CANADA
DEPARTMENT OF MINES AND RESOURCES

MINES AND GEOLOGY BRANCH
BUREAU OF MINES

INVESTIGATIONS IN ORE DRESSING AND METALLURGY

(Testing and Research Laboratories)

January to June, 1938

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I

REVIEW OF INVESTIGATIONS

C. S. Parsons
Chief of Division of Metallic Minerals

A review of the investigations undertaken and reported on during the first half of the year 1938, January to June inclusive, shows that seventy-five reports of investigation were issued to the parties concerned, sixteen of which are printed in full and are shown in this report in Section II; and the remainder are listed by title only, in Section III. Printing in full is confined only to such reports as may appear of general interest to the mining industry at large; Section IV contains a brief résumé of the special investigations and research completed, in progress, or under consideration.

Much of the work of the Research Section has been connected with special problems arising in the investigations on samples submitted by the industry. Other work that may be considered as of a more general or fundamental character will, if of interest, be published upon completion, in some separate form.

The functions of the Division are twofold, namely, to investigate the character of ore-bodies and deposits of metallic minerals in reference to their exploration and development; to investigate methods of winning the ores, of processing, utilizing, and marketing the ores, metals, alloys, and metallic products and other related matters; and to maintain ore dressing and metallurgical laboratories and facilities for such mineralogical, physical, chemical, mechanical, and metallurgical investigations, tests and researches as are found expedient to determine the character and treatment of ores, improve plant practices, increase recoveries, improve the quality of metallic products, establish new operations, and in general, to aid the mining and metallurgical industry in Canada.

The present staff is sufficient only to handle investigations and research on work connected with ore dressing and metallurgical problems, submitted by the industry.

The investigations shown in this report required some 8,721 chemical and assay determinations by the chemical laboratories' staff, and the mineragraphic laboratory prepared and examined 361 polished sections of ores and products. In addition, the mineragraphic laboratory prepared many photomicrographs for attachment to reports going to the interested parties. This laboratory also made 34 spectrographic analyses.

The metallurgical laboratory, in addition to carrying out special investigational and research projects, performed a considerable amount of work for other Government departments. An important project in this connection was field investigation into the die-casting process as related to aluminium alloys. The Naval and Air Services and the Departments of Transport and Public Works have also had occasion to submit numerous problems, both material and consultative, in the course of the period reviewed.

An increasing amount of correspondence is reaching the Division relative to plant operating problems, treatment processes for ores, alloys, chemical and metallurgical problems and enquiries as to suitable types of equipment for certain mill operations.

The chemical laboratories are also frequently requested to make check analyses and assays for certain of the operating mining and other laboratories, and to provide methods of analysis for the purpose of controlling and determining mill effectiveness.

Several senior members of the staff have served as members of special committees under the sponsorship of the National Research Council.

Investigations. Of the seventy-five reports of major investigation, forty-one were concerned with ore treatment, nine embraced special microscopic examinations and twenty-five were related to metallurgical products or problems. Thirty-five investigations involved treatment of gold ores originating provincially as follows: Nova Scotia 2, Quebec 7, Ontario 11, Manitoba 3, British Columbia 9, and Northwest Territories 3. Particular attention is directed to Investigation No. 741 on "Gold-Silver Ore from Berens River Mines, Limited, at Favourable Lake, Ontario". This ore is of a particularly refractory type and exceedingly sensitive to operating conditions, particularly with respect to alkalinity control.

Reference may also be directed to the ores received from the new fields in the Northwest Territories. The results of two of these investigations are published in this report, namely, No. 737 on "Gold Ore from the Camlaren mine at Gordon Lake, N.W.T." and No. 742 on "Gold-silver ore from the Negus Mines, Limited, Yellowknife River Area, N.W.T.".

Mineragraphic Laboratory. The following is a summary of the work in the Mineragraphic Laboratory:

A. Investigations:	
Examination of gold ores	28
Examination of base metal ores	1
Miscellaneous examinations	3
Special studies	9
Total	41
B. Spectrographic analyses	34
C. Polished sections prepared:	
For Mineragraphic Laboratory	352
For others	9
Total	361

Chemical Laboratories. The following is a summary of the work done in the chemical laboratories:

The staff of the chemical laboratories analysed 3,603 samples of ores, minerals, and metal products and complete reports were issued thereon.

This work included a total of 8,721 chemical and assay determinations, 47 different mineral constituents being involved.

The samples were made up from the following:

Metallic ore mill products	2,916
Pyrometallurgical Laboratory	123
Fuel Testing Laboratory	29
Industrial Minerals Division mill products	261
Field samples	67
Custom assays	207
Total samples	3,603
Total determinations	8,721
Total gold assays	2,798
Total silver assays	749

Staff. The work on ore dressing was carried out under the supervision of A. K. Anderson, senior engineer, J. D. Johnston, W. R. McClelland, H. L. Beer, B. Robinson, W. S. Jenkins, and J. Kostash.

The associated special microscopic and spectrographic work was performed by M. H. Haycock assisted by W. E. White.

All special research investigation was under the general supervision of R. J. Traill, senior engineer. P. B. Coyne and L. S. Macklin assisted in special ore problems.

The metallurgical and research work on iron and steel and alloys, including mechanical and physical testing was performed by G. S. Farnham assisted by N. B. Brown.

The staff and work of the chemical laboratories were supervised by J. A. Fournier, Chief Chemist. The staff comprised R. A. Rogers, A. Sadler, T. T. Merrifield, R. W. Cornish, J. S. McCree, S. R. M. Badger, A. E. Larochelle, J. A. Rivington, chemists; and L. Lutes and C. H. Derry, assayers.

New Ore Dressing Laboratory. This building, started late in the fall of 1937, is well under way and is expected to be completed by September, 1938. The necessary mechanical draughting required in the design and layout of equipment has been done by W. E. Ellis, assisted by W. J. Flood of the Draughting Division.

II

INVESTIGATIONS THE RESULTS OF WHICH ARE RECORDED IN DETAIL

Ore Dressing and Metallurgical Investigation No. 732

MILL PRODUCTS FROM THE NAYBOB GOLD MINES, LIMITED, TIMMINS, ONTARIO

Shipment. One five-gallon can each of flotation concentrate and filter cake residue and one bottle of pregnant solution were received on November 1, 1937, from the Naybob Gold Mines, Limited, Timmins, Ontario. These were submitted by R. V. Neily, mine manager.

The flotation concentrate had been taken directly from the discharge of the diaphragm pump that pumps from the bottom of the concentrate wash thickener and contained whatever flotation reagents still remained in the concentrate. This sample represented the concentrate that was fed to the regrinding unit where cyanide solution is added as classifier dilution.

The filter cake residue was taken directly from the filter. Barren solution only was used to wash the filter cake.

Purpose of the Investigation. This investigation was undertaken to determine if possible the reasons for the lowered extraction and difficulties in precipitation at the mill in cyaniding a flotation concentrate; and whether the concentrate after cyanidation could be further concentrated to produce a product of shipping grade and a tailing of low enough value to discard.

Sampling and Analysis. The flotation concentrate was sampled and assayed as follows:

Gold	0·76 oz./ton
Nickel	0·09 per cent

The filter cake residue was sampled and was found to contain 0·25 ounce of gold per ton.

The barren solution, which was decanted from the filter cake residue, contained 0·009 ounce of gold per ton.

The sample of pregnant solution was examined in the chemical laboratory and was found to contain:

Thiocyanate, KCNS	3·14	grm./litre
Ferrocyanide, $K_4Fe(CN)_6$	1·77	"
Copper	0·405	"
Total calcium reported as CaO	1·22	grm./litre
	=2·44	lb./ton
Ferric iron	Trace	
Nickel	0·07	grm./litre
Reducing power	2800 c.c.	$\frac{N}{10}$ $KMnO_4$/litre

Titrations of the solution gave the following:

	Lb./ton
Available KCN	2·68
Total KCN	3·4
Protective alkalinity, CaO	2·8

These titrations were carried out in the ordinary way with silver nitrate. Further examination showed that the high alkalinity caused part of the combined cyanide to report as available cyanide, and that instead of 2·68 pounds of potassium cyanide per ton there was but 0·8 pound.

An examination of the alkaline constituents showed that the protective alkalinity consisted of 3·36 pounds of caustic soda and 0·7 pound of lime per ton.

The total calcium reported in terms of lime was found to be 2·44 pounds per ton. As 0·7 pound of lime was found by titration to be present, the remainder of the calcium is probably there as sulphate.

Experimental Tests

Filter Cake Residue

Procedure. Part of the residue was repulped in water and the slime separated by decantation. The slime was assayed and a screen test was made.

The sand was treated by flotation to determine the effect of selective concentration.

Samples of the residue were repulped in cyanide solution and agitated for different periods of time.

DECANTATION OF SLIME AND FLOTATION OF SAND

Test No. 1

A sample of the residue was repulped in water and the slime separated by decantation. The sand and slime were filtered, and a screen test was made of each, the two products were assayed for gold.

A sample of the sand was repulped in a flotation cell and conditioned for 10 minutes with 4·0 pounds of soda ash per ton and for 5 minutes with 0·2 pound of sodium cyanide per ton. Then 0·1 pound of amyl xanthate was added and allowed 3 minutes' contact. Pine oil, 0·05 pound per ton, was added and a concentrate was removed.

Decantation Test:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·225	100·00	
Slime	26·88	0·13	15·53	3·72 : 1
Sand	73·12	0·26	84·47	

Screen Tests:

Slime

Mesh	Weight, per cent
−100+150......	0·10
−150+200......	0·10
−200+325......	0·23
−325...........	99·57
	100·00

Sand

Mesh	Weight, per cent
− 65+100......	0·1
−100+150......	2·0
−150+200......	11·0
−200..........	86·9
	100·0

Flotation Concentration of Sand:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed..	100·00	0·275	100·00	
Flotation concentrate..........................	11·84	0·09	3·87	8·45 : 1
Flotation tailing..............................	88·16	0·30	96·13	

No appreciable concentration was obtained.

CYANIDATION OF FILTER CAKE RESIDUE

Test No. 2

Samples of the filter cake residue were water-washed and repulped in cyanide solution, 3·0 pounds of potassium cyanide per ton, at a dilution of 1 : 3 (one part residue to 3 parts solution). Lime was used for protective alkalinity. The samples were agitated in duplicate for 24, 48, and 72 hours. Two similar samples (Tests Nos. 2H and 2K) were cyanided overnight (18 hours), the pulps were filtered, washed and repulped in fresh solution and agitated for 24 hours longer in a solution of 3·0 pounds of potassium cyanide per ton at 1 : 3 dilution.

Results of Cyanidation:

Test No.	Agitation, hours	Dilution ratio, solids to liquids	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton residue		Final titrations of solution, lb./ton of solution	
			Feed	Tailing		KCN	CaO	KCN	CaO
2A..........	24	1 : 3	0·25	0·24	4·0	4·82	14·69	2·65	0·20
2B..........	24	1 : 3	0·25	0·24	4·0	4·30	14·61	2·80	0·175
2C..........	48	1 : 3	0·25	0·22	12·0	5·72	19·79	3·20	0·20
2D..........	48	1 : 3	0·25	0·225	10·0	5·61	19·39	3·20	0·20
2E..........	72	1 : 3	0·25	0·225	10·0	6·69	22·85	3·30	0·20
2G..........	72	1 : 3	0·25	0·225	10·0	7·00	22·80	3·20	0·20
2H..........	18+24	1 : 3	0·25	0·225	10·0	6·26	19·42	2·60	0·25
2K..........	18+24	1 : 3	0·25	0·225	10·0	6·31	19·69	2·50	0·25

Conclusions. The decanted slime contained appreciable amounts of gold.

The attempt to separate a shipping grade of concentrate from the sand product was not successful.

Flotation Concentrate

Procedure. The investigative procedure on the flotation concentrate included several series of the following cyanidation tests:

Straight cyanidation of the concentrate as received.
Straight cyanidation of water-washed samples.
Regrinding in cyanide solution and cyaniding.
Regrinding in water, aerating, and cyaniding.
Regrinding in cyanide and cyaniding with lead nitrate added.
The determination of the minimum time of agitation required.

STRAIGHT CYANIDATION OF THE CONCENTRATE AS RECEIVED

Test No. 3

To note the effect of varying the ratio of dilution in the agitators, samples of the flotation concentrate were repulped in cyanide solution, 3·0 pounds of potassium cyanide per ton, at dilutions of 1 : 1·5 and 1 : 3, and agitated for 24, 48, and 72 hours. Lime and cyanide were added as required to keep the solutions up to strength.

Results:

Test No.	Agitation, hours	Dilution ratio, solids to liquids	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton concentrate		Final titration of solutions, lb./ton of solution	
			Feed	Tailing		KCN	CaO	KCN	CaO
3A..........	24	1 : 1·5	0·76	0·385	49·34	3·04	12·26	3·15	1·20
3B..........	48	1 : 1·5	0·76	0·36	52·63	4·31	12·91	3·0	0·60
3C..........	72	1 : 1·5	0·76	0·315	58·55	6·32	15·70	2·70	0·25
3D..........	24	1 : 3	0·76	0·36	52·63	3·06	8·23	3·0	0·45
3E..........	48	1 : 3	0·76	0·33	56·58	4·44	10·00	3·0	0·50
3F..........	72	1 : 3	0·76	0·33	56·58	5·58	11·10	3·0	0·40

There is a slight increase in extraction when the ratio of solids to solutions is 1 : 3.

A screen test on the concentrate as received showed the following:

Mesh	Weight, per cent
− 48+ 65..........	0·7
− 65+100..........	3·8
−100+150..........	16·9
−150+200..........	21·9
−200..........	56·7
	100·0

STRAIGHT CYANIDATION OF WASHED CONCENTRATE

Test No. 4

A sample of the flotation concentrate was repulped in water, filtered, and washed several times.

Samples were then repulped in cyanide solution, 3·0 pounds of potassium cyanide per ton, at dilutions of 1 : 1·5 and 1 : 3 and agitated for periods of 24, 48, and 72 hours.

Results:

Test No.	Agitation, hours	Dilution ratio, solids to liquids	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton concentrate		Final titration of solutions, lb./ton of solution	
			Feed	Tailing		KCN	CaO	KCN	CaO
4A..........	24	1 : 1·5	0·76	0·31	59·21	3·43	9·35	2·80	0·55
4B..........	48	1 : 1·5	0·76	0·34	55·26	4·02	9·20	2·40	0·50
4C..........	72	1 : 1·5	0·76	0·305	59·87	6·42	9·92	2·70	0·35
4D..........	24	1 : 3	0·76	0·32	57·89	3·49	8·27	2·90	0·55
4E..........	48	1 : 3	0·76	0·33	56·58	4·38	8·38	2·60	0·50
4F..........	72	1 : 3	0·76	0·28	63·16	7·72	9·83	2·0	0·20

Thorough washing of the concentrate has a slight beneficial effect on the extraction.

REGRINDING IN CYANIDE SOLUTION, USING MILL WATER FROM THE SHIPMENT

Test No. 5

The mill water accompanying the shipment was decanted, filtered, and used for grinding and agitating.

The sample was ground in a ball mill, dilution of 4 parts concentrate to 3 parts solution (3·0 pounds of potassium cyanide per ton) to give a product 86 per cent through 200 mesh. After grinding, the pulp was split into six portions and agitated, three at a dilution of 1 : 1·5 and three at 1 : 3 dilution. The solution consisted of the grinding solution made up to the required volume with mill water, potassium cyanide 3·0 pounds per ton. The agitation was continued for 24, 48, and 72 hours.

Results:

Test No.	Agitation, hours	Dilution ratio, solids to liquids	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton of concentrate		Final titration of solutions, lb./ton of solution	
			Feed	Tailing		KCN	CaO	KCN	CaO
5A..........	24	1 : 1·5	0·76	0·31	59·21	5·12	19·78	2·90	0·25
5B..........	48	1 : 1·5	0·76	0·29	61·84	6·94	26·69	2·70	0·55
5C..........	72	1 : 1·5	0·76	0·30	60·53	8·09	26·52	3·60	0·25
5D..........	24	1 : 3	0·76	0·31	59·21	4·96	19·09	2·20	0·35
5E..........	48	1 : 3	0·76	0·295	61·18	6·85	23·27	3·00	0·65
5F..........	72	1 : 3	0·76	0·295	61·18	7·46	23·80	2·80	0·50
Solution from grind..........								0·30	0·03

The test shows that long agitation was not beneficial to the extraction.

A similar test using fresh water in place of mill water showed about the same results. The grind was 91 per cent through 200 mesh. The mill water was apparently not harmful to the reagent consumption or the extraction.

REGRINDING IN WATER, AERATING, AND CYANIDING

Test No. 6

To study the effect of aeration, a sample of the concentrate was ground in a ball mill, with lime 10 pounds per ton, dilution 4 : 3 with water, to 96 per cent through 200 mesh.

The ground pulp was aerated in a Wallace super-agitator for 20 hours, then filtered, washed, and repulped in cyanide solution, 3·0 pounds of potassium cyanide per ton, at dilutions of 1 : 1·5 and 1 : 3. The pulps were agitated for 24, 48, and 72 hours.

Results:

Test No.	Agitation, hours	Dilution ratio, solids to liquids	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton of concentrate		Final titration of solutions, lb./ton of solution	
			Feed	Tailing		KCN	CaO	KCN	CaO
6A..........	24	1 : 1·5	0·76	0·29	61·84	4·05	10·12	3·2	0·15
6B..........	48	1 : 1·5	0·76	0·29	61·84	5·44	10·29	3·2	0·15
6C..........	72	1 : 1·5	0·76	0·29	61·84	7·93	10·17	2·4	0·10
6D..........	24	1 : 3	0·76	0·285	62·50	4·22	9·33	3·0	0·30
6E..........	48	1 : 3	0·76	0·280	63·16	4·62	9·53	3·3	0·25
6F..........	72	1 : 3	0·76	0·29	61·84	8·32	9·77	2·5	0·15

Lime used during grind..............................	7·73	lb./ton of	concentrate
Lime used during aeration..........................	15·46	"	"
Total....................................	23·19	"	"

The aeration and washing show a lower consumption of reagents but no appreciable increase in extraction of gold. Long agitation periods apparently are not necessary.

Test No. 7

This test was made to note the effect of lead nitrate.

A sample of concentrate was ground in a ball mill, dilution 4 : 3, in cyanide solution, 3·0 pounds of potassium cyanide per ton, to 85 per cent through 200 mesh. The grinding solution was saved and made up to volume for the agitation with wash water from washing the pulp into the filter. This was done in order to retain any cyanicides that might be formed during the grind, and to note the effect produced on the extraction of gold.

Three samples were repulped in cyanide solution, 3·0 pounds of potassium cyanide per ton, dilution 1 : 1·5, and agitated for 24, 48, and 72 hours. Three other samples were repulped at 1 : 3 dilution and agitated for the same periods. To each test, lead nitrate was added at the rate of 0·5 pound per ton.

Two more samples, 7G and 7H, were repulped at 1 : 1·5 and 1 : 3 dilutions in cyanide solution, 3·0 pounds of potassium cyanide per ton, and agitated, without lead nitrate, for 18 hours. The pulp was then filtered, washed, and repulped in fresh solution and agitated for 24 hours longer.

