	Assay, Au, oz./ton	Calculated assay of tailing
+100	20·83	+100	0·08	0·056 oz./ton
−100+150	20·98	−100	0·05	
−150+200	10·74			
−200	47·45			

STRAIGHT CYANIDATION

Test Nos. 7 and 8

Samples of raw ore were crushed dry −48, −65, −100 mesh and treated by cyanidation in a pulp containing 3 parts of solution to one part of ore. The strength of the sodium cyanide solution used was equivalent to 1·0 pound of potassium cyanide per ton of solution, and lime equivalent to 12 pounds per ton of ore was added to supply a protective alkalinity.

The cyanide tailings were screened for a sizing test. The +200, −200-mesh products were assayed.

Results:

Test No.	Mesh	Time, hours	Products	Weight, per cent	Assay Au. oz./ton		Distribution of gold, per cent	Total extraction, per cent	Reagent consumption	
					Feed	Tailing			KCN	CaO
									lb./ton	lb./ton
1......	−48	24	+200 −200	62·0 38·0	1·50 1·50	0·05 0·01	89·08 10·92	98·0	0·30	11·8
2......	−48	48	+200 −200	60·8 39·2	1·50 1·50	0·035 0·015	78·31 21·69	98·0	0·15	11·1
3......	−65	24	+200 −200	53·6 46·4	1·50 1·50	0·75 0·05	94·54 5·46	72·0	0·45	11·8
4......	−65	48	+200 −200	52·4 47·6	1·50 1·50	0·02 0·015	59·66 40·34	98·67	0·15	11·1
5......	−100	24	+200 −200	30·5 69·5	1·50	0·07 0·025	55·15 44·85	97·33	0·45	11·3
6......	−100	48	+200 −200	26·4 73·6	1·50	0·03 0·01	51·63 48·37	98·67	0·45	11·4

Screen Tests on Cyanide Tailings:

A screen test on the minus 48-mesh tailing showed the ore to be crushed as follows:—

Mesh	Weight, per cent
+ 65	18·30
− 65+100	19·76
−100+150	14·26
−150+200	9·68
−200	38·00

Screen test on minus 65-mesh tailing.

Mesh	Weight, per cent
+100	19·58
−100+150	20·84
−150+200	13·15
−200	46·43

Screen test on minus 100-mesh tailing.

Mesh	Weight, per cent
+150	13·69
−150+200	16·84
−200	69·47

Test No. 9

Samples of raw ore were crushed dry −48, −65, −100 mesh and treated by cyanidation in a pulp containing 3 parts of solution to one part of ore. The strength of the sodium cyanide solution used was equivalent to 1·0 pound potassium cyanide per ton of solution, and lime equivalent to 12 pounds per ton of ore was added to supply a protective alkalinity. The results were as follows:

Test No.	Time, hours	Mesh	Assay, Au, oz./ton		Extraction, per cent	Reagent consumption	
			Feed	Tailing		KCN	CaO
						lb./ton	lb./ton
1	24	−48	1·50	0·035	97·67	0·45	9·4
2	48	−48	1·50	0·01	99·33	0·45	9·4
3	48	−65	1·50	0·008	99·47	0·60	9·8
4	24	−100	1·50	0·02	98·67	0·45	8·7
5	48	−100	1·50	0·07	95·33	0·45	9·0

AMALGAMATION AND FLOTATION

Test No. 10

A sample of ore was crushed minus 48 mesh and amalgamated. The amalgam was separated, and the tailing was treated by flotation with the following reagents.

	Lb./ton
Soda ash	2·0
Coal-tar creosote	0·20
Sodium ethyl xanthate	0·3
Copper sulphate	0·5
Pine oil	0·10

The results were as follows:

Products	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	1·57		
Amalgam			92·36	
Amalgamation tailing		0·12		
Flotation feed		0·12		
Flotation concentrate	2·58	3·21	70·83	38·76 : 1
Flotation tailing	97·42	0·035		

	Per cent
Recovery by amalgamation	92·36
Recovery by flotation	5·41
Total recovery	97·77

Total recovery of gold as bullion:

	Per cent
Amalgamation of ore	92·36
Amalgamation of flotation concentrate	3·59
Total recovery	95·95

Screen Test

Mesh	Weight, per cent
+65	....
− 65+100	3·73
−100+150	26·94
−150+200	13·31
−200	56·02

Test No. 11

A sample of ore was crushed dry minus 65 mesh and amalgamated. The amalgam was separated and the tailing was treated by flotation with the following reagents:

	Lb./ton
Soda ash	2·0
Coal-tar creosote	0·20
Sodium ethyl xanthate	0·30
Copper sulphate	0·50
Pine oil	0·10

The results were as follows:

Products	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	1·53		
Amalgam			88·89	
Amalgamation tailing		0·17		
Flotation feed		0·17		
Flotation concentrate	3·16	4·36	82·56	31·65 : 1
Flotation tailing	96·84	0·03		

	Per cent
Recovery by barrel amalgamation	88·89
Recovery by flotation	9·17
Total recovery	98·06

	Per cent
Recovery as bullion—	
Amalgamation of feed	88·89
Amalgamation of flotation concentrate	6·09
Total recovery	94·98

Screen Test on Tailing

Mesh	Weight, per cent
+65	0·07
− 65+100	1·68
−100+150	5·96
−150+200	16·62
−200	75·67

AMALGAMATION OF FLOTATION CONCENTRATE

Test No. 12

A sample of flotation concentrate was amalgamated by barrel amalgamation.

The results were as follows:

Products	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	2·56	100·00
Amalgam		66·41
Tailing	0·86	33·59

Recovery.......66·41 per cent

SUMMARY AND RECOMMENDATIONS

If the sample submitted is representative of the ore which will be mined the results of the test work show that the treatment of this ore is relatively simple.

Two different methods of treatment are indicated by the test work. (1) Straight cyanidation and (2) amalgamation followed by flotation of the amalgamation tailing. The results of the test work show that a cyanide plant will recover more than 95 per cent of the gold. The flow-sheet suggested for the plant would be similar to the one used at the Dome Mines mill at Timmins, Ontario. The ore should be ground in ball mills in cyanide solution, and blankets used to take out the coarse gold before the pulp goes to the agitators or thickeners and the gold caught on the blankets to be recovered by barrel amalgamation. The use of blankets is recommended on account of the large proportion of coarse gold present in the ore. Some of the coarse gold would undoubtedly pass through the cyanide plant before it had time to be entirely dissolved. For cyanidation the ore should be crushed to pass 48 mesh, so that about 60 per cent is minus 200 mesh.

The second method of treatment involves amalgamation on plates, also the use of blankets to catch any coarse, rusty gold not amalgamated on the plates. The tailing after amalgamation would be concentrated by flotation. The test work indicates that in practice at least 85 per cent of the gold can be recovered as bullion by the plates and that an additional amount making a total of over 95 per cent or better can be recovered in a flotation concentrate. The ratio of concentration is excellent being about 38 : 1, and the grade of the concentrate was between 3·00 ounces and 4·00 ounces per ton. This concentrate can be either treated at the mine or shipped to a smelter. Test No. 12 shows that 66 per cent of the gold in the concentrate can be extracted by amalgamation in the clean-up barrel. No cyanide tests were made on the concentrate as there was not sufficient ore to produce the amount of concentrate required for such tests.

The flow-sheet suggested for the second method would be crushing in stamps to 35 mesh using inside and outside amalgamation, thickening the amalgamation tailing and concentrating it by flotation in mechanical flotation machines such as the "Fahrenwald" cell. An alternative to this would be to grind the ore to minus 48 mesh in a ball mill and amalgamate in closed circuit between the ball mill and the classifier. The classifier overflow to be passed over blankets before flotation in order to remove rusty free gold. The advantage of the ball mill over stamps would be that the ore would be ground finer and that it would not be necessary to thicken before flotation.

The decision as to which of the two methods should be used can be arrived at only after a study of the economic factors involved, such as ore reserves, daily tonnage of ore to be milled, and capital available, etc. The mill using amalgamation and flotation will cost much less to build than a cyanide plant and would be more adaptable for a small mill of 25 to 50 tons. If a 100-ton mill is contemplated it is probable that a cyanide plant would be advisable.

Ore Dressing and Metallurgical Investigation No. 485

GOLD ORE FROM THE COCHENOUR-WILLANS PROPERTY AT RED LAKE, ONTARIO

Shipment. A shipment of five lots of ore samples, total weight 5,000 pounds, was received November 9, 1932. The numbers and weights of the individual samples were as follows:

Sample No.	Weight, pounds
1	1,000
3	500
5	1,000
13	500
21	2,000

The samples were submitted by Col. N. F. Parkinson, Secretary-Treasurer, Ventures Limited, 100 Adelaide Street West, Toronto.

Characteristics of the Ore. Two of the samples were studied microscopically. The metallic minerals observed in the polished sections are as follows:

Sample No. 1—Pyrite, arsenopyrite, "limonite".
Sample No. 21—Pyrite, arsenopyrite, chalcopyrite, a mineral resembling pyrrhotite, native gold, "limonite".

The gangue of both samples consists of impure quartz which contains numerous small irregular grains of carbonate. Specimens from Sample No. 1 exhibited a rather uniform distribution of the disseminated pyrite and arsenopyrite, but those from Sample No 21 showed these minerals to be arranged in stringers with exceedingly variable proportions of the two. Many of the larger crystals of pyrite in Sample No. 21 contain small irregular grains of chalcopyrite and pyrrhotite(?), this being the only occurrence of these two minerals. The only native gold observed in the sections occurs in those from Sample No. 21 as small irregular grains in the quartz. "Limonite" is present locally as a result of weathering.

An average analysis of the samples was as follows:

Lot No.	Gold, oz./ton	Silver, oz./ton	Iron, per cent	Arsenic, per cent
1	0·62	0·07	3·85	0·83
3	0·59	0·10	5·70	1·15
5	0·29	0·03	5·35	1·40
13	0·57	0·02	3·45	0·23
21	0·565	0·15	7·25	0·52

Experimental Tests

The five lots making up the shipment were sampled separately and assayed. Then Lots Nos. 1 and 5 were mixed together and assayed, as was also done with Lots Nos. 3 and 13. The analyses of these two composite samples were as follows:

—	Lots Nos. 1 and 5	Lots Nos. 3 and 13
Gold, oz./ton	0·46	0·545
Silver, oz./ton	0·07	0·065
Iron, per cent	4·70	4·75
Lime, per cent	1·31	19·90
Arsenic, per cent	1·05	0·73
Silica, per cent	81·15	25·15

These two composite samples, along with Lot No. 21, as received, were used for small-scale test work which included tests by amalgamation, cyanidation, and flotation. Finally, all the ore was mixed together in one sample for a larger scale mill run using a unit of 100 pounds per hour capacity.

None of the results of the small-scale tests was good and most of them were decidedly poor. Good grade flotation concentrates were produced on all three lots after grinding about 95 per cent through 200 mesh, but recoveries were low ranging from 84 to 90 per cent approximately. Recoveries by amalgamation and cyanidation were low on all samples, particularly Lots Nos. 1 and 5 and Lots Nos. 3 and 13. Although almost 50 per cent of the gold could be amalgamated from Sample No. 21, cyanidation of the amalgamation tailing brought the total recovery up to only 74 per cent and this was possible by cyanidation alone.

In the mill run an overall recovery of 89·4 per cent of the gold was obtained, assuming that the gold held up in the mill and classifier is recoverable. The flotation concentrate actually contained 65·3 per cent of the total gold in the feed.

Part A, Lot No. 21

GRINDING

Test No. 1

This test was run to find out how the ore responded to grinding in a ball mill.

Charge to Ball Mill:

Ore, 1,000 grammes −14 mesh.
Water, 750 c.c.

This charge was ground for 15 minutes.

Screen Analysis:

Mesh	Weight, grammes	Weight, per cent	Cumulative weight, per cent	Assay, Au, oz./ton
+65	21·2	2·1	2·1	....
− 65+100	37·8	3·8	5·9	0·625
−100+150	104·7	10·6	16·5	0·63
−150+200	111·2	11·2	27·7	0·55
−200	721·8	72·3	100·0	0·36
Totals	996·7	100·0	....	....

HYDRAULIC CLASSIFICATION

Test No. 2

In this test the ore at minus 14 mesh was ground for 15 minutes in a jar mill. The pulp was then run to a hydraulic classifier and the coarse gold and heavy sulphides were allowed to settle against a slowly rising current of water. The oversize and the overflow from the classifier were assayed for gold. Some fairly coarse free gold was noticed in the oversize with the aid of a pocket glass.

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Classifier oversize	0·18	40·824	16·8
Classifier overflow	99·82	0·365	83·2
Feed (cal.)	100·00	0·438	100·0

AMALGAMATION

Tests Nos. 3 and 4

In Test No. 3 the ore at minus 14 mesh was amalgamated with mercury without any grinding. In Test No. 4 the ore at minus 14 mesh was ground for 17 minutes in a ball mill and then amalgamated. In both cases 1,000 grammes of ore was amalgamated with 100 grammes of mercury in 1 : 1 pulp for 30 minutes.

Summary:

Feed sample, Au 0·565 oz./ton.

Test No.	Tailing assay, Au, oz./ton	Recovery, per cent
3	0·355	37·2
4	0·290	48·7

CYANIDATION

Test No. 5

In this test the ore was ground for 17 minutes in a ball mill, then agitated in cyanide solution for 24 hours at 2·5 : 1 dilution. The solution strength was KCN one pound per ton.

Summary:

Product	Assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
Feed	0·565	74·3	0·38	9·4
Tailing	0·145			

AMALGAMATION AND CYANIDATION

Test No. 19

The ore was ground for 15 minutes in a ball mill and amalgamated with mercury in the usual way. (See Test No. 4.) The amalgamation tailing was then agitated at 2·5 : 1 dilution in solution running 2 pounds per ton in KCN. At the end of 24 hours the pulp was filtered and washed and a sample taken out for assay. The remainder was re-pulped and agitated for another 24 hours in fresh solution at approximately the same dilution and solution strength. The final cyanidation tailing was also assayed for gold.

This test was made to decide whether the low recovery by cyanidation obtained in Test No 5 was due to the presence of coarse gold. The result of the test indicated that this was not the case.

Summary:

Feed sample, Au 0·565 oz./ton

Product	Assay, Au, oz./ton	Recovery, per cent amalgamation and cyanidation	Reagent consumed, lb./ton	
			KCN	CaO
24-hour cyanide tailing from amalgamation tailing	0·148	73·8	0·61	7·85
48-hour cyanide tailing from amalgamation tailing	0·125	77·9	0·70	9·50

It will be noticed that at the end of 24 hours the recovery by amalgamation and cyanidation approximates very closely that by cyanidation alone for the same period. (See Test No. 5.)

FLOTATION

Test No. 18

This test was carried out on a sample of ore ground comparatively coarse. The tailing was screened and the products assayed for gold. When the coarser sizes were ground down in preparation for assaying, free gold liberated during the process was looked for in the metallics which were assayed separately. (None, however, was found.)

In this way it was hoped that the amount of grinding necessary to liberate the gold could be determined but unfortunately all the fractions were much too high in gold to be discarded, which meant that fine grinding of the entire sample held out the only hope of making a satisfactory recovery by flotation. The same condition was found to exist in the other two lots tested. (See Tests Nos. 16 and 17).

Charge to Ball Mill:

Ore	2,000 grammes
Water	1,500 grammes
Na_2CO_3	3·0 lb./ton
Oil mixture	0·1 lb./ton

The reagent "oil mixture" contains: Coal-tar creosote 60 parts, coal tar 20 parts, and cresylic acid 20 parts.

Reagents to Cell:

	Lb./ton
Potassium amyl xanthate	0·10
Pine oil	0·08
Tarol No. 1	0·05

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	13·65	2·44	66·6
Average tailing	86·35	0·193	33·4
Feed (cal.)	100·00	0·50	....

Screen Analysis Flotation Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
+ 35	9·3	0·57	27·4
− 35+ 48	12·4	0·33	21·2
− 48+ 65	15·3	0·215	17·0
− 65+100	13·1	0·16	10·9
−100+150	8·8	0·135	6·2
−150+200	5·3	0·125	3·4
−200	35·8	0·075	13·9
Total	100·0	0·193	....

Test No. 22

In this test the ore was ground finer than in Test No. 18 to see the effect that finer grinding would have on the recovery.

Charge to Ball Mill:

Ore	2,000 grammes −14 mesh
Water	1,500 grammes
Na_2CO_3	3·0 lb./ton
Oil mixture	0·1 lb./ton.

This charge was ground for 15 minutes.

Reagents to Cell:

	Lb./ton
Potassium amyl xanthate	0·10
Pine oil	0·08

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	15·1	2·56	84·3
Tailing	84·9	0·085	15·7
Feed (cal.)	100·0	0·459	....

Test No. 25

In this test the fine grinding was carried still further and copper sulphate was added to the cell. Otherwise the test was the same as Test No. 22.

Charge to Ball Mill:

Ore	2,000 grammes −14 mesh
Water	1,500 grammes
Na_2CO_3	3·0 lb./ton
Oil mixture	0·1 lb./ton

This charge was ground for 30 minutes.

Reagents to Cell:

	Lb./ton
Potassium amyl xanthate	0·1
$CuSO_4$	0·5
Pine oil	0·08

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	17·0	2·63	86·4
Tailing	83·0	0·085	13·6
Feed (cal.)	100·0	0·518	100·0

Screen Analysis Flotation Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
+200	5·9	0·635	44·3
−200	94·1	0·05	55·7
Average tailing	100·0	0·085	100·0

Summary

Tests Nos. 18, 22, and 25 show clearly that fine grinding increases the recovery obtained by flotation of this ore. There is still room for improvement in both the recovery and the grade of tailing, but this could probably be brought about by floating a classified feed and using a hydraulic gold trap between the ball mill and classifier to catch the coarse gold. The ratio of concentration also falls off somewhat as the amount of fines increases.

Part B, Lots Nos. 1 and 5

Small-scale tests similar to those made on Lot No. 21 were made on the composite sample resulting from the mixing together of Lots Nos. 1 and 5.

GRINDING

Test No. 6

A sample of the ore was ground for 15 minutes in a ball mill and a screen analysis made on the product. The fractions were weighed and assayed for gold.

Charge to Ball Mill:

Ore .... 1,000 grammes −14 mesh
Water .... 750 grammes

Screen Analysis:

Mesh	Weight, grammes	Weight, per cent	Cumulative weight, per cent	Assay, Au, oz./ton
+ 65	141·3	14·2	14·2	0·36
− 65+100	168·1	16·9	31·1	0·38
−100+150	145·7	14·6	45·7	0·39
−150+200	93·9	9·5	55·2	0·40
−200	447·9	44·8	100·0	0·58
Total	996·9	100·0	....	....

AMALGAMATION

Tests Nos. 8 and 10

Two tests were made on this sample of ore, one at minus 14 mesh and the other ground as was done in Test No. 1. In the first case no recovery was obtained at all, and in the second case the recovery was 6·5 per cent.

HYDRAULIC CLASSIFICATION

Test No. 12

The ore at minus 14 mesh was ground for 15 minutes in a ball mill and then run to a hydraulic classifier to allow the coarse gold and heavy sulphides to settle. Results were poor as 98·5 per cent of the ore overflowed, carrying with it 94·5 per cent of the gold.

CYANIDATION

Test No. 14

The ore responded very poorly to cyanidation, only 29·3 per cent of the gold being extracted in 24 hours, leaving a tailing that assayed gold 0·325 ounce per ton.

FLOTATION

Test No. 16

This test is similar to Test No. 18, Lot 21.

Charge to Ball Mill:

Ore .... 2,000 grammes −14 mesh
Water .... 1,500 grammes
Na_2CO_3 .... 3·0 lb./ton
Oil mixture .... 0·1 lb./ton

The charge was ground for 10 minutes.

Reagents to Cell:

	Lb./ton
Potassium amyl xanthate	0·10
Pine oil	0·08

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	4·84	4·76	48·0
Tailing	95·16	0·262	52·0
Feed (cal.)	100·0	0·48	100·0

Screen Analysis:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
+ 35	2·0	0·37	2·8
− 35+ 48	11·9	0·35	15·9
− 48+ 65	12·6	0·32	15·4
− 65+100	22·8	0·32	27·9
−100+150	6·2	0·29	6·9
−150+200	6·7	0·255	6·5
−200	37·8	0·17	24·6
Total	100·0	0·262	....

Test No. 20

The same reagent combination was used in this test as in No. 16 but the ore was ground much finer.

Charge to Ball Mill:

Ore	2,000 grammes −14 mesh
Water	1,500 grammes
Na_2CO_3	3 lb./ton
Oil mixture	0·1 lb./ton

The charge was ground 25 minutes.

Reagents to Cell:

	Lb./ton
Potassium amyl xanthate	0·10
Pine oil	0·08

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	10·1	4·54	86·0
Tailing	89·9	0·083	14·0
Feed (cal.)	100·0	0·533	....

Screen Analysis:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
+200	15·6	0·18	33·9
−200	84·4	0·065	66·1
Average tailing	100·0	0·083	....

Test No. 23

In this test grinding was carried still further than in Test No. 20 and copper sulphate was added to the reagent combination.

Charge to Ball Mill:

Ore	2,000 grammes −14 mesh
Water	1,500 grammes
Na_2CO_3	3·0 lb./ton
Oil mixture	0·1 lb./ton

The charge was ground for 40 minutes.

Reagents to Cell:

	Lb./ton
Potassium amyl xanthate	0·10
$CuSO_4$	0·50
Pine oil	0·08

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	9·4	4·82	89·9
Tailing	90·6	0·056	10·1
Feed (cal.)	100·0	0·504	100·0

Screen Analysis Flotation Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
+200	5·9	0·155	16·3
−200	94·1	0·05	83·7
Average tailing	100·0	0·056	....

In summing up Tests Nos. 16, 20, and 23 it will be observed that finer grinding results in a steady increase in recovery. The ratio of concentration, which is quite high in Test No. 16, falls off considerably in Test No. 20 but improves again in Test No. 23.

Part C, Lots Nos. 3 and 13

A series of small-scale tests was run on a composite sample made up from Lots Nos. 3 and 13.

GRINDING

Test No. 7

The ore was ground in a ball mill and a screen analysis made.

The charge was ground for 15 minutes.

Charge to Ball Mill:

Ore	1,000 grammes −14 mesh
Water	750 grammes

Screen Analysis:

Mesh	Weight, grammes	Weight, per cent	Cumulative weight, per cent	Assay, Au, oz./ton
+ 65	2·6	0·3	0·3	....
− 65+100	22·4	2·2	2·5	....
−100+150	77·7	7·9	10·4	0·46
−150+200	105·8	10·7	21·1	0·48
−200	787·8	78·9	100·0	0·59
Total	996·3	100·0	....	....

NOTE.—The assay reading 0·46 ounce per ton in gold is an assay of everything coarser than 150 mesh.

AMALGAMATION

Tests Nos. 9 and 11

Two tests were made on this sample of ore, the first one at minus 14 mesh, and the second one after 15 minutes' grinding or roughly 79 per cent minus 200 mesh. In the first test no recovery at all was obtained and in the second one only 9·2 per cent of the gold was recovered.

HYDRAULIC CLASSIFICATION

Test No. 13

The ore was ground for 15 minutes and then put through a hydraulic classifier to allow the coarse gold and heavy sulphides to settle. Although the classifier oversize assayed approximately 6 ounces per ton in gold the ratio of concentration was very high and the recovery amounted only to 5·4 per cent of the gold, the other 94·6 per cent of it remaining with the classifier overflow.

CYANIDATION

Test No. 15

In this test the ore, ground to approximately 79 per cent minus 200 mesh, was agitated for 24 hours in cyanide solution. Extraction was only 35·8 per cent of the gold.

Test No. 17

This test, similar to Tests Nos. 16 and 18, was made with the object of finding out, if possible, the limit to which grinding should be carried in order to liberate the gold-bearing minerals. The ore was ground for 5 minutes, and the reagent combination was the same as that used in Tests Nos. 16 and 18.

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	5·1	5·52	45·2
Tailing	94·9	0·359	54·8
Feed (cal.)	100·0	0·622	100·0

Screen Analysis:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
+35	4·9	0·82	11·2
− 35+48	9·5	0·62	16·4
− 48+65	18·4	0·46	23·6
− 65+100	13·3	0·36	13·3
−100+150	10·2	0·285	8·1
−150+200	8·0	0·25	5·6
−200	35·7	0·22	21·8
Total	100·0	0·359	100·0

Test No. 21

In this test grinding was carried on further, the ore being reduced to 79 per cent minus 200 mesh, and then floated with the same reagent combination as was used in Test No. 17.

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	13·1	3·50	80·2
Tailing	86·9	0·13	19·8
Feed (cal.)	100·0	0·57	100·0

Test No. 24

In this test the ore was reduced to 94·5 per cent minus 200 mesh, after 30 minutes' grinding in a ball mill. Reagents were the same as used in Tests Nos. 23 and 25.

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	9·3	5·24	84·3
Tailing	90·7	0·10	15·7
Feed (cal.)	100·0	0·578	100·0

Screen Analysis:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
+200	5·5	0·265	14·6
−200	94·5	0·09	85·4
Average tailing	100·0	0·10	100·0

Recovery increases with finer grinding.

Part D, Mill Run

All the ore representing the five original samples was mixed together in one lot and ground in a rod mill in closed circuit with an Akins classifier. The classifier overflow went to flotation. The final concentrate was taken from the first cell in a series of six. A rougher concentrate from the last five cells was returned to the first cell to be cleaned.

Reagents used were as follows:

	Lb./ton
Na_2CO_3	3·0
Oil mixture	0·10
Potassium amyl xanthate	0·10
Pine oil	0·08

The first two reagents were fed into the rod mill, the third into the conditioning tank, and the fourth into the fourth cell in the series of six. The feed rate was kept at 90 pounds per hour throughout the test and the rod charge was 392 pounds. Samples were taken every 15 minutes of the feed, rod mill discharge, classifier overflow, flotation concentrate, and flotation tailing.

The assays and per cent of the total gold contained in the products are as follows:

—	Au, oz./ton	Distribution of gold, per cent
Mill feed	0·54	100·0
Rod mill discharge	0·475	88·0
Classifier overflow	0·41	75·9
Flotation concentrate	3·08	65·3
Flotation tailing	0·065	10·6

The ratio of concentration is 8·7 : 1.

From the above it is obvious that 24·1 per cent of the gold is retained in the mill and classifier. The flotation concentrate contains 86 per cent of the gold in the classifier overflow or 65·3 per cent of the total gold in the original feed.

When the mill and classifier were cleaned out large and small particles of free gold well flattened out were found.

A cyanidation test on the concentrate gave an extraction of 41·6 per cent of the gold contained in it.

CONCLUSIONS

This ore cannot be treated by amalgamation nor cyanidation nor any combination of the two processes. Concentration by flotation is the only process that holds out any hope at all and that is only possible after grinding practically everything through 200 mesh. However, the problem of what to do with the concentrate which will not respond to cyanidation in its raw state still remains. Any further investigation undertaken on this ore might include cyanidation tests on a roasted concentrate.

A good flow-sheet for concentrating the ore would be to grind in a rod mill and have the mill discharge through a hydraulic gold trap or over amalgamation plates into a classifier. The idea of the trap or plates is to catch the free gold discharged from the mill and prevent its building up in the classifier. In this way it is presumed that after a period of continuous running the mill discharge would correspond closely to the feed. The classifier overflow would go to flotation and the oversize would be returned to the mill for regrinding. If a hydraulic trap were used, the trap cleanings could be treated by barrel amalgamation, and the amalgamation tailing united with the flotation concentrate for further treatment. If plates are used nothing remains but to dress them and treat the concentrate by itself.

Ore Dressing and Metallurgical Investigation No. 486

GOLD ORE FROM THE MICHAEL-BOYLE PROPERTY, ALDEN, ALGOMA DISTRICT, ONTARIO

Shipment. A shipment consisting of five bags of gold ore weighing 500 pounds was received January 3, 1933. The consignment was made by Dr. Paul Armstrong, 1469 Drummond Street, Montreal, from the Michael-Boyle property, Alden, Algoma District, Ont. The material was said to have been taken from depth.

Characteristics of the Ore. Under the microscope the gangue was seen to consist chiefly of milky to glassy quartz which contains small irregular stringers of a dark, undetermined gangue mineral with small amounts of associated carbonate. Irregular large masses of coarsely granular pyrite, by far the predominating metallic mineral, occur in the quartz. Locally the pyrite contains irregular grains of galena up to 1 mm. in diameter.

The other metallic minerals—native gold, chalcopyrite, and a telluride, possibly sylvanite (?)—along with considerable galena, tend to occur in the above-mentioned stringers in the quartz. Galena is perhaps the most abundant of these. In rare spots the galena is intimately intergrown with a bright yellow mineral which microscopic, microchemical, and spectrographic tests show to be a gold-silver telluride, possibly sylvanite (?). Spectrographic analyses show further that a small quantity of gold-silver telluride also occurs with the galena in the pyrite.

Native gold is rare in the sections studied, and its occurrence as small grains in the fine stringers only was observed.

Chalcopyrite is developed very locally as grains up to 1 millimetre or more in size in the stringers. It was not associated with other metallic minerals in the sections examined.

Purpose of Experimental Tests. The object of the investigation was to determine what recovery of gold could be obtained by concentration with a view to shipping the product so obtained to a smelter.

Experimental Tests

The investigation included blanket concentration, flotation and amalgamation singly and in combination. Tests were made to determine the recoveries at various screen sizes of grinding.

The results indicate that 80 per cent of the gold can be caught on blankets and an additional 18·8 per cent in a flotation concentrate, a total recovery of 98·8 per cent. Amalgamation recovers 85·6 per cent and an additional 12·8 per cent is recovered in a flotation concentrate, thus making a total recovery of 98·4 per cent.

Flotation of the finely ground ore recovers 96·1 per cent of the gold.

After crushing, sampling and assaying, the shipment was found to contain 2·34 ounces gold per ton.

BLANKET CONCENTRATION

Test No. 1

A sample of the ore was ground to pass 48 mesh with 59·8 per cent minus 200 mesh and passed over a corduroy blanket.

Mesh	Weight, per cent
+65	0·3
− 65+100	4·5
−100+150	16·5
−150+200	18·9
−200	59·8

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·0	2·36	100·0
Blanket concentrate	11·6	14·88	73·1
Blanket tailing	88·4	0·72	26·9

BLANKET AND FLOTATION CONCENTRATION

Test No. 2

A sample was ground in water to pass 77 per cent through 200 mesh and run over a blanket. The tailing was then conditioned 15 minutes with 3 pounds soda ash, 0·06 pound Aerofloat No. 25 per ton; 0·15 pound amyl xanthate and 0·06 pound pine oil were added and a flotation concentrate removed. The blanket concentrate was amalgamated to note the recovery of free gold from this product.

Mesh	Weight, per cent
− 65+100	0·6
−100+150	5·4
−150+200	16·6
−200	77·4

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	2·34	....
Blanket concentrate	....	35·76	....
Blanket concentrate amalgamated	4·43	1·82	3·4
Amalgam from blanket concentrate	....	....	64·4
Blanket tailing	....	0·79	....
Flotation concentrate	17·69	4·16	31·4
Flotation tailing	77·88	0·025	0·8

These results show that 67·8 per cent of the gold was recovered on the blankets and that 94·9 per cent of this can be removed by amalgamating the concentrate. A further recovery of 31·4 per cent is obtained by flotation. Combining the residue left after amalgamation of the blanket concentrate with the flotation concentrate gives a shipping product equal to 22·12 per cent of the weight of ore milled, and containing 3·65 ounces gold per ton. A total recovery of 99·2 per cent of the gold is obtained.

Test No. 3

A sample of the ore was coarse ground as in Test No. 1 and passed over a corduroy blanket. The tailing was then conditioned for 15 minutes in a flotation machine with 3 pounds soda ash; 0·15 pound sodium xanthate and 0·06 pound pine oil then added and a concentrate removed.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed (cal.)	100·0	2·54	100·0
Blanket concentrate	13·8	14·70	80·0
Blanket tailing	....	0·50	....
Flotation concentrate	10·0	4·76	18·8
Flotation tailing	76·2	0·04	1·2

A recovery of 98·8 per cent of the gold is made in a combined blanket and flotation concentrate which has an assay value of 10·52 ounces per ton. A ratio of concentration 4·2 : 1 is noted.

FLOTATION

Test No. 4

A sample of the ore was coarsely ground as in Test No. 1 with 3 pounds soda ash and 0·06 pound Aerofloat per ton. Following the additions of 0·15 pound amyl xanthate and 0·12 pound pine oil, a concentrate was removed.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed (cal.)	100·0	2·05	100·0
Flotation concentrate	19·4	9·82	92·9
Flotation tailing	80·6	0·18	7·1

Test No. 5

A test was made with fine grinding and floated as in Test No. 4. The concentrate was cleaned once.

Mesh	Weight, per cent
+150	0·2
−150+200	8·9
−200	90·9

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed (cal.)	100·00	2·05	100·0
Flotation concentrate	15·76	12·30	94·6
Flotation middling	4·49	0·70	1·5
Flotation tailing	79·75	0·10	3·9

The concentrate had the following analysis:

Gold	12·30 oz./ton
Iron	39·5 per cent
Sulphur	44·3 "
Lime	0·15 "
Magnesia	0·15 "
Silica	13·7 "
Alumina	1·05 "
Arsenic	Nil

Fine grinding raises the recovery and grade of the concentrate. The rougher concentrate contains 96·1 per cent of the gold.

Test No. 6

A test was made similar to the above with the exception that grinding was 77·4 per cent minus 200 mesh with 0·6 per cent on 100 mesh. The flotation concentrate was then amalgamated.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	2·34	100·0
Concentrate		11·31	
Concentrate amalgamated	18·38	1·96	15·4
Amalgam			73·4
Flotation tailing	81·62	0·32	11·2

Amalgamation recovers 82·7 per cent of the gold in the flotation concentrate, leaving a product containing 1·96 ounces per ton.

Test No. 7

A sample was treated as in Test No. 6. After flotation, the tailing was passed over a corduroy blanket.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed (cal.)	100·00	2·13	100·0
Flotation concentrate	17·59	11·40	94·0
Flotation tailing (cal.)		0·128	
Blanket concentrate	3·09	3·63	5·3
Blanket tailing	79·32	0·02	0·7
Combined concentrates	20·68	10·24	

Ratio of concentration of the combined concentrates is 4·8 : 1.