Results:

Cyanidation with Lead Nitrate:

Test No.	Agitation, hours	Dilution ratio, solids to liquids	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton of concentrate			Final titration of solutions, lb./ton of solution	
			Feed	Tailing		KCN	CaO	$Pb(NO_3)_2$	KCN	CaO
7A......	24	1 : 1·5	0·76	0·285	62·50	6·00	15·37	0·5	3·15	0·25
7B......	48	1 : 1·5	0·76	0·290	61·84	7·78	19·26	0·5	2·95	0·20
7C......	72	1 : 1·5	0·76	0·290	61·84	8·79	22·18	0·5	3·20	0·25
Grinding solution		4 : 3				1·87	4·90	....	0·9	0·15
7D......	24	1 : 3	0·76	0·26	65·79	5·76	13·79	0·5	2·90	0·35
7E......	48	1 : 3	0·76	0·295	61·18	6·91	16·37	0·5	3·00	0·45
7F......	72	1 : 3	0·76	0·290	61·84	7·98	18·75	0·5	3·15	0·30

Cyanidation without Lead Nitrate—First Solution:

7G......	18	1 : 1·5	....	....	....	5·37	13·71	....	0·80	0·1
7H......	18	1 : 3	....	....	....	5·52	13·97	....	2·05	0·3

Cyanidation without Lead Nitrate—Second Solution:

7G......	24	1 : 1·5	0·76	0·28	63·15	2·35	4·67	....	2·50	0·1
7H......	24	1 : 3	0·76	0·295	61·18	1·37	4·02	....	2·60	0·3

Total Reagents Consumed in Tests Nos. 7G and 7H:

—	KCN	CaO
	(lb./ton)	
7G..	7·72	18·38
7H..	6·89	17·99

The addition of lead nitrate shows no appreciable benefit except in Test No. 7D, which had the lowest tailing obtained in any of the tests. The cyanide consumption was high. No increase in extraction was obtained by breaking the agitation period and repulping with fresh solution.

Test No. 8

In this test the effect of various amounts of lead nitrate added to the cyanide solution was investigated.

A sample of concentrate was ground in a ball mill, dilution 4 : 3, with water, to 96 per cent through 200 mesh. Four samples were repulped in cyanide solution, 3·0 pounds of potassium cyanide per ton, at a dilution of 1 : 2. Lead nitrate in various amounts was added to each charge. The agitation was discontinued after 24 hours.

Results:

Test No.	Agitation, hours	Dilution ratio, solids to liquids	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton of concentrate			Final titration of solutions, lb./ton of solution	
			Feed	Tailing		KCN	CaO	$Pb(NO_3)_2$	KCN	CaO
8A......	24	1 : 2	0·76	0·290	61·84	6·52	13·0	0·5	2·80	0·10
8B......	24	1 : 2	0·76	0·295	61·18	6·31	12·8	1·0	2·90	0·15
8C......	24	1 : 2	0·76	0·295	61·18	6·13	13·1	1·5	3·00	0·10
8D......	24	1 : 2	0·76	0·295	61·18	6·15	13·0	2·0	3·00	0·15

No increase in extraction was noted by using lead nitrate in the solutions.

GRINDING IN CYANIDE SOLUTION AND AGITATING FOR VARIOUS PERIODS OF TIME

Test No. 9

A sample of the flotation concentrate was ground in a ball mill, dilution 4 : 3, with cyanide solution, 3 pounds of potassium cyanide per ton, to give a product 95 per cent through 200 mesh.

The pulp was filtered and the solution was saved and made up to the required volume with wash water from the mill. Samples of the ground concentrate were repulped at 1 : 3 dilution in cyanide solution (3·0 pounds of potassium cyanide per ton) and agitated for the following periods: 1, 2, 3, 4, 8, and 16 hours.

A sample of the ground concentrate was assayed prior to agitation to determine the extraction occurring during the grind.

Results:

Test No.	Agitation, hours	Dilution ratio, solids to liquids	Assay, Au, oz./ton		Extraction, per cent
			Feed	Tailing	
Grind..........................	0·5	4 : 3	0·76	0·57	25·00
9A..............................	1	1 : 3	0·76	0·34	55·26
9B..............................	2	1 : 3	0·76	0·30	60·53
9C..............................	3	1 : 3	0·76	0·30	60·53
9D..............................	4	1 : 3	0·76	0·30	60·53
9E..............................	8	1 : 3	0·76	0·30	60·53
9F..............................	16	1 : 3	0·76	0·28	63·16

The test shows that 25 per cent of the gold was extracted during the grind.

The 2-hour period of agitation shows an extraction equal to that obtained in the 24-hour tests.

Results of Cyanidation from 1 to 72 Hours:

(Material ground and agitated in cyanide solution)

Test No.	Agitation, hours	Dilution ratio, solids to liquids	Assay, Au, oz./ton		Extraction, per cent
			Feed	Tailing	
9A	1	1 : 3	0·76	0·34	55·26
9B	2	1 : 3	0·76	0·30	60·53
9C	3	1 : 3	0·76	0·30	60·53
9D	4	1 : 3	0·76	0·30	60·53
9E	8	1 : 3	07·6	0·30	60·53
9F	16	1 : 3	0·76	0·28	63·16
5D	24	1 : 3	0·76	0·31	59·21
5E	48	1 : 3	0·76	0·295	61·18
5F	72	1 : 3	0·76	0·295	61·18

SUMMARY AND CONCLUSIONS

The minus 325-mesh portion of the filter cake residue contains 0·13 ounce of gold per ton and represents 15·5 per cent of the gold; 11·8 per cent of the weight of the plus 325-mesh portion assays 0·09 ounce per ton, leaving a concentrate containing 0·30 ounce per ton.

It is apparent that the filter cake residue is not amenable to further concentration.

Cyanidation of this material shows that a minimum tailing of 0·22 ounce of gold per ton is to be expected.

Tests made on the thickener underflow show that thorough washing prior to cyanidation is of some slight benefit.

A thin pulp in the agitators is to be preferred. Although there was no marked difference in the results obtained between agitation in a 1 : 1·5 pulp and a 1 : 3 pulp, the higher dilution, especially on sulphide ores, gives more consistent results.

The use of lead salts does not appear to have any beneficial effect.

In all the tests on the product of the thickener underflow a tailing as low as the filter cake residue from the mill was not obtained. This probably is due to the difference in the grinding, or other conditions in mill practice not duplicated in a laboratory mill.

The outstanding feature of the investigation is that shown by the tests in which the pulps were agitated from 1 to 72 hours. The dissolution of gold remains stationary from 2 to 8 hours, and then increases at 16 hours, and falls off after 16 hours. This indicates the advisability of studying the time of agitation in the mill with a view to obtaining about 16 hours' agitation.

The rise in the tailing after 16 hours of agitation also suggests the possibility of precipitation of dissolved gold.

The analysis of the pregnant solution from the mill shows many of the causes of the difficulties being met with there. The high caustic soda content of the solution reacts with the sulphides with consequent formation of ferrocyanide, which in some instances may be a precipitant of dissolved gold. The solution was extremely foul, of a reducing character that would retard the dissolution of gold by oxygen depletion.

Improved results can be expected by agitation in a dilute pulp. This in itself will have the effect of shortening the time of agitation, which has been shown to be advisable. Bleeding the barren solution and replacing by fresh water will tend to keep the solutions in active condition.

Careful control of the alkalinity of these solutions will be found highly beneficial, both in improved extraction and more stable thickener and filter operations.

Ore Dressing and Metallurgical Investigation No. 733

CHROMITE FROM THE CHROMIUM MINING AND SMELTING CORPORATION, LIMITED, COLLINS, ONTARIO

Shipment. One carload of chromite ore, weighing roughly 29 tons, was received on January 3, 1938, from the property of the Chromium Mining and Smelting Corporation, Limited, at Obonga Lake, Collins, Ontario. This shipment was made by H. H. Merritt, mine manager, on advice from Professor W. G. McBride, McGill University, Montreal, Quebec.

Purpose of Investigation. This large shipment was made for the purpose of continuing on a large scale the preliminary investigation, the results of which were reported on November 19, 1937, and also to obtain information that would aid in the design of a 1,000-ton concentrator.

Investigative Procedure. No sample of the entire carload was taken, but one of the feed for each day's run was obtained. The average of thirteen such samples showed the shipment to contain 8·87 per cent of Cr_2O_3, 12·94 per cent of iron.

The ore, crushed to $-\frac{3}{4}$ inch, was fed at the rate of from 800 to 1,000 pounds per hour from a bin to a Marcy rod mill. An automatic sampler cut out a feed sample during the run.

All previous investigations on this Obonga Lake ore showed that the chromite slimes very readily. To reduce as much as possible the formation of slime, a light load of rods was carried in the mill with a consequent heavy circulating load. The mill discharge was elevated and discharged over a "Hummer" screen and the plus 10-mesh size was returned to the grinding mill.

Unless otherwise stated in the details of the mill runs, the minus 10-mesh material was passed through a Richards launder hydraulic classifier, where two spigot products were made. The coarser spigot product was concentrated by jigs or tables and their middlings and tailings were sent to a dewatering classifier, where the oversize was returned to the rod mill for further grinding. The overflow of this classifier contained a very small amount of slime and was united with the overflow of the Richards hydraulic classifiers and was thickened in a Callow cone.

The discharge from the second spigot of the hydraulic classifier was concentrated on a table, the middlings of which were sent to the dewatering classifier and the table tailing was discarded.

The discharge of the thickening cone that received the classifier overflows, unless otherwise stated, was concentrated on a Wilfley table. The middling from this table was returned to the dewatering classifier and the table tailing was discarded. The overflow from the thickening cone was run to waste. All these waste materials were united into one stream and a sample, called "mill tailing", was obtained.

Mill Run No. 1

In this run, the product from No. 1 spigot was tabled on a ¼-deck Butchart table, and its middling was reground. No. 2 spigot product was concentrated on a ¼-deck Plato table and its middling reground. The tailing was run to waste. The hydraulic classifier overflow was thickened and concentrated on a full-size Wilfley table.

Screen analyses of the various products are as follows:

Screen Analyses:

Mesh	Cumulative weight, per cent			
	−10-mesh classifier feed	Butchart table feed	Plato table feed	Wilfley table feed
+ 14	....	1·1	....	....
− 14+ 20	1·2	4·3	....	....
− 20+ 28	4·8	10·9	0·2	....
− 28+ 35	14·4	32·2	1·4	....
− 35+ 48	19·3	62·0	2·6	....
− 48+ 65	41·6	81·0	15·7	1·4
− 65+100	54·3	92·2	34·7	5·0
−100+150	68·3	97·4	58·4	18·1
−150+200	76·6	99·1	74·3	34·1
−200	100·0	100·0	100·0	100·0

Assays:

Product	Cr_2O_3, per cent
Feed	9·65
−10-mesh classifier feed	9·94
Butchart table feed	13·01
Butchart table tailing	2·65
Plato table feed	10·33
Plato table tailing	3·50
Wilfley table feed	8·53
Wilfley table tailing	3·26
Middlings to regrind	16·27
Mill tailing	3·18
Combined concentrates	28·42

Mill Run No. 2

In the preceding run, the Wilfley table treating the fine was overloaded. To relieve this, the coarse spigot product was sent to two James jigs in series. The Butchart table, which formerly treated the coarse product, now treated the intermediate size, and the fine was split between the Plato and the Wilfley tables. The first jig made a hutch product and the concentrate from the second was called a middling product and sent to regrind together with the table middlings. The jig tailing was run to waste.

Measurements made on hydraulic classifier showed the weights of the three products to be in the ratios of:

Spigot No. 1	*Spigot No. 2*	*Overflow*
1·5	1·0	1·5

Screen Analyses:

Mesh	Cumulative weight, per cent				
	−10-mesh classifier feed	Jig feed	Butchart table feed	Plato table feed	Wilfley table feed
+ 14	....	1·0	....	....	....
− 14+ 20	0·7	4·6	....	....	....
− 20+ 28	2·7	13·4	....	....	....
− 28+ 35	10·5	36·1	1·2	....	....
− 35+ 48	21·3	65·3	6·8	....	....
− 48+ 65	39·3	83·3	25·1	0·4	0·2
− 65+100	51·1	95·6	44·8	1·1	1·1
−100+150	70·1	99·3	76·7	15·1	16·3
−150+200	79·1	99·8	89·5	32·0	33·0
−200	100·0	100·0	100·0	100·0	100·0

Assays:

Product	Cr_2O_3, per cent
Feed	9·09
−10-mesh classifier feed	11·22
Jig feed	17·27
Jig concentrate	26·73
Jig middling	18·58
Jig tailing	6·63
Butchart table feed	8·73
Butchart table concentrate	28·72
Butchart table tailing	1·39
Plato table feed	6·57
Plato table concentrate	26·38
Plato table tailing	5·19
Wilfley table feed	6·54
Wilfley table concentrate	27·10
Wilfley table tailing	3·65
Middlings to regrind	8·05
Mill tailing	4·41

Mill Run No. 3

This run is the same as the preceding one. The grind was somewhat finer, as shown by the following screen analyses:

Mesh	Cumulative weight, per cent				
	−10-mesh classifier feed	Jig feed	Butchart table feed	Plato table feed	Wilfley table feed
+ 20	1·0	1·7	....	....	....
− 20+ 28	3·2	6·8	....	....	....
− 28+ 35	11·4	22·0	0·7	....	....
− 35+ 48	21·9	40·2	5·0	....	....
− 48+ 65	37·9	65·6	20·5	....	....
− 65+100	46·5	75·3	42·2	0·2	0·2
−100+150	66·5	94·6	72·1	11·5	12·7
−150+200	76·0	97·0	85·5	29·8	26·2
−200	100·0	100·0	100·0	100·0	100·0

Assays:

Product	Cr_2O_3, per cent
Feed	8·73
−10-mesh classifier feed	10·22
Jig feed	14·57
Jig concentrate	27·37
Jig middling	22·37
Jig tailing	6·67
Butchart table feed	7·61
Butchart table concentrate	27·15
Butchart table tailing	3·38
Plato table feed	5·90
Plato table concentrate	26·32
Plato table tailing	4·87
Wilfley table feed	6·13
Wilfley table concentrate	26·25
Wilfley table tailing	3·47
Combined middlings	15·38
Mill tailing	5·58

A sample of the jig tailing was reground to pass 48 mesh with 39 per cent minus 200 mesh, and tabled. The tailing from this operation contained 2·25 per cent of Cr_2O_3 as against 6·67 per cent.

Mill Run No. 4

In this run, the jig tailing was sent to the dewatering classifier and then returned to the grinding mill. The grinding was considerably coarser than in the preceding run.

Screen Analyses:

Mesh	Cumulative weight, per cent				
	−10-mesh classifier feed	Jig feed	Butchart table feed	Plato table feed	Wilfley table feed
+ 10	1·2	6·0	....	....	....
− 10+ 14	5·8	20·1	....	....	....
− 14+ 20	11·2	36·8	0·4	....	....
− 20+ 28	18·7	53·1	0·9	....	....
− 28+ 35	28·4	73·2	4·2	....	....
− 35+ 48	36·6	84·0	13·0	0·1	0·3
− 48+ 65	50·3	95·1	34·1	1·2	1·7
− 65+100	61·7	97·9	65·5	2·9	10·7
−100+150	72·1	99·8	80·1	20·2	26·6
−150+200	79·9	99·9	89·8	35·9	38·3
−200	100·0	100·0	100·0	100·0	100·0

Assays:

Product	Cr_2O_3, per cent
Feed	9·02
−10-mesh classifier feed	8·64
Jig feed	11·41
Jig concentrate	26·92
Jig tailing	8·97
Butchart table feed	7·61
Butchart table concentrate	24·58
Butchart table tailing	3·29
Plato table feed	5·55
Plato table concentrate	24·76
Plato table tailing	3·47
Wilfley table feed	5·64
Wilfley table concentrate	25·30
Wilfley table tailing	2·53
Middlings to regrind	7·43
Mill tailing	2·62

Mill Run No. 5

This run is similar to the preceding one, with the exception that the grind was considerably coarser.

Screen Analyses:

Mesh	Cumulative weight, per cent				
	−10-mesh classifier feed	Jig feed	Butchart table feed	Plato table feed	Wilfley table feed
+ 8	....	0·6	....	....	....
− 8+ 10	1·8	6·7	....	....	....
− 10+ 14	10·6	31·9	....	....	....
− 14+ 20	19·5	54·1	2·9	....	....
− 20+ 28	30·3	71·7	7·6	....	....
− 28+ 35	44·0	85·8	19·3	0·7	....
− 35+ 48	50·0	92·2	36·9	2·6	0·7
− 48+ 65	60·6	97·6	62·7	8·1	6·2
− 65+100	68·6	98·4	85·1	19·7	19·9
−100+150	76·5	99·8	95·9	37·1	38·3
−150+200	81·5	99·9	98·9	58·5	56·3
−200	100·0	100·0	100·0	100·0	100·0

Assays:

Product	Cr_2O_3, per cent
Feed	8·61
−10-mesh classifier feed	8·06
Jig feed	12·13
Jig concentrate	29·68
Jig tailing	10·37
Butchart table feed	9·16
Butchart table concentrate	28·23
Butchart table tailing	1·58
Plato table feed	7·53
Plato table concentrate	26·70
Plato table tailing	2·98
Wilfley table feed	7·17
Wilfley table concentrate	25·89
Wilfley table tailing	2·30
Table middling to regrind	7·16
Mill tailing	2·34

Mill Run No. 6

It was observed that the concentrate was quite magnetic. To lighten the load on the tables treating the fine from the hydraulic classifier, this product was first passed over a Roche magnetic separator. The tailing and middling from this machine went to the dewatering cone. The thickened underflow was concentrated on the Wilfley table. The coarse sand product from the first spigot of the hydraulic classifier was jigged, a hutch and gate concentrate was taken off the first jig, and a hutch concentrate made in the second jig. The jig tailing was returned to the grinding mill. The second spigot product of the hydraulic classifier was concentrated on the Plato table.

The grinding was about the same as in Mill Run No. 5.

Assays:

Product	Cr_2O_3, per cent
Feed	8·77
−10-mesh classifier feed	8·96
Jig feed	11·46
Jig concentrate No. 1	25·20
Jig concentrate No. 2	28·80
Jig tailing	10·01
Plato table feed	9·14
Plato table concentrate	30·64
Plato table tailing	1·85
Roche feed	7·30
Roche magnetic concentrate	17·00
Roche middling	4·97
Roche tailing	3·33
Wilfley table feed	4·83
Wilfley table concentrate	33·20
Wilfley table tailing	2·30
Mill tailing	2·03

The larger part of the mineral in the fine from the launder classifier was caught by the magnetic separator. The remainder, concentrated on the Wilfley table, was also magnetic. The magnetic field in the Roche separator was not powerful enough to lift all the mineral.

Mill Run No. 7

Instead of passing the classifier fine over the magnetic separator, in this run the minus 10-mesh screen product was sent to the Roche machine. The middling and tailing from this separator were classified, and the products jigged, concentrated on the Plato table, and the fine thickened and concentrated on the Wilfley table.

Screen Analyses:

Mesh	Cumulative weight, per cent		
	−10-mesh Roche feed	Jig feed	Wilfley table feed
+ 10	2·8	5·0	....
− 10+ 14	14·7	25·9	....
− 14+ 20	29·7	47·1	....
− 20+ 28	44·0	64·4	....
− 28+ 35	57·7	83·6	0·5
− 35+ 48	65·2	90·5	1·0
− 48+ 65	75·8	97·8	5·3
− 65+100	82·5	98·8	15·7
−100+150	86·6	99·5	29·7
−150+200	90·2	99·9	48·9
−200	100·0	100·0	100·0

Assays:

Product	Cr_2O_3, per cent	Iron, per cent
Feed	8·28	....
−10-mesh Roche feed	8·51	....
Roche magnetic concentrate	17·10	30·64
Roche tailing	4·86	....
Jig feed	6·18	....
Jig concentrate No. 1	28·86	32·04
Jig concentrate No. 2	30·94	31·64
Jig tailing	4·78	....
Plato table feed	4·86	....
Plato table concentrate	36·72	29·65
Plato table tailing	1·58	....
Wilfley table feed	6·22	....
Wilfley table concentrate	33·73	35·82
Wilfley table tailing	2·39	....
Table middling to regrind	7·04	....
Mill tailing	2·07	....

Mill Run No. 8

In this run, the minus 10-mesh screen product was classified in the hydraulic classifier. The coarse product, instead of being jigged, was concentrated on the Butchart table and the table tailing returned to the grinding mill. The intermediate hydraulic classifier product was sent to the Plato table and its tailing discarded. The classifier fine was sent to the magnetic separator, and its middling and tailing thickened in the cone and concentrated on the Wilfley table.

Screen Analyses:

Mesh	Cumulative weight, per cent				
	Classifier feed	Butchart table feed	Plato table feed	Roche feed	Mill tailing
+ 10	1·8	4·4	0·8	....	....
− 10+ 14	10·2	23·0	6·7	....	1·1
− 14+ 20	20·5	45·0	17·7	....	3·4
− 20+ 28	32·8	67·2	32·8	0·2	6·4
− 28+ 35	44·8	86·6	53·4	0·9	10·4
− 35+ 48	54·3	94·6	67·5	3·6	12·8
− 48+ 65	62·1	98·9	82·1	10·3	19·0
− 65+100	69·0	99·9	90·5	23·0	29·0
−100+150	75·0	100·0	94·3	36·6	37·7
−150+200	82·7	....	97·6	44·8	42·8
−200	100·0	....	100·0	100·0	100·0

A screen analysis of the cone overflow showed 99 per cent minus 325 mesh with 0·4 per cent plus 200 mesh.

Measurements of the products of the hydraulic classifier showed 33·7 per cent of the weight in No. 1 spigot, 25·3 per cent in the intermediate size, and 41·0 per cent in the fine.

Assays:

Product	Cr_2O_3, per cent
Feed	8·87
−10-mesh classifier feed	8·69
Butchart table feed	10·46
Butchart table concentrate	27·51
Butchart table tailing	8·84
Plato table feed	7·04
Plato table concentrate	29·50
Plato table tailing	2·66
Roche feed	5·81
Roche magnetic concentrate	16·95
Roche tailing	3·69
Wilfley table feed	6·18
Wilfley table concentrate	35·18
Wilfley table tailing	1·85
Table middling to regrind	8·01
Cone overflow	2·66
Mill tailing	2·39

Mill Run No. 9

This day's run was made under the same conditions as was Mill Run No. 8; 51·4 per cent of the minus 10-mesh screen product was discharged from No. 1 spigot and concentrated on the Butchart table; 18·9 per cent went to the Plato table; and the remaining 29·7 per cent constituted the classifier fine.

Screen Analyses:

Mesh	Cumulative weight, per cent				
	−10-mesh classifier feed	Butchart table feed	Plato table feed	Roche feed	Mill tailing
+ 10	1·9	5·6	1·7	....	....
− 10+ 14	10·8	28·1	12·2	....	4·0
− 14+ 20	21·5	52·6	28·4	....	9·0
− 20+ 28	33·2	70·1	45·9	0·3	15·6
− 28+ 35	44·7	85·6	68·8	1·0	22·4
− 35+ 48	51·9	91·9	79·5	2·4	26·0
− 48+ 65	62·3	97·2	90·9	11·4	34·3
− 65+100	68·0	99·3	95·4	22·2	42·2
−100+150	72·6	99·7	96·9	34·7	48·5
−150+200	77·3	99·9	99·3	42·8	53·8
−200	100·0	100·0	100·0	100·0	100·0

Assays:

Product	Cr_2O_3, per cent
Feed	8·64
−10-mesh classifier feed	8·46
Butchart table feed	11·82
Butchart table concentrate	26·56
Butchart table tailing	8·84
Plato table feed	7·97
Plato table concentrate	29·79
Plato table tailing	2·84
Roche feed	6·44
Roche magnetic concentrate	18·18
Roche tailing	4·68
Wilfley table feed	6·99
Wilfley table concentrate	32·32
Wilfley table tailing	2·03
Table middling to regrind	7·49
Cone overflow	2·75
Mill tailing	2·34

Mill Run No. 10

This concluding run was made under the same conditions as the preceding one.

The split on the hydraulic classifier was 41 per cent of the weight of feed from No. 1 spigot, 26·1 per cent from No. 2 spigot, and 32·8 per cent in the fine.

Screen Analyses:

Mesh	Cumulative weight, per cent					
	Classifier feed	Butchart table feed	Plato table feed	Roche feed	Wilfley table feed	Mill tailing
+ 10	1·5	7·7	....	....	....	....
− 10+ 14	10·8	36·9	8·7	....	....	4·3
− 14+ 20	22·2	63·8	22·3	....	....	11·8
− 20+ 28	33·9	81·7	39·8	....	....	22·7
− 28+ 35	44·6	92·8	61·4	0·2	0·8	34·4
− 35+ 48	51·9	97·1	77·8	1·3	3·0	41·7
− 48+ 65	59·3	99·3	90·3	7·2	10·0	46·8
− 65+100	66·9	99·8	95·9	17·5	22·7	51·8
−100+150	73·3	99·9	97·9	32·8	35·8	57·4
−150+200	78·2	100·0	99·5	44·8	62·8	63·0
−200	100·0	....	100·0	100·0	100·0	100·0

Of the cone overflow 99·6 per cent was minus 325 mesh.

Assays:

Product	Cr_2O_3, per cent	Iron, per cent
Feed	8·65	12·94
− 10-mesh classifier feed	9·11	....
Butchart table feed	11·64	....
Butchart table concentrate	24·80	30·24
Butchart table tailing	10·87	....
Plato table feed	7·31	....
Plato table concentrate	31·30	35·42
Plato table tailing	2·07	....
Roche feed	6·54	....
Roche magnetic concentrate	18·22	43·18
Roche tailing	4·42	....
Wilfley table feed	10·19	....
Wilfley table concentrate	35·45	34·42
Wilfley table tailing	1·89	....
Table middling to regrind	14·52	....
Cone overflow	2·93	....
Mill tailing	2·26	....

An analysis of the mill tailing showed the following:

Mesh	Weight, per cent	Assay, Cr_2O_3 per cent	Distribution, per cent
+ 65	46·7	1·76	34·5
− 65+100	5·0	1·17	2·5
−100	48·3	3·11	63·0
	100·0		100·0

Settling tests on the ore at a dilution of 1 : 5 showed the pulp to settle at the rate of 0·7 foot per hour.

CONCLUSIONS

The ore contains a considerable amount of magnetite. Microscopic examination of polished sections of concentrates showed the magnetite and chromite grains to be intimately associated. This close relationship makes it difficult to obtain a high-grade concentrate.

The magnetic machine produced a concentrate containing about 18 per cent of Cr_2O_3. The tailing from this operation was concentrated to about 35 per cent of Cr_2O_3. Direct table concentration of the ore produced concentrates containing from 27 to 28 per cent Cr_2O_3.

As found in previous shipments from Obonga Lake, the ore grinds easily and the chromite slimes quite readily. Coarse sand tailings containing less than 2 per cent of Cr_2O_3 were produced in some of the runs and the solids in the overflow from the dewatering cone, 99 per cent minus 325 mesh, contained from 2·6 to 3 per cent of Cr_2O_3.

Jigs were used in most of the runs to relieve the load on the concentrating tables. The results obtained by jigging the coarse classified product compare very favourably with those of table concentration.

Magnetic separators also were incorporated in the flow-sheet to relieve the load on the tables treating the fines. These magnetic concentrates were about 10 per cent lower in Cr_2O_3 than those produced by the tables. With sufficient table capacity, the magnetic separator could be omitted.

The average feed sample of the shipment contains 8·87 per cent of Cr_2O_3 and about 12·9 per cent of iron. An average grade of concentrate of 28 per cent of Cr_2O_3 is to be expected and with sufficient table capacity to treat the fine material, a mill tailing of 2 per cent of Cr_2O_3 or lower should be realized. This represents a recovery of 83·4 per cent with a ratio of concentration of 3·8 : 1.

In the design of a mill to treat this type of ore, particular attention should be paid to the recovery of the very fine chromite, for it is here that the losses mainly occur. The ore should be given a quick pass through the grinding mill, and liberated grains of chromite removed as soon as freed. This will mean heavy circulating loads of very coarse material from both the ball mill screen or classifier and the coarse sand tables. Pumps or elevators to handle these returns should be carefully chosen. The product for concentration should be split into four classified fractions, and tabled. The very fine chromite appears to cling to the linoleum table-top and does not travel so fast as, say, clean iron pyrite. Cross wash water on the table if in excess carries the fine chromite into the tailing. The design of the table top should be such that it will minimize these losses.

At least three times as much table capacity should be provided for the fine as for the coarser sand products. Provision should be made for the return of middling products from each table to the grinding circuit, or if deemed more satisfactory, to a separate regrind mill.

If thickeners be incorporated in the flow-sheet, they should be of ample size to recover the minus 325-mesh chromite particles.

On this grade of ore, with a balanced plant and careful operation, a recovery of from 80 to 85 per cent of the chromite should be realized.

Ore Dressing and Metallurgical Investigation No. 734

CONE OVERFLOW FROM THE ALDERMAC COPPER CORPORATION, LIMITED, ARNTFIELD, QUEBEC

Sample. The sample was received on March 14, 1938, from the Aldermac Copper Corporation, Limited, Arntfield, Quebec. It is stated to have been taken from the overflow of cone 2, the second of two 4-foot cones operated in series to wash the copper tailing (i.e. pyrite concentrate). The problem was to determine the condition of the copper in the product, as an indication of how practice might be improved. To this end two polished sections were prepared and studied microscopically.

Microscopic Analysis of the Product. The following table, showing the quantities and degree of combination of the minerals, is based upon microscopic data. The specific gravities assumed for purposes of calculating the percentages by weight are: pyrite, 5·0; pyrrhotite, 4·6; chalcopyrite, 4·2; and gangue, 2·7.

Quantities and Distribution of the Minerals

(Percentage by weight)

	Per cent	
Pyrite—		
Free (largely minus 200 mesh)	94·0	
Combined with chalcopyrite (averaging about 200 mesh)	1·1	
		95·1
Pyrrhotite—		
Free (largely minus 280 mesh)	1·5	
Combined with chalcopyrite (all minus 280 mesh)	0·2	
		1·7
Gangue—		
Free (averages coarser than the sulphides, plus 200 mesh)	1·5	
Combined with chalcopyrite (largely plus 200 mesh)	0·7	
		2·2
Chalcopyrite—		
Free (largely minus 10 microns)	0·3	
Combined with pyrite (minus 280 mesh)	0·3	
Combined with gangue (minus 280 mesh)	0·4	
Combined with pyrrhotite (minus 280 mesh)	Trace	
		1·0
Total		100·0

Distribution of the Copper. The distribution of the copper, based on microscopic data, is shown below:

Distribution of the Copper in the Overflow from Cone 2

(Percentage by weight)

Condition of copper	Per cent of product	Per cent of total copper
As free chalcopyrite	0·10	30
Combined with pyrite	0·10	29
Combined with gangue	0·12	36
Combined with pyrrhotite	0·02	5
Totals	0·34	100

CONCLUSIONS

Several points are apparent from a study of the microscopic data.

(1) Somewhat less than one-third of the copper content is present as free chalcopyrite, and this is largely very finely divided.

(2) Somewhat over one-third of the copper content is present as relatively small grains of chalcopyrite combined with large grains of gangue.

(3) Most of the remaining copper (somewhat less than one-third) occurs as small grains of chalcopyrite combined with the coarser sizes of the pyrite.

(4) Very little copper is combined with pyrrhotite.

Ore Dressing and Metallurgical Investigation No. 735

ZINC CONCENTRATES FROM THE NORMETAL MINING CORPORATION, LIMITED, DUPUY, QUEBEC

Shipment. Seven samples of zinc concentrate were received on December 13, 1938, from the Normetal Mining Corporation, Limited, Dupuy, Quebec. The zinc contents of the samples were stated to be as follows:

Sample No.	Zinc, per cent
1	50·00
2	29·79
3	48·75
4	43·42
5	45·06
6	58·43
7	52·74

Purpose of Examination. The purpose of the examination was to determine the cause of the wide variation in zinc content of the concentrate. Two possibilities were apparent: *(a)* that it might be caused by variation in the iron content of the sphalerite itself, in which case the concentrate must necessarily be high in iron and low in zinc when the sphalerite was the high-iron variety (marmatite); and *(b)* that it might be caused by dilution of the concentrate with other minerals, notably pyrite.

Character of the Minerals. The samples of concentrate were examined microscopically, both in unmounted form and in polished sections. Under the binocular microscope it was immediately apparent that the sphalerite varied considerably in colour and transparency, all gradations being present from the light translucent straw-coloured sphalerite to the opaque black sphalerite usually referred to as "black jack". In so far as could be estimated, however, there appeared to be no significant difference in the quantities of light and dark sphalerite in the various samples. In an attempt to determine the variation in iron content of the sphalerite, several samples were picked out grain by grain under the microscope, some representing the light and others representing the dark sphalerite. As such samples were very small (about 0·001 gramme is the largest practical size owing to the time required) the analyses were carried out microchemically and are therefore only a rough indication. They showed the sphalerite to carry from about 8 per cent to somewhat over 14 per cent of iron. It is probable that both figures should be considerably lower, because it was difficult to pick out a large number of sphalerite grains without selecting some at least with adhering pyrite particles.

In polished sections the other minerals were seen to be pyrite, chalcopyrite, and gangue. No variations could be observed in their character.

All the minerals are largely freed from one another, the gangue showing the highest degree of combination, usually with pyrite.

Quantities of the Minerals. Three samples were selected for quantitative determination of the minerals, as follows:

	Per cent
Sample No. 2	Zn, 29·79
Sample No. 4	Zn, 43·42
Sample No. 7	Zn, 52·74

The results of microscopic analyses of the three polished sections are shown in Table I. The percentages are calculated by weight, using the following specific gravities: sphalerite, 4·0; pyrite, 5·0; chalcopyrite, 4·2; and gangue, 2·7.

TABLE I

Microscopic Analyses of Three Samples of Concentrate

(Percentages calculated by weight)

Mineral	Sample No. 2	Sample No. 4	Sample No. 7
Sphalerite	41·9	58·1	82·0
Pyrite	44·7	30·2	8·7
Chalcopyrite	1·7	2·5	3·1
Gangue	11·7	9·2	6·2
Totals	100·0	100·0	100·0

If the quantities of sphalerite that should be present in the concentrate are calculated on the basis of the chemical analyses and the assumption that the sphalerite is pure ZnS, the figures in the first column of Table II are obtained. The third column shows the errors calculated for the microscopic analyses.

TABLE II

Comparison of Calculated and Microscopically Determined Sphalerite Contents

(Percentages)

Sample No.	Sphalerite present calculated from chemical analyses	Sphalerite determined microscopically	Calculated error in microscopic determination
2	44·56	41·9	− 5·7
4	64·80	58·1	−10·3
7	78·65	82·0	+ 4·4

DISCUSSION

As already noted, the variation in the zinc content of the samples of concentrate might be due to:

(a) Variation in the iron content of the sphalerite.

(b) Dilution of the concentrate by other minerals.

Although the iron content of the sphalerite is known to vary within any one sample, no preponderance of either the iron-rich or the iron-lean varieties could be observed in the low-zinc and high-zinc concentrates respectively. Rather it seems certain that such variation is of little or no consequence in determining the grade of the concentrate. On the other hand, the grade as determined chemically is roughly proportional to the quantity of sphalerite as determined microscopically, and as the zinc content rises the quantities of pyrite and gangue decrease. As would be expected, there is a slight increase in the quantity of chalcopyrite with increase in zinc. Although the errors in the microscopic determination are not excessive, Table II shows that they are the opposite from what would be expected if the low-zinc concentrate were due to high-iron sphalerite. In other words, more iron-rich sphalerite would be required to furnish a certain amount of zinc than would be the case with iron-lean sphalerite; therefore, the microscope should be expected to show an excess of sphalerite in the low-zinc concentrate and a dearth of sphalerite in the high-zinc concentrate.

CONCLUSIONS

The above discussion leads to the following conclusions:

1. There is much variation in the iron content of the sphalerite; the lower limit is not known, but the upper limit is probably somewhat less than 14 per cent.

2. The variation in grade of the zinc concentrate is not due to the presence in varying quantity of iron-rich and iron-lean varieties of sphalerite.

3. The variation in the grade of the concentrate is directly due to dilution by other minerals, notably pyrite and gangue. In this connection pyrite is the more important.

4. The pyrite is largely free from the sphalerite, and it should thus be possible to improve the grade by improving the concentration practice.

Ore Dressing and Metallurgical Investigation No. 736

GOLD-LEAD-ZINC ORE FROM THE CARIBOO-HUDSON GOLD MINES, LIMITED, CUNNINGHAM, BARKERVILLE DISTRICT, BRITISH COLUMBIA

Shipment. A shipment of gold-lead-zinc ore consisting of 4 bags, weight 312 pounds, and one box of specimens, was received on November 3, 1937, from the Cariboo-Hudson Gold Mines, Limited, Wells, British Columbia.

Characteristics of the Ore. Twenty-four polished sections of the ore were prepared and examined microscopically.

The *gangue* is white vein quartz with a minor quantity of coarsely crystalline, light brown ferruginous dolomite(?).

The *metallic minerals* are:

Pyrite: abundant.
Sphalerite: considerable quantity.
Galena: considerable quantity, less than sphalerite.
Marcasite: small quantity.
Pyrrhotite: small quantity, locally abundant.
Native gold.

Pyrite is present as finely crystalline masses and medium to fine disseminated grains; the masses contain numerous inclusions of gangue and are somewhat fractured with both sphalerite and galena present in fractures.

Sphalerite and galena occur as masses and medium to fine grains; they often form masses together with the sphalerite predominating, and this assemblage usually contains numerous small disseminated grains of pyrite.

Marcasite occurs as irregular grains and small masses associated with pyrite, which it appears to have replaced. In most sections pyrrhotite is present only as small grains in sphalerite; in a few, however, it is present as large masses associated with massive galena.

Native gold is present as irregular grains in quartz, sphalerite, galena, and pyrite. The grain analysis is shown in the following table:

Mesh	In quartz, per cent	In sphalerite, per cent	In galena, per cent	In pyrite, per cent		Totals, per cent
				In cracks	In dense pyrite	
+ 65........	21·9	8·2	....	....	....	30·1
− 65+ 100........	5·2	2·8	2·7	....	....	10·7
− 100+ 150........	3·6	3·3	1·6	....	....	8·5
− 150+ 200........	7·3	4·5	3·2	1·4	....	16·4
− 200+ 280........	5·5	1·6	....	....	....	7·1
− 280+ 400........	1·2	1·2	....	2·6	....	5·0
− 400+ 560........	3·0	2·3	....	2·7	....	8·0
− 560+ 800........	2·1	1·0	....	2·1	0·4	5·6
− 800+1100........	2·5	0·8	0·6	0·6	0·2	4·7
−1100+1600........	1·2	0·7	0·4	0·5	0·3	3·1
−1600+2300........	0·2	....	0·1	0·1	0·1	0·5
−2300..............	....	....	0·1	0·1	0·1	0·3
Totals.......	53·7	26·4	8·7	10·1	1·1	100·0
				11·2		

Sampling and Assaying. The ore was crushed and sampled by standard methods and the analysis is as follows:

Gold..	0·97 oz./ton
Silver..	0·88 "
Lead..	2·12 per cent
Zinc..	2·42 "
Iron..	9·57 "

Purpose of Investigation and Results. The purpose of the investigation was to determine the most suitable flow-sheet for the ore, with special attention to the possible use of gravity table concentration.

The results indicate that the ore is not amenable to satisfactory table concentration. The loss of valuable metals in the slime is high and it is not possible to make a reasonable recovery. The gold is distributed in all the sulphide minerals, thus rendering its concentration in any one sulphide concentrate impossible. By selective flotation lead, zinc, and pyrite concentrates were made, the lead and zinc being of shipping grade.

Barrel amalgamation of the raw ore indicated that at a grinding of 60 per cent minus 200 mesh 67 per cent of the gold was free-milling.

The report is divided into two parts: Part I, covering concentration, and Part II, covering cyanidation.