Flotation followed by blanket concentration recovers 99·3 per cent of the gold.

Test No. 8

A sample was coarse ground as in Test No. 1 and amalgamated. The amalgamation tailing was then conditioned for 15 minutes with 3 pounds soda ash, and 0·06 pound Aerofloat No. 25; and floated with 0·15 pound amyl xanthate and pine oil.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	2·34	100·0
Amalgam tailing (cal.)	....	0·34	....
Recovered as amalgam	....	....	85·6
Flotation concentrate	19·05	1·58	12·8
Flotation tailing	80·95	0·045	1·6

Amalgamation followed by flotation recovers 98·4 per cent of the gold. After removing free amalgamable gold, 19 tons of concentrate assaying 1·58 ounces per ton is recovered from each 100 tons of feed.

SUMMARY AND CONCLUSIONS

Blanket concentration recovers from 73 to 80 per cent of the gold in the ore as shown in Tests Nos. 1 and 3, and 94·9 per cent of the gold in this concentrate can be amalgamated.

A recovery of 93 to 96 per cent can be made by straight flotation as indicated in Tests Nos. 4 and 5, and 83 per cent of the gold in this flotation concentrate can be amalgamated.

Blanket concentration coupled with flotation results in a total recovery of 98·8 to 99·3 per cent as shown in Tests Nos. 2, 3, and 7.

For a small mill treating 25 tons a day, the proper plant to install would be a grate-discharge mill in circuit with a classifier grinding to approximately 70 per cent minus 200 mesh. To avoid trapping much gold in the classifier, a trap might be installed between the mill and classifier. The classifier overflow should pass over blankets to remove particles of gold before entering a mechanically agitated flotation machine. This blanket concentrate should be amalgamated to recover the gold before shipping to a smelter. The residues from the amalgamation added to the flotation concentrate constitute the shipping product.

Ore Dressing and Metallurgical Investigation No. 487

GOLD ORE FROM CANADIAN MINERALS, LIMITED, MORTON LAKE, MANITOBA

Shipment. A shipment consisting of five bags of gold ore, weighing approximately 500 pounds, was received January 17, 1933. The shipment was from the North Star group of claims near Morton Lake, Manitoba, and was submitted by Thos. Young, President Canadian Minerals, Limited, The Pas, Manitoba.

Characteristics of the Ore. Under the microscope the ore is seen to consist of stringers of pyrrhotite, arsenopyrite, pyrite, sphalerite, and chalcopyrite in a quartz gangue. Native gold occurs as fine grains in both the sphalerite and the quartz, and occasionally as fine distinct veinlets in the quartz. The relationships of the minerals indicate that the gold is contemporaneous with the sphalerite-chalcopyrite mineralization and that gold is thus to be expected in this association.

Purpose of Experimental Tests. The object of the investigation was to determine the value of the sample and to indicate the best process for treating the ore to advantage.

Experimental Tests

Sampling and assaying of the ore showed the shipment to contain 0·57 ounce gold, 0·06 ounce silver per ton, 0·20 per cent zinc, and 0·08 per cent copper. Amalgamation recovers 85 to 88 per cent of the gold.

The investigation included tests by amalgamation at different grindings, amalgamation and flotation, amalgamation and cyanidation, straight cyanidation, and blanket concentration.

AMALGAMATION AND CYANIDATION

Test No. 1

A sample of the ore was ground dry to pass 48 mesh and amalgamated. After separating amalgam, the tailing was cyanided for 48 hours, 1 : 3 dilution with a 1·0 pound KCN per ton solution and 3 pounds lime per ton of ore.

Mesh	Weight, per cent
− 48+ 65	19·4
− 65+100	21·4
−100+150	15·5
−150+200	12·5
−200	31·2
	100·0

Results:

Amalgamation—

Feed	0·57 Au oz./ton
Amalgamation tailing	0·13 "
Recovery	77·2 per cent

Cyanidation—

Feed—Amalgamation tailing	0·13 Au oz./ton
24-hour cyanide tailing	0·02 "
48-hour cyanide tailing	0·02 "
Extraction	84·6 per cent
Cyanide consumption	1·05 lb./ton ore
Recovery by amalgamation and cyanidation	96·5 per cent

Test No. 2

A sample was ground minus 100 mesh and treated as in Test No. 1.

Results:

Feed	0·57 Au oz./ton
Amalgamation tailing	0·085 "
Recovery	85·2 per cent
24-hour cyanide tailing	0·01 Au oz./ton
48-hour cyanide tailing	0·005 "
Total recovery by amalgamation and cyanidation	99·1 per cent

AMALGAMATION AND FLOTATION

Test No. 3

A sample was ground wet in a jar mill loaded with iron balls until 60 per cent passed 200 mesh. After removing the balls, the pulp was amalgamated. The amalgam was then removed, the pulp conditioned in a flotation machine for 15 minutes with 4 pounds soda ash per ton and floated with 0·14 pound amyl xanthate and 0·10 pound pine oil per ton.

Mesh	Weight, per cent
— 48+ 65	0·62
− 65+100	5·46
−100+150	14·20
−150+200	19·62
−200	60·10

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	0·57	100·0
Amalgamation tailing		0·065	
Amalgam			88·6
Flotation concentrate	1·92	1·86	6·3
Flotation tailing	98·08	0·03	5·1

The results indicate that 88·6 per cent of the gold can be recovered as amalgam with an additional 6·3 per cent by flotation in a product 1·92 per cent of the weight of ore milled.

BLANKET CONCENTRATION

Test No. 4

A sample was ground 66 per cent solids in a jar mill containing iron balls until 75 per cent passed 200 mesh. The pulp was then passed over a corduroy blanket.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed (cal.)	100·00	0·64	100·0
Blanket concentrate	3·27	15·36	78·2
Blanket tailing	96·73	0·10	21·8

The blanket recovers 78·2 per cent of the gold in a product 3·27 per cent of the weight of ore milled. The tailing contains 0·10 ounce per ton.

FLOTATION

Test No. 5

A sample was ground 75 per cent minus 200 mesh in a jar mill with 4 pounds soda ash, 0·07 pound Aerofloat No. 25. Following the addition of 0·14 pound sodium xanthate and 0·06 pound pine oil, a concentrate was removed.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	0·58	100·0
Concentrate	3·54	5·68	34·8
Tailing	96·46	0·39	65·2

STRAIGHT CYANIDATION

Test No. 6

Samples of the ore were ground through different meshes and agitated 1 : 3 dilution with 1·0 pound per ton solution; 6 pounds of lime per ton of ore was added to maintain protective alkalinity.

Mesh grind	Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Consumption, lb./ton	
					KCN	CaO
− 48	24	0·57	0·14	75·4	0·9	4·8
− 48	48	0·57	0·05	91·2	0·6	4·8
−100	24	0·57	0·145	74·5	0·9	5·2
−100	48	0·57	0·04	93·0	0·9	4·8
−150	24	0·57	0·09	84·3	1·2	5·5
−150	48	0·57	0·035	93·8	1·05	5·3
−200	24	0·57	0·06	89·4	1·35	5·7
−200	48	0·57	0·01	98·2	1·2	5·3

SUMMARY AND CONCLUSIONS

Much of the gold in the ore is free and can be recovered by amalgamation. Tests Nos. 1, 2, and 3 show recoveries ranging from 77 per cent to 88 per cent, depending on the fineness of grinding.

Cyanidation extracts 98 per cent of the gold when the ore is ground minus 200 mesh. When applied to the tailing from amalgamation of coarsely ground ore, a final residue containing 0·02 ounce per ton is obtained. Similar treatment of a somewhat more finely ground product gives a total recovery of 99·1 per cent as in Test No. 2.

Test No. 3 indicates that amalgamation followed by flotation gives a recovery of 94·9 per cent.

Test No. 4 shows that following an average grind 78 per cent of the gold can be caught on blankets, thus removing the bulk of the gold and larger particles.

Flotation alone does not yield high recoveries due no doubt to heavy particles of free gold in the ore.

The process indicated by this investigation is cyanidation. The ore should be ground in cyanide solution in a grate-discharge ball mill in closed circuit with a classifier to give an overflow between 60 and 70 per cent minus 200 mesh. A hydraulic trap should be installed between the ball mill discharge and classifier. This will ensure heavy particles of gold being removed from the classifier circuit. The classifier overflow should pass over blankets before entering the cyanide agitators, where approximately 24 hours' agitation should be given.

Trap cleanings and blanket concentrate should be barrel-amalgamated to recover the free gold and the residue added to the blanket tailing entering the cyanide agitators.

The remainder of the plant should follow standard practice, including thickeners, filters, Merrill-Crowe precipitation and refinery.

However, for a pilot mill treating 25 tons to 50 tons of ore per day, the initial installation to avoid costly equipment could be a ball mill and classifier as outlined above, followed by blankets and flotation cells. The blanket concentrate should be amalgamated and the residues added to the flotation concentrate which could be sold to a smelter.

This will enable the operators to install a plant at a small cost and to recover the major part of the gold in the ore mined during development of the property.

Ore Dressing and Metallurgical Investigation No. 488

GOLD ORE FROM WINE HARBOUR GOLD DISTRICT, GUYSBOROUGH COUNTY, NOVA SCOTIA

Shipment. A shipment of four barrels of ore samples was received December 20, 1932. The samples were numbered 1, 2, 3, and 4, and weighed 165, 183, 123, and 186 pounds respectively. The samples were submitted by E. H. Henderson, Inverness, Nova Scotia.

Characteristics of the Ore:

Sample No. 1. The gangue consists of grey to glassy quartz, with some slate. Due to the weathered condition the metallic iron-bearing minerals are at present represented only by their product "limonite".

Sample No. 2. The gangue is composed of white quartz and some sericitic material in films in the quartz. The metallic minerals observed are coarse-textured crystals of arsenopyrite and considerable "limonite".

Sample No. 3. The gangue is white quartz and slate. This sample also represents much weathered material and contains considerable "limonite".

Sample No. 4. Sample No. 4 represents much fresher material than the preceding three samples. The gangue is greyish to white quartz.

The metallic minerals include considerable arsenopyrite, moderate amounts of pyrrhotite, a small amount of galena, and, rarely, small grains of chalcopyrite.

Native gold was observed under the microscope in some of the specimens and the samples are characteristic of many Nova Scotia occurrences which carry this metal.

An average analysis of each of the samples was as follows:—

Sample No.	1	2	3	4
Gold (Au), oz./ton	0·11	0·12	0·32	0·545
Silver (Ag), oz./ton	0·055	0·027	0·045	0·265
Iron (Fe), per cent	4·45	3·95	3·90	3·30
Arsenic (As), per cent	2·25	0·98	1·27	1·55
Sulphur (S), per cent	1·25	1·33	0·70	0·91
Insol., per cent	88·20	88·85	88·25	89·45

Experimental Tests

The four lots comprising the shipment were separately sampled and assayed.

Small-scale amalgamation and flotation tests were made on each of the samples and a cyanidation test was made on Sample No. 4.

Recoveries by amalgamation ranged from 85 to 88 per cent approximately.

Flotation gave very poor results due to the highly oxidized condition of the ore and to the presence of coarse gold which was lost in the machine.

Cyanidation gave good recovery on the one sample that was treated by this method.

AMALGAMATION

Tests Nos. 1 to 4

In this series of tests the ore at minus 20 mesh was ground in a small ball mill for 15 minutes, and then amalgamated with mercury for 30 minutes in 1 : 1 pulp.

Screen Analyses of Amalgamation Tailings:

Sample No.	Test No.	Mesh	Weight, per cent	Assay, Au, oz./ton	Average tailing
1	1	+200 −200	36·9 63·1	0·005 0·02	0·014
2	2	+200 −200	32·2 67·8	0·005 0·015	0·018
3	3	+200 −200	23·1 76·9	0·04 0·05	0·048
4	4	+200 −200	42·3 57·7	0·06 0·07	0·066

Summary:

Sample No.	Test No.	Feed assay, Au, oz./ton	Tailing assay, Au, oz./ton	Recovery, per cent
1	1	0·11	0·014	87·3
2	2	0·12	0·018	85·0
3	3	0·32	0·048	85·0
4	4	0·545	0·066	87·9

FLOTATION

Tests Nos. 1 to 4

In this series of tests the ore at minus 20 mesh was ground in a small ball mill for 15 minutes and then floated.

Charge to Ball Mill:

Ore	1,000 grammes
Water	750 c.c.
Soda ash	8·0 lb./ton
Aerofloat No. 25	0·07 "

Reagents to Cell:

Potassium amyl xanthate..0·10 lb./ton
Pine oil..0·20 "
Tarol No. 1...0·10 "

Results:

Sample No.	Test No.	Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
1..................	1	Concentrate......	12·4	0·34	65·8
		Tailing...........	87·6	0·025	34·2
		Feed (cal.).......	100·0	0·064	
2..................	2	Concentrate......	14·4	0·29	76·5
		Tailing...........	85·6	0·015	23·5
		Feed (cal.).......	100·0	0·054	
3..................	3	Concentrate......	12·7	0·95	90·8
		Tailing...........	87·3	0·14	9·2
		Feed (cal.).......	100·0	0·133	
4..................	4	Concentrate......	10·6	2·10	57·4
		Tailing...........	89·4	0·185	42·6
		Feed (cal.).......	100·0	0·388	

CYANIDATION

In this test the ore at minus 20 mesh was ground for 15 minutes in water in a ball mill. The pulp was then made up to 2·5 : 1 dilution and agitated for 48 hours in solution running 2 pounds per ton in KCN.

Summary of Cyanidation Test, Sample No. 4:

Product	Assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
Feed................................	0·545	96·3	0·62	5·80
Tailing..............................	0·02			

CONCLUSIONS

This ore may be treated by straight cyanidation or by cyanidation after amalgamation. A good flow-sheet would be to grind the ore in cyanide solution and pass the pulp over blankets to catch the coarse gold. The blanket concentrate could be washed and barrel-amalgamated, and the amalgamation tailing re-united with the blanket tailing in the cyanide plant. The blankets would prevent coarse gold settling out in the agitators

Amalgamation and flotation would not be feasible with this ore, because it is so highly oxidized that it is not amenable to flotation. This, however, would probably not apply to ore mined at depth where the highly oxidized condition would not exist.

Ore Dressing and Metallurgical Investigation No. 489

GOLD ORE FROM THE HANTS GOLD MINES, LTD., CENTRAL RAWDON, NOVA SCOTIA

Shipment. A shipment consisting of three barrels, marked A, B, and C, and containing approximately 400 pounds of ore, was received on December 16, 1932. The shipment was sent by the Hants Gold Mines, Ltd., Central Rawdon, Nova Scotia, on instructions from E. H. Henderson, Halifax.

Characteristics of the Ore. Samples A and B consisted of vein matter, chiefly grey to white quartz. Considerable sericitic material was present as films on the quartz. Oxidation products such as limonite and weathered arsenopyrite were noted. Sample C was said to be taken over a width of 7 feet and consisted of semi-banded slates mixed with quartz, quartz predominating. The material was considerably weathered. In addition to arsenopyrite, minute quantities of galena and chalcopyrite were observed.

Experimental Tests

As Samples A and B were of quartz from narrow veins and could not be representative of the material constituting a mill feed, all tests were made on Sample C.

Sampling and assaying showed the three lots to contain:

—	Gold, oz./ton	Arsenic, per cent
A	0·365	0·19
B	3·23	0·63
C	1·255	0·80

The investigation made on Sample C showed that 86·8 per cent of the gold could be recovered by amalgamation at minus 48-mesh grinding.

Cyanidation of the unground amalgamation tailing leaves a tailing containing 0·01 ounce gold per ton, an overall recovery of 99·2 per cent.

Straight cyanidation at minus 48 mesh gives the same recovery.

Amalgamation followed by flotation shows 86·5 per cent recovered by amalgamation and an additional 10·2 per cent in a concentrate containing 1·76 ounces gold per ton.

AMALGAMATION AND CYANIDATION

Test No. 1

A sample of lot "C" was ground dry to pass 48 mesh with 35·8 per cent minus 200 mesh and then amalgamated.

Results:

Feed..Au 1·255 oz./ton
Tailing...Au 0·17 "
Recovery by amalgamation....................................86·8 per cent

Samples of the amalgamation tailing were cyanided for 48 hours, 1 : 3 dilution with a KCN solution 1·0 pound per ton and 4 pounds lime per ton of ore.

Results:

24-hour cyanide tailing.....................................Au 0·02 oz./ton
Recovery by amalgamation and cyanidation....................98·4 per cent
48-hour cyanide tailing.....................................Au 0·01 oz./ton
Recovery by amalgamation and cyanidation....................99·2 per cent
Cyanide consumption...KCN 0·9 lb./ton ore

Test No. 2

A sample was ground minus 100 mesh and treated as in Test No. 1.

Results:

Recovered by amalgamation...................................90·0 per cent
24-hour cyanide tailing.....................................Au 0·01 oz./ton
Recovery by amalgamation and cyanidation....................99·2 per cent

STRAIGHT CYANIDATION

Test No. 3

Samples of the ore were ground to pass varying meshes and cyanided 1 : 3 dilution with a 1·0 pound KCN per ton solution; 4 pounds of lime per ton of ore was added to the coarser sizes, and 6 pounds to the finer.

Mesh grind	Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption	
					KCN	CaO
48........	24	1·255	0·02	98·4	0·3	2·9
48........	48	1·255	0·01	99·2	0·6	3·2
100........	24	1·255	0·02	98·4	0·9	4·2
100........	48	1·255	0·015	98·8	1·05	4·2
150........	24	1·255	0·015	98·8	0·75	5·7
150........	48	1·255	0·01	99·2	0·9	5·0
200........	24	1·255	0·01	99·2	0·9	5·8
200........	48	1·255	0·01	99·2	1·2	5·8

These results show that the gold is very soluble in cyanide solution.

AMALGAMATION AND FLOTATION

Test No. 4

A sample of the ore ground minus 48 mesh with 73 per cent minus 200 mesh was amalgamated and the tailing floated with 0·10 pound amyl xanthate, 0·06 pound coal-tar creosote, and 0·06 pound pine oil per ton after being conditioned for 15 minutes with 3 pounds soda ash.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·0	1·255	....
Amalgamation tailing	....	0·17	....
Amalgam	....	....	86·5
Flotation concentrate	7·27	1·76	10·2
Flotation tailing	92·73	0·045	3·3

A total recovery of 96·7 per cent of the gold is obtained by this method. From each 100 tons of ore milled, 7·27 tons concentrate was produced assaying 1·76 ounces gold per ton. The tailing contains 0·04 ounce per ton.

SUMMARY AND CONCLUSIONS

The microscopic examination of the ore shows much limonite, a product of oxidation of iron sulphides. This indicates that the samples furnished are from the upper region of the ore-body. There would be a higher percentage of free gold in this oxidized material than in fresh unaltered ore below the zone of oxidation. Consequently the results of these tests cannot be assumed to apply to fresh ore.

The samples contain a very large percentage of quartz from narrow veins and stringers and do not include the gangue dilution that would ensue from mining these veins. The recovery as indicated would not apply to lower grade run-of-mine ore.

The tests of Sample C indicate a recovery of 86 to 90 per cent of the gold by amalgamation of a coarsely ground feed. The remainder is easily extracted by cyaniding, giving an overall recovery of 99·2 per cent. This same recovery is obtained by straight cyanidation.

Amalgamation followed by flotation does not give as high a recovery, the flotation tailing contains 0·04 ounce per ton.

To determine the most suitable plant to treat this class of ore, it would be necessary to decide what tonnage was to be treated daily. For operations of from 25 to 100 tons per day, the one best suited for highest profit would be crushing in stamps to minus 48 mesh with inside and outside amalgamating plates followed by flotation of the plate tailing in mechanically agitated flotation machines. The concentrate secured could be reground in a barrel amalgamator to recover a small additional amount of gold and the residue sold to a smelter.

For larger tonnages, the ore should be cyanided. Grinding should be done in cyanide solution in a grate discharge mill. Blankets should be installed between the mill and cyanide agitators. This will ensure the removal of coarse particles of gold which otherwise would tend to lodge in the agitators and dissolve slowly.

Blanket concentrate should be barrel-amalgamated and the tailing added to the cyanide treatment plant for further extraction of gold.

Before any decision as to the type of mill to install is made, ore representative of that to be milled should be tested to note its behaviour under the conditions of this investigation.

Ore Dressing and Metallurgical Investigation No. 490

GOLD ORE FROM THE BEATTIE GOLD MINES, LIMITED, DUPARQUET TOWNSHIP, ABITIBI COUNTY, QUEBEC

Shipment. The shipment consisted of a carload of ore from the Beattie Gold Mines, Limited, containing approximately 30 tons from ore mined at, and adjacent to, the 175-foot level. In addition the car contained about 5 tons of sacked ore taken from one round below the 300-foot level in No. 2 shaft. It is stated by the company that the samples tested during this run are more representative of the ore that will be milled at the mine than samples submitted in 1931 and 1932, which were taken from surface ore.

Purpose of Test. Owing to the decision of the directors of the company to market a gold-bearing pyrite concentrate, instead of attempting to treat it at the mine, it was desirable that they have reasonable assurance that the 2-ounce concentrate indicated in former tests can be maintained from ore comparable to that now submitted, and also an attempt be made to improve the grade formerly indicated, as the grade of concentrate produced has an important bearing on the profit when shipping concentrates.

Characteristics of the Ore. With the exception of native gold, no additional metallic minerals were observed under the microscope in the ore from the shipment of January 3, 1933. The sulphides, pyrite and arsenopyrite, are finely disseminated in a quartz-carbonate gangue. Native gold was observed in fine grains in a section of ore from the shaft below the 300-foot level, and in the following relationships:

(1) As a portion of the filling of skeleton crystals of arsenopyrite (*See* Plate IA), and (2) associated with chalcopyrite contained in a mass of fine crystals of arsenopyrite (*See* Plate IB). It is to be noted that in all observed cases the gold is in contact with the arsenopyrite, and apparently later than this mineral.

The grain size of the sulphides of the two shipments is shown below. In general there appears to be a distinct increase in grain size at the 175-foot level, whereas a slight decrease is indicated in the ore from below the 300-foot level. The finest material observed occurs in the ore from the surface.

Grain Size of the Sulphides:

Mesh	A	B	C
+ 35	....	....	....
− 35+ 48	4·9	3·9	10·7
− 48+ 65	3·8	14·8	5·7
− 65+100	8·1	11·5	11·3
−100+150	12·5	11·5	15·7
−150+200	7·1	11·6	8·9
−200+325	17·6	15·2	11·6
−325	46·0	31·5	36·1
Totals	100·0	100·0	100·0

A. Grain analysis of ore from the surface; shipment of September, 1931.
B. Grain analysis of ore from 175-foot level; shipment of January 3, 1933.
C. Grain analysis of ore from shaft below 300-foot level; shipment of January 3, 1933.

Experimental Tests

FLOTATION

Large-scale Continuous Pilot Plant Tests

About 30 tons of ore was run at an average rate of 500 to 900 pounds per hour. Eleven separate runs were made, using four different flow-sheets and various quantities of reagents. Ore was crushed to approximately ½ inch for all runs.

Run No. 1

Feed rate, approximately 550 lb./hr.
Length of run, 8 hours.
Note.—See Figure 1 for flow-sheet.

Reagents, lb./ton:

—	Na_2CO_3	$CuSO_4$	B4	Xanthate	Pine oil	R.B.
Ball mill	2·6	0·46	0·24	....	....	....
Primary cell No. 2	....	....	....	0·005	0·012	....
Rougher cell No. 5	....	....	....	0·005	0·024	....
" No. 11	....	....	....	0·004	0·028	....
" No. 14	....	....	....	....	....	0·01

R.B.—A mixture of 'Rhodamine B' in weak soap solution. Xanthate, all tests—Potassium amyl xanthate.

Screen Tests on Products:

Mesh	Primary classifier overflow	Primary concentrate	Primary cleaner tailing	Rougher concentrate	Regrind cleaner concentrate	Regrind cleaner tailing	Final tailing
+200	32·80	2·1	1·3	4·2	....	....	34·1
−200	67·20	97·9	98·4	95·6	....	....	65·8
+325	40·00	11·0	3·0	7·0	1·0	1·0	49·0
−325	60·00	89·0	97·0	93·0	99·0	99·0	51·0

Dilution:

Primary classifier overflow	37 per cent solids
Final tailing	50 " "

Assays for entire run:

	Oz. Au.
Feed	0·22
Primary concentrate	3·66
Primary cleaner tailing	1·04
Rougher concentrate	0·18
Regrind cleaner tailing	0·13
Regrind cleaner concentrate	3·96
Final tailing	0·05
Ratio of concentration	22·17 : 1
Recovery percentage	78·3 per cent
Average grade of concentrate produced	3·81 oz./ton

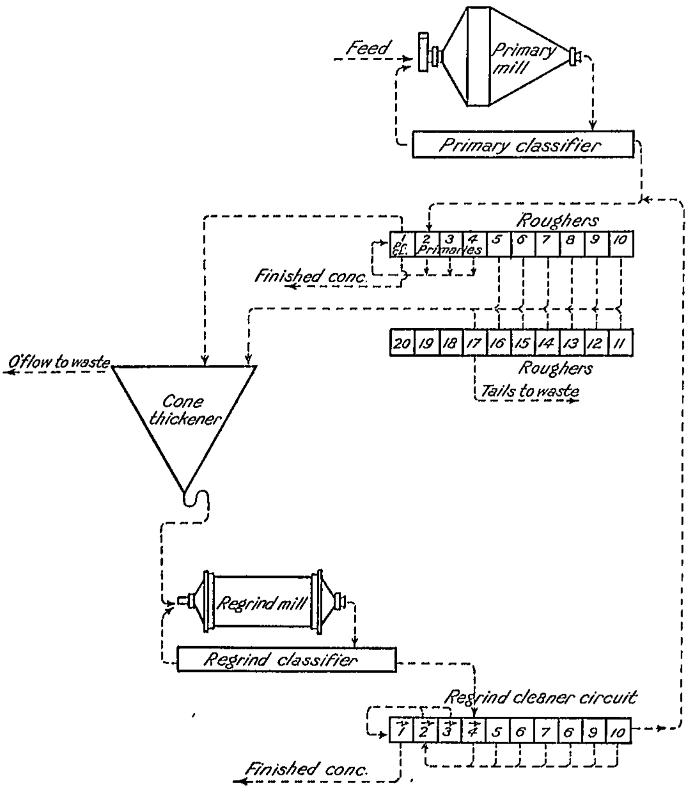


Figure 1. Flow-sheet used in Runs Nos. 1 and 2 on Beattie ore.

Run No. 2

A continuous run was made at the rate of 550 pounds per hour. The same flow-sheet was used as that of Figure 1.

Reagents, lb./ton:

—	Na_2CO_3	$CuSO_4$	B4	Xanthate	Pine oil	R.B.
Ball mill	2·6	0·46	0·15	...	...	...
Primary cell No. 2	...	...	...	0·008	0·006	...
Rougher cell No. 5	...	...	...	0·006	0·04	...
" No. 11	...	...	...	0·006	0·05	...
" No. 14	...	...	...	...	...	0·01
Regrind mill	0·26	0·20	...	...	...	...
Cleaner cell No. 6	...	...	...	0·005	0·005	...

Screen Tests:

Mesh	Primary classifier overflow	Primary concentrate	Primary cleaner tailing	Rougher concentrate	Regrind cleaner concentrate	Regrind cleaner tailing	Final tailing
+200	8·8	0·2	0·9	1·7	...	...	2·2
−200	91·0	99·6	98·6	98·2	...	...	97·8
+325	39·0	12·0	3·0	5·0	5·0	4·0	42·0
−325	61·0	88·0	97·0	95·0	95·0	96·0	58·0

Special tailing check sample—
+200 .... 27·1
−200 .... 73·0

Dilution:

Primary classifier overflow .... 33 per cent solids
Regrind cleaner tailing .... 20 " "
Final tailing .... 52 " "

Assays:

	Oz. Au/ton
Feed	0·215
Primary concentrate	3·280
Primary cleaner tailing	0·825
Rougher concentrate	0·140
Regrind concentrate	2·280
Regrind cleaner tailing	0·110
Final tailing	0·035
Special final tailing	0·030
Cone overflow	0·090

Average grade of concentrate produced .... 2·8 oz./ton
Ratio of concentration .... 15·7 : 1
Concentrate produced per 100 tons milled .... 6·369 tons

Run No. 3

A continuous run of 8 hours was made at the rate of 750 pounds per hour. (See Figure 2 for flow-sheet used.)

Reagents, lb./ton:

—	Na_2CO_3	$CuSO_4$	B4	Xanthate	Pine oil	R.B.
Ball mill	2·66	0·42	0·16	...	...	...
Primary cell No. 2	...	...	...	0·02	0·01	...
Rougher cell No. 5	...	...	...	0·019	0·10	...
" No. 11	...	...	...	0·05	0·05	...
" No. 14	...	...	...	...	...	0·005
" No. 15	...	...	...	...	...	...
Regrind mill	0·35	0·15	...	...	...	...
Cleaner cell No. 6	...	...	...	0·01	0·01	...

Screen Tests:

Mesh	Primary classifier overflow	Primary concentrate	Primary cleaner tailing	Rougher concentrate	Regrind cleaner concentrate	Regrind cleaner tailing	Final tailing
+200...	14·4	5·7	2·5	9·7	....	....	12·1
−200...	85·6	94·3	97·0	89·7	....	....	87·6
+325...	40·0	23·0	7·0	9·0	5·0	1·0	35·0
−325...	60·0	77·0	93·0	91·0	95·0	99·0	65·0

Dilution:

Primary classifier overflow....................................	37 per cent solids
Rougher concentrate..	5 " "
Regrind cleaner tailing...	13 " "
Final tailing..	43 " "
Cone overflow..	0·5 " "
Cone underflow..	30 " "

Assays:

	Oz. Au/ton
Feed..	0·20
Primary concentrate...	2·80
Primary cleaner tailing..	0·77
Rougher concentrate...	0·10
Regrind cleaner concentrate...................................	1·38
Regrind cleaner tailing..	0·035
Final tailing..	0·013

Average grade of concentrate produced..........................	2·33 oz./ton
Ratio of concentration..	12·4 : 1
Concentrate produced per 100 tons milled.......................	8·06 tons

Run No. 4

A continuous run of 8 hours was made at the rate of 750 pounds per hour. (See Figure 2 for flow-sheet used.)

Reagents, lb./ton:

—	Na_2CO_3	$CuSO_4$	W.G.T.	Xanthate	Pine oil	R.B.	H_2SO_4
Ball mill........	1·7	0·17	0·12	....	....	....	....
Primary cell—							
No. 2........	....	....	....	0·02	0·02	....	....
No. 5........	....	....	....	0·02	0·13	....	....
No. 11........	....	....	....	0·02	0·05	....	....
No. 17........	....	....	....	....	0·06	0·01	....
Regrind mill....	0·08	0·17	....	....	....	....	....
Cleaner cell—							
No. 2........	....	....	....	....	....	....	0·14
No. 6........	....	....	....	0·01	0·01	....	....

Screen Tests:

Mesh	Primary classifier overflow	Primary concentrate	Primary cleaner tailing	Rougher concentrate	Regrind cleaner concentrate	Regrind cleaner tailing	Final tailing
+200...	14·2	1·8	2·7	6·9	....	....	22·4
−200...	85·6	98·2	97·3	92·7	....	....	77·2
+325...	18·0	3·0	2·0	6·0	35·0	6·0	34·0
−325...	82·0	97·0	98·0	94·0	65·0	94·0	66·0

Dilution:

Primary classifier overflow	35 per cent solids
Regrind classifier overflow	28 " "
Cone underflow	38 " "
Final tailing	50 " "

Assays:

	Oz. Au/ton
Feed	0·21
Primary concentrate	2·96
Primary cleaner tailing	0·75
Rougher concentrate	0·12
Regrind concentrate	1·22
Regrind cleaner tailing	0·038
Final tailing	0·017

Average grade of concentrate produced	2·15 oz./ton
Ratio of concentration	11·06 : 1
Concentrate produced per 100 tons milled	9·04 tons

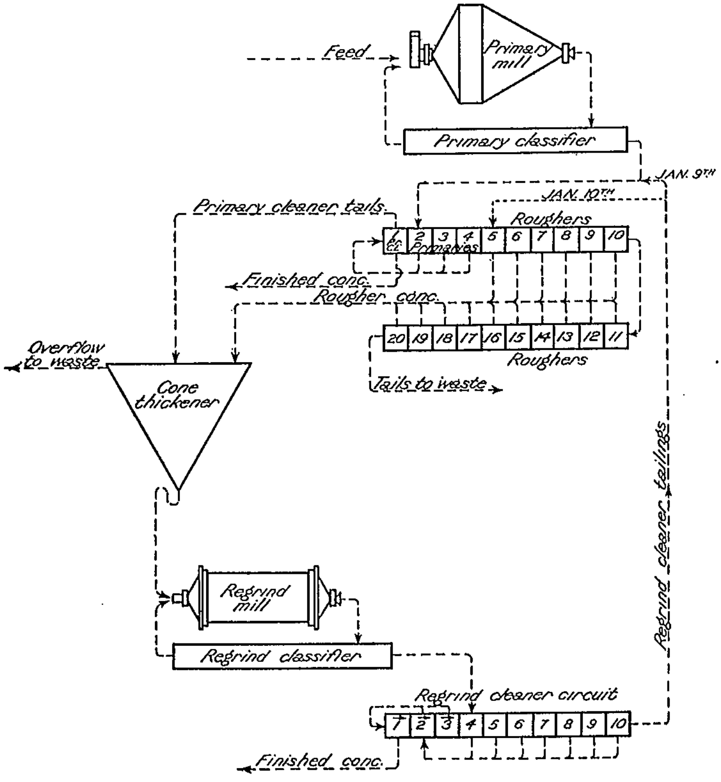


Figure 2. Flow-sheet used in Runs Nos. 3, 4, 5, and 6 on Beattie ore.

Run No. 5

An eight-hour run was made on the continuous testing unit at the rate of 750 pounds per hour. The primary cells and the regrind cleaner cells were pulled slowly and the rougher cells very rapidly in an effort to make both a high-grade concentrate and a low-grade tailing. (See Figure 2 for flow-sheet used.)