Experimental Investigation

Part I

GRAVITY TABLE CONCENTRATION

The results of tabling the ore on a small laboratory Wilfley table using an unsized feed were unsatisfactory.

A series of tests using a sized feed resulted in low-grade bulk concentrates and high slime losses on the finest product. A 100-pound sample was run over a standard Wilfley table, resulting in a fair grade gold-lead concentrate, but the recoveries were low.

The results of these tests are given below:

Series of Table Tests on Sized Feed

A sample of ore was crushed in rolls to minus 35 mesh and screened to yield four products. The percentage weight of each screened product was as follows:

Mesh	Weight, per cent
− 35+ 48	22·2
− 48+ 65	23·0
− 65+100	16·1
−100	38·7
	100·0

Each product was tabled separately with the following results:

Test A: −35 +48 Mesh

Product	Weight, per cent	Assay				Distribution, per cent		
		Oz./ton		Per cent				
		Au	Ag	Pb	Zn	Au	Pb	Zn
Feed	100·00	0·83	....	1·45	1·89	100·0	100·0	100·0
Concentrate	16·22	4·26	3·70	7·85	8·60	83·3	87·7	73·4
Middling	16·62	0·55	0·40	0·87	2·63	11·0	10·0	23·0
Tailing	67·16	0·07	....	0·05	0·10	5·7	2·3	3·6

At this size no mineral separation was observed on the deck of the table.

Test B: −48 +65 Mesh

Product	Weight, per cent	Assay				Distribution, per cent		
		Oz./ton		Per cent				
		Au	Ag	Pb	Zn	Au	Pb	Zn
Feed	100·00	0·71	....	1·66	2·00	100·0	100·0	100·0
Concentrate	11·89	3·91	3·33	10·91	4·95	65·3	78·0	29·4
Middling	16·54	1·21	0·60	1·43	8·10	28·1	14·2	67·0
Tailing	71·57	0·065	....	0·18	0·10	6·6	7·8	3·6

A slight mineral separation was noted on the table.

Test C: −65 +100 Mesh

Product	Weight, per cent	Assay				Distribution, per cent		
		Oz./ton		Per cent				
		Au	Ag	Pb	Zn	Au	Pb	Zn
Feed	100·0	0·78	....	2·09	2·30	100·0	100·0	100·0
Concentrate	7·2	4·96	4·02	14·22	5·72	45·9	48·9	17·9
Middling	25·7	1·34	0·78	2·50	6·58	44·2	30·6	73·4
Tailing	67·1	0·115	....	0·64	0·30	9·9	20·5	8·7

Test D: −100 Mesh

Product	Weight, per cent	Assay					Distribution, per cent		
		Oz./ton		Per cent					
		Au	Ag	Pb	Zn	Fe	Au	Pb	Zn
Feed	100·00	0·90	....	3·36	2·97	....	100·0	100·0	100·0
Concentrate	10·67	5·12	4·60	18·52	4·40	35·50	60·7	58·8	15·8
Middling	11·82	1·06	0·75	2·96	10·37	18·90	13·9	10·4	41·3
Tailing	48·78	0·28	....	0·92	0·46	2·41	15·2	13·4	15·8
Slime	28·73	0·32	....	2·04	2·79	5·23	10·2	17·4	27·1

Screen Test on Table Tailing, −100 Mesh

Mesh	Weight, per cent
+100	1·2
−100+150	35·1
−150+200	26·3
−200	37·4
Total	100·0

Combining the results of these four tests, the distribution of metals in the table products is as follows:

Combined table products	Distribution, per cent		
	Gold	Lead	Zinc
Feed	100·0	100·0	100·0
Concentrate	64·4	68·1	32·1
Middling	21·4	14·4	48·3
Tailing	14·2	17·5	19·6

The combined concentrates have the following analysis:

Gold	4·63 oz./ton
Lead	13·71 per cent
Zinc	5·67 "

Test MR-1

A sample of minus 14-mesh ore, 100 pounds in weight, was screened on a 48-mesh screen and the oversize reduced in rolls to minus 48 mesh. The minus 48-mesh product was fed to a standard Wilfley table. All products were recovered and sampled, with the exception of the slime, which was sampled only.

A high-grade gold-lead concentrate was made, but a selective concentrate of the sphalerite was not effected. The results are tabulated as follows:

Product	Weight, per cent	Assay					Distribution, per cent		
		Oz./ton		Per cent					
		Au	Ag	Pb	Zn	Fe	Au	Pb	Zn
Feed	100·00	0·97	0·88	2·12	2·42	9·57	100·0	100·0	100·0
Concentrate No.1	0·42	29·20	22·90	59·56	1·72	13·70	15·3	13·7	0·4
Concentrate No.2	1·84	6·38	5·34	19·00	4·90	32·65	14·7	19·2	4·9
Concentrate No.3	12·00	3·44	....	6·22	10·02	34·00	51·5	41·0	65·9
Middling	2·50	0·47	....	0·68	4·40	6·10	1·5	0·9	6·0
Tailing	63·00	0·12	....	0·25	0·03	1·28	9·4	8·7	1·0
Slime	20·24*	0·30	....	1·48	1·95	4·21	7·6	16·5	21·8

*By difference.

Screen Test on Table Tailing:

Mesh	Weight, per cent
+ 48	5·3
− 48+ 65	30·8
− 65+100	27·5
−100+150	18·6
−150+200	8·6
−200	9·2
Total	100·0

FLOTATION TESTS

A series of flotation tests was carried out on the ore with the object of making selective concentrates of the galena, sphalerite, and pyrite. In each test the coarse free gold was first removed in a hydraulic classifier and the overflow floated. Concentrates of fair grade were made. Most of the gold was recovered in the lead concentrate.

The results of the flotation tests are summarized in the following tables:

Free Gold Recoveries by Hydraulic Classification:

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
1	Underflow	0·19	101·11	18·1	526 : 1
	Overflow	99·81	0·87	81·9	
2	Underflow	0·03	755·68	25·6	3333 : 1
	Overflow	99·97	0·66	74·4	
3	Underflow	0·07	313·99	20·0	1428·5 : 1
	Overflow	99·93	0·88	80·0	
4	Underflow	0·04	450·93	18·0	2500 : 1
	Overflow	99·96	0·82	82·0	

The fineness of grinding for the respective tests is indicated by the following screen tests on the flotation tailings:

Screen Tests:

Mesh	Weight, per cent			
	Test No. 1	Test No. 2	Test No. 3	Test No. 4
+ 65	1·7	0·1	0·4	....
− 65+100	8·7	1·4	4·1	1·6
−100+150	17·5	9·1	12·4	8·9
−150+200	17·4	14·6	15·7	14·4
−200	54·7	74·8	67·4	75·1
Total	100·0	100·0	100·0	100·0

Flotation Results:

Test No.	Product	Weight, per cent	Assay, Oz./ton, Au	Assay, Oz./ton, Ag	Assay, Per cent, Pb	Assay, Per cent, Zn	Assay, Per cent, Fe	Distribution, per cent, Au	Distribution, per cent, Pb	Distribution, per cent, Zn	Ratio of concentration
1	Feed	100·00	0·87	0·91	2·51	2·60	....	100·0	100·0	100·0	
	Lead concentrate	5·93	12·22	13·68	39·77	6·78	17·37	83·6	94·0	15·5	16·86 : 1
	Zinc concentrate	5·89	0·97	1·13	1·27	34·50	19·78	6·6	3·0	78·2	16·98 : 1
	Pyrite concentrate	11·39	0·61	0·27	0·25	1·11	39·68	8·0	1·1	4·9	
	Tailing	76·79	0·02	....	0·06	0·05	1·45	1·8	1·9	1·4	
2	Feed	100·00	0·66	....	2·42	2·50	....	100·0	100·0	100·0	
	Lead cleaner concentrate	7·07	7·08	7·64	31·42	4·96	24·76	76·1	91·8	14·0	14·14 : 1
	Lead middling	0·56	2·20	....	6·88	4·21	24·00	1·9	1·6	0·9	
	Zinc cleaner concentrate	3·37	1·65	1·71	1·78	55·76	7·14	8·5	2·5	75·2	29·67 : 1
	Zinc middling	0·62	0·77	....	1·84	9·21	10·25	0·7	0·5	2·3	
	Pyrite concentrate	11·71	0·62	....	0·36	1·16	40·84	11·0	1·7	5·4	
	Tailing	76·67	0·015	....	0·06	0·07	1·20	1·8	1·9	2·2	
3	Feed	100·00	0·88	....	2·70	2·62	....	100·0	100·0	100·0	
	Lead cleaner concentrate	3·70	16·12	19·22	61·00	5·80	8·60	67·4	83·6	8·2	27 : 1
	Lead middling	0·48	11·29	....	17·44	11·38	20·60	6·1	3·1	2·1	
	Zinc cleaner concentrate	3·80	2·18	2·28	4·60	52·22	7·20	9·4	6·5	75·8	26 : 1
	Zinc middling	0·36	2·78	....	4·00	7·72	17·80	1·1	0·5	1·1	
	Pyrite concentrate	15·40	0·67	....	0·61	1·83	41·40	11·7	3·5	10·8	
	Tailing	76·26	0·05	....	0·10	0·07	....	4·3	2·8	2·0	
4	Feed	100·00	0·82	....	2·70	2·60	....	100·0	100·0	100·0	
	Lead cleaner concentrate	4·75	13·86	15·72	47·63	10·00	11·20	80·4	83·8	18·2	21·05 : 1
	Lead middling	0·22	7·14	....	9·68	11·72	....	1·9	7·9	9·9	
	Zinc cleaner concentrate	3·18	1·06	1·40	1·32	53·44	8·28	4·2	1·6	65·4	31·45 : 1
	Zinc middling	0·46	1·13	....	2·00	4·60	14·10	0·6	0·3	0·8	
	Pyrite concentrate	7·49	0·96	....	0·97	1·62	37·40	8·8	2·7	4·7	
	Tailing	83·90	0·04	....	0·12	0·03	....	4·1	3·7	1·0	

Reagent Consumption:

Test No.	—	Reagents consumed, lb./ton of ore										Remarks
		Soda ash	Lime	Zinc sulphate	Sodium cyanide	Aerofloat No. 25	Butyl xanthate	Amyl xanthate	Copper sulphate	Cresylic acid	Pine oil	
1	Grinding	2·0	...	...	...	...	...	...	...	...	...	
	Lead concentrate	3·0	...	1·0	...	...	0·04	...	...	0·264	...	
	Zinc concentrate	...	2·0	...	...	...	...	...	0·5	0·264	0·025	
	Pyrite concentrate	...	...	...	...	...	...	0·1	1·0	...	...	
2	Grinding	...	...	...	...	...	...	...	...	...	...	
	Lead rougher	5·0	...	1·0	...	...	0·04	...	...	0·176	...	
	Lead cleaner	...	...	0·1	...	...	...	...	...	0·040	...	
	Zinc rougher	...	2·0	...	...	...	...	...	0·5	...	...	
	Zinc cleaner	...	0·1	...	...	...	...	...	...	...	0·025	
	Pyrite concentrate	...	...	...	...	...	...	0·2	1·0	...	0·025	
3	Grinding	...	...	...	...	...	...	...	...	...	...	
	Lead rougher	5·0	...	1·0	0·1	...	0·04	...	...	0·264	...	Cyanide drops pyrite.
	Lead cleaner	...	...	0·1	...	...	...	...	...	...	...	
	Zinc rougher	...	2·0	...	...	...	...	...	0·5	...	0·050	
	Zinc cleaner	...	0·1	...	...	...	...	...	...	...	...	
	Pyrite concentrate	...	...	...	...	...	...	0·3	1·0	...	0·025	
4	Grinding	5·0	...	...	...	0·07	...	...	...	...	...	Aerofloat lowers gold in tailing.
	Lead rougher	1·0	...	1·0	0·1	...	...	...	...	0·176	...	
	Lead cleaner	...	...	0·1	...	...	...	...	...	0·044	...	
	Zinc rougher	...	2·0	...	...	...	...	...	0·5	...	0·050	
	Zinc cleaner	...	0·1	...	...	...	...	...	...	0·044	...	
	Pyrite concentrate	...	...	...	...	...	...	0·3	1·0	0·044	0·025	

Summary of Gold Recovery (in Percentage):

Products	Test No. 1	Test No. 2	Test No. 3	Test No. 4
Gold recovered in hydraulic classifier	18·1	25·6	20·0	18·0
Gold recovered in lead concentrate	68·5	58·0	58·8	67·5
Gold recovered in zinc concentrate	5·4	6·8	8·4	3·9
Gold recovered in pyrite concentrate	6·6	8·2	9·4	5·5
Overall recovery	98·6	98·6	96·6	94·9

In the above table the percentage of gold shown in the lead and zinc concentrates represents the gold in the combined cleaner concentrates and middlings.

AMALGAMATION AND CYANIDATION TESTS

Amalgamation and cyanidation tests were carried out on the raw ore to determine the relative amount of free-milling gold and to determine the extraction of gold by cyanide on the amalgamation tailing and the consumption of reagents involved.

The results indicate that cyanidation of the raw ore at moderately fine grinding is satisfactory and should provide a possible method of treatment if the lead content of the ore falls off as development of the mine progresses.

Amalgamation Test

A sample of ore was ground to have 60 per cent minus 200 mesh and then barrel-amalgamated with mercury for one hour.

Gold in feed	0·97 oz./ton
Gold in amalgamation tailing	0·32 "
Recovery	67·0 per cent

Cyanidation Test No. 1

Part of the above amalgamation tailing was agitated in cyanide for 24 hours. The strength of the solution was equivalent to 1 pound of potassium cyanide per ton and the pulp dilution was 1·5 : 1.

The results were as follows:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
Feed	Tailing		KCN	CaO
0·32	0·035	89·0	0·62	3·80

Cyanidation Test No. 2

The rest of the amalgamation tailing was reground to have 75·9 per cent minus 200 mesh and two lots were cyanided for 24 and 48 hours respectively.

The results were as follows:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
24..........	0·32	0·03	90·6	0·61	3·88	1·5 : 1
48..........	0·32	0·025	92·2	0·77	4·18	1·5 : 1

DISCUSSION OF RESULTS

The results obtained from the sample of ore submitted indicate that gravity concentration on tables is not satisfactory. From a large-scale table test (*see* Test MR-1) although a good grade lead concentrate can be made, the recovery of lead is only 13·7 per cent. The slime loss is high and there is no concentration of the sphalerite.

Selective flotation offers a satisfactory method of treatment. From 20 to 25 per cent of the free gold can be removed in jigs or traps prior to flotation. Moderate grinding, around 75 per cent minus 200 mesh, is sufficient to dissociate the different sulphides.

Three products were made by flotation, lead, zinc, and pyrite concentrates. The tailings were as low as 0·015 ounce of gold per ton and the ratios of concentration were satisfactory. The first two sulphides provide shipping concentrates, and the pyrite concentrate is amenable to treatment by cyanide.

The microscopic examination shows that the gold is associated with all three sulphides. This is verified by the results obtained by selective flotation.

The flotation results show that from 60 to 65 per cent of the gold is recovered in the lead concentrate, 5 to 7 per cent in the zinc concentrate, and from 6 to 8 per cent in the pyrite concentrate. The tailing loss amounts to from 1·5 to 3·5 per cent.

Reagent control is an important consideration as the table on reagent consumption indicates. Addition of 0·1 pound of cyanide per ton in the lead rougher float helps to depress more pyrite and thus makes a cleaner lead concentrate. It probably tends to increase slightly the pyrite in the tailing, thus accounting for an increased gold loss in the tailing as shown in Tests Nos. 3 and 4.

The results obtained, especially in regard to flotation, would not necessarily apply should the grade of the ore change at some future time.

The results of amalgamation and cyanidation of the raw ore indicate the possibilities of such a method of treatment.

Amalgamation of the free-milling gold and the making of a lead concentrate for shipment would give a gold recovery of from 80 to 85 per cent. The remaining 15 to 20 per cent of gold is contained in the zinc and pyrite concentrates and tailings. (*See* table on page 38.)

Amalgamation of the free-milling gold followed by cyanidation shows an overall recovery of 97 per cent on ore of the grade submitted for investigation.

Part II

The treatment of the ore by direct cyanidation is subject to several considerations. The most important of these is control of lime. Tests according to the normal practice of grinding in a lime-cyanide pulp showed a very low extraction. If no lime were added the extractions were high, but settling was poor and cyanide consumption high. It is evident that the essential requirement is to determine the minimum amount of lime necessary to promote satisfactory settling and provide protective alkalinity without lowering the extraction of gold. The tests carried out to meet these conditions gave fairly satisfactory results.

Why the lime should lower the gold extraction is not definitely known, but it is probably due to the formation of a protective coating on the gold particles caused by the combination of free lime with one or more of the ore constituents.

The mineragraphic examination of the ore discloses the presence of marcasite and pyrrhotite, both of which are cyanicides and add to the difficulties of direct cyanidation. Tests of the pulp showed the presence of sulphates, and a pH of the pulp in distilled water was 6·0, indicating a slight acidity.

The results of experimental tests are shown in detail below:

GRINDING WITH CYANIDE AND LIME

Samples of ore were ground with cyanide, 1 pound of potassium cyanide per ton and 5 pounds of lime per ton ore, at a dilution of 0·75 : 1. The pulp after grinding was made up to a dilution of 1·5 : 1 and agitated for 24 and 48 hours. The cyanide strength during agitation was maintained at 1 pound of potassium cyanide per ton.

Results of Tests:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, of gold, per cent	Titration, CaO lb./ton	Reagents consumed, lb./ton ore	
		Feed	Tailing			KCN	CaO
S-1.....	24	0·97	0·71	26·8	0·62	0·45	4·07
S-2.....	48	0·97	0·67	30·9	0·50	0·57	4·25
S-3.....	24	0·97	0·70	27·8	0·48	0·57	4·28
S-4.....	48	0·97	0·67	30·9	0·46	0·71	4·31

The fineness of grinding is indicated by the following screen tests:

Mesh	Weight, per cent	
	Tests Nos. S-1 and S-2	Tests Nos. S-3 and S-4
+ 65	0·5	....
− 65+100	5·8	1·0
−100+150	11·5	6·0
−150+200	24·1	18·7
−200	58·1	74·3
	100·0	100·0

An analysis of the solutions from the 48-hour tests gives the following results:

—	Test No. S-2	Test No. S-4
Reducing power	160 c.c. $\frac{N}{10}$ $KMnO_4$/litre	
KCNS	0·19 grm./litre	0·19 grm./litre
Ferrous iron	Nil	Nil

Test No. S-5

A sample of ore, weight 2,000 grammes, was ground in water and the pulp fed to a laboratory Denver gold jig. The hutch product, weight 4·2 grammes, assayed in gold 111·12 ounces per ton. The overflow was dewatered, and a sample cut out for assay gave 0·79 ounce of gold per ton. A portion of the jig overflow was agitated in cyanide at a 1·5 : 1 dilution for 48 hours. The solution strength was 1 pound of potassium cyanide per ton and 2 pounds of lime per ton was added as protective alkalinity.

The rest of the overflow was reground and two lots were cyanided, one for 24 hours using no lime and one for 48 hours using 1 pound of lime per ton.

The results are as follows:

Lime, lb./ton	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, CaO lb./ton	Reagents consumed, lb./ton		Pulp dilution
		Feed	Tailing			KCN	CaO	
0·0	24	0·79	0·03	96·2	0·0	2·40	....	1·5 : 1
1·0	48	0·79	0·025	96·8	0·25	1·52	1·63	1·5 : 1
2·0	48	0·79	0·03	96·2	0·30	1·02	2·55	1·5 : 1

The first two tests are on reground ore.

The fineness of grinding is indicated by the following screen tests:

Screen Tests:

Mesh	Weight, per cent	
	Primary grind	Secondary grind
+ 65	0·1	....
− 65+100	1·2	0·4
−100+150	9·7	6·2
−150+200	14·8	12·1
−200	74·2	81·3
	100·0	100·0

The above tests show a very decided improvement in the gold extraction by using no lime in the grinding circuit. It is evident that lime in the agitation circuit in amounts not exceeding 2 pounds per ton of ore and titrating not over 0·3 pound of lime per ton does not affect the extraction of gold adversely.

Test No. S-6 (Grinding in Cyanide)

In order further to check cyanidation without lime a sample of ore was ground in a 0·75 : 1 pulp with 1 pound of potassium cyanide per ton to a fineness of 76·2 per cent minus 200 mesh. The pulp was made up to a dilution of 1·5 : 1 and agitated for 48 hours. High consumption of cyanide is to be expected owing to the acidity of the pulp.

Results:

Assay, Au, oz./ton		Extraction of gold, per cent	Cyanide consumption, lb./ton ore
Feed	Tailing		
0·97	0·025	97·4	4·87

Test No. S-7 (Grinding in Water)

The ore was ground in a water pulp and the pulp filtered and the cake washed. The washed cake was repulped at a dilution of 1·5 : 1 and cyanided without lime for 24 hours in solution of strength equivalent to 2 pounds of potassium cyanide per ton.

The results are as follows:

Assay, Au, oz./ton		Extraction of gold, per cent	Cyanide consumption, lb./ton ore
Feed	Tailing		
0·97	0·025	97·4	3·35

The gold extraction is the same as in Test S-6. The settling in both tests was very poor owing to the absence of lime. The results indicate that water washing of the pulp has no influence on the extraction, but lowers somewhat the cyanide consumption.

Test No. S-8

For the purpose of examining the free gold a sample of ore (minus 14 mesh) was run over the Denver gold jig and the hutch product was panned on the Haultain superpanner. One coarse grain and several fine grains were observed, and under the binocular microscope they were seen to be dull and tarnished.

Test No. S-9

In order to determine the minimum of lime necessary for promoting satisfactory settling, 0·1-gramme additions of lime were made to a 3 : 1 pulp and titrations for lime were made after each addition.

For the first three additions the titrations for lime were zero. Settling was fairly rapid and the pH of the solution was 8·6. An additional 0·20 gramme of lime showed a titration of 0·025 pound of lime per ton of solution and a pH of 8·8.

Similar tests were made at a dilution of 1·5 : 1 and a total addition of 0·40 gramme of lime gave a titration of 0·05 pound of lime per ton of solution.

These tests indicated that from 0·8 to 1·0 pound lime per ton of ore was sufficient to promote satisfactory settling and provide protective alkalinity.

Test No. S-10

This was a cycle test to determine the extraction of gold, settling conditions of the pulp, and fouling of the solution. No free lime was added to the grinding circuit in the initial cycle. Lime was added during the run in small amounts to promote settling. In the second and third cycles barren solution was used, but no free lime was added during grinding. This is equivalent to adding lime at the agitation and washing thickeners in plant operation.

The ore was ground to a fineness of 78·3 per cent minus 200 mesh in cyanide solution at a dilution of 0·75 : 1. The pulp was transferred to a bottle and diluted to 2·29 : 1 and 0·4 gramme of lime (0·8 pound per ton of ore) was added. The solution titrated 0·05 pound of lime per ton and had a pH of 9·1. Cyanide was added to give a strength of 2 pounds of potassium cyanide per ton and agitation was carried out for 48 hours.

The final solution titrated as follows:

	Lb./ton
KCN	2·0
CaO	0·12

Settling was good. Overflow slightly cloudy. The solution was treated with zinc dust and lead acetate to precipitate the gold, and filtered.

The second cycle was carried out similarly to the first. Fresh ore was ground in barren solution from Cycle No. 1 and dilutions made with barren solution. Lime was added at the beginning of the agitation period and once during the 48-hour run.

The third cycle was carried out similarly to the second.

The following is a summary of the three cycles:

Cycle No.	Lime, lb./ton ore	Final titration, lb./ton solution		Assay, Au, oz./ton		Extraction, of gold, per cent
		KCN	CaO	Feed	Tailing	
1	1·6	2·00	0·12	0·97	0·03	96·7
2	1·6	1·80	0·10	0·97	0·035	96·2
3	0·8	1·84	0·05	0·97	0·04	95·8

Analysis of Final Solution (3rd Cycle):

Reducing power	476 c.c. $\frac{N}{10}$ $KMnO_4$/litre
KCNS	0·51 grm./litre
Iron (ferrous)	0·17 "

There is a slight indication of lowered extraction with each cycle.

Test No. S-11

A further cycle test was carried out, following the same practice of keeping the lime at a minimum and adding no free lime to the grinding circuit. Agitation was for 24-hour periods and before filtering the dilution was increased to 3 : 1. The solution from the first cycle was de-aerated and the gold precipitated. The barren solution was then aerated for 1 hour. The aerated barren solution was used in grinding of fresh ore and subsequent dilutions.

A similar procedure was carried out on the solutions of each cycle.

To the barren solution from Cycle No. 3, 0·55 gramme of ammonium chloride was added. This is roughly 0·36 pound per ton of solution. The object of adding the ammonium chloride was to reduce the free lime content of the solution, as follows:

$$2\,NH_4Cl + Ca(OH)_2 \longrightarrow 2\,NH_4OH + CaCl_2\,.$$

In Cycles Nos. 5 and 6, soda ash (Na_2CO_3) was added to the grinding circuit in amounts equivalent to 0·5 pound per ton of ore. Soda ash will react with lime as follows:

$$Na_2CO_3 + Ca(OH)_2 \longrightarrow CaCO_3 + 2\,NaOH.$$

The use of soda ash made settling very difficult and increased additions of lime were necessary to bring about moderately satisfactory settling.

The results of the cycle tests are as follows:

Cycle No. 1

Total cyanide added	1·74	grms.
Total lime added	0·40	"
Final titration: KCN	1·40	lb./ton
CaO	0·1	"
Gold in tailing	0·035	oz./ton
Extraction	96·2	per cent

Cycle No. 2

Total cyanide added	1·50	grms.
Total lime added	1·00	"
Final titration: KCN	1·92	lb./ton
CaO	0·16	"
Gold in tailing	0·035	oz./ton
Extraction	96·2	per cent

Settling slower than in Cycle No. 1.

Cycle No. 3

Total cyanide added	1·26	grms.
Total lime added	1·40	"
Final titration: KCN	2·68	lb./ton
CaO	0·56	"
Gold in tailing	0·04	oz./ton
Extraction	95·8	per cent

Settling was slow.

Titration of de-aerated barren solution:	KCN	1·55	lb./ton
	CaO	0·35	"
Titration of aerated barren solution:	KCN	1·40	"
	CaO	0·40	"

Cycle No. 4

Ammonium chloride added to barren solution	0·55 grm.
Total cyanide added	1·26 "
Total lime added	0·8 "
Final titration: KCN	2·08 lb./ton
CaO	0·28 "
Gold in tailing	0·04 oz./ton
Extraction	95·8 per cent

Settling slow.

Titration of de-aerated barren solution:	KCN	1·40 lb./ton
	CaO	0·30 "
Titration of aerated barren solution:	KCN	1·35 "
	CaO	0·25 "

Cycle No. 5

Soda ash added to grinding circuit	0·5 lb./ton ore
Total cyanide added	0·78 grm.
Total lime added	1·40 "
Final titration: KCN	1·92 lb./ton
CaO	0·52 "
Gold in tailing	0·04 oz./ton
Extraction	95·8 per cent

Settling very poor.

Cycle No. 6

Soda ash added to grinding circuit	0·5 lb./ton ore
Total cyanide added	0·78 grm.
Total lime added	0·80 "
Final titration: KCN	1·96 lb./ton
CaO	0·24 "
Gold in tailing	0·04 oz./ton
Extraction	95·8 per cent

Settling was slow, but lime added was sufficient to promote fair settling.

The solutions in all cycles filtered clear.

The analysis of the final solution was as follows:

Titration of pregnant solution:	KCN	1·48 lb./ton
	CaO	0·30 "
Titration of aerated solution:	KCN	1·26 "
	CaO	0·32 "

Reducing power	343 c.c. $\frac{N}{10}$ $KMnO_4$/litre
KCNS	0·55 grm./litre
Total iron	0·049 "
Zinc	0·364 "
H_2S metals determined by spectroscope:	
Copper	Moderately strong
Lead, silver, gold	Traces
Tin	Faint trace

The results from the above cycles show that after the second cycle there is no rise in the final tailing. The total lime added for settling and protective alkalinity is just under 2·0 pounds per ton. This confirms the results shown in a previous test on the lime necessary for settling the pulp. The titrations for lime on Cycles Nos. 3 and 5 are slightly over 0·5 pound per ton, but in these tests the lime additions were also higher. In all the other cycles the lime titrates under 0·30 pound per ton of solution.

The addition of soda ash increased the normally poor settling property of the pulp, but did not raise the gold in the tailing. Although more lime was required to promote settling, its subsequent removal by the soda ash keeps it within the limits mentioned above.

The analysis of the final solution shows a lower reducing power than the previous cycle test (S-10). This is probably due to the aeration of the barren solution carried out during the cycles of this test.

CONCLUSIONS

The treatment of this ore will require very careful control in mill operation. Free lime should not be added to the grinding circuit and the free lime content of the grinding solution should be kept under 0·5 pound of lime per ton of solution.

Lime may be added for settling prior to thickening and clarifying, but in amounts not to exceed 2 pounds per ton of ore feed. It must not be assumed that this amount of lime will promote perfect settling, but a balance will have to be kept between maximum extraction and permissible operating settling.

It has been shown that ammonium chloride or soda ash can be added to the circuit for removal of excess lime without adversely affecting the extraction. It should not be necessary to add these reagents continuously to the circuit but they may be of value in controlling the lime content of the solutions should they exceed the limits as indicated above.

The acidic character of this ore indicates that it is of surface origin and it would be expected that at greater depth this acidity would diminish.

Ore Dressing and Metallurgical Investigation No. 737

GOLD ORE FROM THE CAMLAREN MINE AT GORDON LAKE, N.W.T.

Shipment. A shipment of 29 sacks of ore, net weight 1,750 pounds, was received on January 26, 1938. The shipment was submitted by A. K. Muir, Manager, Camlaren Mines, Limited, Gordon Lake, N.W.T.

Location of Property. This property is located at Gordon Lake, in the Yellowknife area of the Northwest Territories.

Characteristics of the Ore. Six polished sections of the ore were prepared and examined microscopically to determine the character of the ore.

The gangue of the ore is white vein quartz, which in places is somewhat rusty.

Figure 1. Drawing of polished section of gold ore from Camlaren Mines, Yellowknife, N.W.T. Pyrite—crossed; galena—dotted; pyrrhotite—waved lines; gold—black; gangue—white. Magnification: X 50 approximately.

The metallic minerals are: pyrite, pyrrhotite, marcasite, arsenopyrite, galena, sphalerite, chalcopyrite, and native gold. The sulphides are not abundant throughout most of the ore, but occur admixed along irregular stringers in which they form small masses. The total quantity of sulphides is small, pyrite, pyrrhotite, marcasite, arsenopyrite, galena, and sphalerite predominating. Chalcopyrite is comparatively rare.

The modes of occurrence of the native gold are quite varied. It is present as irregular grains and veinlets along fine sinuous fissures in the quartz; as grains bordering and enclosed by galena, sphalerite, and pyrrhotite; as grains and films in contact with the surface of pyrite grains; as veinlets in pyrite; and as small grains that appear to be enclosed in dense pyrite.

All of the evidence points to the fact that the deposition of the gold took place almost contemporaneously with the galena-sphalerite deposition and later than the arsenopyrite-pyrite deposition, and it is probable that the small quantity of gold that appears to be contained within dense pyrite may have been introduced along fractures that are not apparent on the polished surface. The grain size of the gold is given in the following table:

Grain Size of the Native Gold:

Mesh	Isolated from sulphides in quartz, per cent	Associated with sulphides, per cent						Total, per cent
		With galena	With sphalerite	With pyrrhotite	With pyrite			
					As veinlets	In dense pyrite	Along borders of grains	
+ 65	9·0	13·8	....	....	....	....	2·3	25·1
− 65+ 100	3·2	5·8	1·8	....	1·8	....	1·7	14·3
− 100+ 150	3·4	....	1·4	1·7	1·2	....	2·0	9·7
− 150+ 200	6·2	3·8	....	....	0·8	0·7	1·5	13·0
− 200+ 280	5·3	2·4	0·6	....	1·2	0·7	1·0	11·2
− 280+ 400	5·1	1·6	0·7	....	0·8	0·8	0·9	9·9
− 400+ 560	4·0	0·3	0·7	....	1·2	0·9	0·9	8·0
− 560+ 800	2·4	0·2	0·7	0·2	0·6	0·8	0·8	5·7
− 800+1100	0·8	0·1	0·1	0·1	0·4	....	0·3	1·8
−1100+1600	0·6	0·2	....	....	0·2	....	....	1·0
−1600	....	0·1	....	....	....	0·2	....	0·3
Totals	40·0	23·3	6·0	2·0	8·2	4·1	11·4	100·0
					23·7			

Sampling and Assaying. The sample was crushed and assayed as follows:

Gold	2·65 oz./ton
Silver	0·76 "
Copper	0·03 per cent
Iron	1·28 "
Sulphur	0·24 "
Arsenic	Trace

Experimental Tests

As the ore contains a lot of coarse free gold this was collected by the use of a gold jig followed by blanket concentration. The blanket tailings at various grindings were treated by cyanidation and the jig and blanket concentrates were barrel-amalgamated and the amalgamation tailing treated by cyanidation.

In this manner as much as 86 per cent of the gold was recovered in the jig and blanket concentrates and nearly all is recoverable as bullion by amalgamation. By cyanidation of the residue total extraction exceeds 99 per cent.

When the ore is ground 1·5 per cent on 65 mesh and 52 per cent through 200 mesh 91·5 per cent of the gold is free and recoverable by barrel amalgamation and 99 per cent of it will dissolve in cyanide solution.

The tests are described in detail as follows:

CYANIDATION

Tests Nos. 1 to 4

Samples of the ore were ground in cyanide solution, 1·0 pound per ton of potassium cyanide, to give a product 1·5 per cent on 65 mesh and 50 per cent through 200 mesh. The pulps were agitated for 72 hours and 120 hours, after which the cyanide tailings were filtered, washed, and assayed for gold. A screen analysis was made of one of the 72-hour cyanide tailings and two of them were examined on a superpanner to see if any undissolved free gold remained. None was found.

Summary:

Feed sample: gold, 2·65 oz./ton

Test No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton ore	
				KCN	CaO	KCN	CaO
1.......	72	0·025	99·06	1·16	0·22	0·31	2·27
2.......	72	0·025	99·06	1·12	0·22	0·37	2·27
3.......	120	0·025	99·06	1·08	0·20	0·43	3·30
4.......	120	0·025	99·06	1·08	0·22	0·43	3·27

A screen analysis of the cyanide tailing from Test No. 1 was as follows.

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold		Extraction, per cent
			Per cent content	Per cent total	
+ 65........................	1·56	0·06	3·63	0·04	
− 65+100........................	8·15	0·04	12·63	0·12	
−100+150........................	22·06	0·035	29·92	0·29	
−150+200........................	18·28	0·035	24·79	0·24	
−200........................	49·95	0·015	29·03	0·28	
Average tailing..................	100·00	0·026	100·00	0·98	99·02

BARREL AMALGAMATION AND CYANIDATION

Tests Nos. 5 and 6

Samples of the ore were ground in water for 20 minutes to give a product of 1·5 per cent on 65 mesh and about 50 per cent through 200 mesh. The pulps were amalgamated with mercury for 1 hour in a jar mill. The amalgamation tailings were sampled and assayed and portions of each agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for periods of 24 and 48 hours. The products were assayed for gold. The 24-hour cyanide tailing was examined on the superpanner for the presence of free gold and none was found in it.

Summary:

Feed sample: gold, 2·65 oz./ton.

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent		Final titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Amalgamation tailing	Cyanide tailing	Amalgamation	Cyanidation	KCN	CaO	KCN	CaO
5........	24	0·225	0·031	91·51	7·32	0·94	0·15	0·19	2·27
6........	48	0·225	0·028	91·51	7·43	1·04	0·16	0·24	2·42

A screen analysis made on the 24-hour cyanide tailing was as follows:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold		Extraction, per cent
			Per cent content	Per cent total	
+ 65........................	1·40	0·07	3·18	0·04	
− 65+100........................	7·40	0·05	12·02	0·14	
−100+150........................	21·78	0·04	28·30	0·33	
−150+200........................	17·54	0·04	22·79	0·27	
−200........................	51·88	0·02	33·71	0·39	
Average cyanide tailing..........	100·00	0·031	100·00	1·17	98·83

A screen analysis on the 48-hour cyanide tailing showed much the same result.

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold		Extraction, per cent
			Per cent content	Per cent total	
+ 65........................	1·58	0·075	4·23	0·05	
− 65+100........................	8·14	0·045	13·07	0·14	
−100+150........................	22·24	0·035	27·77	0·29	
−150+200........................	17·92	0·03	19·18	0·20	
−200........................	50·12	0·02	35·75	0·38	
Average cyanide tailing..........	100·00	0·028	100·00	1·06	98·94

These tests show that at a comparatively coarse grind 91·5 per cent of the gold is free and that 99 per cent of the total gold is soluble in cyanide solution.

CONCENTRATION BY JIG AND BLANKETS FOLLOWED BY BARREL AMALGAMATION OF CONCENTRATES AND CYANIDATION OF THE TAILINGS

Test No. 7

Fifty pounds of the ore at minus 14 mesh was put through a jig to remove the coarse gold that was free at that grind. The jig overflow was dried carefully and after being well mixed was riffled into 8 fractions of about equal size. These fractions were reground in water in ball mills to various sizes and again put through the jig with the jig overflow passing over a blanket. All jig and blanket concentrates were bulked together and were later reground and barrel-amalgamated and the amalgamation tailing treated by cyanidation. The blanket tailings at various sizes were assayed separately and portions of each agitated in cyanide solution containing the equivalent of 1·0 pound of potassium cyanide per ton for stated periods of time. All products were assayed for gold and total extractions calculated on the basis of the final cyanide residues from the individual blanket tailings and from the amalgamation tailing from the combined concentrates. The distribution of the gold between the concentrates and blanket tailing was calculated from the individual blanket tailing assays using as a ratio of concentration the average figure obtained from the combined weight of all the concentrates and the total weight of sample used.

The results obtained in this test are tabulated in the following table:

Summary of Concentration Tests, Test No. 7:

Test No.	Grind, per cent minus 200 mesh	Blanket slope, in./ft.	Distribution of gold, per cent		Assay, Au, oz./ton		Agitation, hours	Total extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton ore	
			Combined concen-trates	Blanket tailing	Blanket tailing	Blanket tailing cyanided			KCN	CaO	KCN	CaO
7A.......	29·5	1·5	75·82	24·18	0·656	0·075	24	97·07	0·96	0·16	0·13	1·71
					0·656	0·05	48	98·00	1·16	0·14	0·19	2·24
7B.......	43·0	1·5	81·72	18·28	0·496	0·038	24	98·42	0·82	0·12	0·26	1·99
					0·496	0·037	48	98·47	1·04	0·18	0·39	2·25
7C.......	49·0	2·0	85·63	14·37	0·390	0·025	24	98·89	0·86	0·11	0·21	1·98
					0·390	0·025	48	98·89	1·12	0·14	0·26	2·59
7D.......	52·0	2·0	84·52	15·48	0·42	0·02	24	99·07	0·76	0·10	0·38	1·90
					0·42	0·02	48	99·07	1·04	0·15	0·36	2·67
7E.......	57·0	2·0	82·31	17·69	0·48	0·02	48	99·08	0·97	0·12	0·57	2·35
					0·48	0·015	72	99·27	0·90	0·12	0·69	2·24
7F.......	63·0	2·0	86·00	14·00	0·38	0·015	48	99·26	0·92	0·11	0·65	2·46
					0·38	0·015	72	99·26	0·90	0·11	0·67	2·61
7G.......	70·0	2·0	83·78	16·22	0·44	0·015	24	99·27	0·96	0·23	0·60	2·15
					0·44	0·015	48	99·27	0·94	0·10	0·62	2·50
7H.......	31·0	2·0	76·78	23·22	0·63	0·05	48	97·99	0·96	0·14	0·23	2·33
					0·63	0·035	72	98·54	1·00	0·13	0·24	2·13

The combined jig and blanket concentrates were reground 84 per cent through 200 mesh with some lime added, and were amalgamated with mercury in a jar mill for 2 hours. The amalgam and excess mercury were retorted and found to contain 1·549 grammes of gold, corresponding to an assay value of 86·60 ounces per ton of combined concentrates. The amalgamation tailing assayed 1·42 ounces per ton in gold. These two assay values added together give the value of 88·02 ounces per ton for the combined concentrates, which weighed 521·7 grammes or 2·34 per cent of the original feed.