Reagents, lb./ton:

—	Na_2CO_3	W.G.T.	Xanthate	Pine oil	H_2SO_4
Ball mill	1·75	0·12			
Primary cell No. 2			0·02	0·02	
Rougher cell No. 5			0·09	0·12	
" No. 11			0·07	0·07	
" No. 17			0·03		
" No. 18				0·04	
Cleaner cell No. 1					0·07
" No. 6			0·007	0·01	

The copper sulphate feed was changed to the conditioner ahead of No. 2 primary cell instead of being fed to the ball mill. Rate of feed: 0·5 pound per ton.

Screen Tests:

Mesh	Primary classifier overflow	Primary concentrate	Primary tailing	Primary cleaner tailing	Rougher concentrate cone overflow	Regrind cleaner concentrate	Regrind cleaner tailing	Final tailing
+200	16·6	3·0	24·7	2·0	21·2	8·0		32·1
−200	83·2	97·0	75·3	97·8	78·7	92·0		67·8
+325	35·0	8·0	36·0	7·0	31·0	23·0	11·0	45·0
−325	65·0	92·0	64·0	93·0	69·0	77·0	89·0	55·0

Dilution:

Primary classifier overflow	35 per cent solids
Cone underflow	35 " "
Regrind cleaner tailing	12 " "
Final tailing	44 " "

Assays:

	Oz. Au/ton
Feed	0·225
Primary concentrate	2·80
Primary tailing	0·05
Primary cleaner concentrate	3·16
Primary cleaner tailing	0·935
Rougher concentrate	0·210
Regrind cleaner concentrate	1·340
Regrind cleaner tailing	0·080
Final tailing	0·020
Final tailing check	0·020
Final tailing check sample	0·020
Final tailing check on above	0·020

Primary circuit. Ratio of concentration, 15·7 : 1; 750 lb. feed per hour produced 47·74 lb. primary concentrate.

Primary cleaner circuit. Ratio of concentration, 1·19 : 1; or 40·11 lb. primary cleaner concentrate was produced and 7·63 lb. cleaner tailing.

75719—4½

Rougher circuit. Ratio of concentration, 6·33 : 1; 809 lb. primary tailing (primary tailing plus circulating load) = 127·7 lb. material to regrind circuit.

Regrind cleaner circuit. Ratio of concentration, 9·69 : 1, or 13·17 lb. regrind cleaner concentrate produced.

Average grade of concentrate produced, 2·77 ounces per ton.

For each 100 tons milled, 15·7 tons of material would be sent to the regrind circuit. This material is 69 per cent minus 325 mesh and the tailing from circuit is 89 per cent minus 325 mesh. Therefore 20 per cent of this material or 3·14 tons of sand per 100 tons milled was reground.

Reground Cleaner Concentrate:

—	Gold	Iron	Sulphur	Arsenic	Insoluble
Assay	1·34	20·51	21·11	1·76	50·00

Primary Concentrate:

—	Gold	Iron	Sulphur	Arsenic	Insoluble
Assay	2·80	35·02	39·33	2·54	16·25

Average as content of combined concentrates........................ 2·34 per cent

Run No. 6

An eight-hour continuous run at the rate of 750 pounds per hour. Rougher cells were pulled very slowly in an effort to reduce the amount of material being sent to the regrind circuit. Same flow-sheet as Figure 2.

Reagents, lb./ton:

—	Na_2CO_3	$CuSO_4$	W.G.T.	Xanthate	Pine oil	H_2SO_4	R.B.
Ball mill	1·75	0·50	0·16				
Primary cell No. 2				0·03	0·02		
Rougher cell No. 5				0·03	0·02		
" No. 11				0·07	0·02		
" No. 14							0·0026
" No. 15					0·04		
Cleaner cell No. 1						0·10	
" No. 6					0·02	0·01	

Screen Tests:

Mesh	Primary classifier overflow	Primary concentrate	Primary tailing	Primary cleaner tailing	Rougher concentrate cone underflow	Regrind cleaner concentrate	Regrind cleaner tailing	Final tailing
+200	21·8	9·4	24·8	6·2	11·4			20·6
−200	78·2	90·6	75·0	93·8	88·4			79·2
+325	32·0	7·0	32·0	5·0	11·0	35·0	3·0	31·0
−325	68·0	93·0	68·0	95·0	89·0	65·0	97·0	69·0

The screen tests checked well. Dry froth on primaries. It will be noted that the amount of water being removed from the rougher circuit was greatly reduced from that of previous runs. Final tailing is 35 per cent solids, whereas previous runs were from 45 to 50 per cent solids. This would mean that the thickener capacity for rougher concentrate would be greatly reduced.

Assays:

	Oz. Au/ton
Feed	0·22
Primary concentrate	2·48
Primary tailing	0·035
Primary cleaner concentrate	2·94
Primary cleaner tailing	0·71
Rougher concentrate	0·12
Regrind cleaner concentrate	0·92
Regrind cleaner tailing	0·04
Final tailing	0·22

Primary circuit. Ratio of concentration, 13·21 : 1; 750 lb. primary feed produce 56·77 lb. primary concentrate.

Primary cleaner circuit. Ratio of concentration, 1·25 : 1; 56·77 lb. primary concentrate give 45·4 lb. primary cleaner concentrate and 11·36 lb. primary cleaner tailing.

Rougher circuit. Ratio of concentration, 7·8 : 1; 785·22 lb. primary tailing (feed plus circulating load) 100·6 lb. of rougher concentrate sent to regrind circuit.

Regrind cleaner circuit. Ratio, 11 : 1. Feed 100·6 lb. produce 9·14 lb. regrind cleaner concentrate.

This indicates a total production of 54·55 lb. of concentrate having an average grade of 2·6 oz. per ton.

Ratio of concentration (total), 13 : 1.

For each 100 tons milled there would be 12·82 tons of material sent to the regrind circuit. This material is 89 per cent minus 325 mesh and is reduced to 97 per cent minus 325 mesh.

Run No. 7

A continuous run of 8 hours was made at the rate of 780 pounds per hour. See Figure 3 for flow-sheet used. It will be noted that the rougher concentrate from cells Nos. 14 to 20 inclusive was returned to cell No. 5. This was done in order to reduce the amount of material being sent to the regrind circuit.

Reagents, lb./ton:

—	Na_2CO_3	$CuSO_4$	W.G.T.	Xanthate	Pine oil	R.B.	H_2SO_4
Ball mill	1·7	0·48	0·10				
Primary cell No. 2				0·04	0·02		
Rougher cell No. 5				0·08	0·04		
" No. 17						0·002	
" No. 18					0·05		
" No. 19						0·002	
Regrind mill		0·08					
Cleaner cell No. 6				0·01	0·01		
" No. 1							0·13

Screen Tests:

Mesh	Primary classifier overflow	Primary concentrate	Primary tailing	Primary cleaner concentrate	Primary cleaner tailing	Rougher concentrate	Regrind cleaner concentrate	Regrind cleaner tailing	Final tailing	Second regrind concentrate, cells 14–20
+200	26·2	5·2	24·4	4·8	6·8	15·1	....	....	21·1	13·0
−200	72·7	94·3	73·1	94·3	92·4	84·3	....	....	78·9	86·0
+325	....	....	25·0	11·0	5·0	11·0	18·0	1·0	27·0	....
−325	....	....	65·0	89·0	95·0	89·0	82·0	99·0	73·0	....

Assays:

	Oz. Au/ton
Feed	0·025
Primary concentrate	2·700
Primary tailing	0·050
Primary cleaner concentrate	3·000
Primary cleaner tailing	0·820
Rougher concentrate	0·220
Regrind cleaner concentrate	1·080
Regrind cleaner tailing	0·045
Final tailing	0·020
Second rougher concentrate	0·080

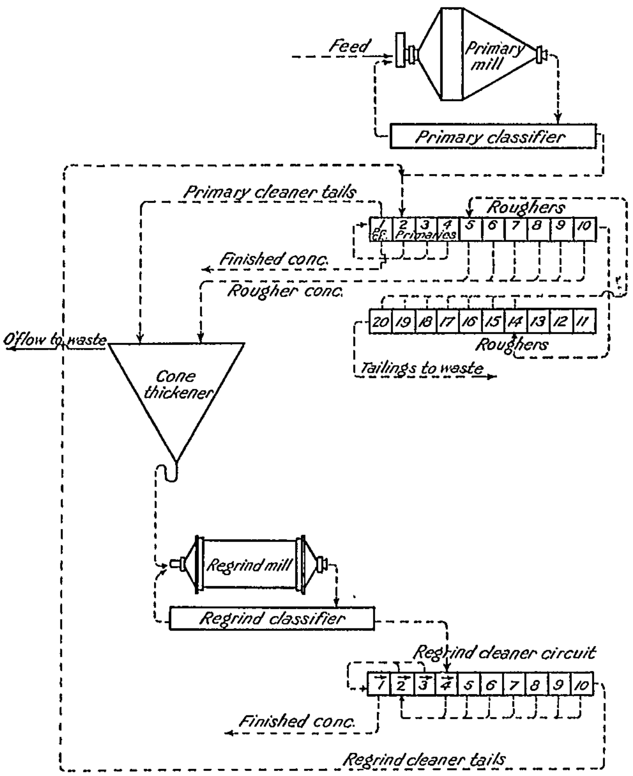


Figure 3. Flow-sheet used for Runs Nos. 7 and 8 on Beattie ore.

Primary circuit: Ratio of concentration, 15·14 : 1; 780 : 15·14= 51·51 lb primary concentrate produced. Primary cleaner circuit: ratio, 1·159 : 1. 51·51 : 1·159 = 44·44 lb. primary cleaner concentrate produced, and 7·07 lb. primary cleaner tailing produced.

Rougher circuit: Ratio of concentration, 6·66 : 1. 826·37 lb. primary tailing (feed plus circulating load). 124·0 lb. rougher concentrate to regrind circuit.

Cleaner circuit: Ratio of concentration, 5·91 : 1. 124 : 5·91 = 20·98 lb. regrind cleaner concentrate produced; 65·42 lb. total concentrate produced having an average value of 2·37 oz. per ton. Total ratio of concentration, 11·9 : 1.

For each 100 tons milled 15·89 tons of material was sent to the regrind circuit.

Rougher concentrate: 89 per cent −325.

Rougher cleaner tailing: 99 per cent −325.

Indicating that 10 per cent of the total sent to regrind circuit was reground, or 1·6 tons of sand was reground for each 100 tons milled.

Run No. 8

A seven-hour continuous run was made on the ore from below the 300-foot level, at the rate of 750 pounds per hour. The flow-sheet was the same as that for Run No. 7.

Reagents, lb./ton:

—	Na_2CO_3	$CuSO_4$	W.G.T.	Xanthate	Pine oil	R.B.	H_2SO_4
Ball mill	1·7	0·46	0·10				
Primary cell No. 2				0·04	0·02		
Rougher cell No. 5				0·08	0·04		
" No. 8						0·002	
" No. 11				0·08	0·03		
" No. 15						0·002	
Regrind mill		0·13					
Cleaner cell No. 1							0·10
" No. 4				0·02	0·01		

Screen Tests:

Mesh	Primary classifier overflow	Primary concentrate	Primary tailing	Primary cleaner concentrate	Primary cleaner tailing	Rougher concentrate	Regrind cleaner concentrate	Regrind cleaner tailing	Final tailing
+200	20·0	3·2	19·2	2·1	7·3	14·4			18·8
−200	79·3	96·6	80·5	97·3	92·2	85·1			80·9
+325	31·0		29·0	12·0		22·0	21·0	2·0	27·0
−325	69·0		71·0	88·0		78·0	79·0	98·0	73·0

Dilution:

Primary classifier overflow	30–33 per cent solids
Cone underflow rougher concentrate	20 " "
Regrind cleaner tailing	8 " "
Final tailing	25–29 " "

NOTE.—100 pounds more ½-inch balls put in regrind mill and classifier overflow raised to try to get a better grinding in regrind circuit. Material sent to regrind circuit is very difficult to classify.

Assays:

	Oz. Au/ton
Feed	0·260
Primary concentrate	2·650
Primary tailing	0·055
Primary cleaner concentrate	3·020
Primary cleaner tailing	0·920
Rougher concentrate	0·026
Regrind cleaner concentrate	1·200
Regrind cleaner tailing	0·065
Final tailing	0·0275

Primary circuit: Ratio of concentration, 12·65 : 1; primary concentrate produced = 59·28 lb.

Primary cleaner circuit: Ratio of concentration, 1·213 : 1; primary cleaner concentrate produced = 48·87 lb.

Rougher circuit: Ratio of concentration, 8·45 : 1.

Primary tailings (feed + circulating load) = 768·78 lb.; rougher concentrate to regrind circuit = 90·97 lb.

Cleaner circuit: Ratio of concentration, 5·82: 1; regrind cleaner concentrate produced = 15·63 lb.

Total production = 64·5 lb. concentrate which assayed 2·58 oz. per ton on average.

Assay of check time sample for cyanidation test = 2·59 oz. per ton.

Material sent to regrind circuit = 12·1 tons per 100 tons milled.

Rougher concentrate: 78 per cent −325.

Regrind cleaner tailing: 98 per cent −325.

Twenty per cent of material sent to the regrind circuit, or 2·42 tons of sand was reground for each 100 tons milled.

Run No. 9

A six-hour run was made on the ore from below the 300-foot level. (See Figure 4 for flow-sheet used.) All the rougher concentrate was sent to the thickener and the regrind cleaner tailing was returned to cell No. 5, which is the first of the rougher circuit. The roughers were pulled rapidly in an effort to reduce the tailing on the previous run.

Reagents, lb./ton:

—	Na_2CO_3	$CuSO_4$	B4	Xanthate	Pine oil	R.B.
Ball mill	1·7	0·45	0·10			
Primary cell No. 2				0·019	0·01	
Rougher No. 5				0·026	0·10	
" No. 11				0·03	0·08	
" No. 14						0·004
" No. 15					0·05	
" No. 16						0·004
Regrind mill	0·12	0·12				
Cleaner cell No. 4					0·005	
" No. 6				0·007		

Screen Tests:

Mesh	Primary classifier overflow	Primary concentrate	Primary tailing	Primary cleaner concentrate	Primary cleaner tailing	Rougher concentrate	Regrind cleaner concentrate	Regrind cleaner tailing	Final tailing
+200	21·9	4·5	20·4	1·5	11·7	1·3	3·5	18·0	18·0
−200	78·0	94·8	79·0	98·5	87·9	98·6	96·1	81·3	81·6
+325	26·0	11·0	30·0	9·C	17·0	3·0	15·0	17·0	31·0
−325	74·0	89·0	70·0	91·0	83·0	97·0	85·0	73·0	69·0

Dilution:

Primary classifier overflow	32–35 per cent solids
Cone underflow rougher concentrate	29 " "
Regrind cleaner tailing	14 " "
Final tailing	35–40 " "

Alkalinity:

Rougher tailing	0·6 lb./ton solution
Cleaner	0·3 " "

Assays:

	Oz. Au/ton
Feed	0·265
Primary concentrate	2·400
Primary tailing	0·055
Primary cleaner concentrate	3·240
Primary cleaner tailing	1·115
Rougher concentrate	0·250
Regrind cleaner concentrate	1·170
Regrind cleaner tailing	0·060
Final tailing	0·035

	Oz. Au/ton
Primary cleaner concentrate	1·480

Primary circuit: Ratio of concentration, 10·28 : 1.
Feed, 750 lb. per hour. Primary concentrate produced = 72·95 lb.

Primary cleaner circuit: Ratio of concentration, 1·65 : 1.
Primary cleaner concentrate produced = 44·21 lb. (by weight, 45·50 lb.)
Primary cleaner tailing produced = 28·74 lb. (by weight, 29·00 lb.)

Rougher circuit: Ratio, 10·75 : 1.
Primary tailing produced = 758·95 lb.
(Feed plus circulating load = Primary tailing).
Rougher concentrate produced = 70·6 lb.

Cleaner circuit: Ratio, 6·41 : 1.

Regrind cleaner concentrate produced = 11·01 lb.

Total production: 55·22 lb. of concentrate averaging 2·82 oz. per ton in value.

Taking rougher concentrate by weight, time sample shows 90 lb. per hour of rougher concentrate being returned to regrind circuit, which would change results as follows:—

90 : 6·41 : 14·04 lb. regrind cleaner concentrate.

44·21 lb. primary cleaner concentrate.

Making total production = 58·25 lb. of a concentrate averaging 2·74 oz. per ton.

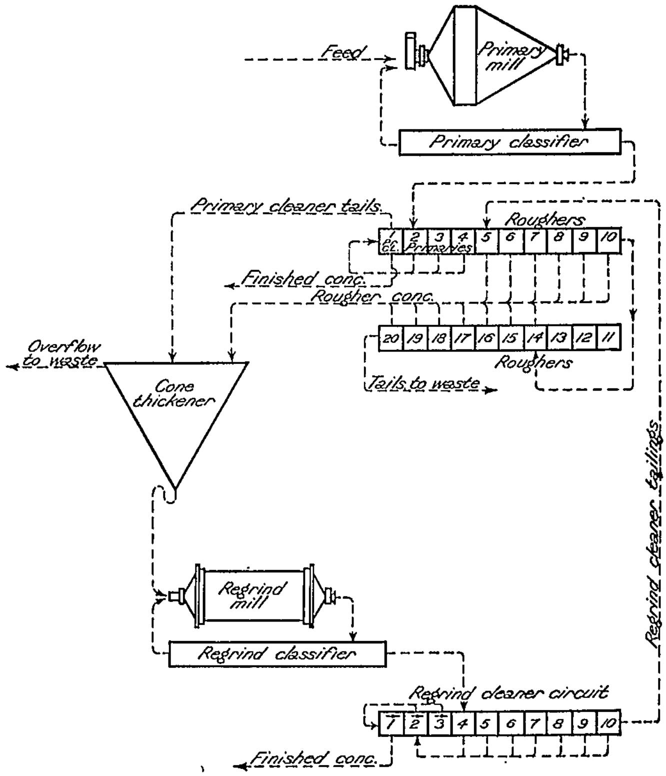


Figure 4. Flow-sheet used in Runs Nos. 9, 10, and 11 on Beattie ore.

Run No. 10

An eight-hour run was made on the ore from above the 175-foot level at the rate of 600 pounds per hour. The same ball load was used in the mill. Feed was reduced from 750 to 600 pounds per hour, the dilution in the mill was reduced and the dilution of the primary classifier overflow was reduced from 35 per cent solids to 30 per cent in an effort to grind finer than preceding tests. It will be noted by the screen tests that the effort was not successful, but remained substantially the same grind.

Reagents, lb./ton:

—	Na_2CO_3	$CuSO_4$	B4	Xanthate	Pine oil	R.B.
Ball mill	1·4	0·50	0·14	....	....	....
Primary cell No. 2	....	....	....	0·024	0·015	....
Rougher No. 5	....	....	....	0·016	0·10	....
" No. 11	....	....	....	0·036	0·06	....
" No. 14	....	....	....	....	....	0·001
" No. 15	....	....	....	....	0·05	....
" No. 16	....	....	....	....	....	0·001
Regrind mill	0·20	0·10	....	....	....	....
Cleaner cells	....	....	....	0·005	....	....

Screen Tests:

Mesh	Primary classifier overflow	Primary concentrate	Primary tailing	Primary cleaner concentrate	Primary cleaner tailing	Rougher concentrate	Regrind cleaner concentrate	Regrind cleaner tailing	Final tailing	Special final tailing
+200	21·9	3·8	22·0	2·2	13·7	10·4	7·0	1·2	27·1	19·2
−200	78·1	95·7	77·4	97·1	85·6	89·5	92·5	96·8	72·3	80·3
+325	34·0	....	37·0	25·0	22·0	8·0	28·0	8·0	38·0	36·0
−325	66·0	....	63·0	75·0	78·0	92·0	72·0	92·0	62·0	64·0

Dilution:

Primary classifier overflow....................29–30 per cent solids
Cone thickener discharge....................28–30 " "
Regrind cleaner tailing....................13–15 " "
Final tailing....................37–42 " "

Assays:

	Oz. Au/ton
Feed	0·265
Primary concentrate	3·160
Primary tailing	0·050
Primary cleaner concentrate	3·420
Primary cleaner tailing	1·430
Rougher concentrate	0·140
Regrind cleaner concentrate	1·270
Regrind cleaner tailing	0·050
Final tailing	0·020
Final tailing (check)	0·020

Primary circuit: Ratio of concentration, 14·46 : 1; primary concentrate produced = 41·49 lb.

Primary cleaner circuit: Ratio of concentration, 1·15 : 1.
Primary cleaner concentrate produced = 36·07 lb. (by weight) = 37·0.
Primary cleaner tailing produced = 5·42 lb.

Rougher concentrate circuit: Ratio of concentration, 3·66 : 1.
Primary tailing produced (feed plus circulating load) 705·26 lb.
Rougher concentrate to regrind circuit (by assay) = 152·59 lb. (bv weight) = 135·00 lb.

Cleaner circuit: Ratio of concentration, 13·55 : 1.

Regrind cleaner concentrate produced = 19·26 lb.

From 600 lb. per hour feed, 55·33 lb. of concentrate was produced, having an average value of 2·77 oz. per ton.

Gold in concentrate—1·478.

Gold in feed—1·590 oz. per ton.

Gold in tailing—0·112 oz. per ton by calculation, recovery 93 per cent.

Gold in tailing—0·108 oz. per ton by assay.

There has probably been a concentration of gold in the primary mill, due to operating with a thin pulp. When pulp was thickened this gold would unload, which would account for the high feed reported in this run.

Run No. 11

A five-hour run was made on ore from above the 175-foot level. Feed, 600 pounds per hour. An attempt was made to do finer grinding in view of improving the grade of concentrate and lowering that of the tailing. The flow-sheet was the same as that used in Run No. 9.

Reagents, lb./ton:

—	Na_2CO_3	$CuSO_4$	B4	Xanthate	Pine oil	R.B.
Ball mill	1·5	0·50	0·15			
Primary cell No. 2				0·02	0·01	
Rougher cell No. 5				0·01	0·10	
" No. 11				0·03	0·06	
" No. 14						0·001
" No. 15					0·02	
" No. 16						0·001

No reagents on cleaner circuit.

$ZnSO_4$ to settle concentrate —0·14 lb. per ton added to rougher concentrate produces clear overflow with excess froth in roughers.

Screen Tests:

Mesh	Primary classifier overflow	Primary concentrate	Primary tailing	Primary cleaner concentrate	Primary cleaner tailing	Rougher concentrate	Regrind cleaner concentrate	Regrind cleaner tailing	Final tailing	Special final tailing
+200	18·7	2·8	19·6	1·6	7·0	9·5	10·2	2·6	17·4	17·6
−200	80·9	96·4	79·8	98·3	93·0	90·5	89·4	96·9	82·0	81·9
+325	32·0	2·0	3·0	14·0	12·0	13·0	24·0	5·0	32·0	31·0
−325	68·0	98·0	97·0	86·0	88·0	87·0	76·0	95·0	68·0	69·0

Dilution:

Primary classifier overflow .... 24–29 per cent solids
Regrind cleaner tailing .... 25 " "
Cone discharge .... 41 " "
Final tailing .... 32 " "

Alkalinity:

Rougher tailing .... 0·5 lb. per ton
Regrind cleaner tailing .... 0·3 " "

Assays:

	Oz. Au/ton
Feed	0·23
Primary concentrate	2·98
Primary tailing	0·045
Primary cleaner concentrate	3·27
Primary cleaner tailing	1·215
Rougher concentrate	0·170
Regrind cleaner concentrate	1·420
Regrind cleaner tailing	0·040
Final tailing	0·020
Special final tailing	0·0225

Primary circuit: Ratio of concentration, 15·86 : 1.

Primary concentrate produced = 37·8 lb.

Primary cleaner circuit: Ratio of concentration, 1·16 : 1.

Primary cleaner concentrate produced = 32·5 lb.

Primary cleaner tailing produced = 5·3 lb.

Recovery—77 per cent of total gold recovered in primary cleaner concentrate.

Rougher circuit: Ratio of concentration = 6·26 : 1.

Primary tailing (feed plus circulating load) = 648·8 lb.

Rougher concentrate sent to regrind circuit = 103·6 lb.

Regrind circuit: Ratio of concentration = 10·61 : 1.

Regrind cleaner concentrate produced = 9·7 lb.

Total concentrate produced = 42·2 lb., having an average value of 2·84 oz. per ton.

Ratio of concentration produced = 14·21 : 1.

Concentrate produced per 100 tons milled = 7 tons.

Rougher concentrate = 17·2 tons sent to regrind circuit for each 100 tons milled.

Special Tests

A series of special tests was conducted at the request of J. J. Denny, consulting engineer.

The first test outlined was to cyanide the flotation concentrate, thereby removing about 65 per cent of the contained gold; then refloat the cyanide tailing on the supposition that the refractory part containing the bulk of the gold would report in the flotation concentrate which would be of a grade profitable to ship to the smelter.

A brief summary of the tests follows. The concentrates obtained from Run No 5 were used for this test.

CYANIDATION

A lot, 1,451·7 grammes, dry weight, of concentrate was ground for 10 minutes in a ball mill. The pulp was then made up to 5·42 : 1 dilution and lime added at the rate of 20·7 pounds per ton. Enough sodium cyanide was added to bring the solution up to 3 pounds per ton in KCN and it was kept there approximately by further additions of the salt from time to time. The pulp was agitated for 24 hours, then it was filtered and washed twice with water. A sample of the cyanide tailing was assayed for gold and the remainder of it was floated in two parts in an attempt to concentrate its contained gold into a smaller bulk.

NOTE.—Considerable frothing occurred during agitation.

Summary:

Feed	2·59 oz. Au/ton
Tailing	0·91 "
Recovery	64·9 per cent
KCN consumed	7·3 lb./ton concentrate
CaO consumed	18·4 " "

FLOTATION

The cyanide tailing was filtered, washed twice, repulped, and divided in two parts for flotation. One part was floated in a 1,000-gramme test machine, with the following reagents: 1 pound $CuSO_4$, 0·10 pound potassium amyl xanthate, and 0·05 pound pine oil; and a black greasy froth was removed for five minutes.

Product	Weight	Assay, Au, oz./ton	Metal, weight	Distribution of gold, per cent
Feed	603·2	0·91	0·274	100
Concentrate	502·5	0·89	0·223	81
Tailing	100·7	1·21	0·060	21
			0·283	102

The second part of cyanide tailing was floated with 1 pound $CuSO_4$, 1·5 pound Na_2CO_3, 0·14 pound potassium amyl xanthate; and a light voluminous froth was given off, which was removed until a light grey froth appeared.

Product	Weight	Assay, Au, oz./ton	Metal, weight	Distribution of gold, per cent
Feed	698·5	0·91	0·318	100
Concentrate	335·4	1·10	0·184	57
Tailing	363·1	0·78	0·141	44
			0·325	101

Only a limited amount of work was done by this method because little hope was held of any success. The process would be expensive and the tests made do not indicate any possible separation between a barren and a gold-bearing sulphide.

The second test was made owing to the suggestion that arsenopyrite could be separated from the other sulphides by table concentration.

A table test was, therefore, made and the flotation concentrate obtained from Runs Nos. 3 and 4 was used.

The product was run over a Wilfley table. The products were sampled and screen analyses were made.

The following table shows the results obtained.

Screen Tests:

Mesh	Table concentrate	Table middling	Table tailing
+200	5·1	9·7	....
−200	94·9	90·3	....
+325	4·0	44·0	4·0
−325	96·0	56·0	96·0

Assays:

—	Gold	Iron	Arsenic	Sulphur
	oz./ton	%	%	%
Table concentrate	2·67	37·00	3·71	41·24
Table middling	1·24	16·40	2·23	16·78
Table tailing	1·87	28·13	4·02	27·80

It is quite evident from the results that no separation can be obtained by tabling. The microscopic slides already made have shown that the arsenopyrite occurs as a skeleton crystal, all minus 325 mesh, and it is so intimately associated with other minerals it is doubtful if any separation either by gravity concentration or flotation could be made.

Study of photomicrograph Plate I B well illustrates the above point.

TABLE I

Microscopic Study of Distribution of Gold in Table Concentrate from Run No. 5

Ratios (at 100)	Concentrate	Middling	Tailing	Total
	per cent	per cent	per cent	per cent
Gold	46·19	21·46	32·35	100
Pyrite	47·47	19·63	32·90	100
Arsenopyrite	37·23	22·41	40·36	100

DISCUSSION OF RESULTS

The microscopic study of reground cleaner concentrate shows this product to contain 67·2 per cent middling, in which 7·4 per cent is sulphide, all minus 325, and 59·8 per cent is gangue. Improvement of the grinding and liberation of more of the sulphides would result in a better grade of concentrate. Twenty-nine per cent of this middling is plus 325 mesh and it is on this material that there is a chance for slight improvement in grinding. Also it is shown that the regrind cleaner concentrate contains 19 per cent free gangue and the best chance of improvement of grade lies in the reduction of this figure. Taking into consideration the small proportion of regrind cleaner concentrate produced, compared with primary cleaner concentrate, it is apparent that the grade will not have to be greatly increased to produce a combined product containing 3 ounces per ton.

TABLE II

Reground Cleaner Concentrate

Run No. 8 (See Plate III A)

Mesh	Free sulphides %	Combined		Free gangue %	Total %
		Sulphides %	Gangue %		
− 65+100	....	....	....	....	....
−100+150	....	....	3·26	....	3·26
−150+200	....	....	8·20	1·10	9·30
−200+325	0·67	....	13·16	2·81	16·64
−325	12·70	7·42	35·22	15·46	70·80
Total	13·37	7·42	59·84	19·37	100·00

Table III shows there is little free sulphide left in the regrind cleaner tailing, but it does contain 11·26 per cent of middling that is plus 325, indicating finer grinding to lower the gold content of this product.

TABLE III

Reground Cleaner Tailing

Run No. 8 (See Plate IIIB)

Mesh	Free sulphides %	Combined		Free gangue %	Total %
		Sulphides %	Gangue %		
− 65+100	....	....	....	....	....
−100+150	....	....	2·34	....	2·34
−150+200	....	....	3·97	1·80	5·77
−200+325	....	....	4·95	4·50	9·45
−325	2·98	2·16	9·70	67·60	82·44
Total	2·98	2·16	20·96	73·90	100·00

Table IV, final tailing, shows the extremely fine dissemination of the sulphides left in the tailing and well illustrates the exceedingly fine grinding that would be necessary to reduce the tailing further and still maintain a satisfactory grade of concentrate.

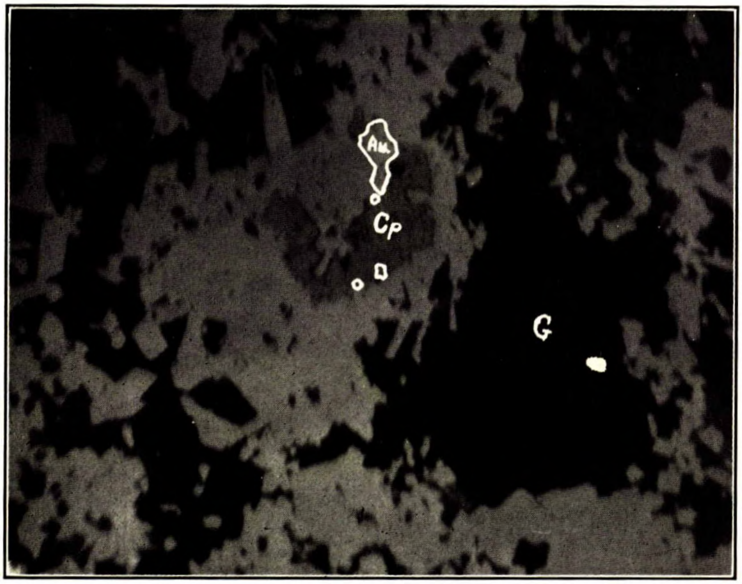
TABLE IV

Final Tailing

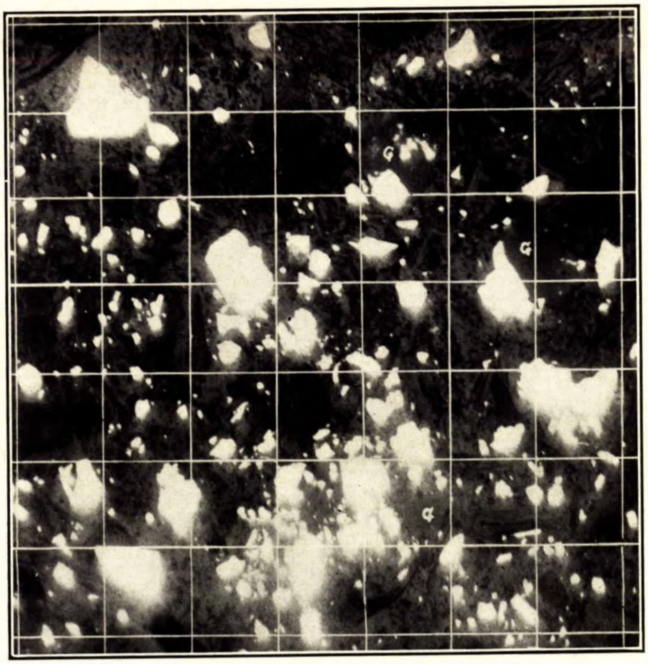
Run No. 9 (See Plate IV)

Mesh	Sulphides %	Combined		Gangue %	Total %
		Sulphides %	Gangue %		
− 65+100	....	....	1·55	....	1·55
−100+150	....	....	7·30	....	7·30
−150+200	....	....	11·60	5·58	17·18
−200+325	....	....	8·25	7·72	15·97
−325	0·17	1·29	9·64	46·90	58·00
Total	0·17	1·29	38·34	60·20	100·00

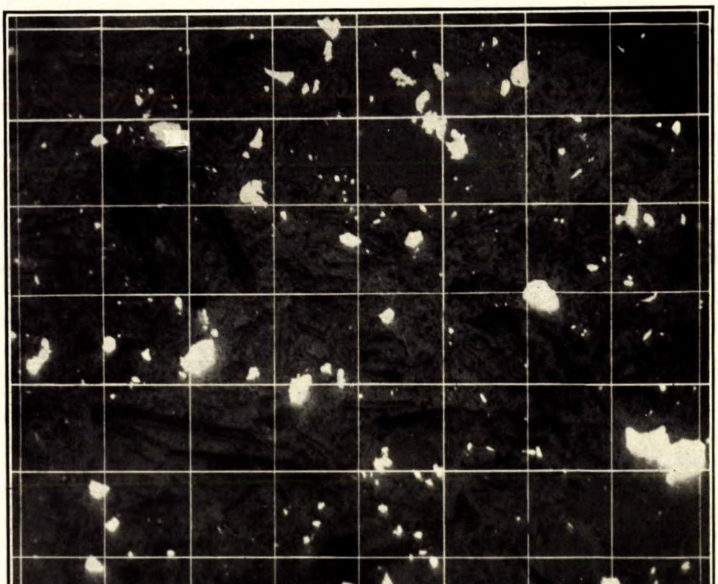
A. Skeletal growth of arsenopyrite with gold (Au) and gangue (G) occurring within the skeleton. Magnification, 1250 X, oil immersion.