The average value of the blanket tailings calculated from their weights and assays was 0·487 ounce per ton.

Summary of Concentration Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent of gold in ore	Extraction	
				Per cent contained gold	Per cent total gold
Combined concentrates	2·34	88·02	81·24	....	....
Blanket tailing	97·66	0·487	18·76	....	....
Feed (cal.)	100·00	2·535	100·00	....	....
Concentrates amalgamated	....	1·42	1·31	98·39	79·93
Amalgamation tailing cyanided	....	0·19	0·18	1·39	1·13

Combined ratio of concentration .... 42·74 : 1

From the above table it is seen that 98·39 per cent of the gold in the combined concentrates is recoverable by amalgamation and an additional 1·39 per cent of it can be extracted by cyanidation of the amalgamation tailing. Total extraction from the combined concentrates is, therefore, 99·78 per cent of the contained gold.

This figure and the one for ratio of concentration were used to calculate recoveries and extractions at the various grinds in the individual tests, shown in the first table of Test No. 7, by working back from the blanket tailing assay and the feed sample assay which checks very well against the figure calculated from the products of this test.

Example: Test No. 7E—

Product	Weight, per cent	Assay, Au, oz./ton	Units	Distribution of gold, per cent
Feed	100·00	2·65	265·0000	100·00
Blanket tailing	97·66	0·38	37·1108	14·00
Combined concentrates (by difference)	2·34	....	227·8892	86·00

	Per cent of total gold
Extraction from concentrate, $86·00 \times 0·9978$	85·81
Extraction from blanket tailing (48 or 72 hours), $\frac{0·39 - 0·015}{0·38} \times 14·00$	13·45
Total extraction	99 26

Screen analyses were made on some of the products from these tests as follows:

Blanket Tailing, Test No. 7A:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold		Recovered in concentrate, per cent
			Per cent content	Per cent total	
+ 35	3·40	2·0	10·37	2·51	
− 35+ 48	6·98	0·86	9·15	2·21	
− 48+ 65	14·39	0·77	16·89	4·08	
− 65+100	19·83	0·68	20·56	4·97	
−100+150	14·57	0·55	12·22	2·95	
−150+200	11·28	0·39	6·71	1·62	
−200	29·55	0·535	24·10	5·84	
Average tailing	100·00	0·656	100·00	24·18	75·82

A sample of this blanket tailing was panned by hand and found to contain fine free gold, the largest particle being near 150 mesh.

48-Hour Cyanide Tailing from Blanket Tailing, Test No. 7A:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold		Total extraction, per cent
			Per cent content	Per cent total	
+ 35	1·98	0·41	16·36	0·30	
− 35+ 48	5·31	0·105	11·24	0·21	
− 48+ 65	13·79	0·085	23·63	0·43	
− 65+100	19·70	0·055	21·84	0·40	
−100+150	14·87	0·04	11·99	0·22	
−150+200	15·22	0·02	6·14	0·11	
−200	29·13	0·015	8·80	0·16	
Average cyanide tailing	100·00	0·0496	100·00	1·83	98·00

Blanket Tailing, Test No. 7B:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold		Recovered in concentrate, per cent
			Per cent content	Per cent total	
+ 48	1·62	1·55	5·06	0·92	
− 48+ 65	4·90	0·36	3·56	0·65	
− 65+100	14·98	0·50	15·11	2·76	
−100+150	20·02	0·43	17·36	3·17	
−150+200	15·46	0·47	14·66	2·68	
−200	43·02	0·51	44·25	8·10	
Average tailing	100·00	0·496	100·00	18·28	81·72

Hand-panning did not reveal any free gold in this sample.

24-Hour Cyanide Tailing from Blanket Tailing, Test No. 7B:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold		Total extraction, per cent
			Per cent content	Per cent total	
+ 48	1·56	0·33	13·49	0·19	
− 48+ 65	5·49	0·09	12·95	0·18	
− 65+100	14·91	0·065	25·40	0·36	
−100+150	20·42	0·04	21·41	0·30	
−150+200	15·60	0·025	10·22	0·14	
−200	42·02	0·015	16·53	0·23	
Average cyanide tailing	100·00	0·038	100·00	1·40	98·42

CONCENTRATION BY FLOTATION AND BLANKETS

Test No. 8

A sample of the ore was ground 7·6 per cent on 65 mesh and 37·7 per cent through 200 mesh and floated. The flotation tailing was sampled for assay and the remainder treated on a blanket. All the products were assayed for gold.

Charge to Ball Mill:

Ore	2,000 grms. minus 14 mesh
Water	1,500 c.c.
Soda ash	1·0 lb./ton
Aerofloat No. 31	0·07 "

Reagents to Cell:

No. 208	0·10 lb./ton
Pine oil	0·10 "

Screen Analysis of Flotation Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold	
			Per cent content	Per cent total
+ 65	7·64	1·08	25·02	3·30
− 65+100	17·96	0·51	27·77	3·66
−100+150	22·22	0·325	21·90	2·89
−150+200	14·48	0·29	12·73	1·68
−200	37·70	0·11	12·58	1·66
Average tailing	100·00	0·33	100·00	13·19

Screen Analysis of Blanket Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold	
			Per cent content	Per cent total
+ 65	7·15	0·285	18·83	0·82
− 65+100	18·78	0·160	27·77	1·21
−100+150	21·22	0·115	22·55	0·98
−150+200	17·35	0·09	14·43	0·63
−200	35·50	0·05	16·42	0·70
Average tailing	100·00	0·108	100·00	4·34

Summary:

Flotation Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Flotation concentrate	1·62	131·9	86·81
Flotation tailing	98·38	0·33	13·19
Feed (cal.)	100·00	2·46	100·00

Flotation and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Flotation concentrate	1·62	131·9	86·81
Blanket concentrate (cal.)	1·15	19·10	8·92
Blanket tailing	97·23	0·108	4·27
Feed (cal.)	100·00	2·46	100·00

Combined ratio of concentration.. 36·1 : 1

The feed sample assay was calculated from the flotation products alone and by difference from this the blanket concentrate assay was calculated.

BLANKET CONCENTRATION AND FLOTATION OF BLANKET TAILING

Test No. 9

This was a repetition of Test No. 8 except that the order was reversed and flotation became the last step in the operation. Copper sulphate was also added to the reagent combination and greatly reduced the ratio of concentration without any appreciable decrease in recovery. The combined concentrates were reground 97 per cent through 200 mesh and agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 48 hours. The cyanide tailing was assayed for gold. A screen analysis was made of the flotation tailing and the fractions were assayed for gold. Percentage of total gold extracted by cyanidation was calculated from the average of the two tailing assays.

Screen Analysis, Flotation Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent content
+ 65	8·33	0·27	24·05
− 65+100	20·50	0·13	28·49
−100+150	25·19	0·09	24·24
−150+200	13·31	0·065	9·25
−200	32·67	0·04	13·97
Average tailing	100·00	0·094	100·00

The cyanide tailing from the flotation concentrate assayed 0·20 ounce per ton in gold and the average overall tailing was calculated as follows:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent total Au	Extraction, per cent total Au	Reagents consumed, lb./ton ore	
					KCN	CaO
Flotation tailing.........	87·53	0·094	3·12	....	....	....
Cyanide tailing from concentrate............	12·47	0·20	0·94	....	....	....
Average tailing..........	100·00	0·107	4·06	95·96	0·43	0·87

Combined ratio of concentration......................................8·02 : 1

CONCLUSIONS

All the coarse gold is soluble in cyanide solution and the ore can be treated by cyanidation. If this be done, however, a jig or blankets should be put between the mill and classifier to prevent accumulation of gold in the grinding circuit. The jig or blanket concentrate could be barrel-amalgamated and the amalgamation tailing sent on to the cyanidation circuit or it could simply be reground to reduce the size of the gold particles and sent to the cyanide agitators, tests having shown that this gold is all soluble in cyanide solution. If the ore be ground 60 to 70 per cent through 200 mesh, about 99 per cent of the gold should be extracted in this way.

By treating the ore with jigs followed by blankets at this same grind about 85 per cent of the gold can be recovered in the form of concentrates, from which more than 98 per cent of the contained gold can be recovered as bullion by amalgamation (*See* Test No. 7).

The blanket tailing and amalgamation tailing could be impounded for treatment by cyanidation.

Owing to the abundance of coarse gold in the ore and the scarcity of sulphide minerals to act as carriers for the free gold it is not possible to produce a flotation tailing low enough in gold to be discarded.

A flotation concentrate would probably have to be treated by cyanidation, because the reagents might sicken mercury if amalgamation were to be tried. The presence of soda ash in the flotation tailing would also complicate its subsequent treatment by cyanidation.

The screen analyses of the products of Tests Nos. 8 and 9 show the nature of the flotation and blanket tailings obtained and it is obvious that further treatment is required.

A temporary treatment plant using jig and blanket concentration with amalgamation of the concentrates can be operated satisfactorily to recover about 85 per cent of gold and the cyanide plant could be added later without discarding any of the temporary equipment.

If a flotation plant be used, additional reagents and equipment would be required before the tailing could be satisfactorily treated in a cyanide plant.

Ore Dressing and Metallurgical Investigation No. 738

GOLD ORE FROM THE COCHENOUR WILLANS GOLD MINES, LIMITED, McKENZIE ISLAND, ONTARIO

Shipment. A shipment of ore, consisting of 15 bags and weighing 2,425 pounds, was received on December 17, 1936, from the Cochenour Willans Gold Mines, Limited, McKenzie Island, Ontario.

A second shipment, comprising 6 bags and weighing 600 pounds, was received on June 29, 1937.

Location of Property. The mine is situated in the Red Lake area, Patricia portion of Kenora District, Ontario.

Characteristics and Analysis of Ore:

Shipment No. 1. The *gangue* is largely a fine-textured, greenish grey siliceous rock. This contains vein quartz, and a considerable quantity of finely disseminated carbonate, which may be dolomitic rather than calcite.

The *metallic minerals* present in the sections examined microscopically are, in their order of abundance: pyrite, arsenopyrite, tetrahedrite, grey mineral (possibly galena), chalcopyrite, pyrrhotite, sphalerite, "limonite", and native gold. Of these pyrite is the major mineral and arsenopyrite the minor. The remainder of the sulphide minerals are rare.

All the metallic minerals are disseminated in the gangue. Pyrite is moderately coarse to very fine, and arsenopyrite is medium to very fine, the average grain size of the latter being considerably finer than the pyrite. The remaining minerals are very finely divided.

Only seven grains of native gold were seen. These were of normal colour and occurred as small irregular grains, both in pyrite and in gangue. Those in the gangue were around 560 mesh, those along cracks in the pyrite were between 800 and 280 mesh, and those in dense pyrite were all below 1600 mesh. Hydraulic classification and blanket concentration showed a few comparatively coarse grains of gold.

The ore was crushed and sampled by standard methods and assayed as follows:

Gold	0·83 oz./ton
Silver	0·30 "
Arsenic	0·44 per cent
Iron	4·30 "

Shipment No. 2. The *gangue* is largely composed of grey carbonate and some silicates, which were not determined. Very little quartz is present.

The *ore minerals* in the polished sections are: pyrite, stibnite, arsenopyrite, chalcopyrite, and sphalerite. Pyrite and stibnite are moderately abundant. The former occurs as medium to small crystals disseminated in the gangue, the latter as medium to fine irregular grains and stringers, which in places where the mineral is abundant form a lacy network. The small quantity of arsenopyrite occurs as fine disseminated crystals. A small quantity of chalcopyrite and much less sphalerite occur as medium to small irregular grains in the gangue; the chalcopyrite is commonly associated with pyrite, and the sphalerite usually is associated with small amounts of stibnite.

The analysis is as follows:

Gold	0·54 oz./ton
Silver	0·06 "
Arsenic	0·56 per cent
Antimony	0·55 "
Copper	Trace
Sulphur	1·48 per cent

Results of Experimental Research. The ore of the first shipment is refractory, and extractions by cyanide at a fineness of grinding of 90·8 per cent minus 325 mesh only accounted for 57·83 per cent of the gold.

About 37 per cent is free-milling, as indicated by barrel amalgamation at a grinding of 62·6 per cent minus 200 mesh.

The making of a flotation concentrate gave considerable difficulty, which was attributed to the carbonate or talcy constituents in the gangue. The use of caustic soda instead of soda ash as a modifying agent in the grinding circuit overcame this difficulty to a considerable extent, and with fine grinding an overall gold recovery of slightly over 95 per cent was obtained as concentrates by blankets and flotation.

By chloridizing roasting and cyanidation of the calcine about 89 per cent of the gold in the calcine was extracted.

The second shipment of ore, in its response to treatment, resembled closely that of the first shipment.

Experimental Tests

The data of the experimental work follow in detail:

AMALGAMATION

Test No. 1

A sample of minus 14-mesh ore, 1,000 grammes in weight, was barrel-amalgamated with mercury for 1 hour.

Gold in feed	0·83 oz./ton
Gold in tailing	0·68 "
Recovery	18·07 per cent

A screen test indicated 27·3 per cent minus 200 mesh.

Test No. 2

A sample of ore was ground in a porcelain grinding mill for 20 minutes and then barrel-amalgamated with mercury for 1 hour.

Gold in feed	0·83 oz./ton
Gold in tailing	0·52 "
Recovery	37·35 per cent

Screen Test on Tailing:

Mesh	Weight, per cent
+ 48	0·9
− 48+ 65	2·3
− 65+100	8·1
−100+150	12·5
−150+200	13·6
−200	62·6
	100·0

HYDRAULIC CLASSIFICATION

Test No. 3

A sample of ore was ground for 15 minutes and run over a hydraulic classifier. The oversize was panned and a number of coarse grains of free gold were observed in the panned concentrate.

CYANIDATION TESTS

Tests Nos. 4 to 9

A number of cyanidation tests were carried out on ore ground to different degrees of fineness. The strength of cyanide solution was equivalent to 1 pound of potassium cyanide per ton and the pulp dilution was 2 : 1.

The ore was ground in a water pulp, filtered, and again repulped and cyanide added to the required strength in each bottle.

Screen Tests:

Tests Nos. 4 and 5:

Mesh	Weight, per cent
+ 65	0·9
− 65+100	4·5
−100+150	10·4
−150+200	14·2
−200	70·0
	100·0

Tests Nos. 6 and 7:

Mesh	Weight, per cent
+100	0·3
−100+150	2·8
−150+200	8·3
−200	88·6
	100·0

Tests Nos. 8 and 9:

Mesh	Weight, per cent
+150	0·6
−150+200	4·1
−200	95·3
	100·0

Results of Cyanidation:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
4	24	0·83	0·39	53·01	0·02	5·20
5	48	0·83	0·38	54·22	0·02	5·40
6	24	0·83	0·365	56·02	0·02	5·30
7	48	0·83	0·37	55·42	0·02	5·50
8	24	0·83	0·375	54·82	0·17	5·48
9	48	0·83	0·355	57·23	0·17	5·65

Test No. 10

The ore was ground to have 90·81 per cent pass a 325-mesh screen and was cyanided for 24 hours in a solution equivalent to 1 pound of potassium cyanide per ton.

Results:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
Feed	Tailing		KCN	CaO	
0·83	0·35	57·83	0·58	5·50	2 : 1

The prolonged grinding accounted for only a slight increase in the extraction of gold.

Test No. 11

This was a similar test to No. 10, on ore ground to the same fineness, but no lime was added to the pulp. The results showed an increased consumption of cyanide and a lower extraction of gold.

Results:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
Feed	Tailing		KCN	CaO	
0·83	0·365	56·02	0·84	Nil	2 : 1

Test No. 12

In this, 0·5 pound of lead oxide (PbO) was added to the pulp. The results showed no apparent difference from any previous tests.

Test No. 13

A sample of ore was ground to have 88 per cent pass a 200-mesh screen and was cyanided for 24 hours. The tailing was filtered, washed, and roasted, and the calcine cyanided in an endeavour to see if roasting would render the remaining gold amenable to the action of cyanide. The results showed an overall recovery of 80·7 per cent.

Cyanidation of Raw Ore:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
Feed	Tailing		KCN	CaO	
0·83	0·365	56·02	0·32	5·90	2 : 1

Gold in roasted tailing.. 0·385 oz./ton
Gold in cyanide tailing of calcine.............................. 0·16 "
Overall recovery, $\frac{0 \cdot 83 - 0 \cdot 16}{0 \cdot 83} \times 100$.......................... 80·7 per cent

FLOTATION TESTS

A number of flotation tests were carried out using a variety of reagents in order to obtain the best conditions for making a bulk concentrate. In every case the free gold was removed prior to flotation.

Test No. 14

A sample of ore was ground in a water pulp and was fed to a hydraulic classifier to remove free gold. The classifier overflow was dewatered and floated in a laboratory Denver Sub-A flotation cell. The pulp was conditioned in the cell with soda ash 2 pounds per ton, butyl xanthate 0·2 pound per ton, and pine oil 0·075 pound per ton.

Hydraulic Classification:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·83	100·00	
Oversize	0·10	127·33	15·34	1000 : 1
Overflow	99·90	0·70	84·66	

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·70	100·00	
Concentrate	5·38	7·04	54·07	18·59 : 1
Tailing	94·62	0·34	45·93	

Combined concentrates, assay	9·23 Au, oz./ton
Gold recovery as concentrates	61·12 per cent

Screen Analysis of Flotation Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent
+150	9·59	0·465	12·29
−150+200	10·68	0·33	9·71
−200	79·73	0·355	78·00
	100·00		100·00

Test No. 15

This was similar to Test No. 14 but finer grinding was carried out on the ore.

The results were not satisfactory.

Hydraulic Classification:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·83	100·00	
Oversize	0·13	71·17	11·15	769 : 1
Overflow	99·87	0·74	88·85	

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed (cal.)	100·00	0·71	100·00	
Concentrate	5·37	6·72	51·09	18·62 : 1
Tailing	94·63	0·365	48·91	

Combined concentrates, assay .... 8·245 Au, oz./ton
Gold recovery as concentrates .... 56·54 per cent

Screen Analysis of Flotation Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent
+150	1·58	0·26	3·90
−150+200	3·88		
−200	94·55	0·37	96·10
	100·00		100·00

Test No. 16

The overflow from the hydraulic classifier was reground with 2 pounds of soda ash per ton and 0·07 pound of Aerofloat No. 31 per ton. The pulp was floated with 0·10 pound of potassium amyl xanthate and 0·05 pound of pine oil per ton.

Hydraulic Classification:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·83	100·00	
Oversize	0·27	81·04	26·36	370 : 1
Overflow	99·73	0·61	73·64	

Flotation:

Feed (cal.)	100·00	0·73	100·00	
Concentrate	9·95	4·98	67·91	10·05 : 1
Tailing	90·05	0·26	32·09	

Combined concentrates, assay .... 6·99 Au, oz./ton
Gold recovery as concentrates .... 76·27 per cent

The flotation tailing was lowered slightly.