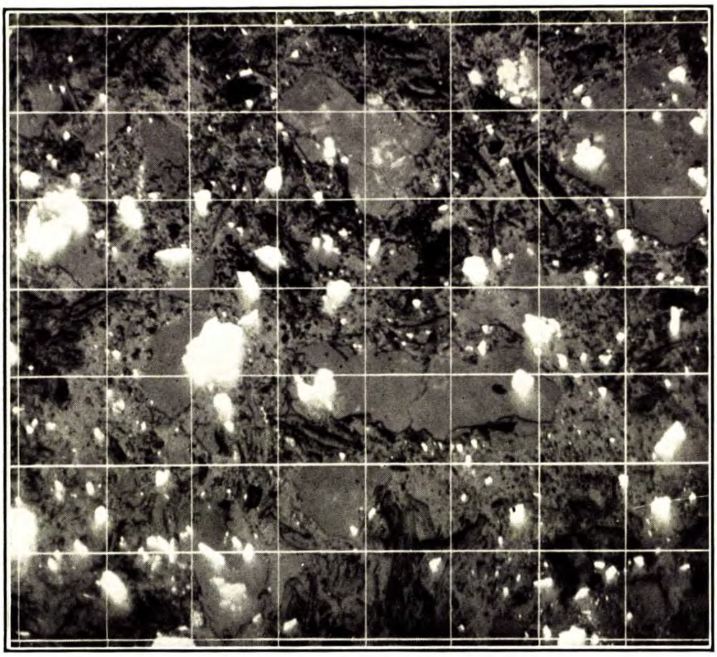
B. Mass of arsenopyrite crystals with chalcopyrite (Cp) and four grains of gold (Au) outlined. Gangue—black. Magnification, 350 X, oil immersion.



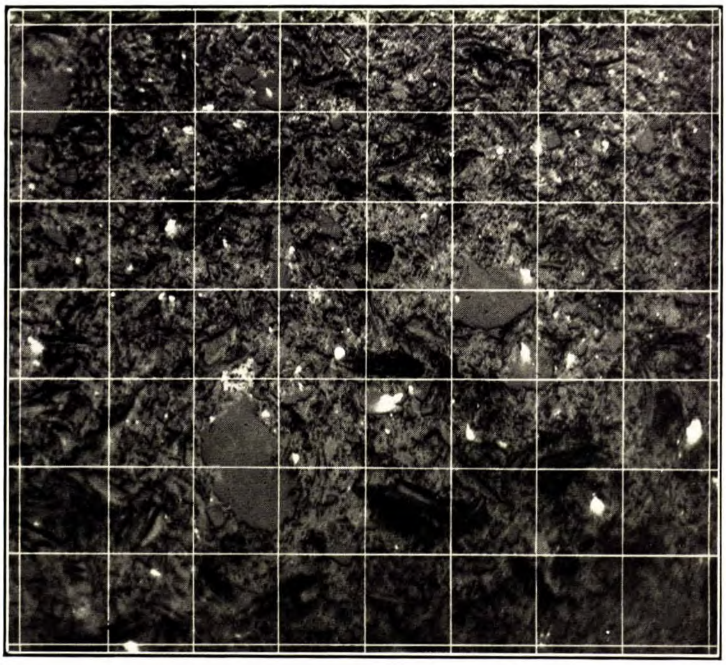
A. Primary cleaner concentrate, Run No. 8. Sulphides—white; gangue—smooth grey (G); mounting medium—rough grey. Magnification, 150 X. A 200-mesh grid is superimposed.



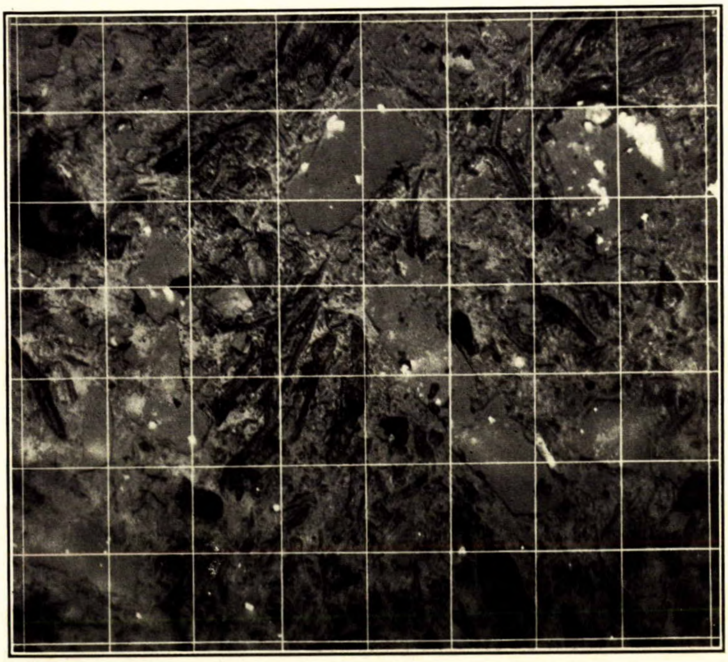
B. Primary cleaner tailing, Run No. 8. Sulphides—white; gangue—smooth grey; mounting medium—rough grey. Magnification, 150 X. A 200-mesh grid is superimposed.



A. Reground cleaner concentrate, Run No. 8. Sulphides—white; gangue—smooth grey; mounting medium—rough grey. Magnification, 150 X. A 200-mesh grid is superimposed.



B. Reground cleaner tailing, Run No. 8. Sulphides—white; gangue—smooth grey; mounting medium—rough grey. Magnification, 150 X. A 200-mesh grid is superimposed.



Final tailing, Run No. 9. Sulphides—white; gangue—smooth grey; mounting medium—rough grey. Magnification, 150 X. A 200-mesh grid is superimposed.

The question of cleaner tailing or middling building up in the circuit and finally increasing the gold in the tailing, has been a debatable point. Referring again to Table II and Plate III A, it will be noted that the regrind cleaner concentrate is principally a middling product and accounts for the elimination of the bulk of this material from the circuit. It is probable that building-up does occur to a certain extent. When making a 2·5-ounce concentrate, as Runs Nos. 3 and 4 indicate, the 0·02-ounce tailing could be slightly improved by accepting a lower grade concentrate, i.e. taking more of the middling product in the concentrate.

Table V and Plate IIA show that the primary cleaner concentrate carried only 2·76 per cent free gangue but 21·16 per cent of the middling product. A slight improvement in grade could be made by eliminating some of this middling from the product.

Table VI and Plate IIB show that in the primary cleaner tailing the sulphides compose 21·2 per cent of the whole product, and of that amount 28·1 per cent is locked in the middling. But this 28·1 per cent sulphide locked as middling consists of particles that are finer than 325 mesh. This again illustrates the exceedingly fine grinding required.

TABLE V

Primary Cleaner Concentrate

Run No. 8 (See Plate IIA)

Mesh	Sulphides %	Combined		Free gangue %	Total %
		Sulphides %	Gangue %		
− 65+100	....	....	....	....	....
−100+150	....	....	1·93	....	1·93
−150+200	1·38	....	3·40	....	4·78
−200+325	12·70	0·97	4·50	....	18·17
−325	62·00	2·76	7·60	2·76	75·12
Total	76·08	3·73	17·43	2·76	100·00

TABLE VI

Primary Cleaner Tailing

Run No. 8 (See Plate IIB)

Mesh	Sulphides %	Combined		Free gangue %	Total %
		Sulphides %	Gangue %		
− 65+100	....	....	2·88	....	2·88
−100+150	....	....	6·25	....	6·25
−150+200	....	....	11·60	....	11·60
−200+325	0·96	....	11·82	2·24	15·02
−325	14·17	6·08	17·30	26·70	64·25
Total	15·13	6·08	49·85	28·94	100·00

SUMMARY OF RESULTS

The result of this series of tests can be said definitely to assure the Beattie management of a 2-ounce concentrate together with a 0·02-ounce tailing from ore similar to that now tested and by the application of the process described, and that this result can be attained by grinding in the primary ball mill circuit to 80 per cent through 200 mesh but only if the flotation middling is reground. These tests further indicate that a 2·5-ounce concentrate can be produced without sacrificing recovery, and it is probable that a 3-ounce concentrate together with a 0·02-ounce tailing will eventually be obtained in the mill.

Ore Dressing and Metallurgical Investigation No. 491

GOLD ORE FROM HALCROW-SWAYZE MINES, LIMITED, IN HALCROW TOWNSHIP, ONTARIO

Shipment. A shipment of 160 pounds of ore was received January 4, 1933, from Horace F. Strong, Manager, Halcrow-Swayze Mines, Limited, Haileybury, Ontario.

Characteristics of the Ore. The metallic minerals observed under the microscope are pyrite, chalcopyrite, and native gold. The pyrite is disseminated in the gangue as irregular grains and rather well formed cubes; the largest size observed was 1·4 mm. in diameter, and the smallest less than 0·01 mm. in diameter, but by far the greater portion of the pyrite varies in size between 0·12 mm. and 0·20 mm. or between 150 and 65 mesh approximately.

Both the chalcopyrite and the gold occur only within the pyrite in the sections examined. Chalcopyrite is very rare as tiny irregular grains. The gold is very fine, varying in diameter of grains from 0·006 mm. to 0·023 mm., or all below 325 mesh.

The gangue consists chiefly of quartz with considerable finely dissem inated carbonate. A greenish grey colour is imparted, possibly by the presence of chloritic (?) material, but this was not determined.

An average assay of the sample was as follows:—

Gold (Au)	0·245 oz./ton
Silver (Ag)	0·053 "

Experimental Tests

A series of small-scale tests was made on this ore for the purpose of finding out how best it might be treated in practice for the recovery of its gold. The work included tests by cyanidation, amalgamation, and concentration, both alone and in combination with each other.

Recovery by cyanidation was excellent, amounting to as much as 98 per cent when the ore was ground 95 per cent minus 200 mesh. Recovery by amalgamation at the same grinding was only 55 per cent. Concentration alone produced good grade concentrate, but high tailing with resulting low recovery.

CYANIDATION

Tests Nos. 1 to 4

Four tests were made on the ore ground wet in a ball mill for 15, 20, 25, and 30 minutes respectively. Screen tests on the above products showed them to be 78·4, 87·1, 94·7, and 95·0 per cent minus 200 mesh respectively. The pulp was agitated for 48 hours in KCN solution of 2 pounds per ton strength. Pulp dilution was 2·5 : 1 and protective alkalinity was maintained by the addition of lime.

Summary:

Feed, Au, 0·245 oz./ton

Test No.	Tailing assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
1	0·015	93·9	0·40	2·9
2	0·010	95·9	0·40	3·0
3	0·005	98·0	0·50	3·5
4	0·005	98·0	0·50	3·5

AMALGAMATION AND FLOTATION

Test No. 5

In this test the ore was ground to 96 per cent minus 200 mesh in a Denver ball mill and amalgamated with mercury. The amalgamation tailing was then floated with the following reagents:—

	Lb./ton
Na_2CO_3	6·0
Potassium amyl xanthate	0·1
Pine oil	0·2

Summary:

Feed, Au, 0·245 oz./ton

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	8·2	2·10	88·2
Tailing	91·8	0·025	11·8
Amalgamation tailing (cal.)	100·0	0·195	100·0

AMALGAMATION AND CYANIDATION

Test No. 6

In this test the ore was ground to 96 per cent minus 200 mesh in a Denver ball mill, as in Test No. 5, and amalgamated. The amalgamation tailing was agitated for 24 hours in cyanide solution, KCN 2 pounds per ton, at 2·5 : 1 dilution. Protective alkalinity was maintained by the addition of lime.

Summary:

Feed to amalgamation	Au 0·245 oz./ton
Amalgamation tailing	Au 0·195 "
Recovery by amalgamation	20·4 per cent
Cyanidation tailing	Au 0·005 oz./ton
Recovery by cyanidation	77·6 per cent
Total recovery	98·0 "

Reagents Consumed—

	Lb./ton
KCN	1·3
CaO	5·3

The low recovery by amalgamation in Tests Nos. 5 and 6 and the high cyanide and lime consumption in Test No. 6 are, no doubt, due to the extremely fine grinding in the Denver ball mill.

AMALGAMATION

Tests Nos. 7 to 10

Grinding was the same as for cyanidation Tests Nos. 1 to 4. The pulp, at 1 : 1 density, was then amalgamated with mercury for 30 minutes. The amalgamation tailings were assayed for gold.

Summary:

Feed, Au, 0·245 oz./ton

Test No.	Tailing assay, Au, oz./ton	Recovery, per cent
7	0·131	46·5
8	0·118	51·8
9	0·11	55·1
10	0·11	55·1

BLANKET CONCENTRATION AND FLOTATION

Test No. 11

An attempt was made to concentrate the ore by passing it over blankets to catch the coarse gold and by floating the blanket tailing to recover the gold remaining in it. The blanket slope was $3\frac{1}{2}$ inches to the foot.

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Blanket concentrate	3·9	2·68	41·6
Flotation concentrate	6·3	1·76	44·1
Flotation tailing	89·8	0·04	14·3
Feed (cal.)	100·0	0·25	100·0

CONCLUSIONS

The ore is comparatively easy to grind and responds remarkably well to cyanidation. All the test work carried out indicates that this process is the one to be adopted. A maximum recovery of 55 per cent was obtained by amalgamation and this process may hold possibilities for use in conjunction with a cyanide plant. Concentration appeared to be out of the question as it was not possible to make a low tailing.

Ore Dressing and Metallurgical Investigation No. 492

GOLD-SILVER-COPPER-ZINC ORE FROM THE LYNX PROPERTY AT OXFORD LAKE, NORTHERN MANITOBA

Shipment. A shipment of 100 pounds of ore was received January 24, 1933, from Nels Mattson, 266 Furby Street, Winnipeg, Manitoba.

Characteristics of Ore. Under the microscope the gangue is seen to be chiefly glassy quartz. Micaceous minerals, which are in many places arranged in parallel orientation, impart a schistose texture to the ore.

The metallic minerals are pyrite, sphalerite, chalcopyrite, galena, and pyrrhotite. Pyrite, sphalerite, and chalcopyrite are in coarse grains and tend to form granular aggregates up to 5 mm. in diameter. Sphalerite and chalcopyrite are usually intimately associated, the latter forming tiny grains in the sphalerite. Chalcopyrite also occurs as fine grains in the quartz. Galena is present in minor amounts and pyrrhotite is very rare.

Spectrographic analysis of pyrite, chalcopyrite, and sphalerite did not reveal the presence of gold but indicated traces of silver in each of these minerals, especially in the sphalerite.

No free gold was observed in any of the polished sections.

An average analysis of the ore is as follows:—

Gold	0·40 oz./ton;
Silver	1·52 "
Copper	1·69 per cent;
Zinc	4·46 "

Experimental Tests

Three small-scale tests were made on this ore in an attempt to recover the gold and silver. Amalgamation did not result in any appreciable recovery and the high copper content made it impossible to cyanide the ore in its raw state. When the copper was floated off it was found that most of the gold and silver as well as a large portion of the zinc floated with it. The tailing from this operation contained 0·065 ounce per ton in gold, and after cyanidation contained 0·015 ounce per ton in gold, but cyanide consumption was prohibitively high.

AMALGAMATION AND FLOTATION

Test No. 1

In this test 1,000 grammes of the ore at minus 14 mesh was ground for 15 minutes in a ball mill. The pulp was then amalgamated with 100 grammes of mercury for 30 minutes. The amalgam was separated and the amalgamation tailing floated with the following reagents:

	Lb./ton
Soda ash	6·0
Potassium amyl xanthate	0·1
Copper sulphate	1·0
Pine oil	0·15

Summary:

Product	Weight, per cent	Assays				Distribution of metals, per cent			
		Au oz./ton	Ag oz./ton	Cu %	Zn %	Au	Ag	Cu	Zn
Concentrate	28·5	0·66	4·98	5·92	14·90	56·8	90·8	95·9	97·5
Tailing	71·5	0·20	0·20	0·10	0·15	43·2	9·2	4·1	2·5
Amalgamation tailing (cal.)	100·0	0·33	1·56	1·76	4·35	100·0	100·0	100·0	100·0

SELECTIVE FLOTATION

Test No. 2

The object of this test was to separate the various minerals of which the ore is composed and determine by assay which of them carried the gold. The test was a failure as far as separations are concerned and it is impossible to say from this test whether the gold is associated with the copper or the zinc. The ore was ground in a ball mill for 20 minutes.

Charge to Ball Mill:

Ore	1,000 grammes −14 mesh
Soda ash	6·0 lb./ton
Thiocarbanalide	0·10 "
Sodium cyanide	0·20 "

Reagents to Cell:

Copper float	Cresylic acid	0·15 lb./ton
Zinc float	$CuSO_4$	1·0 lb./ton
	Sodium Aerofloat	0·1 "
	Cresylic acid	0·07 "
Iron float	Potassium amyl xanthate	0·10 lb./ton
	Pine oil	0·20 "

Summary:

Product	Weight, per cent	Assays					Distribution of metals, per cent			
		Au	Ag	Cu	Zn	Fe	Au	Ag	Cu	Zn
Cu concentrate	11·4	2·44	15·78	13·18	20·93	......	77·0	82·0	92·2	55·8
Zn concentrate	13·9	0·13	1·38	0·28	12·10	28·18	5·0	8·7	2·4	39·2
Fe concentrate	4·5	0·20	1·84	0·40	0·79	8·29	2·5	3·8	1·1	0·8
Tailing	70·2	0·08	0·17	0·10	0·27	......	15·5	5·5	4·3	4·4
Feed (cal.)	100·0	0·36	2·19	1·63	4·29	......	100·0	100·0	100·0	100·0

FLOTATION AND CYANIDATION

Test No. 3

The object of this test was to float off the copper selectively and cyanide what was left. This gave a concentrate containing 96·5 per cent of the copper, 80·6 per cent of the gold, 75·1 per cent of the silver, and 26·8 per cent of the zinc. The tailing assayed 0·065 ounce per ton in gold which after cyaniding was reduced to 0·015 ounce per ton in gold. Cyanide consumption, however, was excessive. The ore at minus 14 mesh was ground in a ball mill for 20 minutes.

Charge:

Ore	2,000 grammes
Soda ash	6·0 lb./ton
Thiocarbanalide	0·10 "
Sodium cyanide	0·40 "

Reagents to Cell:

Cresylic acid	0·15 lb./ton

The flotation tailing was divided in two parts, one being sent for assay without further treatment and the other after being agitated for 48 hours in cyanide solution, 4 pounds per ton KCN. These two products and the concentrate were assayed for gold, silver, copper, and zinc.

Summary:

Product	Weight, per cent	Assays				Distribution of metals, per cent			
		Au	Ag	Cu	Zn	Au	Ag	Cu	Zn
Flotation concentrate	8·2	3·03	14·50	18·60	15·20	80·6	75·1	96·5	26·8
Flotation tailing	91·8	0·065	0·43	0·06	3·70	19·4	24·9	3·5	73·2
Flotation tailing cyanidation	91·8	0·015	0·13	0·05	3·24	4·5	7·5	2·9	64·1
Feed (cal.)	100·0	0·31	1·58	1·58	4·64	......	......	......	......

Reagents Consumption—

KCN	6·25 lb./ton tailing
CaO	7·00 "

CONCLUSIONS

The ore will be difficult to treat economically, partly on account of its nature and partly on account of its geographical location. It does not amalgamate and cannot be cyanided in its raw state owing to its high copper content; moreover it does not seem possible to float the copper separately from the gold and the zinc, 57 per cent or more of the gold and 25 per cent or more of the zinc always being found in the copper concentrate.

The mode of occurrence of the chalcopyrite and sphalerite, as explained in the section "Characteristics of Ore," is responsible for the difficulty in separating them. The same section explains the occurrence of the silver, although microscopic examination failed to determine the association of gold. Results of the tests seem to indicate that the gold is more closely associated with chalcopyrite than with the sphalerite.

The grade of concentrates produced will not allow them to be shipped to a smelter in view of the great distance they would have to be sent by rail.

Ore Dressing and Metallurgical Investigation No. 493

MILL TAILINGS FROM BUSSIÈRES MINING COMPANY, LTD., SENNETERRE, QUEBEC

Shipment. A shipment of three bags of mill tailing, net weight 365 pounds, was received February 2, 1933, from J. P. Norrie, Senneterre, Quebec. The shipment represented a sample of mill tailing covering a considerable period.

Experimental Tests

A head sample was cut by standard methods and a screen analysis made.

Results:

Mesh	Weight, per cent	Assays	
		Au, oz./ton	Cu, per cent
+ 65	8·00	0·105	0·04
+100	16·09	0·10	0·05
+150	14·26	0·12	0·06
+200	25·32	0·09	0·08
−200	36·31	0·09	0·15

The head sample analysed—

Gold.......0·097 oz./ton
Copper.......0·095 per cent

A series of small-scale tests was made to determine a possible method of gold recovery from the tailing. These tests included tabling, flotation, amalgamation of flotation concentrate, and cyanidation.

TABLE CONCENTRATION

A sample of 5,000 grammes of the tailing was run over a laboratory Wilfley table. The middling product was re-run. Continuous samples were cut from the tailing overflow.

The results of the test are as follows:

Product	Weight, per cent	Assays		Distribution of gold, per cent
		Au, oz./ton	Cu, per cent	
Concentrate	6·98	0·90	0·63	60·01
Tailing	93·02	0·045	0·05	39·99

FLOTATION

Test No. 1

The tailing was conditioned in the cell for three minutes and then floated. The concentrate and tailing were filtered, washed, and assayed for gold and copper.

Charge to Cell:

Tailing	2,000 grammes
Water	1,000 c.c.
Soda ash	3·0 lb./ton
Potassium xanthate	0·2 "
Pine oil	0·05 "

Summary:

Product	Weight, per cent	Assays		Gold recovery, per cent
		Au, oz./ton	Cu, per cent	
Concentrate	4·29	1·82	1·54	89·0
Tailing	95·71	0·01	0·01	

Test No. 2

This test was a duplicate of Test No. 1, with exception that 0·075 pound per ton of pine oil was added to the cell. The concentrate was not weighed and assayed but subjected directly to amalgamation.

Summary:

Product	Weight, per cent	Assays		Gold recovery, per cent
		Au, oz./ton	Cu, per cent	
Concentrate	5·25	1·66	....	90·2
Tailing	94·75	0·01	0·01	

AMALGAMATION

The flotation concentrate from Test No. 2 was amalgamated in a weak cyanide solution with approximately 40 grammes of mercury for one hour. The mercury was panned and the amalgamation tailing filtered, washed, and assayed for gold and copper.

Results:

Product	Weight, per cent	Assays		Gold recovery, per cent
		Au, oz./ton	Cu, per cent	
Tailing	98·4	0·345	2·06	79·2

The total recovery of gold by flotation and barrel amalgamation of the flotation concentrate is 71·4 per cent.

CYANIDATION

Tests Nos. 1 to 3

The three tests were all carried out under the same conditions, no grinding having been done on any of them. The three samples of mill tailing were agitated for 24 hours in cyanide solution, 3 pounds per ton in KCN, at 3 : 1 dilution. The cyanide tailing was assayed for gold.

Summary:

Feed, Au 0·097 oz./ton

Test No.	Tailing assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
1	0·015	84·5	0·3	5·00
2	0·010	89·7	0·3	5·00
3	0·010	89·7	0·3	5·00

CONCLUSIONS

A summary of the gold recoveries by the different methods of testing is as follows—

Method of Test	Gold recovery, per cent
Tabling	60·01
Flotation	90·2
Flotation and amalgamation	71·4
Cyanidation	89·7

In the flotation tests the ratio of concentration is approximately 20 : 1.

Ore Dressing and Metallurgical Investigation No. 494

GOLD ORE FROM KOOTENAY BELLE MINE, SALMO, BRITISH COLUMBIA

Shipment. A shipment of two bags of gold ore was received February 25, 1933. The sample consisted of an oxidized gold quartz ore from the Kootenay Belle Mine in the Sheep Creek district, Salmo, B.C., and was submitted by F. M. Black, Trustee, Kootenay Belle Mine, 701 Rogers Building, Vancouver, B.C.

Characteristics. The ore was a highly oxidized gold-quartz ore in which gold was finely disseminated.

Experimental Tests

The sample was crushed to minus 14 mesh and sampled by standard methods. A series of cyanidation tests on different sizes of ore was made and a flotation and a tabling test were also carried out.

The feed analysed as follows:—

Gold	2·12 oz./ton
Silver	0·59 "
Lead	Nil
Zinc	Trace
Copper	0·01 per cent

CYANIDATION

Test No. 1

Samples, 200 grammes, of the ore crushed to minus 48, 65, and 100 mesh respectively were treated in a dilution of 2·5 : 1, with potassium cyanide, 5 pounds per ton, and lime, 12·5 pounds per ton, for 24 hours.

Product	Assay tailing Au, oz./ton	Recovery, per cent	Consumption, lb./ton	
		Au	KCN	CaO
− 48	0·43	79·7	0·125	8·725
− 65	0·33	84·4	0·25	8·875
−100	0·05	97·6	0·25	9·25

Test No. 2

This test was a duplicate of Test No. 1 with exception that treatment was continued for 48 hours.

Product	Assay tailing, Au, oz./ton	Recovery, per cent	Consumption, lb./ton	
		Au	KCN	CaO
− 48	0·135	93·6	0·75	8·81
− 65	0·055	97·4	1·00	9·19
−100	0·04	98·1	1·50	9·56

Screen tests were made on each of the three products.

−48 mesh		−65 mesh	
Mesh	Weight, per cent	Mesh	Weight, per cent
+ 48	0·2	+100	0·8
+ 65	11·5	+150	11·8
+100	19·1	+200	16·8
+150	25·8	−200	70·5
+200	10·8		
−200	32·6		
Total	100·0	Total	100·0

−100 mesh

Mesh	Weight, per cent
+150	2·2
+200	12·9
−200	84·9
Total	100·0

Summary. The results of cyanidation show that minus 100-mesh ore of which 84·9 per cent is minus 200 mesh gives a gold recovery of 97·6 per cent, a 0·05-ounce tailing, a potassium cyanide and lime consumption of 0·25 and 9·25 pounds per ton respectively with 24 hours' treatment. With 48 hours' treatment the gold recovery is 98·1 per cent with a 0·04-ounce tailing and a cyanide and lime consumption of 1·50 and 9·56 pounds per ton respectively.

FLOTATION AND TABLING

A lot of 2,000 grammes of minus 14-mesh ore was ground in a ball mill for 15 minutes with 1,000 c.c. water and floated. The flotation tailing was run over a Wilfley table.

Reagents to Ball Mill:

Soda ash	5·0 lb./ton
Potassium xanthate	0·2 "
Barrett No. 4	0·06 "

Reagents to Flotation Cell:

Pine oil .... 0·05 lb./ton

Results

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	5·13	27·28	79·2
Tailing	94·87	0·385	20·8

Table:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	10·25	1·66	65·4
Tailing	89·75	0·10	34·6

Gold recovery by flotation and tabling=20·8 x 0·654=13·6 per cent.
Overall recovery=79·2+13·6=92·8 per cent.

CONCLUSIONS

The results of the test work would indicate that the ore is adaptable to cyanidation, giving a high recovery and a 0·04-ounce tailing. Flotation and tabling tests show a lower recovery and a 0·10-ounce tailing.

Ore Dressing and Metallurgical Investigation No. 495

GOLD ORE FROM THE SULLIVAN CONSOLIDATED MINES, LIMITED, DUBUISSON TOWNSHIP, ABITIBI COUNTY, QUEBEC

Shipment. A shipment of 2,420 pounds of gold ore was received February 8, 1933. The shipment consisted of one lot of mine ore and was submitted by A. K. Muir, Superintendent, Sullivan Consolidated Mines, Limited, Siscoe P.O., Quebec.

Experimental Tests

The following tests were carried out on the 100-pound per hour, small-scale amalgamation and flotation unit. These consisted of amalgamation followed by flotation, and blanket concentration followed by flotation.

It was found that 50 per cent of the gold was recovered on the plates, 22 per cent of the gold was held in the rod mill. The possible recovery of gold by amalgamation is 72 per cent.

The plate tailing was treated by flotation and 82 per cent of the gold content recovered. The flotation concentrate was cyanided and a recovery of 97 per cent of the contained gold obtained. The overall recovery was 94 per cent. The ratio of concentration was 123 : 1.

The blanket concentration test showed that 26 per cent of the gold was recovered on the blankets and that 26 per cent of the gold was held in the rod mill. The possible recovery of gold in the mill plus recovery on blankets is 52 per cent. The blanket tailing was treated by flotation and 87 per cent of the gold was recovered in the concentrate. This concentrate was cyanided and 97 per cent of the contained gold recovered, giving a total overall recovery of 93 per cent. The ratio of concentration was 125 : 1.

AMALGAMATION AND FLOTATION

Mill Run No. 1

Assay of Products:

Product	Assay, oz./ton
Feed	0·90
Rod mill discharge	0·70
Plate tailing	0·25
Flotation concentrate	25·36
Flotation tailing	0·045

Amalgamation:

Feed assay	0·90 oz./ton
Rod mill discharge	0·70 "
Plate tailing	0·25 "

Recovery by amalgamation $\frac{0{\cdot}45}{0{\cdot}90} \times 100 = 50{\cdot}00$ per cent.

Gold held in the rod mill $\frac{0{\cdot}20}{0{\cdot}90} \times 100 = 22{\cdot}22$ per cent.

Possible recovery of gold in rod mill plus recovery on plates=72·22 per cent.

Reagents for Flotation:

Soda ash	1·9 lb./ton
Sodium ethyl xanthate	0·11 "
Barrett No. 4	0·13 "
Pine oil	0·07 "

Gold in flotation feed (plate tailing) 100−72·22=27·78 per cent.
Grind 70 per cent minus 200 mesh.
Ratio of concentration of flotation concentrate 123 : 1.
Recovery by flotation 82·14 per cent of flotation feed.
Recovery by flotation 82·14 x 27·78 = 22·82 per cent of feed.

Cyanidation of Concentrates. The flotation concentrates were mixed, sampled, and cyanided.

Assay of Products:

Feed	25·51 oz./ton
Tailing	0·73 "
Recovery by cyanidation	97·14 per cent of gold in concentrate.

Consumption of Reagents:

KCN	24·0 lb./ton concentrate
CaO	27·0 " "
PbO	1·0 " "

Recovery of gold in concentrate by cyanidation in terms of the feed 97·14 x 22·82=22·17%

Summary:

Possible gold recovery as bullion by amalgamation	72·2 per cent
Possible gold recovery by cyanidation of flotation concentrate	22·2 "
Overall recovery	94·4 "

Mill Run No. 2

Assay of Products:

Product	Assay, oz./ton
Feed	0·84
Rod mill discharge	0·835
Plate tailing	0·70
Flotation concentrate	42·96
Flotation tailing	0·05

Amalgamation:

Feed	0·84	oz./ton
Rod mill discharge	0·835	"
Plate tailing	0·70	"

Recovery by amalgamation $\frac{0\cdot135}{0\cdot84} \times 100 = 16\cdot07$ per cent

Gold held in rod mill $\frac{0\cdot005}{0\cdot84} \times 100 = 0\cdot59$ per cent

Possible recovery of gold in rod mill plus recovery on plates=16·66 per cent

Flotation:

Reagents for flotation—	Lb./ton
Soda ash	1·84
Sodium ethyl xanthate	0·09
Barrett No. 4	0·41
Cresylic acid	0·20

Gold in flotation feed (plate tailing) 100−16·66 = 83·34
Grind 65 per cent minus 200 mesh.
Ratio of concentration of flotation concentrate 66·23 : 1.
Recovery by flotation 92·7 per cent of flotation feed.
Recovery by flotation 92·7 x 83·34=77·26 per cent of feed.
Recovery by cyanidation 97·14 per cent of gold in flotation concentrate.
Recovery by cyanidation 97·14 x 77·26=75·05 per cent of feed.

Summary:

Possible gold recovery as bullion by amalgamation	16·66	per cent
Possible gold recovery as bullion by cyanidation of the flotation concentrate	75·05	"
Overall recovery	91·71	"

Note.—Although this test does not show the same recovery by amalgamation as does Mill Run No. 1, the greater part of the gold escaping the amalgamation is recovered by flotation.

BLANKET CONCENTRATION AND FLOTATION

Mill Run No. 3

Assay of Products:

	Assay, oz./ton
Feed	0·80
Rod mill discharge	0·59
Blanket concentrate	141·98
Blanket tailing	0·38
Flotation concentrate	41·46
Flotation tailing	0·05

Blanket Concentration:

Feed assay	0·80	oz./ton
Rod mill discharge	0·59	"
Blanket tailing	0·38	"

Gold held in mill $\frac{0\cdot21}{0\cdot80} \times 100 = 26\cdot25$ per cent

Recovery of gold by blanket concentration $\frac{0\cdot21}{0\cdot80} \times 100 = 26\cdot25$ per cent

Possible recovery of gold in mill plus recovery on blankets = 52·50 per cent

Flotation:

Reagents for Flotation:

Soda ash	2·11 lb./ton
Sodium ethyl xanthate	0·11 "
Cresylic acid	0·47 "
Barrett No. 4	0·23 "

Gold in flotation feed (blanket tailing) 100—52·5=47·50
Grind 65 per cent minus 200 mesh.
Ratio of concentration of flotation concentrate 125 : 1.
Recovery by flotation 87·29 per cent of the gold in flotation feed.
Recovery by flotation 87·29 x 47·50=41·46 per cent of feed.
Recovery by cyanidation 97·14 per cent of gold in flotation concentrate.
Recovery by cyanidation 97·14 x 41·46=40·27 per cent of feed.