Test No. 17

The classifier overflow was reground finer than in the previous test. The results were not satisfactory.

Test No. 18

In this and subsequent tests blankets were used in place of the hydraulic classifier to remove the coarse free gold. Copper sulphate was used in the flotation.

Reagents Added to Regrinding (87·3 per cent −200 mesh):

	Lb./ton
Soda ash	1·0
Aerofloat No. 31	0·105

To Flotation Cell:

Potassium amyl xanthate	0·10
Copper sulphate	1·0
Pine oil	0·025

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·83	100·00	
Concentrate	0·45	41·40	22·45	222·2 : 1
Tailing	99·55	0·65	77·55	

Flotation:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	As	Fe	Au	As	Fe	
Feed (cal.)	100·00	0·60	0·45	4·28	100·00	100·00	100·00	
Concentrate	5·36	6·42	3·56	11·37	57·84	41·87	14·23	18·6 : 1
Tailing	94·64	0·265	0·28	3·88	42·16	58·13	85·77	

Combined concentrates, assay	9·13 Au, oz./ton
Gold recovery as concentrates	67·30 per cent

Test No. 19

The grinding was the same as in Test No. 18. The reagents added were:

	Lb./ton
Sodium silicate	1·0
Barrett No. 4	0·176
Lime	1·0
Copper sulphate	1·0
Potassium amyl xanthate	0·2
Pine oil	0·025

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·83	100·00	
Concentrate	0·54	59·35	38·61	185·2 : 1
Tailing	99·46	0·51	61·39	

Flotation of Blanket Tailing:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	As	Fe	Au	As	Fe	
Feed (cal.).........	100·00	0·56	0·46	4·31	100·00	100·00	100·00	
Concentrate........	7·12	5·70	3·76	9·95	71·99	57·85	16·43	14·04 : 1
Tailing............	92·88	0·17	0·21	3·88	28·01	42·15	83·57	

Combined concentrates, assay.............................. 9·48 Au, oz./ton
Gold recovery as concentrates.............................. 82·80 per cent

A sample of the flotation tailing was panned on a Haultain superpanner and the sulphides concentrated. Under the binocular microscope they were found to be practically all free from gangue. Some of the sulphide grains were extremely fine.

Test No. 20

An attempt was made to float off the talcy material in a primary product from the blanket tailing and to regrind the talc tailing and make a sulphide concentrate.

The results were not satisfactory as the talc concentrate carried 1·50 ounces of gold per ton and the sulphide concentrate 6·05 ounces of gold per ton.

Test No. 21

This was carried out similarly to Test No. 19. The amounts of sodium silicate and Barrett No. 4 were doubled and the amount of soda ash added was 0·5 pound per ton. No improvement was made in lowering the flotation tailing.

Test No. 22

The blanket tailing was reground for a longer period than in the previous tests. The fineness of grind was 90·7 per cent minus 200 mesh. Potassium amyl xanthate and Barrett No. 4 were added to the grinding circuit.

The results showed an improvement in the grade of concentrate and a tailing carrying 0·14 ounce of gold per ton, which was the lowest so far made. It would appear that the addition of xanthate to the grinding combined with finer grinding made a definite improvement in the flotation.

Test No. 23

Xanthate was used without any addition of Barrett No. 4. The results were not good, which would indicate that the use of a heavy oil collector with the xanthate is necessary.

Test No. 24

In this and the two following tests, caustic soda was used in the grinding circuit instead of soda ash. A marked improvement occurred both in grade of concentrate and lowered tailing.

Investigations carried out using caustic soda have shown that in the flotation of certain ores containing carbonates, sodium hydroxide has advantages over sodium carbonate. The reason may be the action of the caustic soda in promoting a condition of the carbonate material in the pulp, which would not be brought about by the addition of another carbonate, such as soda ash.

The results of these tests follow in detail.

A sample of ore (2,000 grammes) was ground for 10 minutes and the pulp passed over a blanket strake. The tailing was reground for 20 minutes using the following reagents:

	Lb./ton
Potassium amyl xanthate	0·2
Caustic soda	0·5
Barrett No. 4	0·088

The pulp was floated with the following reagents:

	Lb./ton
Copper sulphate	1·0
Potassium amyl xanthate	0·1
Pine oil	0·05

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·83	100·00	
Concentrate	0·17	120·90	24·76	588·2 : 1
Tailing	99·83	0·63	75·24	

Flotation:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	As	S	Au	As	S	
Feed (cal.)	100·00	0·58	0·38	0·63	100·00	100·00	100·00	
Concentrate	9·13	5·18	3·17	5·66	81·90	76·11	81·39	10·95 : 1
Tailing	90·87	0·115	0·10	0·13	18·10	23·89	18·61	

Combined concentrates, assay	7·29 Au, oz./ton
Gold recovery in concentrates	86·38 per cent

Test No. 25

The caustic soda was increased to 1 pound per ton and the Barrett No. 4 to 0·176 pound per ton. The pine oil was 0·025 pound per ton.

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·83	100·00	
Concentrate	0·39	80·50	37·83	256·4 : 1
Tailing	99·61	0·52	62·17	

Flotation:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	As	S	Au	As	S	
Feed (cal.)	100·00	0·50	0·35	0·55	100·00	100·00	100·00	
Concentrate	6·39	6·34	3·92	6·43	80·48	70·87	74·53	15·6 : 1
Tailing	93·61	0·105	0·11	0·15	19·52	29·13	25·47	

Combined concentrates, assay .... 10·61 Au, oz./ton
Gold recovery as concentrates .... 87·86 per cent

Test No. 26

The caustic soda was increased to 2 pounds per ton and the pine oil to 0·05 pound per ton, the other reagents being the same as in Test No. 25.

Blanket Concentration:

Product	Weight, per cent	Assay, Au oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·83	100·00	
Concentrate	0·10	201·20	24·24	1000 : 1
Tailing	99·90	0·63	75·76	

Flotation:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	As	S	Au	As	S	
Feed (cal.).........	100·00	0·59	0·37	0·75	100·00	100·00	100·00	
Concentrate........	7·35	7·24	4·60	8·00	89·83	92·40	77·90	13·6 : 1
Tailing.............	92·65	0·065	0·03	0·18	10·17	7·60	22·10	

Combined concentrates, assay.............................. 9·84 Au, oz./ton
Gold recovery as concentrates.............................. 91·4 per cent

The results of these tests show that grinding the ore in caustic soda definitely improves the flotation results.

Test No. 27

This was for determining whether the gold remaining in the flotation tailing is tied up with the sulphides or the quartz gangue.

A portion of the tailing from Test No. 26 was cyanided for 72 hours in a solution of strength equivalent to 1 pound of potassium cyanide per ton and after filtering and washing the cyanide tailing was treated with hot aqua regia (400 c.c. HCl and 45 c.c. HNO_3). The final tailing was assayed for gold.

Results:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
Feed	Tailing		KCN	CaO	
0·065	0·035	46·15	0·62	5·30	2 : 1

The extraction from 72 hours' agitation accounts for any free gold and such other gold in the flotation tailing that is amenable to the action of cyanide.

The remaining gold is either enclosed in sulphides or in the gangue.

Results of treatment with aqua regia:

Gold in cyanide tailing.. 0·035 oz./ton
Gold in aqua regia tailing...................................... 0·0025 "

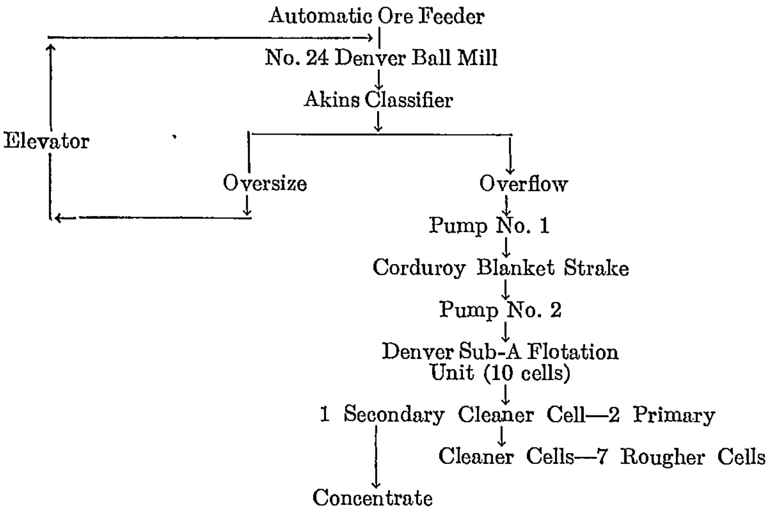
The results of this test show that the refractory gold is practically all associated with the sulphide minerals of the ore.

Many of the pyrite and arsenopyrite grains are extremely fine. It may be assumed therefore that the gold liberated by the action of aqua regia is that contained in dense sulphide grains of extremely fine size.

MILL RUNS

Two continuous mill runs were carried out in a pilot-scale testing unit, embodying the findings from the laboratory-scale tests. The results were satisfactory.

The flow-sheet used was as follows:



The cell feed was fed to the first of the primary cleaner cells, the overflow from which fed to the secondary cleaner cell. The overflow from the seven rougher cells was pumped to the first of the primary cleaner cells. The tailings were run over a Butchart table for checking the efficiency of the flotation operation by a visible examination.

Weight of balls in mill..................................	500 pounds
Average rate of feed....................................	134 pounds per hour

Reagents were fed as follows:

	Lb./ton
To the Mill:	
Barrett No. 4..	0·087
Potassium amyl xanthate..	0·148
Caustic soda...	1·53
To No. 2 Pump:	
Copper sulphate..	0·75
Potassium amyl xanthate..	0·073
To Primary Rougher Cells:	
Pine oil...	0·05

The pH of the cell feed was 9·58.

Samples for assay were taken every 10 minutes.

Test MR-1

Time of run: 7 hours.

Results:

Product	Assay					Weight, pounds	Pulp density, per cent
	Oz./ton		Per cent				
	Au	Ag	As	Fe	S		
Mill feed	0·75	0·12	0·34	4·90	0·75	....	....
Mill discharge	0·73	....	....	....	....	....	....
Classifier overflow	0·56	....	....	....	....	....	23
Blanket feed	0·51	....	....	....	....	....	....
Blanket concentrate	12·60	....	....	....	....	1·69	....
Blanket tailing	0·49	....	....	....	....	....	....
Flotation concentrate	5·26	0·60	4·12	12·77	7·72	53·00	....
Flotation tailing	0·035	....	0·10	3·66	0·07	....	....

Screen Tests:

Mesh	Classifier overflow, weight, per cent	Flotation tailing, weight, per cent
+100	....	0·3
−100+150	2·8	5·8
−150+200	10·9	13·1
−200	86·3	80·8
	100·0	100·0

Test MR-2

Time of run: 8 hours.

Results:

Product	Assay					Weight, pounds	Pulp density, per cent
	Oz./ton		Per cent				
	Au	Ag	As	Fe	S		
Mill feed	0·96	0·16	0·43	4·90	0·66	....	....
Mill discharge	1·08*	....	....	....	....	....	....
Classifier overflow	0·72	....	....	....	....	....	23·1
Blanket feed	0·69	....	....	....	....	....	....
Blanket concentrate	32·00	....	....	....	....	1·75	....
Blanket tailing	0·67	....	....	....	....	....	....
Flotation concentrate	12·40	....	8·31	17·42	10·88	45·0	....
Flotation tailing	0·04	....	0·03	....	0·05	....	....

* Probably due to accumulation of free gold from previous run.

NOTE: The average mill feed for the two days' runs checks very closely the assay on the feed sample of the ore shipment.

Screen Test of Classifier Overflow:

Mesh	Weight, per cent
+150	4·2
−150+200	9·5
−200	86·3
	100·0

Fine free gold was seen in the blanket concentrate.

BARREL AMALGAMATION OF BLANKET CONCENTRATE

Amalgamation tests were carried out on the blanket concentrates from Tests MR-1 and MR-2.

The results are as follows:

Test A:

Gold in blanket concentrate (MR-1)	12·60 oz./ton
Gold in amalgamation tailing	6·83 "
Recovery	45·79 per cent

Test B:

Gold in blanket concentrate (MR-2)	32·00 oz./ton
Gold in amalgamation tailing	12·57 "
Recovery	60·72 per cent

The concentrates were not reground prior to barrel amalgamation. Regrinding might raise the recovery somewhat.

RATIOS OF CONCENTRATION IN MILL RUN

The ratios of concentration obtained by the blankets and by flotation have been calculated from the weights of feed and products and, although only approximate, they should indicate what might be expected from mill operations.

The total feed to the mill was about one ton.

Ratio of concentration by blankets	588·2 : 1
Ratio of concentration by flotation	20·4 : 1

CYANIDATION OF FLOTATION CONCENTRATE

Test No. 28

Assay of bulk flotation concentrate, MR-1:

Gold	4·54 oz./ton
Silver	0·48 "
Arsenic	3·65 per cent
Sulphur	6·00 "

This test was carried out on a sample of raw concentrate (MR-1).

The concentrate was ground in a water pulp for 30 minutes, giving a fineness of grinding of 98·5 per cent minus 200 mesh. The ground concentrate was then dewatered and repulped in a cyanide solution of strength equivalent to 3 pounds of potassium cyanide per ton. Agitation was carried out for 48 hours and 5·0 pounds of lime per ton was added at the start for protective alkalinity. The pulp dilution was 2·5 : 1.

Results:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
Feed	Tailing		KCN	CaO
4·54	2·44	46·25	5·475	7·375

ROASTING OF CONCENTRATES AND CYANIDATION OF CALCIUM

Roast No. 1

A charge of 1,100 grammes of flotation concentrate (Test MR-1), gold, 4·54 ounces per ton, was roasted in a shallow tray in an electric furnace. For the first two hours the temperature was under 500° C., after which the temperature was raised, reaching a maximum of 650° C. in 4½ hours. The calcine assayed as follows:

Gold	4·46 oz./ton
Arsenic	3·04 per cent
Sulphur trioxide	6·98 "
Lime	17·73 "
Magnesia	7·09 "

Four samples of this calcine were cyanided under varying conditions. In each test the pulp dilution was 3 : 1 and the strength of solution equivalent to 3 pounds of potassium cyanide per ton. The variables were as follows:

Test 1. 15-minute grind (99 per cent −200 mesh). 24-hour agitation. No lime added.
Test 2. 15-minute grind. 28-hour agitation. One pound lime per ton.
Test 3. 30-minute grind (99·9 per cent −200 mesh). 24-hour agitation. One pound lime per ton.
Test 4. 15-minute grind. 24-hour agitation. Ground with 5 per cent by weight of ferric sulphate. Ten pounds lime per ton.

Grinding was carried out in a water pulp.

Cyanide Tests on Calcine, Roast No. 1:

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton calcine		Pulp dilution
	Feed	Tailing		KCN	CaO	
1	4·46	2·09	53·1	11·4	0·00	3 : 1
2	4·46	1·86	58·3	13·8	0·55	3 : 1
3	4·46	1·88	55·6	11·1	0·55	3 : 1
4	4·46	1·55	65·2	11·7	10·0	3 : 1

Direct roasting of the concentrate under the above conditions did not promote satisfactory extraction of the gold in the calcine by cyanidation. The results of Test No. 4, in which ferric sulphate was ground with the calcine, would indicate that the gold had been coated during the roasting, probably by the arsenic, which may under certain conditions of roasting chemically combine with the gold, forming a compound which is only slowly attacked by cyanide.

The calcine is distinctly alkaline.

Roast No. 2

A sample of the flotation concentrate from Test MR-2 (gold, 11·42 ounces per ton) was roasted under similar conditions to Roast No. 1.

Two samples of the calcine were cyanided for 24 hours, one being ground for 15 minutes and the other for 30 minutes. In the former no lime was added, while in the latter one pound per ton was added.

The analysis of the calcine was as follows:

Gold	11·32 oz./ton
Arsenic	5·12 per cent
Sulphur trioxide	11·56 "
Lime	13·15 "
Magnesia	5·78 "

Cyanide Tests on Calcine, Roast No. 2:

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton calcine		Pulp dilution
	Feed	Tailing		KCN	CaO	
1	11·32	1·31	88·4	11·7	0·00	3 : 1
2	11·32	1·16	89·7	12·0	0·55	3 : 1

The tailings are lower and the extractions better on the higher grade concentrate. The gold to arsenic ratio is considerably higher in this calcine than in that of Roast No. 1.

Roast No. 3

In this roast, carried out on concentrate MR-1, the tray containing the concentrate was placed in a muffle inside the electric furnace. The temperature of roasting was similar to the two preceding tests. The loss of weight during roasting was 10·6 per cent.

The analysis of the calcine was as follows:

Gold	4·64 oz./ton
Arsenic	2·16 per cent
Sulphur trioxide	3·56 "
Sulphide S	0·42 "

The calcine was ground for 30 minutes in water in a grinding jar, using quartz pebbles. Two portions were cyanided at a pulp dilution of 3 : 1 and a strength of solution equivalent to 3 pounds of potassium cyanide per ton for 24 and 48 hours respectively. No lime was added to the pulp.

Cyanide Tests on Calcine, Roast No. 3:

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton calcine	
	Feed	Tailing		KCN	CaO
1	4·64	2·58	11·39	2·07	0
2	4·64	4·09	11·85	2·67	0

The extraction was low. The results of the 48-hour test indicate re-precipitation of gold.

Roast No. 4

This was a chloridizing roast, adding 3 per cent of sodium chloride by weight to the charge of flotation concentrate MR-1. Roasting was carried out as in the previous test, but at a temperature just under 500° C. For the first hour or so of the roast the temperature was under 400° C. in order to burn off or oxidize the arsenic at a low temperature.

The calcine was washed to remove soluble chloride. Analysis of the calcine was as follows:

Gold	4·30 oz./ton
Arsenic	2·14 per cent
Sulphur trioxide	4·05 "
Sulphide S	0·09 "
Loss of weight	7·8 "

Two lots of the calcine were ground in water and then cyanided for 24 and 48 hours respectively in solutions of strength equivalent to 3 pounds of potassium cyanide per ton. No lime was added.

Cyanide Tests on Calcine, Roast No. 4:

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton calcine		Pulp dilution
	Feed	Tailing		KCN	CaO	
1	4·30	0·70	83·72	4·47	0	3 : 1
2	4·30	0·50	88·37	4·53	0	3·37 : 1

A chloridizing roast makes a marked improvement on the extraction of gold.

Roast No. 5

In this test, carried out on concentrate MR-1, the amount of salt added was increased to 5 per cent. The temperature during the final hour of the roast was raised until a maximum of 600° C. was reached.

The assay of the calcine was as follows:

Gold	5·05 oz./ton
Arsenic	2·34 per cent
Sulphur trioxide	3·14 "
Sulphide S	0·82 "
Loss of weight	11·0 "

Two cyanide tests were carried out on the calcine for 24 and 48 hours respectively. The solution strengths were equivalent to 3 pounds of potassium cyanide per ton. No lime was added. The final solutions, however, showed an alkalinity equivalent to 0·35 pound of lime per ton of solution.

Cyanide Tests on Calcine, Roast No. 5:

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton calcine		Pulp dilution
	Feed	Tailing		KCN	CaO	
1	5·05	0·525	89·40	2·09	0	3·55 : 1
2	5·05	0·71	85·94	2·59	0	3·76 : 1

The rise in the tailing for the 48-hour run may be due to re-precipitation of gold.

Roast No. 6

A sample of concentrate (MR-2, gold, 11·42 ounces per ton) was roasted at a low temperature to drive off the arsenic and then at a higher temperature around 700° C.

The loss in weight during roasting was 20·8 per cent.

The calcine was ground for 15 minutes and washed and a portion cyanided for 24 hours at a solution strength of 2 pounds of potassium cyanide per ton.