Summary:

Possible gold recovery as bullion by blanket concentration	52·50 per cent
Possible gold recovery as bullion by cyanidation of the flotation concentrate	40·27 "
Overall recovery	92·77 "

Ore Dressing and Metallurgical Investigation No. 496

LOW-GRADE GOLD ORE FROM THE YOUNG-DAVIDSON PROPERTY, MATACHEWAN, ONTARIO

Shipment. A carload of ore was received from N. A. Timmins Incorporated, containing 38,458 pounds. The shipment was made up from samples taken from the dump of the Young-Davidson property. The dump consists of ore obtained from the development work carried on by the Porcupine Gold Fields Development and Finance Corporation and had lain exposed to the weather for about 9 years.

Sampling and Analysis. The shipment was crushed to $\frac{3}{8}$ inch and one-tenth cut out as a head sample by a Vezin automatic sampler. This sample was further reduced by alternately crushing and splitting in a Jones sampler to obtain duplicate samples for assay.

Analysis:

Gold	0·11	oz./ton
Silver	0·058	"
Iron	2·53	per cent
Sulphur	1·60	"
Copper	0·01	"

Characteristics of the Ore. Samples of the ore were selected for microscopic examination. The gangue was found to be chiefly feldspathic material with, locally, considerable white quartz. Small amounts of carbonate occur in fine fractures and in irregular grains.

Pyrite was the only important metallic mineral observed and is disseminated throughout the gangue. Native gold is present in the ore both as small grains and also as a coating on the surface of the pyrite. The pyrite itself also contains gold enclosed within the crystals.

The grain size of the pyrite was determined and is approximately as follows:—

Mesh	Per cent pyrite	Cumulative per cent
+ 20	7·4	7·4
+ 28	15·6	23·0
+ 35	8·9	31·9
+ 48	21·0	52·9
+ 65	13·4	66·3
+100	13·8	80·1
+150	7·7	87·8
+200	4·9	92·7
−200	7·3	100·0

It will be observed from this table that the pyrite is relatively coarse, and, therefore, comparatively coarse grinding should liberate it within an economic range.

Purpose of Experimental Tests. The purpose of these experimental tests was to determine the most economic method of recovering the gold from the ore. The fact that the deposit is large and of low grade was kept constantly in mind in all the experimental work. The possibility of sorting the ore was not studied because this can only be intelligently done at the property and requires a long careful investigation extending over a period of months on ore from development faces and test stopes.

Experimental Tests

In 1925 a carload of ore from this property was received from the Porcupine Gold Fields Development and Finance Corporation. This shipment assayed 0·14 ounce gold per ton. A series of tests was made at that time using a stamp mill with inside and outside amalgamation to recover the gold. The tailing after amalgamation was concentrated on Wilfley tables and the table concentrate cyanided. The crushing was through a 40-mesh screen on the stamps. The recovery by amalgamation was 61 per cent and the tables saved an additional 20·5 per cent, making a total of 81·5 per cent. By regrinding the table concentrate to minus 200 mesh 89 per cent of the gold contained in this product could be extracted, but at 150 mesh only 82 per cent was obtained.

The results obtained on the previous shipment, together with the study made on the grain sizes and of the quantity of pyrite present in the recent shipment, showed the possibility of combining a method of either amalgamating, trapping, or catching the free gold on blankets with concentration by flotation at a relatively coarse size, and obtaining a high ratio of concentration, in the neighbourhood of 20 : 1. As such a procedure, if it could be successfully accomplished, promised the lowest possible milling costs the experimental work was commenced with this idea in view.

The first preliminary tests were carried out in a continuous unit at a feed rate of about 100 pounds of ore per hour. The unit consisted of an ore feeder, a small rod mill, amalgamation plates or blanket tables, and a 6-cell flotation machine of the mechanical type.

Three series of tests were made—

A. Amalgamating the rod mill discharge on plates and concentrating the plate tailing by flotation. Represented by Tests Nos. 1, 2A, 2B, and 3.

B. Grinding as in A but using corduroy blankets to replace the plates and concentrating the blanket tailing by flotation. Represented by Tests Nos. 4 and 5.

C. A series of cyanide tests on the flotation concentrates represented by Tests Nos. 1 to 8 inclusive.

Test No. 1

Feed rate, 100 lb. per hour.

Feed to rod mill	0·110 oz./ton gold
Amalgamation (plate) tailing	0·06 "
Recovery of gold by amalgamation	45·2 per cent
Gold remaining in tailing	54·8 "
Feed to flotation	0·06 oz./ton gold
Concentrate	0·73 "
Tailing	0·005 "
Recovery by flotation	92·4 per cent
Percentage of gold recovered by flotation	54·8 x 92·4=50·7 per cent
Total recovery of gold by amalgamation and in flotation concentrate	95·9 "

Ratio of concentration, 13·2 : 1.

Analysis of Flotation Concentrate:

Gold	0·73 oz./ton
Iron	20·35 per cent
Sulphur	20·90 "
Copper	0·03 "
Insoluble	45·8 "

Reagents used for Flotation: All reagents were added in a conditioning tank after amalgamation.

Soda ash	4·5 lb./ton
Sod. xanthate	0·13 "
Coal-tar creosote	0·05 "
Pine oil	0·05 "

Screen Analysis of Products

Rod Mill Discharge:

Mesh	Weight, per cent	Gold, oz./ton
+ 35	0·8	
+ 48		
+ 65	3·8	0·075
+100	14·7	
+200	29·4	0·085
−200	51·2	0·12
Total	99·9	

Amalgamation Tailing:

Mesh	Weight, per cent	Gold, oz./ton
+ 35	0·1	
+ 48	1·12	0·04
+ 65	4·20	
+100	11·9	
+200	30·1	0·08
−200	52·6	0·06
Total	99·9	

Flotation Tailing:

Mesh	Weight, per cent	Gold, oz./ton
+ 48	0·56	0·005
+ 65	3·40	
+100	12·6	
+200	30·2	0·003
−200	53·24	0·005
Total	100·00	

Flotation Concentrate:

Mesh	Weight, per cent
+ 65	2·7
+100	6·4
+150	9·3
+200	7·0
−200	74·5
Total	99·9

Test No. 2A

The same flow-sheet as in Test No. 1 was used.

Feed rate to mill, 100 lb. per hour.

Feed to rod mill	0·12 oz./ton gold
Amalgamation (plate) tailing	0·06 "
Recovery by amalgamation	50·0 per cent
Gold remaining in tailing	50·0 "
Feed to flotation	0·06 oz./ton gold
Concentrate	0·75 "
Tailing	0·005 "
Recovery by flotation	92·4 per cent
Percentage of gold recovered by flotation	46·2 "
Total recovery of gold by amalgamation and in flotation concentrate	96·2 "

Ratio of concentration, 13·5 : 1.

Screen Test of Feed to Amalgamation

Mesh	Weight, per cent
+ 35	0·08
+ 48	0·48
+ 65	3·20
+100	10·88
+150	19·46
+200	13·42
−200	52·48
Total	100·00

Test No. 2B

This test is similar to Tests Nos. 1 and 2A, with the exception that the flotation reagents were changed.

The reagents used were as follows:

Soda ash	2·5 lb./ton
Minerec B	0·03 "
Pine oil	0·08 "

Feed rate, 100 lb. per hour.

Feed to amalgamation	0·12 oz./ton gold
Amalgamation (plate) tailing	0·08 "
Recovery by amalgamation	33·3 per cent
Feed to flotation	0·08 oz./ton gold
Flotation concentrate	1·50 "
Flotation tailing	0·005 "
Recovery by flotation	94·1 per cent
Percentage of total gold recovered by flotation	62·8 "
Total recovery of gold by amalgamation and in flotation concentrate	96·1 "

Ratio of concentration, 19·9 : 1.

Screen Test of Feed to Amalgamation

Mesh	Weight, per cent
+ 35	0·10
+ 48	0·42
+ 65	2·60
+100	12·46
+150	18·22
+200	13·72
−200	52·48
Total	100·00

Test No. 3

This test shows the effect of finer grinding.

The reagents used were as follows:

Soda ash	2·5 lb./ton
Minerec B	0·03 "
Pine oil	0·08 "

Feed rate, 100 lb. per hour.

Feed to amalgamation	0·115 oz./ton gold
Amalgamation tailing	0·06 "
Recovery by amalgamation	47·8 per cent
Gold remaining in tailing	52·2 "
Feed to flotation	0·06 oz./ton gold
Flotation concentrate	0·72 "
Flotation tailing	0·006 "
Recovery by flotation	92·4 per cent
Percentage of total gold recovered by flotation	48·2 "
Total percentage recovered by amalgamation and in flotation concentrate	96·0 "

Ratio of concentration, 13·0 : 1.

Screen Test of Feed to Amalgamation

Mesh	Weight, per cent
+ 48	0·2
+ 65	1·4
+100	8·6
+150	15·9
+200	13·6
−200	60·2
Total	99·9

Tests Nos. 4 and 5

These two tests were made for the purpose of comparing the results obtained by the use of corduroy blankets with those obtained by plate amalgamation.

The results of Test No. 5 only will be given. In this test the grinding corresponds to the grinding used in Tests Nos. 1 and 2.

The flotation reagents were as follows:

Soda ash	2·5 lb./ton
Sodium xanthate	0·2 "
Pine oil	0·08 "

Feed rate, 100 lb. per hour.

Feed to blankets	0·12 oz./ton gold
Blanket concentrate	4·32 "
Recovery on blankets	42·4 per cent
Ratio of concentration	85 : 1.
Feed to flotation	0·07 oz./ton gold
Flotation concentrate	1·50 "
Flotation tailing	0·006 "
Recovery in flotation concentrate	93·2 per cent
Ratio of concentration	23·0 : 1

Percentage of total gold recovered in flotation concentrate..........53·7 per cent
Total percentage recovered in blanket and flotation concentrate......96·1 "

Combined ratio of concentration, 18·3 : 1.

It will be noted that two different grinds were used, one approximately 50 per cent minus 200 mesh and the other about 60 per cent minus 200 mesh. The results of these tests are briefly summarized in the following table which gives a comparison between plates and blankets and the results at the different grinds.

Test No.	Grinding	Recovery by amalgamation	Recovery by blankets	Recovery by flotation	Total recovery	Ratio of concentration
1	−48 mesh; 51·2%−200 mesh	45·2	....	50·7	95·9	13·2 : 1
2A	− 48 mesh; 52·5%−200 mesh	50·0	....	46·2	96·2	13·5 : 1
2B	3·1%+ 65 mesh; 52·5%−200 mesh	33·3	....	62·8	96·1	19·9 : 1
3	1·6%+ 65 mesh; 60·2%−200 mesh	47·8	....	48·2	96·0	13·0 : 1
5	3·1%+ 65 mesh; 54%−200 mesh	....	42·4	53·7	96·1	18·3 : 1

Screen Test on Feed to Blankets

Mesh	Weight, per cent
+ 35	0·02
+ 48	0·50
+ 65	2·80
+100	10·68
+150	17·86
+200	14·14
−200	54·00
Total	100·00

CYANIDATION OF FLOTATION CONCENTRATES

A series of cyanide tests was made on the flotation concentrates from Test No. 1 and Test No. 2B. These tests were made on unground concentrate and on the concentrate ground to various degrees of fineness.

Test No. 1

This test is on unground concentrate produced from flotation Test No. 1.

Assay of concentrate used	0·88 oz./ton gold
Assay of cyanide tailing	0·41 "
Recovery by cyanidation	53·4 per cent
Time of treatment	48 hours
KCN consumption	1·9 lb./ton
CaO consumption	9·5 "
Dilution	2·5 : 1

Screen Analysis of Cyanide Tailing

Mesh	Weight, per cent	Assay, gold, oz./ton
+200	27·5	1·095
−200	72·5	0·15
Total	100·0	0·41

Test No. 2

This test is on concentrate from flotation Test No. 1 which was reground in water, filtered, and then cyanided.

Assay of concentrate used	0·88 oz./ton gold
Assay of cyanide tailing	0·183 "
Recovery by cyanidation	79·2 per cent
Time of treatment	48 hours
KCN consumption	3·1 lb./ton
CaO consumption	9·9 "
Dilution	2·5 : 1

Screen Analysis of Cyanide Tailing

Mesh	Weight, per cent	Assay, gold, oz./ton
+200	6·4	1·32
−200	93·6	0·105
Total	100·0	0·183

Test No. 4

This test is on concentrate from flotation Test No. 1 which was reground in water, filtered, and then cyanided.

Assay of concentrate used	0·88 oz./ton gold
Assay of cyanide tailing	0·137 "
Recovery by cyanidation	84·4 per cent
Time of treatment	48 hours
KCN consumption	3·35 lb./ton
CaO consumption	11·7 "
Dilution	2·44 : 1

Screen Analysis of Cyanide Tailing

Mesh	Weight, per cent	Assay, gold, oz./ton
+200	3·7	1·35
−200	96·3	0·09
Total	100·0	0·137

Test No. 5

This test is on flotation concentrate from Test No. 2B.

Assay of concentrate used	1·00 oz./ton gold
Assay of cyanide tailing	0·17 "
Recovery by cyanidation	82·7 per cent
Time of treatment	48 hours
KCN consumption	2·4 lb./ton
CaO consumption	10·0 "
Dilution	2·64 : 1

Screen Analysis of Cyanide Tailing

Mesh	Weight, per cent	Assay, gold, oz./ton
+200	28·6	0·33
−200	71·4	0·11
Total	100·0	0·173

Test No. 6

This test is on flotation concentrate from Test No. 2B, which was reground in water, filtered, and cyanided at a dilution of 2·8 : 1.

Assay of concentrate used	1·00 oz./ton gold
Assay of cyanide tailing	0·12 "
Recovery by cyanidation	88 per cent
KCN consumption	3·1 lb./ton
CaO consumption	11·9 "

Screen Analysis of Cyanide Tailing

Mesh	Weight, per cent	Assay, gold, oz./ton
+200	8·7	0·23
−200	91·3	0·11
Total	100·0	0·12

Test No. 7

This test is on flotation concentrate from Test No. 2B, which was reground in water, filtered, and cyanided in a 2·51 : 1 pulp.

Assay of concentrate used	1·00 oz./ton gold
Assay of cyanide tailing	0·119 "
Recovery by cyanidation	88·1 per cent
KCN consumption	3·4 lb./ton
CaO consumption	12·9 "
Time of treatment	48 hours

Screen Analysis of Cyanide Tailing

Mesh	Weight, per cent	Assay, gold, oz./ton
+200	5·1	0·29
−200	94·9	0·11
Total	100·0	0·119

Test No. 8

This test is on flotation concentrate from Test No. 2B, which was re-ground in water, filtered, and cyanided in a 2·48 : 1 pulp.

Assay of concentrate used	1·00 oz./ton gold
Assay of cyanide tailing	0·109 "
Recovery by cyanidation	89·1 per cent
KCN consumption	3·5 lb./ton
CaO consumption	14·8 "
Time of treatment	48 hours

Screen Analysis of Cyanide Tailing

Mesh	Weight, per cent	Assay, gold, oz./ton
+200	3·1	0·395
−200	96·9	0·10
Total	100·0	0·109

Remarks

The results of the mill run tests using amalgamation and flotation or blankets and flotation are excellent, and show a recovery of about 50 per cent of the gold by amalgamation and the balance to make a total of 93 per cent recovery obtained in a flotation concentrate. This concentrate can be made to average better than 1·5 ounces per ton without increasing the tailing loss. The ratio of concentration is approximately 20 : 1.

The results of the cyanide tests made on the flotation concentrate show that it must be reground and that the finer it is ground the higher the extraction. The highest extraction shown was 89 per cent. If this figure is applied to the results of mill run Test No. 2B, the following results are obtained:—

Gold recovery as bullion by amalgamation	33·3 per cent
Gold recovery in flotation concentrate	62·8 "
Gold recovery as bullion by cyanidation of the above concentrate 62·8 x 89	55·9 "
Total gold recoverable as bullion	89·2 "

LARGE-SCALE CONTINUOUS TESTS

It was decided to follow the series of tests just described by larger scale continuous tests. These tests although reported as individual "runs" can be compared to the actual operation of a small mill for a period of 14 days because no clean-ups were made between runs. The ball mill, classifiers, conditioning tanks, and flotation cells were shut down and started up again without draining out any of their accumulated charges. This was purposely done in order to settle definitely the question in regard to the possibility of free gold building up in the circuit and then passing through the flotation cells and reporting in the tailing. The results of these runs show that this did not occur. Keeping in mind the necessity of low milling costs on this ore, these tests were made with the additional object in view, that of determining how coarse a feed could be sent to flotation without causing too great a sacrifice of recovery.

A brief outline of the flow-sheets used now follows: The ore crushed to $\frac{3}{8}$ inch was fed to a small ball-rod mill. In the first of these tests rods were used in this mill and in all the later tests balls were used. The size of the mill was 24 by 48 inches and a ball load of 750 pounds was used. The ball mill was operated in closed circuit with a Dorr classifier and when plates or blankets were used they were placed between the ball mill and classifier so that they received the discharge of the ball mill and consequently the circulating load. When a gold trap alone was used this was placed immediately under the ball mill discharge lip so that it also received the circulating load. In the last series of runs a Hummer screen replaced the classifier. The overflow of the classifier or the undersize of the Hummer screen, as the case might be, was conditioned in a Denver conditioning tank for 10 minutes with the flotation reagents before flotation. The overflow of the conditioner then was floated in a 9-cell unit of a mechanically agitated flotation machine; the first three cells making final concentrate, the six successive cells making a rougher concentrate which was returned with the feed to the head of the machine.

Summary of Flow-sheets Used in Each Test

Runs Nos. 1 to 3. Rod mill discharge to amalgamation plates, plate tailing to classifier in closed circuit with rod mill. Classifier overflow to conditioning tank where soda ash, xanthate, Barrett No. 4, and part of the pine oil were added. Conditioning tank discharge to flotation cells. Additional pine oil added to cell No. 6.

Runs Nos. 4 and 5. Ball mill discharge to blanket table covered with special corduroy. Tailing from blankets to classifier. The rest of the flow-sheet as in preceding runs. The pine oil used in Run No. 1 was changed to cresylic acid in order to control more closely the frothing condition in the cells.

Run No. 6. Ball mill discharge to amalgamation plates. The balance of flow-sheet was similar to preceding runs.

Runs Nos. 7 to 10. Ball mill discharge to gold trap similar in design to one used at Siscoe mill. Trap overflow to Dorr classifier in closed circuit with ball mill. The balance of the flow-sheet was similar to the preceding runs.

Run No. 11. Ball mill discharge to gold trap, overflow to 35-mesh Hummer screen, oversize of Hummer returned to ball mill, undersize to flotation conditioning tank as in the other runs.

Runs Nos. 12 and 13. Similar to Run No. 11 with the exception that 28-mesh screen was used in the Hummer.

Run No. 14. Similar to Runs Nos. 12 and 13 with the exception that pine oil was used as the floating reagent in place of cresylic acid and that $\frac{1}{4}$ pound copper sulphate per ton of ore was added to cell No. 6 in order to lift any tarnished pyrite.

The results of these fourteen runs are now given in detail together with screen tests showing the size to which the ore was crushed for flotation.

Run No. 1

Feed rate, 276 lb. per hour.

Feed to rod mill	0·110 oz./ton gold
Amalgamation (plate) tailing	0·045 "
Recovery of gold by amalgamation	59·1 per cent
Gold remaining in tailing	40·9 "
Feed to flotation	0·045 oz./ton gold
Concentrate	0·83 "
Tailing	0·005 "
Recovery by flotation	89·8 per cent
Percentage of gold recovered by flotation, 40·9 x 89·8	36·7 "
Total recovery of gold by amalgamation and in flotation concentrate	95·8 "

Ratio of concentration, 20·6 : 1

Reagents used for Flotation:

Soda ash	1·2 lb./ton
Sodium xanthate	0·21 "
Pine oil	0·015 "
Cresylic acid	0·25 "

Screen Test of Flotation Feed:

Mesh	Weight, per cent
+ 48	9·75
+ 65	14·90
+100	15·81
+150	12·61
+200	10·26
−200	36·67
Total	100·00

Run No. 2

Feed rate, 276 lb. per hour.

Feed to rod mill	0·110 oz./ton gold
Amalgamation (plate) tailing	0·045 "
Recovery of gold by amalgamation	59·1 per cent
Gold remaining in tailing	40·9 "
Feed to flotation	0·045 oz./ton gold
Concentrate	1·06 "
Tailing (salted)	0·05 "
Recovery by flotation	?

Run No. 3—Amalgamation

Feed rate, 441 lb. per hour.

Feed to rod mill	0·110 oz./ton gold
Amalgamation (plate) tailing	0·05 "
Recovery of gold by amalgamation	54·5 per cent
Gold remaining in tailing	45·5 "
Feed to flotation	0·05 oz./ton gold
Concentrate	1·18 "
Tailing	0·0075 "
Recovery by flotation	85·6 per cent
Percentage of gold recovered by flotation, 45·5 x 85·6=	39·0 "
Total recovery of gold by amalgamation and in flotation concentrate	93·5 "

Ratio of concentration, 27·6 : 1.

Reagents used for Flotation:

Soda ash	1·77 lb./ton
Sodium xanthate	0·123 "
Barrett No. 4	0·096 "
Cresylic acid	0·27 "

Screen Test of Classifier Overflow:

Mesh	Weight, per cent
+ 48	11·0
+ 65	11·7
+100	12·9
+150	11·8
+200	11·6
−200	41·0
Total	100·0

Run No. 4—Blanket Concentration

Feed rate, 475 lb. per hour.

Feed to ball mill	0·110	oz./ton gold
Blanket concentrate	4·67	"
Blanket tailing	0·05	"
Recovery of gold on blankets	54·5	per cent
Gold remaining in tailing	45·5	"
Feed to flotation	0·05	oz./ton gold
Concentrate	1·29	"
Tailing	0·0062	"
Recovery by flotation	88·0	per cent
Percentage of gold recovered by flotation, 45·5 x 88·0=	40·04	"
Total recovery of gold by blankets and flotation concentrate	94·54	"

Ratio of concentration, 29·3 : 1.

Reagents used for Flotation:

Soda ash	2·67	lb./ton
Sodium xanthate	0·134	"
Barrett No. 4	0·10	"
Cresylic acid	0·27	"

Screen Test of Classifier Overflow

Mesh	Weight, per cent
+ 48	12·9
+ 65	12·4
+100	14·2
+150	12·4
+200	10·3
−200	37·8
Total	100·00

Run No. 5—Blanket Concentration

Feed rate, 400 lb. per hour.

Feed to ball mill	0·110	oz./ton gold
Blanket concentrate	6·74	"
Blanket tailing	0·055	"
Recovery of gold on blankets	50·0	per cent
Gold remaining in tailing	50·0	"
Feed to flotation	0·055	oz./ton gold
Concentrate	1·42	"
Tailing	0·0062	"
Recovery by flotation	90·2	per cent
Percentage of gold recovered by flotation, 50·0 x 90·2=	45·1	"
Total recovery of gold by blankets and in flotation concentrate	95·1	"

Ratio of concentration, 29·2 : 1.

Screen Test of Classifier Overflow

Mesh	Weight, per cent
+ 48	5·2
+ 65	14·8
+100	13·2
+150	14·1
+200	13·2
−200	39·5
Total	100·0

Reagents used for Flotation:

Soda ash	2·91 lb./ton
Sodium xanthate	0·15 "
Barrett No. 4	0·10 "
Cresylic acid	0·27 "

Run No. 6—Amalgamation

Feed rate, 367 lb. per hour.

Feed to ball mill	0·110 oz./ton gold
Amalgamation (plate) tailing	0·04 "
Recovery of gold by amalgamation	63·6 per cent
Gold remaining in tailing	36·4 "
Feed to flotation	0·04 oz./ton gold
Concentrate	1·07 "
Tailing	0·0062 "
Recovery by flotation	84·9 per cent
Percentage of gold recovered by flotation, 36·4 x 84·9=	30·9 "
Total recovery of gold by amalgamation and in flotation concentrate	94·5 "

Ratio of concentration, 31·5 : 1.

Reagents used for Flotation:

Soda ash	2·9 lb./ton
Sodium xanthate	0·15 "
Barrett No. 4	0·10 "
Cresylic acid	0·27 "

Screen Test of Classifier Overflow:

Mesh	Weight, per cent
+ 48	2·7
+ 65	7·1
+100	11·4
+150	15·4
+200	13·9
−200	49·5
Total	100·0

Run No. 7—Trap at Ball Mill Discharge

Feed rate, 393 lb. per hour.

Feed to ball mill	0·110 oz./ton gold
Trap clean-up	0·49 "
Classifier overflow	0·04 "
Recovery of gold in trap	63·6 per cent
Gold remaining in tailing	36·4 "
Feed to flotation	0·04 oz./ton gold
Concentrate	1·39 "
Tailing	0·0062 "
Recovery by flotation	84·8 per cent
Percentage of gold recovered by flotation, 36·4 x 84·8=	30·9 "
Total recovery of gold in trap and in flotation concentrate	94·5 "

Ratio of concentration, 29·5 : 1.

Reagents used for Flotation:

Soda ash	3·0 lb./ton
Sodium xanthate	0·15 "
Barrett No. 4	0·15 "
Cresylic acid	0·27 "

Screen Tests of Classifier Overflow:

Mesh	Weight, per cent
+ 48	4·6
+ 65	13·2
+100	11·6
+150	15·3
+200	13·7
−200	41·6
Total	100·0

Run No. 8—Trap at Ball Mill Discharge

Feed to ball mill	0·110 oz./ton gold
Trap clean-up	0·39 "
Classifier overflow	0·04 "
Recovery of gold in trap	63·6 per cent
Gold remaining in tailing	36·4 "
Feed to flotation	0·04 oz./ton gold
Concentrate	1·10 "
Tailing	0·0075 "
Recovery by flotation	81·8 per cent
Percentage of gold recovered by flotation, 36·4 x 81·8=	29·8 "
Total recovery of gold in trap and in flotation concentrate	93·4 "

Ratio of concentration, 33·6 : 1.

Reagents Used for Flotation:

Soda ash	3·0 lb./ton
Sodium xanthate	0·15 "
Barrett No. 4	0·15 "
Cresylic acid	0·27 "

Screen Test of Classifier Overflow:

Mesh	Weight, per cent
+ 48	1·5
+ 65	6·0
+100	13·6
+150	14·8
+200	13·7
−200	50·4
Total	100·0

Run No. 9—Trap at Ball Mill Discharge

Feed rate, 301 lb. per hour.

Feed to ball mill	0·110 oz./ton gold
Trap clean-up	1·0 "
Classifier overflow	0·05 "
Recovery of gold in trap	54·5 per cent
Gold remaining in tailing	45·5 "
Feed to flotation	0·05 oz./ton gold
Concentrate	1·32 "
Tailing	0·0062 "
Recovery by flotation	88·0 per cent
Percentage of gold recovered by flotation, 45·5 x 88·0=	40·1 "
Total recovery of gold in trap and in flotation concentrate	94·6 "

Ratio of concentration, 30 : 1.

Reagents Used for Flotation:

Soda ash	4·0 lb./ton
Sodium xanthate	0·22 "
Barrett No. 4	0·14 "
Cresylic acid	0·30 "

Screen Test of Classifier Overflow:

Mesh	Weight, per cent
+ 48	5·3
+ 65	10·0
+100	14·6
+150	12·7
+200	12·0
−200	45·4
Total	100·0

Run No. 10—Trap at Ball Mill Discharge

Feed rate, 386 lb. per hour:

Feed to ball mill	0·110 oz./ton gold
Trap clean-up	1·50 "
Classifier overflow	0·055 "
Recovery of gold in trap	50·0 per cent
Gold remaining in tailing	50·0 "
Feed to flotation	0·055 oz./ton gold
Concentrate	1·96 "
Tailing	0·0062 "
Recovery by flotation	88·9 per cent
Percentage of gold recovered by flotation, 50 x 88·9 =	44·5 "
Total recovery of gold in trap and in flotation concentrate	94·5 "
Ratio of concentration, 40 : 1.	

Reagents Used for Flotation:

Soda ash	4·0 lb./ton
Sodium xanthate	0·2 "
Barrett No. 4	0·12 "
Cresylic acid	0·27 "

Screen Test of Classifier Overflow:

Mesh	Weight, per cent
+ 48	5·8
+ 65	8·2
+100	12·6
+150	8·7
+200	19·1
−200	45·6
Total	100·0

Run No. 11—Trap at End of Ball Mill

In this run a Hummer screen, 35 mesh, replaced the classifier.

Feed rate, 341 lb. per hour.

Feed to ball mill	0·110 oz./ton gold
Undersize from Hummer screen	0·055 "
Recovery of gold in trap	50·0 per cent
Gold in feed to flotation	50·0 "
Feed to flotation	0·055 oz./ton gold
Concentrate	1·41 "
Tailing	0·0066 "
Recovery by flotation	88·4 per cent
Percentage of gold recovered by flotation, 50 x 88·4 =	44·2 "
Total recovery of gold	94·2 "
Ratio of concentration, 29 : 1.	

Reagents used for Flotation:

Soda ash	3·5 lb./ton
Sodium xanthate	0·2 "
Barrett No. 4	0·1 "
Cresylic acid	0·25 "

Screen Test of Feed to Cells:

Mesh	Weight, per cent
+ 48	1·6
+ 65	11·8
+100	9·6
+150	14·3
+200	17·5
−200	45·2
Total	100·0

Run No. 12

In this run a Hummer screen, 28 mesh, was used and the trap placed at end of ball mill.

Feed rate, 505 lb. per hour.

Feed to ball mill		0·110 oz./ton gold
Undersize from Hummer screen		0·055 "
Recovery of gold in trap		50·0 per cent
Gold in feed to flotation		50·0 "
Feed to flotation		0·055 oz./ton gold
Concentrate		1·37 "
Tailing		0·0074 "
Recovery by flotation		87·0 per cent
Table concentrate		0·04 oz./ton gold
Table tailing		0·0066 "
Percentage recovery of gold by flotation,	50 x 87=	43·5 per cent
Total recovery of gold		93·5 "

Ratio of concentration, 28·6 : 1.

Reagents Used for Flotation:

Soda ash	3·5 lb./ton
Sodium xanthate	0·20 "
Barrett No. 4	0·1 "
Cresylic acid	0·15 "

Feed to Flotation Cells:

Mesh	Weight, per cent
+ 48	12·4
+ 65	13·7
+100	12·8
+150	17·8
+200	8·3
−200	35·0
Total	100·0

Flotation Concentrate:

Mesh	Weight, per cent
+ 48	4·6
+ 65	15·9
+100	11·7
+150	14·6
+200	9·0
−200	44·2
Total	100·0

Run No. 13

Screen and trap as in Run No. 12.

Feed rate, 466 lb. per hour.

Feed to ball mill		0·110 oz./ton gold
Undersize from Hummer screen		0·05 "
Recovery of gold in trap		54·5 per cent
Gold remaining in tailing		45·5 "
Feed to flotation		0·05 oz./ton gold
Concentrate		1·43 "
Tailing		0·0083 "
Recovery by flotation		83·8 per cent
Percentage recovery of gold by flotation,	45·5 x 83·8=	38·1 "
Table concentrate		0·045 oz./ton gold
Table tailing		0·0066 "
Trap clean-up		5·20 "
Total recovery of gold in trap and flotation concentrate		92·6 per cent

Ratio of concentration, 34 : 1.

Reagents Used for Flotation:

Soda ash	3·5	lb./ton
Sodium xanthate	0·2	"
Barrett No. 4	0·1	"
Cresylic acid	0·15	"

Screen Test of Products

Feed to Flotation:

Mesh	Weight, per cent
+ 48	15·5
+ 65	15·7
+100	10·5
+150	15·0
+200	11·8
−200	31·5
Total	100·0

Flotation Concentrate:

Mesh	Weight, per cent
+ 48	6·1
+ 65	22·3
+100	19·5
+150	5·6
+200	9·5
−200	37·0
Total	100·0

Run No. 14

Screen as in Run No. 12.

Feed rate, 448 lb. per hour.

Feed to ball mill	0·110	oz./ton gold
Undersize from Hummer screen	0·065	"
Trap clean-up (Runs 12-13-14)	5·20	"
Recovery of gold in trap	40·9	per cent
Gold remaining in tailing	59·1	"
Feed to flotation	0·065	oz./ton gold
Concentrate	1·57	"
Tailing	0·0066	"
Recovery of gold by flotation	90·2	per cent
Percentage recovery of gold by flotation, 59·1 x 90·2=	53·4	"
Table concentrate	0·065	oz./ton gold
Table tailing	0·005	"
Total recovery of gold in trap and flotation concentrate	94·3	per cent

Ratio of concentration, 26·7 : 1.

Reagents Used in Flotation:

Soda ash	3·5	lb./ton
Sodium xanthate	0·2	"
Barrett No. 4	0·1	"
Pine oil	0·02	"
Copper sulphate	0·25	"

Screen Tests of Products

Feed to Flotation:

Mesh	Weight, per cent
+ 48	15·8
+ 65	14·8
+100	12·8
+150	13·1
+200	10·7
−200	32·8
Total	100·0

Flotation Concentrate:

Mesh	Weight, per cent
+ 48	8·6
+ 65	19·8
+100	21·7
+150	5·5
+200	7·7
−200	36·7
Total	100·0

Flotation Tailing:

Mesh	Weight, per cent
+ 48	14·5
+ 65	17·3
+100	15·1
+150	11·4
+200	9·9
−200	31·8
Total	100·0

The following screen analyses were made on the products from Run No. 14.

Mesh	Flotation Feed		Flotation Tailing		*Approximate recovery from sizes, per cent.
	Weight, per cent	Assay, gold oz./ton	Weight, per cent	Assay, gold oz./ton	
+ 48	15·3	0·02	15·1	0·01	50·6
− 48+ 65	15·8	0·005	15·3	0·0075	?
− 65+100	12·0	0·110	11·3	0·005	95·7
−100+150	17·2	0·080	18·6	0·005	93·3
−150+200	8·5	0·075	8·8	0·01	86·3
−200	31·2	0·075	30·9	0·0066	91·4

*These figures are only approximate because the ratio of concentration has not been allowed for. However they should be correct to within 3 per cent.

The accompanying table, page 99, gives a summary of the results of the fourteen large-scale tests. The results show conclusively that the ore can be concentrated at a very low cost. There should be no trouble duplicating and perhaps improving these results in practice provided the proper equipment is used.

In Runs Nos. 12 to 14 the gold trap was not cleaned out between runs, this was for the purpose of watching the effect on both the trap recovery and on the flotation. It will be observed that in Run No. 14 the trap had evidently become loaded and the recovery dropped from 50 to 40 per cent, but it is very reassuring to note that the flotation took care of this; the recovery by flotation jumping from 43 to 53 per cent.

There is only one other observation and it is in connection with the grinding and classification. The ore crushed and ground fairly easily. This may be due to the weathering of the feldspar. Classification of the ground ore in the Dorr classifier operating in closed circuit was difficult. It settled very fast in the classifier which resulted in a large amount of fines being returned to the mill in the rake product. This fact should be considered in connection with the selection of the classifier equipment.

LARGE-SCALE CYANIDE TEST ON CONCENTRATES

A large-scale cyanidation test was made on a composite sample of the flotation concentrates representing the fourteen runs. A sample of 250 pounds was cut from the mixed concentrates by a Jones riffle sampler and the test was made on this sample.

Lime was mixed with the sample in the proportion of 16 pounds to the ton of concentrates and this was fed into a small rod mill in closed circuit with a classifier at the rate of 20 pounds per hour approximately where it was ground in cyanide solution of strength 4·7 pounds KCN per ton. The mill discharged into a classifier, the overflow from which was sent to a Pachuca tank, the oversize being returned to the mill for regrinding.

The pulp was agitated for 72 hours, the solution strength being maintained by additions of sodium cyanide and lime from time to time. Samples of the pulp were taken out for assay at intervals.

Screen analyses showed the rod mill discharge to be 91 per cent minus 200 mesh, and the classifier overflow to be 94 per cent minus 200 mesh.

Summary of Large-Scale Tests

Run No.	Grind: With classifier	Grind: With Hummer S.	Recovery by amalgamation %	Recovery by blankets %	Recovery by trap %	Recovery by flotation %	Total recovery %	Ratio of concentration	Grade of concentrate Au, oz./ton	Assay of tailing Au, oz./ton
1	9·75% + 48 mesh 36·67% −200 "	....	59·1	....	....	36·7	95·8	20·6 : 1	0·83	·005
3	11·0% + 48 mesh 41·0% −200 "	....	54·5	....	....	39·0	93·5	27·6 : 1	1·18	·0075
4	12·9% + 48 mesh 37·8% −200 "	....	....	54·5	....	40·04	94·5	29·3 : 1	1·29	·0062
5	5·2% + 48 mesh 39·5% −200 "	....	....	50·0	....	45·1	95·1	29·2 : 1	1·42	·0062
6	2·7% + 48 mesh 49·5% −200 "	....	63·6	....	....	30·9	94·5	31·5 : 1	1·07	·0062
7	4·6% + 48 mesh 41·6% −200 "	....	....	....	63·6	30·9	94·5	29·5 : 1	1·39	·0062
8	1·5% + 48 mesh 50·4% −200 "	....	....	....	63·6	29·8	93·4	33·6 : 1	1·10	·0075
9	5·3% + 48 mesh 45·4% −200 "	....	....	....	54·5	40·1	94·6	30·0 : 1	1·32	·0062
10	5·8% + 48 mesh 45·6% −200 "	....	....	....	50·0	44·5	94·5	40·0 : 1	1·96	·0062
11	....	1·6% + 48 mesh 45·2% −200 "	....	....	50·0	44·2	94·2	29·0 : 1	1·41	·0062
12	....	12·4% + 48 mesh 35·0% −200 "	....	....	50·0	43·5	93·5	28·6 : 1	1·37	·0066
13	....	15·5% + 48 mesh 31·5% −200 "	....	....	54·5	38·1	92·6	34·0 : 1	1·43	·0083
14	....	15·8% + 48 mesh 32·8% −200 "	....	....	40·9	53·4	94·3	26·7 : 1	1·57	·0066

Summary of Cyanidation Results

Feed sample, Au—1·02 oz./ton

Period of agitation	Tailing assay, Au, oz./ton	Recovery, per cent
22 hours	0·14	86·3
48 "	0·19	81·4
72 "	0·115	88·7

This cyanide test indicates that after the solution has been in contact with the concentrate for a certain length of time, reprecipitation of the gold takes place. In this particular test the same solution was used from the start to the finish of the test, the strength being maintained by the addition of dry cyanide. If more samples had been taken at closer intervals undoubtedly a time would have been found after which very little additional recovery could be obtained, and after which the recovery would fluctuate slightly, in all probability in accordance with the additions of fresh cyanide. The apparent correction for this condition is to thicken or filter and re-agitate in a fresh solution. The necessity of thorough aeration of the barren solution before it is used again is also indicated.

SUMMARY

The following table gives a summary of the results of all the large-scale tests—column one, the test number; column two, the gold recovered before flotation; column three, the gold recovered by cyanidation from the flotation concentrate using the maximum figure of 88·7 per cent; column four, the total recovery.

Run No.	Gold recovered before flotation	Gold recovered by cyanidation of flotation concentrate	Total recovery
1	59·1	32·5	91·6
3	54·5	34·6	89·1
4	54·5	35·5	90·0
5	50·0	40·0	90·0
6	63·6	27·4	91·0
7	63·6	27·4	91·0
8	63·6	26·4	90·0
9	54·5	35·6	90·1
10	50·0	39·5	89·5
11	50·0	39·2	89·2
12	50·0	38·6	88·6
13	54·5	33·8	88·3
14	40·9	47·4	88·3

Ore Dressing and Metallurgical Investigation No. 497

GOLD-SILVER ORE FROM DENTONIA MINES SYNDICATE, GREENWOOD, BRITISH COLUMBIA

Shipment. A shipment of two sacks of ore, net weight 170 pounds, was received February 27, 1933, from Nelson S. Smith, 313 Lancaster Building, Calgary, Alberta.

Characteristics and Analysis of the Ore. Three types of gangue are represented in the samples studied: (1) "chloritic" type, (2) vein quartz, and (3) calcite. All three may be present in a single hand specimen. The "chloritic" type is apparently altered country rock, it is the oldest, and is traversed by either vague zones or well-defined veins of grey to milky quartz containing some sericite. The calcite occurs in fine veinlets that usually cut the "chloritic" gangue, but at times are observed to cut the quartz. In the specimens, the quartz type predominates, and insofar as can be determined microscopically in this laboratory it contains all of the precious metals.

The metallic minerals present in the area are pyrite, chalcopyrite, galena, native gold, hessite(?), and an unknown grey mineral. Pyrite which forms about 94 per cent of the metallic mineral content of the ore is disseminated as irregular grains and well-formed cubes. It is most abundant in the "chloritic" gangue, less abundant in the quartz, and absent from the calcite veinlets.

An average analysis of the ore is as follows:—

Gold	0·73 oz./ton
Silver	4·18 "
Copper	0·15 per cent
Lead	0·37 "
Zinc	Trace

Experimental Tests

Test work on the ore was confined to concentration tests. In all cases good grade concentrates have been produced with a good ratio of concentration in each case, but recoveries were slightly low.

On the sample submitted over 90 per cent of the gold, silver, and copper can be collected in a concentrate amounting to not more than 7 per cent of the weight of feed, and this concentrate will assay about 10 ounces per ton in gold and 60 ounces per ton in silver.

Test No. 1

The ore at minus 14 mesh was ground for 20 minutes in a ball mill, the charge being as follows:—

Ore	2,000 grammes
Water	1,500 "
Na_2CO_3	4 "

The pulp was then transferred to a flotation machine and floated with the following reagents:—

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

The concentrate and tailing were assayed for gold, silver, copper, and lead.

Summary:

Product	Weight, per cent	Assay				Distribution of metals, per cent			
		Au oz./ton	Ag oz./ton	Cu per cent	Pb per cent	Au	Ag	Cu	Pb
Concentrate	4·9	12·86	73·84	1·30	7·0	88·0	89·0	35·8	92·7
Tailing	95·1	0·09	0·47	0·12	0·03	12·0	11·0	64·2	7·3
Feed (cal.)	100·0	0·72	4·06	0·18	0·37	....	....	....	....

Screen Test:

Mesh	Weight, per cent	Cumulative weight, per cent
+ 65	12·3	12·3
− 65+100	15·8	28·1
−100+150	18·7	46·8
−150+200	17·6	64·4
−200	35·6	100·0
Total	100·0	....

Test No. 2

In this test 8,000 grammes of the ore at minus 14 mesh was ground for 30 minutes in a ball mill and floated with the same reagent combination as was used in Test No. 1, except that 0·05 pound per ton of Tarol No. 1 was added to the cell to make the froth more consistent.

The concentrate and tailing were assayed for gold, silver, copper, and lead. A sample of the concentrate was also treated by cyanidation and the cyanide tailing assayed for gold and silver.

Summary:

Product	Weight, per cent	Assays				Distribution of metals, per cent			
		Au oz./ton	Ag oz./ton	Cu per cent	Pb per cent	Au	Ag	Cu	Pb
Concentrate	7·1	9·14	56·18	1·16	3·95	92·1	92·7	90·0	99·0
Tailing	92·9	0·06	0·34	0·01	trace	7·9	7·3	10·0	....
Feed (cal.)	100·0	0·70	4·30	0·09	....	....	....	....	....
Concentrate cyanided	7·1	0·70	26·63	....	....	....	....	....	....

Per cent total gold recovered by cyanidation of flotation concentrate:

$$\frac{9\cdot14-0\cdot70}{9\cdot14} \times 92\cdot1 = 85\cdot04$$

Screen Test:

Mesh	Weight, per cent	Cumulative weight, per cent
+ 65	0·05	0·05
− 65+100	1·50	1·55
−100+150	10·60	12·15
−150+200	22·90	35·05
−200	64·95	100·0
Total	100·0	....

FLOTATION AND BLANKETING

Test No. 3

The ore was ground for 30 minutes in a ball mill and the pulp floated with the same reagent combination as was used in Test No. 2. The flotation tailing was run over blankets with the idea of producing a clean tailing.

Summary:

Product	Weight, per cent	Assay				Distribution of metals, per cent			
		Au oz./ton	Ag oz./ton	Cu per cent	Pb per cent	Au	Ag	Cu	Pb
Flotation concentrate	6·9	9·80	56·90	1·16	4·27	90·0	98·8	....	....
Blanket concentrate	2·3	0·70	0·54	0·02	0·05	2·1	0·3	....	....
Blanket tailing	90·8	0·065	0·038	0·02	Trace	7·9	0·9	....	....
Feed (cal.)	100·0	0·75	3·97	0·10	....	....	....	....	....

Test No. 4

In this test Aerofloat No. 25 was added to the grinding circuit to see if it would bring about an increased recovery in the concentrate and produce a clean tailing. The ore at minus 14 mesh was ground for 30 minutes in a ball mill, the charge being as follows:—

Ore..2,000 grammes
Water..1,500 "
Na_2CO_3.. 4·0 lb./ton
Aerofloat No. 25.. 0·07 "

Reagents to Cell:

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

Summary:

Product	Weight, per cent	Assay				Distribution of metals, per cent			
		Au oz./ton	Ag oz./ton	Cu per cent	Pb per cent	Au	Ag	Cu	Pb
Concentrate........	6·6	10·70	60·80	1·24	5·07	93·2	93·7	100·0	93·2
Tailing............	93·4	0·055	0·29	nil	0·026	6·8	6·3	....	6·8
Feed (cal.)........	100·0	0·76	4·28	0·08	0·36	....	....	....	....

AMALGAMATION AND FLOTATION

Test No. 5

Having in mind that the high tailings from the foregoing tests might be due to free gold, a sample of the ore was ground for 30 minutes and amalgamated before it was floated.

Charge to Ball Mill:

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

Reagents to Cell:

Na_2CO_3..4·0 lb./ton
Aerofloat No. 25..0·07 "
Potassium amyl xanthate.....................................0·1 "
Pine oil..0·05 "
Tarol No. 1...0·05 "

Summary:

Product	Weight, per cent	Assay				Distribution of metals, per cent			
		Au oz./ton	Ag oz./ton	Cu per cent	Pb per cent	Au	Ag	Cu	Pb
Flotation concentrate............	6·1	8·62	62·04	0·84	5·05	90·3	91·6	91·6	92·7
Flotation tailing...	93·9	0·06	0·37	0·005	0·026	9·7	8·4	8·4	7·3
Amalgamation tailing (cal.).....	100·0	0·58	4·13	0·06	0·33	....	....	....	......

Recovery by amalgamation $\frac{0·73-0·58}{0·73}$............................ =20·5 per cent

Per cent total gold in flotation concentrate (100—20·5) 90·3......... =71·8 "

Net recovery.......................................92·3 "

FLOTATION

Test No. 6

This test was made to find out whether finer grinding would liberate any more of the gold-bearing mineral and make possible its recovery in the flotation concentrate.

The ore at minus 14 mesh was ground for 40 minutes in a ball mill the charge being as follows:—

Ore	2,000 grammes
Water	1,500 "
Na_2CO_3	4·0 lb./ton
Aerofloat No. 25	0·07 "

Reagents to Cell:

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

The concentrate and tailing were assayed for gold and silver only.

Summary:

Product	Weight, per cent	Assay		Distribution of metals, per cent	
		Au oz./ton	Ag oz./ton	Au	Ag
Concentrate	6·3	10·76	60·92	92·9	93·6
Tailing	93·7	0·055	0·28	7·1	6·4
Feed (cal.)	100·0	0·73	4·10	....	....

Screen Test:

Mesh	Weight, per cent	Cumulative weight, per cent
+ 65	0·0	0·0
− 65+100	0·2	0·2
−100+150	7·2	7·4
−150+200	13·7	21·1
−200	78·9	100·0
Total	100·0	....

SUMMARY AND CONCLUSIONS

The consistently high tailings produced in the foregoing tests may, perhaps, be explained as follows:—

In the ore, some of the pyrite crystals are enclosed with a shell of secondary limonite resulting from alteration of the pyrite. Flotation tests indicate that most of the gold is closely associated with the pyrite, either in solid solution or as small grains or films precipitated on the crystal faces.

When the alteration to limonite takes place the gold in the pyrite so altered will remain with the limonite formed. Microscopic examination of the flotation tailings showed that the fragments of these limonite shells had not been floated. It therefore seems possible that the high gold content of the tailings may be largely accounted for by the presence in the ore of these shells of limonite around the gold-bearing pyrite and their appearance finally in the tailing in the mill.

If the above be true then this difficulty should not be encountered when ore from greater depth is being milled because the limonitization is undoubtedly a surface feature.

In looking over the results of tests it will be seen that the tailings in Tests Nos. 2 to 6 inclusive are practically uniform in gold in spite of finer grinding, amalgamation, or blanketing, and this seems to lend support to the above explanation.

If further work on the ore is desirable it is suggested that a sample from greater depth be sent to see if a clean tailing can be made from ore free from limonite.

Ore Dressing and Metallurgical Investigation No. 498

INVESTIGATION OF THE PRECIOUS METAL AND VANADIUM CONTENT IN SAMPLES OF MINERALIZED ROCK SUBMITTED BY DELTA MINES SYNDICATE, WORTHINGTON, ONTARIO

Shipment: A carload shipment of mineralized rock, consisting of three lots, was received January 5, 1933, from the Delta Mines Syndicate, Worthington, Ontario.

Lot No. V-1 consisted of 6 boxes, weight 2,410 pounds. This sample was supposed to contain an appreciable quantity of vanadium.

Lot No. V-2 consisted of 5 bags, weight 545 pounds. This sample was also supposed to be vanadium ore.

Lot No. 3 consisted of loose rock, weight 53,355 pounds. This lot was supposed to contain an appreciable quantity of gold and of the platinum group metals.

Purpose of Investigation. The investigation was conducted to concentrate the ores into products containing the vanadium in the case of the V-1 and V-2 lots, and the precious metals in the case of the large lot, No. 3; and to determine whether the sample lots and concentration products contained the high vanadium and precious metal content claimed by the Delta Mines Syndicate.

Examination of Lots Nos. V-1 and V-2. The polished sections examined exhibited similar characteristics with respect to both metallic and non-metallic minerals. The metallic minerals observed were ilmenite, magnetite, pyrite, arsenopyrite, chalcopyrite, and pentlandite. These were present as small disseminated grains in the non-metallic mineral aggregate, consisting chiefly of dark silicates and quartz. Contained in the dark silicate rock material was an appreciable amount of brownish-coloured mica.

A sample of drill core containing this mica was selected by Mr. J. B. W. Hughes, Chemist of the Delta Mines Syndicate. A spectrographic analysis of the mica was made to ascertain whether the mineral was *roscoelite,* the vanadium mica, as supposed by Mr. Hughes to account for the high vanadium content in this type of rock.

The results of the analysis were as follows:—

Magnesium	essential—strong.
Silicon	essential—strong.
Potassium	essential—moderate
Aluminium	essential—moderate.
Iron	trace.
Chromium	trace.
Vanadium	nil.

The composition and physical properties are very similar to *phlogopite* mica.

Sample Lots Nos. V-1 and V-2 were carefully crushed, sampled, and analysed with the following results:—

—		Lot No. V-1	Lot No. V-2
		per cent	per cent
Copper	Cu	0·05	0·20
Nickel	Ni	nil	nil
Sulphur	S	0·60	0·35
Iron	Fe	5·60	5·65
Arsenic	As	0·02	0·02
Vanadium oxide	V_2O_5	traces	traces
Titanium oxide	TiO_2	0·05	0·05
		oz./ton	oz./ton
Gold	Au	0·004	0·007
Platinum group metals		traces	nil

This partial analysis shows these lots to contain only traces of vanadium and of the precious metals. Both lots were concentrated by flotation and the tailings from flotation concentrated on a table. The riffles of the table filled with the mica so that a fair-sized sample of the mica from each lot was collected, which on chemical determination gave only traces of vanadium. This was also checked by spectrographic analysis, confirming the spectrographic analysis given above of the mica from the drill core, that the mica contained no vanadium, and was not roscoelite but ordinary mica, probably phlogopite.

The table concentrate from each lot, representing a very small proportion of the weight of the lots and consisting mostly of magnetite, ilmenite, and chromite, was sampled and analysed for vanadium and chromium with the following results:—

—	Lot No. V-1	Lot No. V-2
	per cent	per cent
Vanadium oxide—V_2O_5	0·50	0·22
Chromium oxide—Cr_2O_3	8·60	4·07

The trace of vanadium in these sample lots is no doubt in association with small amounts of ilmenite. Most titaniferous magnetites and ilmenites contain small amounts of vanadium. Some of the rock formations of the Sudbury area, with which the ore-bodies are associated, contain as high as 1·50 per cent titanic oxide and most of them will contain smaller amounts. It is, therefore, to be expected that traces of vanadium will be found in association with the titaniferous magnetite and ilmenite of these rocks. An analysis of the tailing from the concentration of the Falconbridge disseminated nickel ore showed it to contain traces of vanadium.

Examination of Lot No. 3. This lot, consisting of the bulk of the carload shipment and weighing 53,355 pounds, was reported to contain an appreciable amount of gold and the platinum group metals. The metallic minerals observed in the polished sections were ilmenite, magnetite, pyrite, pyrrhotite, arsenopyrite, chalcopyrite, pentlandite, and violarite. They were present in small amounts of quartz and dark-coloured silicates, disseminated throughout the gangue rock.

The whole lot was crushed, carefully sampled, and analysed with the following results:

Copper	Cu	0·10 per cent
Nickel	Ni	0·10 "
Sulphur	S	0·86 "
Iron	Fe	6·58 "
Arsenic	As	traces
Vanadium oxide	V_2O_5	traces
Titanium oxide	TiO_2	1·00 per cent
Gold	Au	0·003 oz./ton
Platinum group metals		traces

The whole lot, after crushing and sampling, was ground in a ball mill in circuit with a classifier to 75·0 per cent minus 200 mesh and concentrated by flotation. The flotation tailing was concentrated on a table.

The flotation concentrate obtained weighed 1,100 pounds, 2·08 per cent by weight of the lot, a ratio of concentration of 48·5 : 1. The analysis of this concentrate was as follows:—

Copper	Cu	2·74 per cent
Nickel	Ni	2·70 "
Sulphur	S	27·17 "
Iron	Fe	30·87 "
Arsenic	As	0·02 "
Vanadium oxide	V_2O_5	0·02 "
Titanium oxide	TiO_2	0·55 "
Gold	Au	0·083 oz./ton
Platinum group metals		0·040 "

The precious metal determinations were the result from a furnace assay of 2·5 pounds of flotation concentrate.

Duplicate determinations for gold following the methods used by Mr. Hughes of the Delta Mines Syndicate, but omitting the addition of gold as practised by him, gave 0·085 ounce per ton in the flotation concentrate.

The table concentrate obtained weighed 230 pounds, 0·43 per cent by weight of the lot, a ratio of concentration of 234 : 1. The analysis of this concentrate, consisting chiefly of iron oxides, was:—

Copper	Cu	traces
Nickel	Ni	0·09 per cent
Iron	Fe	44·50 "
Arsenic	As	traces
Vanadium oxide	V_2O_5	0·26 "
Titanium oxide	TiO_2	17·42 "
Gold	Au	0·012 oz./ton
Platinum group metals		nil

The analysis of the tailing from the concentration of the lot, representing by weight 97·51 per cent of the lot, was as follows:

Copper	Cu	0·015 per cent
Nickel	Ni	0·027 "
Vanadium oxide	V_2O_5	nil
Titanium oxide	TiO_2	0·40 "
Gold	Au	nil
Platinum group metals		nil

CONCLUSIONS

It is evident from the analysis of the concentration products that the precious metals were concentrated to a large extent in the flotation concentrate with the copper, nickel, and iron sulphides, and that any vanadium was concentrated to a similar extent in the table concentrate with the oxides of iron and titanium.

It is also evident from the analysis of the three lots of mineralized rock and the concentration products therefrom, that the rock material represented by these lots contains no commercial amounts of vanadium or the precious metals.

Ore Dressing and Metallurgical Investigation No. 499

ARSENICAL GOLD ORE FROM THE BLOMFIELD PROPERTY, MARMORA TOWNSHIP, HASTINGS COUNTY, ONTARIO

Shipment. A small shipment of arsenical gold ore, weight 50 pounds, was received from W. E. Leonard, Lakefield, Ontario, on March 23, 1933. The shipment consisted of 18 samples, taken from the Blomfield property, Marmora Township, Hastings County, Ontario.

Characteristics of the Ore. The predominant sulphide mineral is arsenopyrite. The other sulphide minerals observed are pyrite and pyrrhotite. The gangue mineral is chiefly silica with a small amount of carbonate. No gold was observed in the polished section examined.

Experimental Tests

Each sample was crushed and a laboratory sample assayed for gold. The remainder of the ore was mixed, sampled, and assayed for gold, copper and arsenic. The weight of the combined sample was 41 pounds. The mixed ore was used in small-scale amalgamation and cyanidation tests.

Assays:

Sample No.	Assay, Au, oz./ton	Sample No.	Assay, Au, oz./ton	Sample No.	Assay, Au, oz./ton
1	0·04	7	0·01	13	0·58
2	0·535	8	0·41	14	0·12
3	0·01	9	0·47	15	0·03
4	0·015	10	0·945	16	0·32
5	0·01	11	0·96	17	0·83
6	0·015	12	0·365	18	1·17

The assay of the feed sample of the mixed ore was

Gold	0·855 oz./ton
Arsenic	11·6 per cent
Copper	trace

Representative samples of the mixed ore were crushed dry to minus 48, 100, 150, and 200 mesh for the following tests.

AMALGAMATION

A representative sample of minus 100-mesh ore was amalgamated. After removing the amalgam the tailing assayed 0·145 ounce gold per ton.

Results:

Feed assay..0·855 oz./ton Au
Tailing assay...0·145 "

Recovery $\frac{0{\cdot}710}{0{\cdot}855}$ x 100..83·04 per cent

CYANIDATION

Representative samples of the ore −48, −100, −150, −200 were agitated in a sodium cyanide solution containing the equivalent of 1 pound of potassium cyanide per ton of solution, and lime equivalent to 12 pounds per ton of ore was added to give a protective alkalinity. The pulp dilution was in the ratio of 1 part ore to 3 parts of solution.

Results:

The results of the 24- and 48-hour periods of agitation are shown in the following table.

Time, hours	Mesh	Assay, Au, oz./ton		Extraction, per cent	Reagent consumption, lb./ton of ore	
		Feed	Tailing		KCN	CaO
24	− 48	0·855	0·115	86·55	0·84	8·70
24	−100	0·855	0·04	95·32	0·99	9·30
24	−150	0·855	0·03	96·49	0·99	9·51
24	−200	0·855	0·02	97·66	1·14	10·11
48	− 48	0·855	0·04	95·32	2·70	9·90
48	−100	0·855	0·02	97·66	2·70	10·20
48	−150	0·855	0·02	97·66	2·70	10·65
48	−200	0·855	0·01	98·83	2·85	11·10

N.B.—Prolonged agitation increases the consumption of cyanide for only slight increase in recovery.

Tests were made on minus 200-mesh ore using a minimum and maximum of lime, and agitating for 24 hours in a KCN solution, 1 pound per ton. The lime consumed in the minus 200-mesh ore was 10·11 pounds. In the first test lime equivalent to 10·8 pounds per ton of ore was added, and in the second test, 21·6 pounds per ton.

Results:

CaO lb./ton	Assay, Au, oz./ton		Extraction, per cent	Reagent consumption, lb./ton ore		
	Feed	Tailing		KCN	CaO	CaO
10·8	0·855	0·02	97·66	0·99	9·97	Minimum
21·6	0·855	0·065	92·40	1·59	19·95	Maximum

This test indicates a better recovery of gold when a minimum of lime is used in the cyanide pulp.

Ore Dressing and Metallurgical Investigation No. 500

NICKEL-COPPER ORE FROM THE GERSDORFFITE PROPERTY, NEAR WORTHINGTON, ONTARIO

Shipment. A shipment of two bags of nickel-copper ore, net weight 200 pounds, was received February 27, 1933, from J. T. O'Connor, Sudbury, owner of the property, which is located near Worthington, Ontario.

Characteristics of the Ore. In general pyrrhotite, chalcopyrite, pent-, landite, and violarite are the important minerals. The presence of violarite, which is regarded as a supergene replacement of pentlandite and pyrrhotite, indicates that the material represented in the shipment has suffered considerable surface alteration.

Pyrrhotite is the most abundant metallic mineral and occurs both in the massive form and in small grains. Chalcopyrite, second in abundance, is likewise present in massive form and as a portion of the filling of irregular stringers in the quartz gangue. Pentlandite ((Ni, Fe,)S) is common as irregular veinlets and grains in the pyrrhotite. Violarite ($(Ni, Fe)_3 S_4$) is less abundant than pentlandite, and occurs in the same manner as this material.

No cobalt minerals were observed in the microscopic examination.

Experimental Tests

The ore was sampled by standard methods and a sample crushed to minus 14 mesh was used for the test work. Analysis of the feed sample was as follows:—

Nickel	2·66	per cent
Copper	1·28	"
Cobalt	0·16	"
Gold	0·05	oz./ton
Silver	0·42	"
Platinum group	0·494	"

Flotation tests were made to obtain a bulk concentrate of the nickel and copper.

FLOTATION

The ore was ground in a ball mill for 15 minutes with the reagents and then floated.

Charge and Reagents to Mill:

Ore—14 mesh..2,000 grammes
Water...1,000 c.c.
Soda ash...3·0 lb./ton
Potassium xanthate...0·1 "

Pine oil...0·075 lb./ton added to cells

Test No. 3

Product	Weight, per cent	Assay, per cent		Distribution of metals, per cent	
		Ni	Cu	Ni	Cu
Concentrate...........	9·91	2·78	9·32	10·1	71·4
Tailing...............	90·09	2·45	0·41	89·9	28·6

The first three tests showed a poor recovery of nickel and only a fair recovery of copper in the concentrate.

Two other tests were run using the same charge to the mill; 0·5 pound per ton copper sulphate and 0·125 pound per ton pine oil were added to the cell.

Test No. 5

Product	Weight, per cent	Assay, per cent		Distribution of metals, per cent	
		Ni	Cu	Ni	Cu
Concentrate...........	17·68	5·86	6·00	42·6	87·4
Tailing...............	82·32	1·70	0·19	57·4	12·6

Screen Test of Tailing:

Mesh	Weight, per cent
+ 48	1·2
+ 65	0·4
+100	0·5
+150	3·5
+200	8·0
−200	86·4
Total	100·0

The addition of copper sulphate resulted in an improvement in the copper and nickel recovery, although the latter is still low.

Test No. 6

In this test the charge of soda ash to the mill was doubled and 0·25 pound per ton additional of copper sulphate used in the cell.

Charge and Reagents to Mill:

Ore —14 mesh..2,000 grammes
Water...1,000 c.c.
Soda ash...6·0 lb./ton
Potassium xanthate...0·1 "

Addition to Cell:

Copper sulphate..0·75 lb./ton
Pine oil...0·1 "

Product	Weight, per cent	Assays, per cent		Distribution of metals, per cent	
		Ni	Cu	Ni	Cu
Concentrate......................	13·44	6·64	9·35	36·7	93·4
Tailing..........................	86·56	1·78	0·10	63·3	6·6

Screen Test of Tailing:

Mesh	Weight, per cent
+ 48	0·1
+ 65	0·0
+100	1·2
+150	9·1
+200	10·9
−200	78·7
Total	100·0

The altered condition of the ore appears to prevent the nickel-bearing sulphides from floating. Further tests were run increasing the xanthate addition to the mill and extending the time of grinding to thirty minutes. The results showed a decided improvement in metal recovery.

Test No. 7

Charge and Reagents to Mill:

Ore −14 mesh..2,000 grammes
Water..1,000 c.c.
Soda ash...6·0 lb./ton
Potassium xanthate...0·2 "

Reagents to Cell:

Copper sulphate..0·75 lb./ton
Pine oil...0·075 "

Product	Weight, per cent	Assay, per cent		Distribution of metals, per cent	
		Ni	Cu	Ni	Cu
Concentrate......................	15·2	10·27	7·86	60·6	90·2
Tailing..........................	84·8	1·20	0·15	39·4	9·8

Screen Test of Tailing:

Mesh	Weight, per cent
+ 48	1·3
+ 65	0·3
+100	2·1
+150	1·3
+200	2·8
−200	92·2
Total	100·0

Tests Nos. 8 and 9

The xanthate was further increased to 0·4 pound per ton, all other quantities and conditions being the same as Test No. 7.

Test No. 8

Product	Weight, per cent	Assay, per cent		Distribution of metals, per cent	
		Ni	Cu	Ni	Cu
Concentrate	19·9	9·88	6·02	80·8	93·7
Tailing	80·1	0·58	0·10	19·2	6·3

Ratio of concentration is 5 : 1.

Test No. 9

Product	Weight, per cent	Assay, per cent		Distribution of metals, per cent	
		Ni	Cu	Ni	Cu
Concentrate	21·5	10·19	5·56	86·6	94·4
Tailing	78·5	0·43	0·09	13·4	5·6

Ratio of concentration is 4·6 : 1

Screen Tests of Tailings:

Test No. 8

Mesh	Weight, per cent
+ 48	0·8
+ 65	0·9
+100	0·4
+150	3·2
+200	6·7
−200	88·0
Total	100·0

Test No. 9

Mesh	Weight, per cent
+ 48	5·1
+ 65	0·6
+100	0·2
+150	2·0
+200	2·2
−200	89·9
Total	100·0

CONCLUSIONS

The alteration of the nickel mineral renders it difficult to obtain a bulk concentrate high in nickel. The best results were obtained by using a distinctly alkaline pulp, increasing the potassium xanthate and grinding to have 90 per cent pass a 200-mesh screen.

The products from Test No. 9 were assayed for platinum group metals with the following results:—

Concentrate	1·49 oz./ton
Tailing	0·10 "

Palladium is the major metal of the platinum group present in this ore.

Ore Dressing and Metallurgical Investigation No. 501

SILVER-LEAD ORE FROM QUEENSBORO, ONTARIO

Shipment. A shipment of 40 bags of ore, weighing 2,600 pounds, was received January 28, 1933, from D. S. Baird, Queensboro, Ontario.

Characteristics of the Ore. A sample was examined microscopically and was found to consist of grey, smoky quartz containing metallic minerals disseminated in small grains in the quartz and also as small, irregular stringers.

The minerals consist of chalcopyrite, tetrahedrite containing silver in solid solution, galena which does not appear to be intimately associated with the other minerals, and a small amount of sphalerite and iron pyrite.

The shipment was crushed, sampled, and assayed and found to contain:

Lead	2·15 per cent
Copper	0·61 "
Zinc	0·17 "
Gold	0·20 oz./ton
Silver	6·58 "

EXPERIMENTAL TESTS

Concentration by flotation and blankets was undertaken to note what recovery of the minerals would result.

FLOTATION

Test No. 1

A sample of the ore was ground minus 65 mesh with 70 per cent minus 200 mesh. Six pounds soda ash and 0·14 pound Aerofloat No. 25 were added to the mill. The pulp was then transferred to a flotation machine where 0·20 pound sodium xanthate and 0·08 pound pine oil were added.

Product	Weight, per cent	Assay					Distribution of metals, per cent				
		Au oz./ton	Ag oz./ton	Cu per cent	Pb per cent	Zn per cent	Au	Ag	Cu	Pb	Zn
Feed (cal.)	100·0	0·20	7·85	0·70	2·20	0·15	100·0	100·0	100·0	100·0	100·0
Concentrate	9·4	2·00	72·60	6·04	21·29	1·09	87·4	87·0	88·8	90·6	69·3
Tailing	90·6	0·03	1·13	0·15	0·23	0·05	12·6	13·0	21·2	9·4	30·7

These results show that 87·4 per cent of the gold, 87 per cent of the silver, and 90·6 per cent of the lead are recovered in a product assaying 21·3 per cent lead, 72·6 ounces of silver, and 2·00 ounces of gold per ton.

From each 100 tons of ore milled 9·4 tons of this product is obtained.

Test No. 2

In this test a flotation concentrate was taken off as in Test No. 1 and the tailing passed over a blanket to see if any additional saving would result.

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Au oz./ton	Ag oz./ton	Pb per cent	Au	Ag	Pb
Feed (cal.)	100·0	0·245	7·75	2·51	100·0	100·0	100·0
Flotation concentrate	8·5	2·24	78·02	24·74	77·6	85·5	83·7
Blanket concentrate	3·3	1·00	3·08	1·72	13·4	1·3	2·3
Blanket tailing	88·2	0·025	1·16	0·40	9·0	13·2	14·0

A small amount of concentrate was recovered on the blankets, raising the total recovery of gold to 91 per cent. The recovery of silver is the same as in Test No. 1.

Other tests were made using different flotation reagents. Higher grade concentrates were made but the recovery was lower.

Test No. 3

A sample of the ore was ground with 8 pounds soda ash per ton, and floated with 0·30 pound "Minerec A" per ton, and 0·08 pound pine oil was used to produce a froth.

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Au oz./ton	Ag oz./ton	Pb per cent	Au	Ag	Pb
Feed (cal.)	100·0	0·21	7·38	2·40	100·0	100·0	100·0
Concentrate	5·9	2·88	103·24	33·56	81·9	82·6	82·4
Tailing	94·1	0·04	1·37	0·45	18·1	17·4	17·6

CONCLUSIONS

The results secured in this investigation show that 90 per cent of the gold, 87 per cent of the silver, and 90 per cent of the lead can be recovered by flotation with a ratio of concentration of about 10 : 1. As a safeguard, the flotation tailing could be passed over blankets to recover any coarser particles of gold not caught in the flotation circuit.

Ore Dressing and Metallurgical Investigation No. 502

LEAD-ZINC ORE FROM THE MARSOUINS MINING COMPANY, LIMITED, MARSOUINS, GASPE, QUEBEC

Shipment. A shipment consisting of two samples of ore was received February 18, 1933, from the Marsouins Mining Company, Limited, Marsouins, Gaspe, Quebec. These samples were submitted for test work by Arthur Cote, General Manager, Marsouins Mining Company, Limited.

Sample No. 1 weighed 120 pounds.
Sample No. 2 weighed 100 pounds.

Character and Analysis of Samples. The ore in both samples contained galena, sphalerite, and pyrite, with which were associated gold and silver.

The analyses of the shipments were as follows:—

	Vein No. 1	Vein No. 2
Gold	0·08 oz./ton	0·08 oz./ton
Silver	7·38 "	11·72 "
Copper	0·20 per cent	0·58 per cent
Lead	12·0 "	12·48 "
Zinc	4·60 "	5·35 "

Experimental Tests

Three concentration tests were made on these shipments. The only method of concentration applicable to this ore is the selective flotation process.

Vein No. 1 Ore

Test No. 1

A sample of 2,000 grammes of ore previously crushed to 14 mesh was ground in a small ball mill with the following reagents:—

Water	1,500 grammes
Soda ash	3·0 "
Cyanide	0·2 "
Zinc sulphate	1·0 "
Aerofloat No. 25	0·08 "

After grinding to the following screen sizes the charge was washed into a laboratory flotation machine and a lead concentrate removed by the addition of 0·06 gramme of Aerofloat No. 25. After removal of the lead, the zinc was floated by the addition of the following reagents:—

Copper sulphate	1·0 gramme
Xanthate	0·15 "

Screen Test:

Mesh	Per cent
+ 48	0·2
− 48+ 65	5·0
− 65+100	3·4
−100+150	11·3
−150+200	16·3
−200	63·8
Total	100·0

Product	Weight, per cent	Analysis				Distribution of metals, per cent			
		Lead, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton	Lead	Zinc	Gold	Silver
Lead concentrate	12·4	62·95	1·55	0·04	27·70	89·4	5·7	8·0	62·2
Zinc concentrate	4·5	4·60	50·9	0·18	26·92	2·0	55·2	10·6	17·8
Zinc middling	2·0	8·72	14·18	0·10	10·34	1·6	7·0	2·6	3·1
Zinc tailing	76·1	1·01	1·78	0·08	1·53	7·0	32·1	78·8	16·9
Total	100·0	10·9	4·2	0·08	6·9	100·0	100·0	100·0	100·0

Test No. 2

Product	Weight, per cent	Analysis				Distribution of metals, per cent			
		Lead, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton	Lead	Zinc	Gold	Silver
Lead concentrate	14·6	67·02	0·52	0·05	25·40	79·7	1·6	9·5	48·3
Zinc concentrate	8·9	15·81	38·09	0·09	29·96	11·5	76·0	10·4	34·7
Zinc middling	3·1	6·28	8·78	0·09	7·0	1·5	6·0	3·6	2·8
Tailing	73·4	1·22	0·99	0·08	5·87	7·3	16·4	76·5	14·2
Total	100·0	12·3	4·5	0·08	7·7	100·0	100·0	100·0	100·0

Vein No. 2 Ore

Reagents used were the same as used in test on Vein No. 1 ore.

Test No. 1

Product	Weight, per cent	Analysis				Distribution of metals, per cent			
		Lead, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton	Lead	Zinc	Gold	Silver
Lead concentrate	16·8	65·7	1·10	0·27	45·46	85·8	3·2	57·2	65·8
Zinc concentrate	5·9	3·85	50·2	0·05	28·00	1·8	55·0	3·7	14·2
Zinc middling	2·0	8·45	22·12	0·04	17·4	1·3	8·2	1·0	3·0
Tailing	75·3	1·9	2·40	0·04	2·62	11·1	33·6	38·1	17·0
Total	100·0	12·9	5·4	0·08	11·6	100·0	100·0	100·0	100·0

Test No. 2

Reagents used were as follows: ore 2,000 grammes, water 1,500 grammes, ground in laboratory ball mill with soda ash 3 pounds per ton, zinc sulphate 1 pound per ton, cyanide 0·2 pound per ton, xanthate 0·1 pound per ton, and the lead floated with pine oil; the zinc by the addition of 1 pound copper sulphate per ton and additional pine oil.

Product	Weight, per cent	Analysis				Distribution of metals, per cent			
		Lead, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton	Lead	Zinc	Gold	Silver
Lead concentrate..	22·13	53·0	3·47	0·21	47·14	96·9	14·3	50·0	91·8
Zinc concentrate...	7·04	0·31	54·06	0·04	1·76	0·2	70·9	3·0	1·1
Zinc middling.....	2·48	2·11	17·76	0·05	3·70	0·4	8·2	1·3	0·8
Pyrite concentrate.	23·93	0·52	1·10	0·14	2·12	1·0	4·9	36·1	4·5
Tailing............	44·42	0·40	0·20	0·02	0·46	1·5	1·7	9·6	1·8
Total......	100·0	12·1	5·4	0·09	11·35	100·0	100·0	100·0	100·0

Test No. 3

The only change in the reagents used was the use of Minerec B to replace xanthate. The amount used was 0·08 pound per ton.

Product	Weight, per cent	Analysis				Distribution of metals, per cent			
		Lead, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton	Lead	Zinc	Gold	Silver
Lead concentrate..	22·1	51·22	4·21	0·20	47·86	95·7	18·0	54·3	94·2
Zinc concentrate...	8·3	0·31	42·66	0·04	0·78	0·2	68·7	4·1	0·6
Zinc middling.....	2·7	0·90	8·02	0·07	1·70	0·2	4·2	2·3	0·4
Pyrite concentrate.	27·7	0·52	0·92	0·08	0·82	1·2	4·9	27·3	2·0
Tailing............	39·2	0·79	0·54	0·025	0·80	2·6	4·1	12·0	2·8
Total......	100·0	11·8	5·2	0·08	11·2	99·9	99·9	100·0	100·0

Ore Dressing and Metallurgical Investigation No. 503

GOLD ORE FROM LITTLE LONG LAC GOLD MINES, LTD., BANKFIELD, ONTARIO

Shipment. A shipment of gold ore weighing 2,210 pounds contained in 42 sacks, was received February 2, 1933, from S. J. Fitzgerald of the Sudbury Diamond Drilling Co., Ltd., Sudbury, Ontario. This consisted of two lots, 20 bags containing mostly quartz and 22 bags containing schisted material.

Characteristics of the Ore. The quartz shipment consisted of milky vein quartz containing some oxidized matter and small disseminated crystals of arsenopyrite. Very rare, irregular disseminated grains of iron pyrite were also present.

The schisted material consisted of a fine-textured, greenish grey schist with fine quartz stringers. A considerable amount of arsenopyrite was present as disseminated fine crystals. Iron pyrite was very rare and a very small amount of chalcopyrite also was present, usually in the quartz stringers.

Experimental Tests

The consignment was made to determine what recovery of gold could be obtained and by what method of treatment.

Both lots were sampled and assayed as follows:—

Lot No. 1, Quartz = 2·35 ounces gold per ton.
Lot No. 2, Schist = 0·84 ounce gold per ton.

The tests show that 94·9 per cent of the gold in the quartz sample can be recovered by straight cyanidation at 100 mesh and 91 per cent by amalgamation and flotation.

The tests also show that 86 per cent of the gold in the schist can be recovered by cyanidation at 100-mesh grinding.

Both lots were treated separately. This investigation included cyanidation, amalgamation, blanket concentration, and flotation.

Quartz Ore

CYANIDATION

Test No. 1

Samples of the lot were ground to pass various meshes, and cyanided, 1 : 3 dilution, with a KCN solution, 1 pound per ton, and 7 pounds of lime per ton of ore.

Mesh grind	Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption lb./ton	
					KCN	CaO
48	24	2·35	0·145	93·8	0·6	6·7
48	48	2·35	0·125	94·7	0·75	6·6
100	24	2·35	0·135	94·3	0·6	6·6
100	48	2·35	0·12	94·9	0·75	6·6
150	24	2·35	0·14	94·0	0·6	6·7
150	48	2·35	0·12	94·9	0·75	6·4
200	24	2·35	0·12	94·9	0·9	6·7
200	48	2·35	0·115	95·1	0·9	6·7

Practically the same recovery is made from minus 48-mesh material as from minus 200 mesh.

AMALGAMATION AND CYANIDATION

Test No. 2

A sample of the ore ground minus 65 mesh with 71 per cent minus 200 mesh was mixed with an equal weight of water and 10 per cent by weight of mercury. This was rotated in a jar mill for one hour. The mercury and amalgam were then removed and the residue sampled. Part of the amalgamation tailing was cyanided with 1·0 pound KCN solution, 1 : 3 dilution, together with 7 pounds lime per ton.

Results:

Feed...2·35 ounces gold per ton
Amalgamation tailing.......................................0·46 " " "
Recovery by amalgamation...................................80·4 per cent

Cyanide tailing 24 hours' agitation........................0·15 ounce gold per ton
" " 48 " " ....0·125 " "
Recovery by amalgamation and cyanidation...................94·7 per cent

AMALGAMATION AND CONCENTRATION

Test No. 3

A sample was amalgamated as in Test No. 2, and the amalgamation tailing concentrated by flotation using 6 pounds soda ash, 0·14 pound sodium xanthate, and 0·06 pound pine oil.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·0	2·35	100·0
Amalgam tailing	....	0·46	....
Amalgam	....	....	80·6
Flotation concentrate	2·6	7·07	7·8
Flotation tailing	97·4	0·28	11·6

Test No. 4

A sample was ground minus 65 mesh with 72 per cent minus 200, and passed over a corduroy blanket.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	2·35	100·0
Blanket concentrate	5·05	27·12	67·6
Blanket tailing	94·95	0·69	32·4

Test No. 5

A representative portion of the ore was ground 72 per cent minus 200 mesh with 6 pounds soda ash per ton and floated with 0·14 pound sodium ethyl xanthate and 0·08 pound pine oil per ton.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·0	2·35	100·0
Concentrate	3·31	51·34	78·9
Tailing	96·69	0·47	21·1

Mill Run No. 1

The ore was fed at the rate of 100 pounds per hour to a small rod mill where it was ground minus 48 mesh with 62 per cent minus 200 mesh. The mill discharged over an amalgamating plate into a conditioning tank where the pulp was conditioned for 15 minutes with 8 pounds of soda ash and 0·11 pound sodium xanthate per ton. The tank discharged into the first cell of a 6-cell flotation machine where a finished concentrate was taken off. The rougher concentrate from the last five cells was returned to the head of the machine; 0·14 pound pine oil per ton was required to froth.

Assays:

	Au, oz./ton
Feed	2·29
Flotation concentrate	15·68
Plate tailing	0·88
Flotation tailing	0·21

Results:

	Per cent
Recovered by amalgamation	61·6
Recovered by flotation	29·7
Total recovery	91·3

Ratio of concentration, 23 : 1.

Mill Run No. 2

A run was made substituting blankets in place of amalgamating plates. Other details were similar to the preceding run.

Assays:

	Au, oz./ton
Feed	2·48
Blanket concentrate	86·72
Blanket tailing	1·50
Flotation concentrate	38·64
Flotation tailing	0·28

Results:

	Per cent
Recovery on blankets	39·5
Recovery by flotation	49·5
Total recovery	89·0

Ratio of concentration, blankets, 89 : 1; flotation, 31·4 : 1.

Schist Ore

Test No. 1

Samples of the ore ground through different meshes were cyanided 1 : 3 dilution with a 2·0 pound KCN solution and 7 pounds lime per ton ore.

Mesh grind	Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton	
					KCN	CaO
− 48	24	0·84	0·12	85·7	0·9	5·7
− 48	48	0·84	0·12	85·7	1·2	7·8
−100	24	0·84	0·115	86·3	0·9	5·7
−100	48	0·84	0·12	85·7	1·2	7·8
−150	24	0·84	0·105	87·5	0·9	6·4
−150	48	0·84	0·105	87·5	0·9	9·7
−200	24	0·84	0·135	83·9	1·2	6·4
−200	48	0·84	0·11	86·9	0·9	9·7

Test No. 2

A sample of the schist was ground with 11 pounds lime and a 2·0 KCN cyanide solution until 97·7 per cent passed 325 mesh and agitated 48 hours, 1 : 3 dilution, with a 2-pound KCN solution.

Agitation, hours	Feed, Au, oz./ton	Tailing Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton	
				KCN	CaO
Mill discharge	0·84	0·62	26·2	0·7	....
24 hours	0·84	0·09	89·3	2·2	10
48 hours	0·84	0·08	90·5	2·3	10

Mill runs similar to those made on the quartz shipment were carried out.

AMALGAMATION AND FLOTATION

Mill Run No. 3

Assays:

	Au, oz./ton
Feed	0·85
Amalgamation tailing	0·50
Flotation concentrate	14·34
Flotation tailing	0·245

Results:

	Per cent
Recovered by amalgamation	41·1
Recovered by flotation	29·0
Total recovery	70·1

Ratio of concentration, 55·9 : 1.

BLANKET AND FLOTATION CONCENTRATION

Mill Run No. 4

Assays:

	Au, oz./ton
Feed	0·89
Blanket concentrate	48·48
Blanket tailing	0·67
Flotation concentrate	28·92
Flotation tailing	0·305

Results:

	Per cent
Recovered by blankets	25·1
Recovered by flotation	40·7
Total recovery	65·8

Ratio of concentration, blankets, 218 : 1 ; flotation, 78·4 : 1.

SUMMARY AND CONCLUSIONS

Approximately 80 per cent of the gold in the quartz sample is free, as indicated by small-scale amalgamation tests. However, this recovery is not obtained in continuous runs, dropping to 61·6 per cent as shown in Mill Run No. 1; 41 per cent is recovered from the schist by this method.

Blanket concentration does not give high recoveries. Amalgamation followed by flotation results in a total recovery of 91·3 per cent of the gold in the quartz and 70·1 per cent in the schist.

Blanket and flotation concentration gives recoveries of 89 and 65·8 per cent on the quartz and schist. Small-scale flotation tests show lower recoveries. Straight cyanidation of the quartz ground minus 48 mesh gives a recovery of 94·7 per cent of the gold in this grade of ore; 85·7 per cent is recovered from the schist. When ground exceedingly fine, the recovery is slightly higher, 90·5 per cent of the gold in the schist being recovered.

Straight cyanidation of the raw ore gives the most satisfactory results. Coarse grinding is indicated as there is but 0·01 ounce a ton difference between the tailing from 48- and 200-mesh grinding.

Ore Dressing and Metallurgical Investigation No. 504

COPPER-NICKEL ORE FROM CALUMET ISLAND, PONTIAC COUNTY, QUEBEC

Shipment. A shipment consisting of 52 bags, net weight 5,830 pounds, was received on February 27, 1933.

The shipment consisted of a nickel-copper ore from Calumet Island, and was submitted by W. S. Barnhart, Campbell's Bay, Pontiac County, Quebec.

Characteristics. The metallic minerals observed are, in their order of abundance, pyrrhotite, violarite, pentlandite, chalcopyrite, ilmenite, and pyrite.

The gangue, although not determined, appears to be peridotitic in nature and to have been considerably altered. Veinlets of serpentine traverse the gangue and sulphides alike, and are apparently due to supergene alteration.

Of the nickel-bearing minerals, pyrrhotite and pentlandite are hypogene, whereas violarite is regarded as a supergene replacement of pentlandite and pyrrhotite. It is, therefore, apparent from the large proportion of violarite with relation to pentlandite that the material represented in the sections examined has suffered extensive surface alteration.

Experimental Tests

The shipment was sampled by standard methods and a head sample cut for analysis and test work. The head sample analysed as follows:—

Nickel	0·88 per cent
Copper	0·17 "
Cobalt	none
Iron	18·8 "
Sulphur	9·87 "
Insoluble	54·9 "
Gold	0·02 oz./ton
Silver	0·055 "
Platinum group	none

FLOTATION

Test No. 1

A sample of 2,000 grammes of the ore was charged to a ball mill with soda ash, sodium sulphide, and potassium xanthate, and after thirty minutes' grinding was fed to the flotation cell.

Charge and Reagents to Mill:

Ore	2,000 grammes
Water	1,000 c.c.
Soda ash	10·0 lb./ton
Potassium xanthate	0·1 "
Sodium sulphide	1·0 "
Pine oil to cell	0·075 "

Results:

Product	Weight, per cent	Assays, per cent		Distribution of metals, per cent	
		Ni	Cu	Ni	Cu
Concentrate	9·4	3·11	1·54	33·6	79·5
Tailing	90·6	0·65	0·04	66·4	20·5

Screen Test:

Mesh	Weight, per cent
+ 48	1·1
+ 65	0·4
+100	0·6
+150	3·8
+200	4·7
−200	89·4
Total	100·0

The results indicate that the nickel was difficult to float.

By doubling the quantity of soda ash in the pulp the nickel was further suppressed.

Two tests were run discarding the addition of sodium sulphide to the mill and adding copper sulphate to the cell. The results were not satisfactory. A concentrate of 3·27 per cent nickel and 1·90 per cent copper was obtained, containing 36·2 per cent and 77·2 per cent of the nickel and copper respectively. The tailing was 87·6 per cent minus 200 mesh.

Test No. 5

A test was next run increasing the potassium xanthate in the grinding circuit.

Charge and Reagents to Mill:

Ore	2,000 grammes
Water	1,000 c.c.
Soda ash	10·0 lb./ton
Potassium xanthate	0·4 "
Sodium sulphide	1·0 "
Pine oil to cell	0·075 "

Results:

Product	Weight, per cent	Assay, per cent		Distribution of metals, per cent		Per cent −200 mesh
		Ni	Cu	Ni	Cu	
Concentrate	10·15	4·92	1·36	60·0	72·3	....
Tailing	89·85	0·37	0·06	40·0	27·7	86·6

Test No. 6

In this run a change was made in the reagents added.

Charge and Reagents to Mill:

Ore	2,000 grammes
Water	1,000 c.c.
Soda ash	8·0 lb./ton
Amyl xanthate	0·4 "
Sodium pyrophosphate	0·6 "
Grinding time	30 minutes
Addition to cell, amyl alcohol	0·1 lb./ton

Results:

Product	Weight, per cent	Assay, per cent		Distribution of metals, per cent		Per cent −200 mesh
		Ni	Cu	Ni	Cu	
Concentrate	10·7	5·30	1·32	66·5	78·0	....
Tailing	89·3	0·32	0·04	33·5	22·0	87·9

CONCLUSIONS

The extensive surface alteration of the ore, particularly in the case of the nickel minerals, accounts for the low nickel recovery in the concentrates.

The use of increased quantities of xanthate improves the nickel recovery to a limited extent.

The ratio of concentration on the last two tests averaged 9·5 : 1·0.

Ore Dressing and Metallurgical Investigation No. 505

MOLYBDENITE ORE FROM PIGEON LAKE, GLOUCESTER COUNTY, NEW BRUNSWICK

Shipment. A shipment of 700 pounds of molybdenite ore was received on April 6, 1933, from The David Taylor Company, Inc., 52 Broadway, New York, for Peter Bourque, Dalhousie, New Brunswick.

Location of Property. The property from which the ore was taken is situated near Pigeon Lake, north half of Block 13, Range 14, between Papineau River and Little River, Gloucester County, N.B., on Claims Nos. 397-1, 397-2, 397-3, and 397-4.

Characteristics and Analysis. The ore is the typical flake molybdenite variety. The largest flakes observed were about ⅜ inch across.

The shipment assayed 5·02 per cent MoS_2.

Purpose of Experimental Tests. The purpose of this test was to concentrate the ore and produce a molybdenite concentrate that would be over 85 per cent MoS_2 and that did not contain impurities in excess of the amounts specified by The David Taylor Company. The specifications were as follows:—

	Per cent
Molybdenum sulphide	85·00 minimum
Copper	0·50 maximum
Bismuth	0·30 "
Phosphorus	0·10 "
Zinc	0·10 "
Lead	0·10 "

Experimental Test

The ore was concentrated by flotation in a manner similar to that described in Mines Branch "Memorandum Series No. 22," with the exception that mechanical cells were used in place of the Callow pneumatic type and that a sloping stationary screen was used instead of the Callow belt screen. The Callow belt screen is more efficient than the one used, but in this case the capacity of the belt screen was too great for the small amount of material to be concentrated.

The result of the test is summarized as follows:

Weight of ore concentrated	690·00 lb.		
Content of MoS_2 at 5·02 per cent			34·64 lb.
Concentrate recovered	31·75 lb.		
Content of MoS_2 at 88·1 per cent		27·96 lb.	
Tailing made	658·25 lb.		
Content of MoS_2 at 0·13 per cent		0·85 lb.	
Total MoS_2 accounted for in products of test		28·81 lb.	
Loss of MoS_2 due to spills and remaining in circuit after completion of test		5·83 lb.	
		34·64 lb.	34·64 lb.

Theoretical recovery or recovery equivalent to that obtained in actual mill operations—by formula $\frac{100\ (H - T)\ c}{H\ (c - T)} = 97 \cdot 0$ per cent

Actual recovery from weight of concentrate recovered $\frac{27 \cdot 96 \times 100}{34 \cdot 64} = 80 \cdot 7$ per cent

Analysis of Concentrate:

	Per cent
Molybdenum sulphide	88·1
Copper	0·01
Bismuth	0·03
Phosphorus	0·02
Zinc	nil
Lead	0·03
Arsenic	0·01

CONCLUSIONS

Molybdenite ore of this type can readily be concentrated by flotation with the production of a high-grade concentrate assaying 85 per cent to 90 per cent MoS_2, with high recovery of the molybdenite content of the ore.

Ore Dressing and Metallurgical Investigation No. 506

GOLD ORE FROM THE COLUMARIO GOLD MINES, LTD., USK, BRITISH COLUMBIA

Shipment. A shipment of three sacks of ore, weighing 200 pounds, was received March 28, 1933, from the property of the Columario Gold Mines, Ltd., Usk, British Columbia, with Head Office at Room 507, Confederation Life Building, Toronto, Ontario.

Characteristics of the Ore. The ore was examined microscopically and found to consist of massive sulphides of iron in a white quartz gangue. Iron pyrite is the main sulphide with rounded grains of chalcopyrite enclosed in the pyrite. No free gold was seen under the microscope. However, the spectroscope showed gold in the pyrite crystals.

EXPERIMENTAL TESTS

The lot was crushed, ground, and sampled, and found to contain:—

Gold	1·44	oz./ton
Silver	3·51	"
Copper	0·42	per cent
Lead	0·02	"
Arsenic	0·01	"
Antimony	nil	

The test work included amalgamation, concentration, and cyanidation, the results of which showed that approximately 13 per cent of the gold can be recovered by amalgamating minus 48-mesh material.

Cyanidation recovers 89 per cent of the gold, but owing to the copper in the ore a large amount of cyanide is consumed.

Straight flotation gives recoveries of 95·8 per cent of the gold, and 92·8 per cent of the silver, with a ratio of concentration of 2·2 : 1.

Flotation, with a ratio of concentration of 23 : 1, followed by cyanidation of the tailing recovers 93 per cent of the gold with a reasonable cyanide consumption.

AMALGAMATION AND CYANIDATION

Test No. 1

A sample of the ore was ground to pass 48 mesh and amalgamated. The tailing from this operation was cyanided 48 hours.

Results:

Feed..Au 1·44 oz./ton
Amalgamation tailing....................................Au 1·25 "
Recovery by amalgamation................................13·2 per cent

Cyanide tailing...Au 0·25 oz./ton
Recovery by amalgamation and cyanidation................82·6 per cent

Test No. 2

A similar test was made on a sample ground to pass 100 mesh.

Results:

Feed....................................Au 1·44 oz./ton; Ag 3·51 oz./ton
Amalgamation tailing....................Au 1·23 " Ag 3·35 "
Recovery................................Au 14·6 per cent; Ag 4·5 per cent

Cyanide tailing.........................Au 0·23 oz./ton; Ag 1·05 oz./ton
Total recovery..........................Au 84·0 per cent; Ag 70·1 per cent

CYANIDATION

Test No. 3

Samples of the ore were ground to pass various sized screens and cyanided for 48 hours. The solution was maintained at approximately 1 pound KCN per ton. Protective alkalinity was maintained with lime.

Mesh grind	Agitation, hours	Feed		Tailing		Extraction, per cent		Reagent consumption, lb./ton ore	
		Au oz./ton	Ag oz./ton	Au oz./ton	Ag oz./ton	Au	Ag	KCN	CaO
48	24	1·44	3·51	0·52	....	63·9	....	6·3	10·0
48	48	1·44	3·51	0·20	2·76	86·1	21·4	9·2	12·7
100	24	1·44	3·51	0·41	....	71·5	....	6·4	10·3
100	48	1·44	3·51	0·16	2·54	88·9	27·6	9·5	12·4
150	24	1·44	3·51	0·38	....	73·6	....	7·3	11·0
150	48	1·44	3·51	0·17	2·64	88·2	24·8	12·1	13·5
200	24	1·44	3·51	0·34	....	76·4	....	7·9	11·2
200	48	1·44	3·51	0·20	2·75	86·1	21·6	13·8	14·0

Copper in the ore is a strong cyanicide, resulting in an excessive consumption of this reagent.

FLOTATION

Test No. 4

A sample of the ore ground to pass 83 per cent minus 200 mesh was floated with 6 pounds soda ash, 0·20 pound sodium xanthate, and 0·08 pound pine oil per ton.

Results:

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
Feed	100·00	0·48	1·45	3·78	100·0	100·0	100·0
Concentrate	44·52	1·04	3·12	7·88	96·5	95·8	92·8
Tailing	55·48	0·03	0·11	0·49	3·5	4·2	7·2

Ratio of concentration, 2·2 : 1.

Flotation recovers 95·8 per cent of the gold and 92·8 per cent of the silver in a concentrate assaying 1·04 per cent copper, 3·12 ounces gold and 7·88 ounces silver per ton. The ratio of concentration is low. The tailing contains 0·11 ounce per ton in gold.

FLOTATION AND CYANIDATION

Test No. 5

An attempt was made in this test to remove the copper in as small a bulk of concentrate as possible and then to cyanide the flotation tailing.

The ore was ground with 8 pounds lime per ton of ore until 83 per cent passed 200 mesh; and floated with 0·20 pound sodium ethyl xanthate and 0·08 pound pine oil per ton.

The flotation tailing was filtered and cyanided for 48 hours with a 2 pound KCN solution, 1 : 3 dilution; 6 pounds lime per ton was added for protective alkalinity.

Results:

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
Feed	100·00	0·42	1·44	3·51	100·0	100·0	100·0
Concentrate	4·31	10·16	20·62	49·20	90·1	60·7	58·1
Tailing	95·69	0·05	0·60	1·60	9·9	39·3	41·9

Ratio of concentration, 23·2 : 1.

Cyanidation of Flotation Tailing

24-hr. agitation:

Tailing	Au	0·115 oz./ton;	Ag	0·31 oz./ton.
Reagent consumption	KCN	1·8 lb./ton;	CaO	4·0 lb./ton
Total recovery, flotation and cyanidation	Au	92·0 per cent;	Ag	91·1 per cent.

48-hr. agitation:

Tailing	Au	0·10 oz./ton;	Ag	0·30 oz./ton
Reagent consumption	KCN	1·95 lb./ton;	CaO	4·3 lb./ton.
Total recovery, flotation and cyanidation	Au	93·0 per cent;	Ag	91·4 per cent.

Test No. 6

A flotation test was made to note the effect of copper sulphate on the recovery. This test is the same as Test No. 5, with copper sulphate added.

Reagents:

	Lb./ton
Lime	8·0
Copper sulphate	1·0
Sodium ethyl xanthate	0·20
Pine oil	0·08

Product	Weight, per cent	Assays			Distribution of metals, per cent		
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
Feed	100·00	0·50	1·51	3·63	100·0	100·0	100·0
Concentrate	5·94	7·64	17·12	41·00	90·6	67·5	67·1
Tailing	94·06	0·05	0·52	1·27	9·4	32·5	32·9

Ratio of concentration, 16·8 : 1.

Copper sulphate does not increase the flotability of the copper. The same amount remains in the tailing. The recovery of gold and silver is increased with a lower ratio of concentration.

SUMMARY AND CONCLUSIONS

No free gold is visible in a microscopic examination, only 14·6 per cent being recovered by amalgamating ore ground minus 100 mesh.

Straight cyanidation, owing to copper in the ore, is not practicable. A consumption of approximately 10 pounds KCN per ton is indicated in Test No. 3. A tailing of over 0·15 ounce per ton is also shown.

Cyanidation of amalgamation tailing gives low recovery.

Concentration by flotation, Test No. 4, gives a ratio of concentration of 2·2 : 1 with a tailing of 0·11 ounce per ton, a recovery of 95·8 per cent of the gold. Owing to the large amount of sulphides in the sample furnished, the low ratio of concentration makes shipping of concentrate impracticable.

Removing a small bulk of high-grade concentrate, containing as much of the copper as possible, makes possible the cyanidation of the flotation tailing.

A recovery of 60·7 per cent of the gold, 58 per cent of the silver, and 90 per cent of the copper was obtained in a concentrate assaying 10 per cent copper, 20·6 ounces gold, and 49 ounces silver per ton. The ratio of concentration was 23·2 : 1. This makes a shipping product.

Cyanidation of the flotation tailing which has a content of 0·60 ounce gold reduces this to 0·10 ounce, an overall recovery of 93 per cent of the gold, 91·4 per cent of the silver. Cyanide consumed is 1·95 pounds KCN per ton of ore.

Microscopic examination of polished sections shows no free gold, but spectroscopic analysis shows that the pyrite contains gold. The presence of 0·10 ounce per ton in the final tailing is probably due to the sub-microscopic gold in the pyrite, liberated only by extremely fine grinding.

The process indicated by these tests on this class and grade of ore is fine grinding to free the grains of chalcopyrite enclosed within the pyrite, flotation in a high-lime circuit to produce a high-grade copper concentrate for shipment to a smelter, the de-watering of the flotation tailing, followed by cyanidation.

Care should be taken to be certain that the ore-body to be milled is of the same nature as the shipment tested. A material reduction in sulphide content might make the operation purely a concentration proposition with shipment to a smelter.

Ore Dressing and Metallurgical Investigation No. 507

HIGH-GRADE GOLD ORE FROM SWAYZE-DENYES GOLD AREA, NORTHERN ONTARIO

Shipment. A shipment of auriferous quartz ore, net weight 101 pounds, was received May 10, 1933. The sample was from Claim No. 22044, in Denyes Township, Sudbury mining division, Ontario, and was submitted by P. A. Dyment, President, Dyment Mining and Investments, Limited, 245 Carlaw Avenue, Toronto.

Characteristics. Gangue—milky quartz, locally stained brown by iron oxides.

Metallic Minerals. Pyrite, galena, sphalerite, native gold. Pyrite is common but not abundant, and occurs as disseminated cubes and grains in quartz. Galena and sphalerite both occur in irregular grains and patches, and in irregular veinlets in quartz. Native gold occurs in small grains and irregular veinlets in quartz, in some places intimately associated with the galena and sphalerite.

Purpose of Experimental Tests. This shipment of ore was made for the purpose of determining, *first,* if a high-grade product could be hand-picked from the run-of-mine ore, and *second,* provided the first operation could be performed, should this product be shipped to a smelter or could it be treated to better advantage at the property by some method such as barrel amalgamation.

Experimental Tests

The ore was crushed to minus 14 mesh and sampled by standard methods and assayed for gold and silver. Tests by barrel amalgamation and blanket concentration followed by amalgamation of the blanket concentrate were carried out.

The assay of the ore was as follows:—

Gold	18·25 ounces per ton.
Silver	3·08 " "

BARREL AMALGAMATION TESTS

Three tests were run on charges of 1,000 grammes minus 14 mesh each. The ore was ground in 500 c.c. water for 15 minutes, 30 minutes, and 45 minutes, respectively. Amalgamation was carried out using 50 grammes of mercury, and sodium cyanide 1 pound per ton for one hour. The amalgam was recovered in a hydraulic classifier. Screen tests were made on the tailing from each test. Assays for gold were made on the amalgam and the tailing in each test.

Results:

Test No. 1

Product	Grinding time	Assay, oz./ton		Recovery, per cent	
		Au	Ag	Au	Ag
Amalgam	15 min.	16·95	....	91·2	88
Tailing	"	1·63	0·37		

Screen Test on Amalgamation Tailing

Mesh	Per cent
+ 48	1·4
+ 65	6·7
+100	20·1
+150	19·2
+200	10·7
−200	41·9
Total	100·0

Test No. 2

Product	Grinding time	Assay, oz./ton		Recovery, per cent	
		Au	Ag	Au	Ag
Amalgam	30 min.	17·28	....	95·5	93·5
Tailing	"	0·81	0·20		

Screen Test on Amalgamation Tailing

Mesh	Per cent
+ 65	0·1
+100	2·9
+150	12·3
+200	17·3
−200	67·4
Total	100·0

Test No. 3

Product	Grinding time	Assay, oz./ton		Recovery, per cent	
		Au	Ag	Au	Ag
Amalgam	45 min.	18·49	....	96·9	95·1
Tailing	"	0·59	0·15		

Screen Test on Amalgamation Tailing

Mesh	Per cent
+100	0·6
+150	7·3
+200	10·7
−200	81·4
Total	100·0

The results of these tests indicate that fine grinding is essential and show that barrel amalgamation on high-grade ore will yield a recovery of over 96 per cent with a tailing around 0·5 ounce per ton. This tailing should be adaptable to cyanidation.

BLANKET CONCENTRATION

Three 1,000-gramme samples of the ore were ground for 15-, 30-, and 45-minute periods and the pulp run over a small corduroy blanket. The blanket concentrate was washed into a pebble jar and amalgamated for one hour with 30 grammes of mercury. The amalgam was recovered in a hydraulic classifier. The blanket tailing and amalgam tailing were filtered and assayed for gold and silver. Screen tests were made on both products and the amalgam from each test was also assayed for gold.

Test No. 4

Product	Weight, per cent	Assays, oz./ton		Distribution of metals, per cent		Grinding time
		Au	Ag	Au	Ag	
Blanket tailing	70·5	6·43	1·09	32·5	....	15 min.
Amalgam tailing	29·5	2·565	0·52	13·0	....	
Amalgam	....	....	....	54·5	47·7	

Screen Tests:

Blanket Tailing

Mesh	Per cent
+ 48	0·8
+ 65	5·7
+100	19·7
+150	20·2
+200	10·5
−200	43·1
Total	100·0

Amalgam Tailing

Mesh	Per cent
+ 48	0·1
+ 65	1·3
+100	10·0
+150	20·9
+200	14·6
−200	53·1
Total	100·0

Test No. 5

Product	Weight, per cent	Assays, oz./ton		Distribution of metals, per cent		Grinding time
		Au	Ag	Au	Ag	
Blanket tailing	82·8	1·80	0·37	10·1	....	30 min.
Amalgam tailing	17·2	0·68	0·18	4·0	....	
Amalgam	....	....	....	85·9	82·1	

Screen Tests:

Blanket Tailing

Mesh	Per cent
+100	1·6
+150	10·2
+200	15·7
−200	72·5
Total	100·0

Amalgam Tailing

Mesh	Per cent
+100	0·9
+150	6·5
+200	9·3
−200	83·3
Total	100·0

Test No. 6

Product	Weight, per cent	Assays, oz./ton		Distribution of metals, per cent		Grinding time
		Au	Ag	Au	Ag	
Blanket tailing	88·7	1·75	0·37	9·6	....	45 min.
Amalgam tailing	11·3	0·965	0·25	5·3	....	
Amalgam	....	....	....	85·1	79·2	

Screen Tests:

Blanket Tailing

Mesh	Per cent
+100	0·1
+150	2·7
+200	6·8
−200	90·4
Total	100·0

Amalgam Tailing

Mesh	Per cent
+100	0
+150	1·9
+200	4·2
−200	93·9
Total	100·0

Results of Blanket Concentration Tests

Fine grinding, about 90 per cent minus 200 mesh, shows the best results. A high-grade blanket concentrate is obtained from which over 85 per cent of the gold is obtained by barrel amalgamation. The blanket tailing is, however, high.

CONCLUSIONS

The best results are obtained on straight barrel amalgamation with the ore ground to 80 per cent minus 200 mesh. The tailing from either the straight amalgamation or the blanket concentration could be treated later by cyanidation.

Ore Dressing and Metallurgical Investigation No. 508

GOLD ORE AND MILL TAILING FROM PARKHILL GOLD MINES, LTD., WAWA, ONTARIO

Shipment. A shipment of one bag of ore weighing approximately 100 pounds and a bottle containing wet mill tailing was received May 10, 1933, from the Parkhill Gold Mines, Ltd., Wawa, Ont. The shipment was made to determine the effect of various strengths of cyanide solution and time of agitation on gold extraction.

Experimental Tests

MINE RUN OF ORE

A series of tests was made on the ore ground minus 200 mesh, agitated for 12, 24, 36, 48, and 72 hours with 1·0 pound and 3·0 pounds KCN per ton solutions, 1 : 3 dilution.

Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton ore	
				KCN	CaO
1·0-lb. KCN solution:					
12	1·14	0·05	95·6	0·6	3·5
24	1·14	0·035	96·9	0·75	3·4
36	1·14	0·04	96·5	1·05	3·6
48	1·14	0·065	94·3	1·05	3·6
72	1·14	0·035	96·9	2·59	5·3
3·0-lb. KCN solution:					
12	1·14	0·03	97·4	1·05	3·4
24	1·14	0·05	95·6	1·65	3·4
36	1·14	0·055	95·2	1·20	3·4
48	1·14	0·04	96·5	1·65	3·5
72	1·14	0·015	98·7	1·63	5·4

A test was made on the ore ground minus 200 mesh, agitated 72 hours with a 3·0 pound KCN solution, 1 : 3 dilution; 1·5 pound lubricating oil per ton was added to the test to determine if this substance affected extraction.

Feed	1·14 ounce gold per ton
Tailing	0·015 " "
Extraction	98·7 per cent.

MILL TAILING

The sample received was filtered and the solution assayed. It contained 0·6 pound KCN, 0·3 pound CaO, and 0·15 ounce gold per ton.

After filtering, the sample was water-washed and assayed and found to contain 0·025 ounce gold per ton.

A screen analysis showed that the mill tailing had the following grind:—

Mesh	Weight, per cent
+150	1·1
−150+200	6·0
−200	92·9
	100·0

Test No. 1

Samples of the water-washed tailing were agitated with 5·0 pound KCN solution, 1 : 3 dilution, and 4 pounds lime per ton for 24 and 48 hours.

Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton ore	
				KCN	CaO
24	0·025	0·01	60·0	0·6	3·0
48	0·025	0·0066	73·6	0·9	3·1

Test No. 2

Samples of the tailing ground to pass 200 mesh were agitated for 24 and 48 hours with 1·0 pound and 5 pounds KCN solutions, 1 : 3 dilution, and with 4 pounds of lime per ton of ore.

1-lb. KCN solution

Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton ore	
				KCN	CaO
24	0·025	0·01	60·0	0·3	3·5
48	0·025	0·0066	73·6	0·3	3·5

2-lb. KCN solution:

Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton ore	
				KCN	CaO
24	0·025	0·008	68·0	0·45	3·5
48	0·025	0·005	80·0	0·45	3·5

5-lb. KCN solution:

Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton ore	
				KCN	CaO
24	0·025	0·008	68·0	0·9	3·5
48	0·025	0·005	80·0	0·9	3·4

As the sample furnished was in contact with cyanide solution from the time it was taken at the mill until it was filtered, the gold in the residue is probably lower than in the original sample when freshly taken.

The tests on the mine ore indicate that some precipitation of gold from solution takes place during cyanidation.

Ore Dressing and Metallurgical Investigation No. 509

GOLD-SILVER-LEAD-ZINC ORE FROM THE YANKEE GIRL MINE, AT YMIR, BRITISH COLUMBIA

Shipment. The shipment consisted of a sample of ore contained in five sacks, net weight 491 pounds, and was received February 25, 1933.

The sample was submitted by E. P. Crawford, Yankee Girl Mine, Ymir, B.C.

Characteristics of the Ore. The ore represented in this shipment consists of milky to greyish quartz through which the sulphides occur as discontinuous stringers and fine disseminations. The metallic minerals present are pyrite, sphalerite, galena, pyrrhotite, chalcopyrite, and an unknown cobalt-bearing mineral.

Spectrographic analyses indicate that the pyrite contains an appreciable quantity of gold, and as no free gold was observed, it is present in this mineral in either so finely-divided a state as to be sub-microscopic or in a condition of solid solution. A sufficient number of specimens were not examined to determine whether or not a portion of the gold in the ore occurs in the native state.

An average analysis of the ore was as follows:—

Gold	0·625	oz./ton
Silver	3·00	"
Lead	3·14	per cent
Zinc	5·30	"
Arsenic	0·28	"

Experimental Tests

A series of small-scale cyanidation tests was made on this ore to determine the feasibility of this process for extracting the gold. By cyaniding the ore direct the maximum gold extraction obtained was approximately 87 per cent, but by first grinding the ore with lime and washing it with water and then cyaniding the extraction was raised to 95 per cent. Unfortunately, however, a cycle test showed that the solution fouled rapidly and the tailing assays mounted steadily with each contact of the solution with a fresh lot of the ore in spite of prewashing with lime. It will, therefore, be necessary to discard a large portion of the solution, as much as 40 per cent of it, each time it passes through the circuit in order to prevent too great a drop in extraction.

CYANIDATION

Tests Nos. 1, 2, 3, 4, and 11

In this series of tests the ore was ground in jar mills to the following percentages through 200 mesh respectively, 62·1, 73·2, 86·5, 90·0, and 94·3. In each case the pulp was agitated for 48 hours at 2·5 : 1 dilution, the solution strength being 2 pounds per ton in KCN. Protective alkalinity was maintained by the addition of lime.

Summary:

Feed sample, Au 0·625 oz./ton

Test No.	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
1	0·165	73·6	1·36	6·15
2	0·09	85·6	1·38	6·30
3	0·09	85·6	1·62	6·50
4	0·085	86·4	1·63	6·75
11	0·08	87·2	1·88	6·75

FLOTATION AND CYANIDATION

Test No. 6

In this test a sample of the ore was floated and the flotation concentrate cyanided.

Charge to Ball Mill:

Ore..2,000 grammes
Water......................................1,500 "
Na_2CO_3.....................................3·0 lb./ton

This charge was ground for 25 minutes.

Reagents to Cell:

Potassium amyl xanthate....................0·2 lb./ton
Pine oil...................................0·1 "

A sample of the concentrate was agitated in cyanide solution, 5 pounds per ton KCN, for 48 hours.

All products were assayed for gold, silver, lead, and zinc.

Summary:

Product	Weight, per cent	Assays				Distribution of metals, per cent			
		Au oz./ton	Ag oz./ton	Pb per cent	Zn per cent	Au	Ag	Pb	Zn
Concentrate	36·3	1·77	8·66	9·02	14·68	98·05	97·2	97·3	97·8
Concentrate cyanided	36·3	0·36	5·71	9·12	14·78	....	....	....	....
Tailing	63·7	0·02	0·14	0·14	0·19	1·95	2·8	2·7	2·2
Feed (cal.)	100·0	0·655	3·23	3·36	5·45	....	....	....	....

Ratio of concentration	2·72 : 1
Per cent total gold extracted by cyanidation	78·11
" silver " "	33·11

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

Mesh	Weight, per cent	Cumulative weight, per cent
+ 65	0·1	0·1
− 65+100	1·7	1·8
−100+150	7·0	8·8
−150+200	15·0	23·8
−200	76·2	100·0
Total	100·0	

Tests Nos. 7, 8, 9, 10, and 12

Believing that soluble sulphides might be responsible for the low extraction obtained in Tests Nos. 1, 2, 3, 4, and 11, this series of tests was made under the same conditions but with PbO added in the proportion of 1·0 pound per ton ore.

Summary:

Feed, Au 0·625 oz./ton

Test No.	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
7	0·11	82·4	2·13	5·46
8	0·11	82·4	2·11	5·50
9	0·12	80·8	2·11	5·67
10	0·12	80·8	2·36	6·24
12	0·16	74·4	1·88	6·75

Tests Nos. 14 to 17

This series of tests was made to determine whether or not re-precipitation of the gold was taking place. The samples were each ground for 30 minutes in a ball mill and then agitated for varying periods. The tailing assays dropped steadily as the period of agitation was increased and this is reliable evidence that re-precipitation is not taking place.

Summary:

Test No.	Period of agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
14	16	0·15	76·0	1·88	6·25
15	20	0·13	79·2	1·88	6·20
16	24	0·125	80·0	2·10	6·50
17	32	0·105	83·2	2·10	6·25

Tests Nos. 19 to 27

In this series of tests several different schemes were tried, including prewashing and the addition of oxidizing reagents.

Particulars of each test will be given in detail in the following paragraphs.

Test No. 19. In this test the ore was ground in water for 30 minutes in a ball mill and then agitated in an open vessel for 48 hours at 2·5 : 1 dilution. The solution was kept at 2 pounds per ton KCN and protective alkalinity was maintained by the addition of lime. It was hoped that the more thorough aeration resulting from the use of the mechanical agitator would destroy the reducing power that was found by chemical determination to exist in pregnant solutions from previous tests. This did not take place, however, and the tailing assay was high. The results of this test may be summed up as follows.

Results:

Tailing assay, Au	0·18 oz./ton
Extraction	71·2 per cent
KCN consumed	2·9 lb./ton ore
CaO consumed	8·6 "

Test No. 20. In this test 1,000 grammes of the ore was ground for 30 minutes and then agitated for 24 hours in cyanide solution. The pulp was then filtered and washed with fresh cyanide solution. Then it was repulped in more fresh solution and agitated for another 24 hours.

Results:

Tailing assay, Au	0·035 oz./ton
Extraction	94·4 per cent
KCN consumed	2·1 lb./ton ore
CaO consumed	6·5 "

Test No. 21. In this test a sample of the ore was ground for 30 minutes and agitated in cyanide solution for about 18 hours. Then sodium peroxide was added, 0·5 pound per ton ore, and the agitation continued for another 30 hours.

Results:

Tailing assay, Au	0·09 oz./ton
Extraction	85·6 per cent
KCN consumed	2·7 lb./ton ore
CaO consumed	6·25 "

Test No. 22. In this test the ore was ground for 30 minutes and washed with water before treatment with cyanide solution. The pulp was agitated for 48 hours in solution containing 2 pounds per ton in KCN.

Results:

Tailing assay, Au	0·09 oz./ton
Extraction	85·6 per cent
KCN consumed	2·25 lb./ton ore
CaO consumed	6·50 "

Test No. 23. In this test lead acetate was added to the ore while it was being ground in the ball mill. After 30 minutes' grinding the ore was agitated for 48 hours in cyanide solution, 2 pounds per ton in KCN. Lead acetate, being soluble, was used in place of insoluble lead oxide in this test to see if it would more effectively precipitate soluble sulphides and thus increase extraction.

Results:

Tailing assay, Au	0·095 oz./ton
Extraction	84·8 per cent
KCN consumed	2·25 lb./ton ore
CaO consumed	6·60 "

Test No. 24. In this test the ore was ground for 30 minutes with caustic soda, 6 pounds per ton ore, added. The pulp was then filtered and washed with water and repulped in cyanide solution 2 pounds per ton KCN. The pulp was agitated for 48 hours.

Results:

Tailing assay, Au	0·11 oz./ton
Extraction	82·4 per cent
KCN consumed	1·80 lb./ton ore
CaO consumed	5·5 "

Test No. 25. In this test the ore was ground for 30 minutes in a ball mill with lime added at the rate of 8 pounds per ton ore. After grinding the pulp was filtered and washed with water. The cake was then repulped in cyanide solution, 2 pounds per ton KCN, and agitated for 48 hours.

Results:

Tailing assay, Au	0·03 oz./ton
Extraction	95·2 per cent
KCN consumed	1·40 lb./ton of pre-washed ore
CaO consumed	1·1 " "

Test Nos. 26 and 27. These two tests were made using strong cyanide solution, 5 pounds per ton KCN. The ore was ground for 30 minutes in a ball mill and agitated for 48 hours in cyanide solution without any pre-washing.

Results:

Test No.	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
26	0·055	91·2	4·07	6·00
27	0·04	93·6	3·32	6·00

Tests Nos. 28 to 30

In this series of tests the ore at minus 14 mesh was agitated with lime water for 40 minutes, then filtered, washed, and ground in water for 15, 20, and 30 minutes respectively. The ground pulp was then agitated in cyanide solution, 2 pounds per ton KCN, for 48 hours. This was done to see if the prewashing with lime could be effectively done in the coarser sizes of first-stage grinding, thus allowing the finer second-stage grinding to be done in cyanide solution.

Results:

Feed sample, Au 0·625 oz./ton

Test No.	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
28	0·105	83·2	4·2	2·5
29	0·095	84·8	4·5	2·5
30	0·055	81·2	4·2	2·75

Cycle Tests Nos. 31 to 36

In this series of tests the ore was ground in lime water for 30 minutes, filtered, and washed. Fresh cyanide solution, 2 pounds per ton KCN, was put on the first lot of ore and the pulp agitated for 48 hours. The solution was then filtered off, and made up to proper strength and volume and used to treat the next batch of ore. This was continued until six batches of ore had been treated with one batch of solution. This was done to test the fouling properties of the ore after prewashing with lime.

Results:

Test No.	Tailing assay, Au, oz./ton	Extraction, per cent
31	0·035	94·4
32	0·050	92·0
33	0·095	84·8
34	0·12	80·8
35	0·19	69·6
36	0·16	74·4

Owing to the nature of the tests accurate figures for cyanide and lime consumption could not be obtained, but it remained at approximately 1·5 pound KCN per ton of ore and 1·0 pound lime per ton of prewashed ore.

Test No. 37

In this test the ore was ground in lime water for 30 minutes, the pulp diluted to 2·5 : 1 and agitated in a closed container for six hours without cyanide. At the end of six hours the solution was made up to 2 pounds per ton in KCN and agitation continued for another 48 hours. The cyanide tailing was then assayed for gold.

Results:

Tailing assay, Au	0·13 oz./ton
Extraction	79·2 per cent
KCN consumed	1·15 lb./ton ore
CaO consumed	6·75 "

Test No. 38

In this test the ore was ground with lime in a ball mill for 30 minutes. It was then diluted to 6 : 1 and transferred to a flotation cell and agitated for six hours where it was very thoroughly aerated. The pulp was then thickened to 2 : 1 dilution and agitated for 48 hours in cyanide solution, 2 pounds per ton in KCN.

Results:

Tailing assay, Au	0·045 oz./ton
Extraction	92·8 per cent
KCN consumed	1·8 lb./ton ore
CaO consumed	18·8 "

CONCLUSIONS

The results of the foregoing tests show clearly that this ore will have to be given a pre-treatment with lime before it comes in contact with the cyanide solution. The most effective method as shown by the small-scale test work is to grind a mixture of ore and lime in water, then filter and wash the pulp with water and treat it with fresh cyanide solution. Perhaps this cannot be economically carried out in detail in practice but a flow-sheet approaching the above conditions as nearly as possible, in keeping with practical economy, is suggested here below.

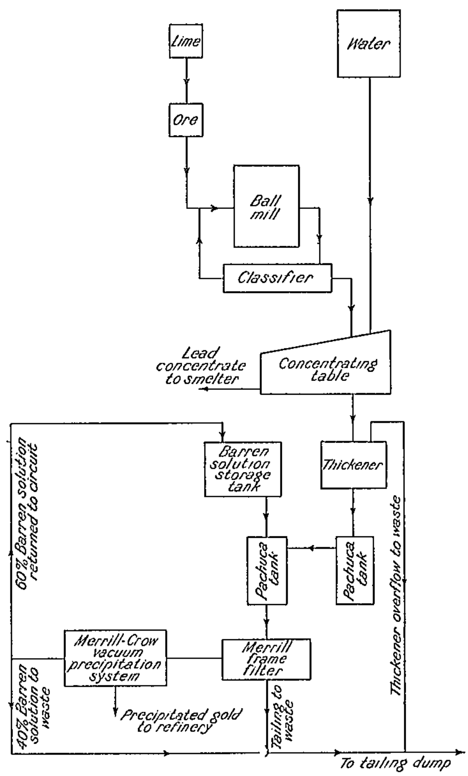


Figure 5. Proposed flow-sheet for cyaniding ore from Yankee Girl mine, Ymir, B.C.

The ore mixed with lime will be ground in water in a ball mill in closed circuit with a classifier. Two-stage grinding can be practised if desirable. The classifier overflow at about 5 : 1, or 6 : 1 dilution will be passed over concentrating tables where a lead concentrate will be taken off and at the same time the ore will be given at least a partial wash. The table tailing will then go to a thickener, the underflow from which should be at least 50 or 60 per cent solids. The thickened pulp, with sufficient lime added to maintain alkalinity, will then be agitated and thoroughly aerated for 6 hours in a Pachuca tank from which it will discharge into another Pachuca tank, where the cyanide will be introduced and agitation carried on for 48 hours or as long as may be found necessary. The pulp will then be filtered in a Merrill frame filter, the pregnant solution going to a Merrill-Crowe precipitation unit. Enough barren solution will have to be discarded to compensate for the water coming into the agitation circuit with the thickener underflow. The remainder of the barren solution, a portion of which will be used to wash the filtered pulp and reunited with the main body, will be made up to strength and returned to the first cyanide agitator. A pressure filter is recommended because it will give maximum washing with a minimum of solution, and this is important when the volume of solution has to be kept within limits with water constantly being added. The lead concentrate will be sold to a smelter and treated for its lead content.

The success of this method will depend chiefly on thorough washing and aeration of the pulp before it comes in contact with the cyanide solution and on the absence of fouling matter in the cyanide solution. If no other means of getting rid of the fouling matter can be found, the solution will have to be wholly or partly discarded at regular intervals. The amount to be disposed of will have to be determined by experiment, the minimum being the equivalent of the water coming into the circuit with the thickener underflow.

This method is virtually the same as that now in use at the Dome Mines, South Porcupine, Ontario.

Ore Dressing and Metallurgical Investigation No. 510

GOLD ORE FROM ALCONA GOLD MINES, LTD., AT ALCONA, ONTARIO

Shipment. The shipment consisted of two sacks of ore, net weight approximately 200 pounds, and was received March 23, 1933. The sample was submitted by Chas. A. Richardson, Manager, Alcona Gold Mines, Ltd., Alcona, Ontario.

Characteristics of the Ore. The gangue is chiefly white quartz. The metallic minerals present are pyrite, chalcopyrite, sphalerite, galena, and native gold.

The pyrite is commonly disseminated as comparatively coarse grains in the quartz. The remaining sulphides form relatively fine stringers in the quartz and are mutually associated. The chalcopyrite, which is not abundant, is associated with rare sphalerite which contains numerous tiny dots of chalcopyrite. Galena is likewise comparatively rare, and is associated mostly with the chalcopyrite and sphalerite, but may occur alone in the quartz.

The only native gold observed was present as small irregular grains enclosed in galena. Its behaviour with etching reagents was misleading, because it was not attacked by KCN; the effect on the polished surface was only to form an extremely thin yellowish film, which was perhaps responsible for protecting the metal against attack. Subsequent microchemical and spectrographic analyses, however, prove these grains to be impure gold.

An average analysis of the sample was as follows:—

Gold	2·94	oz./ton
Silver	10·01	"
Copper	0·43	per cent
Lead	3·10	"
Zinc	0·50	"
Iron	5·10	"
Sulphur	5·00	"

Experimental Tests

A series of small-scale tests was made on the ore to determine the best method of recovering the gold and silver from it. The work included tests by cyanidation, amalgamation, and concentration. About 20 per cent of the gold is recoverable by amalgamation, and from 10 to 15 per cent by cyanidation but with a rather high consumption of lime and cyanide. About 86 per cent of the gold and silver can be recovered in a high-grade flotation concentrate with a ratio of concentration of 6 : 1 or 7 : 1. The flotation tailing will assay about 0·5 ounce per ton in gold and this cannot be appreciably reduced either by amalgamation or tabling, although a small amount of high-grade table concentrate may be produced. Details of the tests follow.

AMALGAMATION AND FLOTATION

Tests Nos. 1 to 4

In this series of tests four lots of the ore at minus 14 mesh were ground in jar mills for 15, 20, 25, and 30 minutes respectively. The pulp was then amalgamated with mercury for 30 minutes and the amalgamation tailings were floated with the following reagents:—

Na_2CO_3	10·0 lb./ton
Potassium amyl xanthate	0·2 "
Pine oil	0·1 "

The flotation concentrates and tailings were assayed for gold and the amalgamation tailing assay was calculated back from these.

Results:

Feed sample, Au 2·94 oz./ton.

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Recovery by amalgamation, per cent	Recovery in flotation concentrate
1	Flotation concentrate	13·7	12·71	20·1	59·1
	Flotation tailing	86·3	0·71		
	Amalgamation tailing (cal.)	100·0	2·35		
2	Flotation concentrate	12·7	12·87	19·4	55·3
	Flotation tailing	87·3	0·845		
	Amalgamation tailing (cal.)	100·0	2·37		
3	Flotation concentrate	10·9	13·98	19·7	51·8
	Flotation tailing	89·1	0·94		
	Amalgamation tailing (cal.)	100·0	2·36		
4	Flotation concentrate	14·8	11·42	18·7	57·5
	Flotation tailing	85·2	0·81		
	Amalgamation tailing (cal.)	100·0	2·39		

CYANIDATION

Tests Nos. 5 to 8

In this series of tests four lots of the ore were ground in jar mills for the same periods of time as those in Tests Nos. 1 to 4. The pulp was then agitated for 48 hours at 2·5 : 1 dilution in cyanide solution. The strength of the solution was kept at 2 pounds per ton in KCN by additions of the salt from time to time. Protective alkalinity was maintained by the addition of lime. A screen test showed the grinding to be 52·9, 67·0, 71·0, and 80·4 per cent minus 200 mesh in Tests Nos. 5 to 8 respectively.

Results:

Feed sample, Au 2·94 oz./ton; Ag 10·01 oz./ton

Test No.	Tailing assay		Extraction, per cent		Reagents consumed, lb./ton	
	Au, oz./ton	Ag, oz./ton	Au	Ag	KCN	CaO
5	2·57	7·10	12·6	29·1	8·6	16·9
6	2·50	6·95	15·0	30·6	9·6	17·0
7	2·60	7·02	11·6	29·9	9·6	17·2
8	2·70	7·08	8·2	29·3	9·1	17·2

The high consumption of cyanide is no doubt due to the copper contained in the ore and the high lime consumption is due to its oxidized condition.

Failing to obtain any worthwhile recovery by amalgamation or cyanidation, an attempt was made to concentrate the ore by flotation. A series of small-scale tests was made for this purpose using different reagent combinations and different grinding. In one case the flotation tailing was tabled, and in another case part of the flotation tailing was amalgamated and cyanided, and part of it cyanided only. Selective flotation of the sulphides, tried in Test No. 10, proved a failure, so the concentrates were mixed together and assayed as a bulk concentrate.

FLOTATION

Test No. 9

The ore at minus 14 mesh was ground for 40 minutes in a jar mill, the charge being as follows:—

Ore	2,000 grammes
Water	1,500 "
Soda ash	20·0 lb./ton
Aerofloat No. 25	0·07 "

Reagents to Cell:

Potassium amyl xanthate	0·01 lb./ton
Pine oil	0·05 "

The products were assayed for gold, silver, copper, lead, and zinc.

Results:

Product	Weight, per cent	Assay					Distribution of metals, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Concentrate	16·2	15·82	57·08	2·80	11·7	2·80	84·8	85·9	87·1	63·5	91·5
Tailing	83·8	0·55	1·81	0·08	1·30	0·05	15·2	14·1	12·9	36·5	8·5
Feed (cal.)	100·0	3·02	10·76	0·52	2·98	0·50	......	......	......	......	......

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

Mesh	Weight, per cent
+100	0·2
−100+150	3·3
−150+200	9·9
−200	86·6
Total	100·0

FLOTATION

Test No. 10

Selective flotation was attempted in this test but did not prove successful, so the concentrates were all united and treated as a bulk concentrate.

The ore at minus 14 mesh was ground for 40 minutes in a jar mill, the charge being as follows:—

Ore	2,000 grammes
Water	1,500 "
Na_2CO_3	30·0 lb./ton
NaCN	0·2 "
Aerofloat No. 25	0·07 "

Reagents to Cell:

Cu, Pb float—Cresylic acid..0·07 lb./ton
Zn float—$CuSO_4$..1·0 "
Sodium Aerofloat..0·1 "
Cresylic acid..0·07 "
Re-float—Potassium amyl xanthate..0·2 "
Pine oil..0·05 "

The products were assayed for gold, silver, copper, lead, and zinc.

Results:

Product	Weight, per cent	Assay					Distribution of metals, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Concentrate......	13·5	17·32	65·18	3·26	13·81	3·30	86·5	87·3	92·7	64·2	91·2
Tailing...........	86·5	0·42	1·48	0·04	1·32	0·05	13·5	12·7	7·3	35·8	8·8
Feed (cal.).......	100·0	2·70	10·08	0·47	2·91	0·49	......	......	......	......	......

FLOTATION WITH CYANIDATION AND AMALGAMATION

Test No. 11

In this test the reagent combination was the same as that for Test No. 9 except that 1·0 pound per ton $CuSO_4$ was added to the cell near the end of the operation to see if it would reduce the tailing assay.

The ore at minus 14 mesh was ground in a jar mill for 40 minutes as in Test No. 9, the charge being as follows:—

Ore..2,000 grammes
Water..1,500 "
Na_2CO_3..20·0 lb./ton
Aerofloat No. 25..0·07 "

Reagents to Cell:

Potassium amyl xanthate..0·1 lb./ton
Pine oil..0·05 "
$CuSO_4$..1·0 "

The concentrate and tailing were assayed for gold and silver. A sample of the tailing was cyanided and another sample was amalgamated and then cyanided, the tailings in these three cases being assayed for gold only.

Summary:

Product	Weight, per cent	Assay		Distribution of metals, per cent		Reagents, lb./ton	
		Au, oz./ton	Ag, oz./ton	Au	Ag	KCN	CaO
Concentrate..........................	20·3	11·82	46·86	85·8	87·0	....	....
Tailing..............................	79·7	0·50	1·78	14·2	13·0	....	....
Feed (cal.)..........................	100·0	2·80	10·93	100·0	100·0	....	....
Amalgamation tailing..............	79·7	0·38	....	10·8	....	....	....
Cyanide tailing......................	79·7	0·205	....	5·8	....	3·2	9·8
Amalgamation tailing cyanided.....	79·7	0·125	....	3·5	....	3·8	10·2

	Per cent
Net recovery of gold by amalgamating flotation tailing..................	3·4
Net recovery of gold by cyaniding flotation tailing.......................	8·4
Net recovery by cyanidation of amalgamation tailing....................	7·3

FLOTATION AND TABLING

Test No. 12

In this test the flotation tailing was tabled to see if its gold content could be reduced in this way.

The ore at minus 14 mesh was ground for 40 minutes in a jar mill, the charge being as follows:—

Ore	2,000	grammes
Water	1,500	"
Na_2CO_3	20·0	lb./ton
Aerofloat No. 25	0·07	"

Reagents to Cell:

Potassium amyl xanthate	0·01	lb./ton
Pine oil	0·05	"

All products were assayed for gold and silver.

Results:

Product	Weight, per cent	Assay		Distribution of metals, per cent	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Flotation concentrate	15·9	15·93	55·52	84·6	84·1
Table concentrate	1·8	7·36	11·18	4·4	1·9
Table sand tailing	65·7	0·375	1·36	8·2	8·5
Table slime tailing	16·6	0·50	3·47	2·8	5·5
Feed (cal.)	100·0	2·99	10·50	100·0	100·0

FLOTATION

Test No. 13

Extremely fine grinding was tried in this test as a means of obtaining a clean flotation tailing.

The ore at minus 14 mesh was ground for one hour in a jar mill, the charge being as follows:—

Ore	2,000	grammes
Water	1,500	"
Na_2CO_3	20·0	lb./ton
Aerofloat No. 25	0·07	"

Reagents to Cell:

Potassium amyl xanthate	0·01	lb./ton
Pine oil	0·05	"

The flotation tailing was screened on a 200-mesh screen, the plus and minus along with the concentrate being assayed for gold and silver.

Results:

	Weight, per cent	Assay		Distribution of metals, per cent	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Concentrate	15·0	15·6	63·62	78·1	83·9
+200-mesh tailing	2·9	7·1	2·75	6·9	0·7
−200-mesh tailing	82·1	0·55	2·14	15·0	15·4
Feed (cal.)	100·0	3·00	11·40	100·0	100·0

Screen Test, Flotation Tailing

Mesh	Weight, per cent
+200	3·4
−200	96·6

CONCLUSIONS

The ore, as represented by the sample submitted, does not respond readily to any ordinary method of treatment. The sample is highly oxidized and therefore consumes a relatively large quantity of lime during cyanidation and owing to its copper content consumes an abnormal amount of cyanide. Extraction by cyanidation is very poor, as may be seen from the results of Tests Nos. 5 to 8. Some grains of free gold were panned from a sample of the ore and although they appeared to be clean and of a nice bright yellow colour they showed no reaction whatever other than the formation of a thin yellowish film when treated with cyanide solution.

Maximum recovery by amalgamation was 20·1 per cent of the gold and this dropped slightly with finer grinding to 18·7 per cent at 80·4 per cent through 200 mesh.

High-grade flotation concentrate can be produced with a fair ratio of concentration, but it has not been possible to produce a flotation tailing lower than 0·5 ounce per ton in gold. This is probably due to the presence of limonite resulting from alteration of the pyrite and containing the gold originally in the pyrite from which it was formed.

A good recovery was obtained in Test No. 11 by floating the ore and then amalgamating and cyaniding the flotation tailing. The final tailing assayed 0·125 ounce per ton in gold, which is high, and there still remains the problem of treating the flotation concentrate to recover the gold from it. Although the overall recovery in this test was 96·5 per cent, this figure might be greatly reduced if ore of the same characteristics but lower in grade were being treated.

It is, therefore, suggested that another sample of ore, taken from depth to avoid the oxidized condition, and more representative of the grade to be milled, be submitted for further test work.

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