The analysis of the calcine was as follows:

Gold	13·96 oz./ton
Silver	0·94 "
Arsenic	2·21 per cent
Total sulphur	2·82 "

Cyanidation of Calcine, Roast No. 6:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton calcine		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing	13·96	9·17	34·31	5·35	0	3·88 : 1

Final titration of solution ... 2·25 CaO, lb./ton

Roast No. 7

A mixture of equal amounts of flotation concentrates MR-1 and MR-2 was roasted first under 540° C., to drive off the arsenic, and then at a higher temperature, 1,000° C.

The calcine was ground for 15 minutes and filtered and washed. Two cyanidation tests were run for 24 and 48 hours respectively. The strength of solution was 3 pounds of potassium cyanide per ton at a pulp dilution of 3 : 1. No lime was added owing to the alkaline nature of the calcine.

The analysis of the calcine was as follows:

Gold	10·28 oz./ton
Arsenic	2·95 per cent
Total sulphur	5·07 "

Cyanidation of Calcine, Roast No. 7:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton calcine		Pulp dilution
	Feed	Tailing		KCN	CaO	
24	10·28	1·32	87·16	6·43	0	3·78 : 1
48	10·28	1·34	86·96	6·73	0	3·74 : 1

The calcine from the final higher temperature roast shows an improvement in extraction.

DISCUSSION OF RESULTS OF ROASTING CONCENTRATES

Straight oxidizing roasting does not render the gold in the calcine amenable to cyanidation, except when final roasting is carried out at high temperature.

Chloridizing roasting gave encouraging results, and the gold extraction obtained by cyanidation of the calcine is shown to be around 89 per cent.

Loss of gold by volatilization will occur in the chloridizing roast, but this difficulty can be partially or altogether overcome by suitable methods of fume recovery.

EXPERIMENTAL RESEARCH ON ORE OF SECOND SHIPMENT

Test No. 2-1

A sample of ore was ground to a fineness of 83·6 per cent −200 mesh and cyanided for 24 hours. The tailing was washed and conditioned and floated. The results show an overall gold recovery of 81·24 per cent.

Cyanidation Results:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
Feed	Tailing		KCN	CaO	
0·54	0·31	42·59	1·80	1·60	2 : 1

Flotation Results:

Reagents added:

	Lb./ton
Caustic soda	2·0
Copper sulphate	1·0
Potassium amyl xanthate	0·2
Pine oil	0·05

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Au, oz./ton	Per cent					
			As	S	Au	As	S	
Feed	100·00	0·31	0·46	1·46	100·0	100·0	100·0	13·35 : 1
Concentrate	7·49	2·80	3·94	7·70	67·3	63·9	39·6	
Tailing	92·51	0·11	0·18	0·95	32·7	36·1	60·4	

Test No. 2-2

This is a duplicate of the previous test.

Cyanidation Results:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
Feed	Tailing		KCN	CaO	
0·54	0·29	48·14	2·00	1·60	2 : 1

Flotation Results:

Reagents added:

	Lb./ton
Caustic soda	2·0
Copper sulphate	2·0
Potassium amyl xanthate	0·15
Pine oil	0·05

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Au, oz./ton	Per cent					
			As	S	Au	As	S	
Feed	100·00	0·29	0·47	1·44	100·0	100·0	100·0	
Concentrate	7·57	2·90	4·12	6·01	73·3	66·4	31·5	13·21 : 1
Tailing	92·43	0·085	0·17	1·07	26·7	33·6	68·5	

Overall Gold Recovery:

	Per cent
By cyanidation	48·14
In flotation concentrate	38·76
	86·90

The high cyanide consumption is probably due to the stibnite in the ore acting as a cyanicide.

Test No. 2-3

This was a straight flotation of the ore at a very fine grinding (95·3 per cent −200 mesh). The concentrate was cleaned and the grade improved, but the tailing is high.

The ore was ground with Barrett No. 4 oil, 0·176 pound per ton, and 2 pounds of caustic soda per ton.

The pulp was conditioned with:

	Lb./ton
Copper sulphate	1·5
Amyl xanthate	0·3
Pine oil	0·075

The concentrate was cleaned with sodium silicate and cresylic acid.

Results:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Au, oz./ton	Per cent					
			As	S	Au	As	S	
Feed	100·00	0·54	0·42	1·54	100·0	100·0	100·0	
Cleaner concentrate	6·97	3·92	3·79	14·48	50·6	63·2	65·6	14·35 : 1
Middling	8·16	2·07	1·26	3·79	31·3	24·6	20·1	
Tailing	84·87	0·115	0·06	0·26	18·1	12·2	14·3	

AMALGAMATION TESTS

Two blanket concentration tests were carried out and the concentrate amalgamated to determine the recovery of free-milling gold.

A barrel amalgamation of the raw ore was carried out to compare the maximum free-milling gold with that of the gold in the first ore shipment.

Test No. 2-4

A sample of ore was ground to have 88·4 per cent minus 200 mesh and the pulp was run over a corduroy blanket set at a slope of 2½ inches in 12 inches. The concentrate was amalgamated.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·54	100·0	
Concentrate	0·77	34·05	48·5	129·8 : 1
Tailing	99·23	0·28	51·5	

Amalgamation of Blanket Concentrate:

Gold in concentrate	34·05 oz./ton
Gold in tailing	5·36 "
Recovery	84·3 per cent
Overall recovery	40·89 "

Test No. 2-5

This is a similar blanket-amalgamation test with the grinding to a fineness of 94 per cent minus 200 mesh.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·54	100·0	
Concentrate	0·73	34·54	46·7	136·9 : 1
Tailing	99·27	0·29	53·3	

Amalgamation of Blanket Concentrate:

Gold in concentrate	34·54 oz./ton
Gold in tailing	4·25 "
Recovery	87·7 per cent
Overall recovery	40·96 "

Test No. 2-6

A sample of ore was ground to a fineness of 89·4 per cent minus 200 mesh and barrel-amalgamated with mercury for 1 hour. The results are as follows:

Gold in feed	0·54 oz./ton
Gold in tailing	0·27 "
Recovery by amalgamation	50·0 per cent

This indicates the free-milling gold and does not represent the recovery that could be expected in mill operation, which would be somewhat lower.

CONCLUSIONS

The results indicate that this is a refractory ore. Not over 50 per cent of the gold is free-milling and from 40 to 45 per cent is refractory to the action of cyanide. The gold unattacked by cyanide is composed of extremely fine grains enclosed in dense pyrite and arsenopyrite; it is not freed at a fineness of grinding in which 90 per cent will pass a 325-mesh screen.

Concentration of the ore by flotation after recovering coarse or free gold on blankets shows a gold recovery in the form of concentrate of 95 per cent.

The carbonate content of the gangue interferes in making a clean concentrate. The substitution of caustic soda for soda ash as a modifying agent overcomes somewhat this difficulty. The depressing action of the caustic soda on the finely disseminated carbonates was instrumental in higher sulphide recovery and better grade of concentrate.

Cyanidation of the raw concentrate is not satisfactory. After oxidizing roasting, the cyanidation of the calcine shows a low recovery; chloridizing roasting was more encouraging and a recovery of 89 per cent of the gold contained in the calcine was accomplished.

From the work carried out on these two shipments of ore, it is apparent that amalgamation and cyanidation of the raw ore are quite unsuitable. Concentration by blankets and flotation offers a method of treatment whereby 95 per cent of the gold is recovered as concentrates. Further treatment of these concentrates by chloridizing roast and cyanidation of calcines is shown to give an overall gold recovery of around 85 per cent.

Ore Dressing and Metallurgical Investigation No. 739

GOLD ORE FROM THE EAST MALARTIC MINES, LIMITED,
AMOS, QUEBEC

Shipment. Two bags of ore, total weight 531 pounds, were received on March 9, 1938, from Jos. R. Norrie, Manager, East Malartic Mines, Limited, Amos, Quebec.

Location of the Property. The property of the East Malartic Mines, Limited, from which the present ore shipment was received is situated in Fournière Township, northwestern Quebec.

Sampling and Analysis. After cutting, crushing, and grinding by standard methods a sample of the ore was obtained which assayed as follows:

Gold	0·235	oz./ton
Silver	0·06	"
Zinc	0·10	per cent
Arsenic	Nil	
Sulphur	1·50	per cent
Iron	6·00	"
Copper	Trace	
Lead	Nil	

Characteristics of the Ore. Six polished sections were prepared and examined microscopically.

The *gangue* is an aggregate of dark grey rock minerals with quartz and finely disseminated carbonate. Also present in the gangue in considerable quantity is a fairly hard light-grey undetermined mineral, which may be an alteration product of magnetite, possibly leucoxene.

Pyrite is by far the most abundant *metallic mineral* and occurs as medium to fine disseminated grains and in small stringers. It is dense and relatively free from inclusions of gangue and other minerals. Magnetite is not abundant; it occurs as medium to fine irregular grains in the gangue and in contact with pyrite, as well as tiny inclusions in pyrite. Rare tiny grains of chalcopyrite occur in both pyrite and in gangue. Native gold is present as irregular grains in the gangue and in association with pyrite. Its mode of occurrence and grain size are shown in the following table:

Grain Size of Gold (72 Grains):

Mesh	In gangue, per cent	Associated with pyrite, per cent			Totals, per cent
		Against pyrite	In cracks	In dense pyrite	
+ 100	6·6	....	....	....	6·6
− 100+ 150	5·2	....	....	....	5·2
− 150+ 200	7·9	3·7	....	....	11·6
− 200+ 280	16·2	3·1	....	....	19·3
− 280+ 400	9·2	2·0	4·0	....	15·2
− 400+ 560	10·3	3·1	....	1·3	14·7
− 560+ 800	7·5	0·9	0·9	2·0	11·3
− 800+1100	8·6	1·3	1·3	....	11·2
−1100+1600	2·6	....	....	0·4	3·0
−1600+2300	1·1	....	....	0·4	1·5
−2300	0·4	....	....	....	0·4
Totals	75·6	14·1	6·2	4·1	100·0

Experimental Tests

The research procedure on this ore consisted of straight cyanidation and tests to determine the rate of settling of the pulp.

Straight cyanidation of the ore gave an extraction of 95·7 per cent of the gold at a grind of 65 to 70 per cent minus 200 mesh, the cyanide tailing assaying 0·01 ounce of gold per ton.

CYANIDATION

Test No. 1

The ore at minus 14 mesh was ground in a ball mill in cyanide solution of 1 pound per ton strength to different degrees of fineness. Three pounds of lime per ton of ore was added to maintain protective alkalinity. After grinding, the pulp was agitated for a 24-hour period.

A screen test on the cyanide tailings showed the grinding as follows:

Mesh	Weight, per cent				
	Test A	Test B	Test C	Test D	Test E
− 35+ 48	0·3	....	....	....	....
− 48+ 65	3·8	0·3	....	....	....
− 65+100	8·9	3·4	0·6	0·3	....
−100+150	12·5	9·3	6·2	3·9	1·6
−150+200	13·0	13·6	11·2	9·5	5·9
−200	61·5	73·4	82·0	86·3	92·5
Totals	100·0	100·0	100·0	100·0	100·0

Cyanidation:

Feed: gold, 0·235 oz./ton

Test No.	Agitation, hours	Grind, per cent −200 mesh	Tailing assay, Au, oz./ton	Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
					KCN	CaO	KCN	CaO
A..........	24	61·5	0·01	95·7	0·96	0·40	0·30	2·2
B..........	24	73·4	0·01	95·7	0·92	0·38	0·35	2·2
C..........	24	82·0	0·01	95·7	1·00	0·35	0·30	2·3
D..........	24	86·3	0·01	95·7	0·90	0·30	0·40	2·4
E..........	24	92·5	0·0075	96·8	0·90	0·25	0·60	2·5

CYANIDATION

Test No. 2

This was to determine whether the addition of extra amounts of lime to the pulp during the grinding and agitation periods would affect the extraction of the gold. The ore at minus 14 mesh was ground in a ball mill in cyanide solution of 1 pound per ton strength to pass 82·0 per cent minus 200 mesh. The pulps were then agitated for a 24-hour period. In Test No. 2A, 5 pounds of lime per ton of ore was added prior to grinding and the titration of the pulp was kept between 0·5 and 0·6 pound per ton of solution during agitation. In Test No. 2B, 7 pounds of lime was added to the grind and the titration was kept at 0·7 to 0·8 pound per ton. A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65+100..........	0·6
−100+150..........	6·2
−150+200..........	11·2
−200..........	82·0
	100·0

Results:

Feed: gold, 0·235 oz./ton.

Test No.	Agitation, hours	Grind, per cent −200 mesh	Tailing assay, Au, oz./ton	Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
					KCN	CaO	KCN	CaO
2A..........	24	81·8	0·01	95·7	0·92	0·56	0·30	3·9
2B..........	24	81·8	0·01	95·7	0·88	0·82	0·30	5·4

CYCLE TEST—CYANIDATION

Test No. 3

In order to determine whether the solutions fouled in the mill circuit, with consequent lowered extraction of the gold, a series of tests was run in cycle formation.

The ore at minus 14 mesh was ground in a ball mill in cyanide solution of 1 pound per ton strength. Three pounds of lime per ton of ore was added. The pulp was agitated for a 24-hour period. After agitation, the pulp was filtered and the same solution used for grinding and agitating a fresh batch of ore. This procedure was carried out for five successive batches of ore.

The grinding was as follows:

Mesh	Weight, per cent
− 65+100	3·0
−100+150	5·4
−150+200	8·4
−200	83·2
	100·0

Results of Cyanidation:

Feed: gold, 0·235 oz./ton

Cycle No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
				KCN	CaO	KCN	CaO
1	24	0·01	95·7	0·96	0·36	0·30	2·3
2	24	0·01	95·7	1·12	0·26	0·30	1·2
3	24	0·01	95·7	1·04	0·24	0·30	1·1
4	24	0·01	95·7	1·00	0·22	0·30	1·1
5	24	0·01	95·7	1·00	0·22	0·30	1·0

An analysis of the final solution resulted as follows:

Reducing power	36 millilitres $\frac{N}{10}$ $KMnO_4$/litre
KCNS	0·02 grm./litre

SETTLING TESTS

Tests Nos. 4 and 5

The ore at minus 14 mesh was ground in a ball mill in cyanide solution of 1 pound per ton strength. In Tests Nos. 4A and 4B, 3 pounds of lime per ton of ore was added, and in Tests Nos. 5A and 5B, 6 pounds. After grinding, the pulp was made up to the required dilution and transferred to a tall glass cylinder of 2-inch diameter. The rate of settling of the pulp, in decimals of feet, was read for a 1-hour period. The clear solution was titrated for alkalinity.

A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65+100	1·9
−100+150	8·1
−150+200	11·9
−200	78·1
	100·0

Results:

—	Test No. 4A	Test No. 4B	Test No. 5A	Test No. 5B
Ratio of solid to liquid	1·5 : 1	2 : 1	1·5 : 1	2 : 1
Lime added per ton of ore, in pounds	3	3	6	6
Alkalinity of solution at end of test, pounds per ton of solution	0·42	0·38	1·04	0·80
Overflow solution	Clear	Clear	Clear	Clear
Rate of settling, in feet per hour	0·40	0·68	0·39	0·68

The pulp settles more slowly than normally.

MILL RUN

The remainder of the shipment, weighing 406 pounds, was ground in a ball mill to pass 69·4 per cent minus 200 mesh. The ball mill discharge was passed over a corduroy blanket and the blanket tailing went to an Akins classifier. The overflow from the classifier passed over a second set of blankets and the coarse product was returned to the ball mill. The blanket concentrate was amalgamated with mercury in a jar mill and the gold bullion finally separated and refined from the amalgam.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·235	100·0	
Blanket concentrate	1·09	11·13	51·6	92 : 1
Tailing	98·91	0·115	48·4	

Amalgamation of Blanket Concentrate:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
11·13	0·76	93·2

The amalgam was retorted and the residue was refined, the final result being a gold button weighing 0·673 gramme.

Summary of Mill Run:

	Per cent
Gold recovered in blanket concentrate	51·6
Gold recovered by amalgamation of blanket concentrate	48·1
Gold recovered as bullion	45·3

SUMMARY AND CONCLUSIONS

The ore is easy to grind and the gold can be extracted by straight cyanidation without any metallurgical difficulties. An extraction of 95·7 per cent of the gold in the ore was made at a grind of 65 to 70 per cent minus 200 mesh, the resulting cyanide tailing assaying 0·01 ounce of gold per ton.

As shown by the microscopic work, 4·1 per cent of the gold is contained in dense pyrite and would require extremely fine grinding to liberate.

The cycle tests showed no appreciable fouling of mill solutions during the grinding and agitation periods.

The pulp settles rather slowly and due allowance in tank capacity should be made in the mill design. Additional amounts of lime do not seem to hasten the rate of settling much. The extraction of gold remains unaltered at 95·7 per cent, irrespective of the quantity of lime added. Reagent consumption is normal.

Ore Dressing and Metallurgical Investigation No. 740

GRAPHITIC GOLD ORE FROM KERR-ADDISON GOLD MINES, LIMITED, LARDER LAKE, ONTARIO

Shipment. A sample of ore, weight 192 pounds, was received from Kerr-Addison Gold Mines, Limited, Larder Lake, Ontario, on March 29, 1938. The sample was from a graphitic ore-body.

On April 19, 1938, a sample of "carbonate" ore, weight 167 pounds, from the main ore-body, was received for mixing with the graphitic ore for the tests.

The shipments were made by W. S. Row, Superintendent, at the request of M. F. Fairlie, consulting engineer, 38 King Street West, Toronto, Ontario.

An investigation of the ore from the main ore-body was carried out in 1937 and the results are reported in Investigation No. 711.

Characteristics of the Ore. Six polished sections were prepared and examined microscopically for determining the character of the ore.

The gangue is a complex aggregate of dark grey minerals containing considerable carbonate and a small quantity of quartz in stringers and patches. Graphite is visible in the hand specimen as films and shear surfaces; it appears quite platy or schistose in some specimens.

Pyrite is the only metallic mineral in abundance. It occurs as coarse to very fine grains disseminated throughout the gangue, and contains small inclusions of gangue, chalcopyrite, and sphalerite. A minor quantity of chalcopyrite is visible as medium to small irregular grains in gangue and in pyrite, and a small amount of sphalerite occurs in the same way.

As no gold was seen in the sections examined no information was obtained as to its mode of occurrence.

Sampling and Assaying. The ore was crushed and sampled by standard methods and the analyses are as follows:—

Gold	0·075	oz./ton
Silver	0·24	"
Arsenic	Nil	
Iron	7·32	per cent
Sulphur	4·21	"
Carbon (graphite)	0·84	"

The analyses of the ore from the main body are:

Gold	0·20	oz./ton
Silver	0·04	"

Purpose of Investigation and Results. The object of this investigation was to ascertain if the graphite in the ore would have a harmful influence on the extraction of gold during cyanidation, and to determine to what extent, if any, extraction would be influenced by mixing the graphitic ore with that from the main ore-body.

The results indicate that in cyanidation of the graphitic ore alone there is a tendency for the graphite to increase the loss of gold in the tailing. The lower tailing obtained by grinding in water compared with that obtained by grinding in cyanide also indicates that there were components in the ore influencing the extraction and reagent consumption.

Mixtures of the two ores do not suffer any appreciable harm from the graphitic ore. By mixing the two ores the factors causing a lowered extraction of gold in the graphitic ore are largely overcome.

The gold in the graphitic ore is very fine and associated largely with the sulphides.

Experimental Tests

Graphitic Ore

The water pulp of this ore is slightly alkaline, having a pH of 8·6 (ore ground in distilled water).

A number of cyanidation tests were carried out in which the ore was ground in cyanide and lime to different degrees of fineness and agitated for different periods of time.

The cyanide strength was equivalent to 1 pound of potassium cyanide per ton and the lime added was 3 pounds per ton of ore. Grinding was carried out at a pulp dilution of 0·75 : 1 and agitation at 1·5 : 1.

The results of the 24- and 48-hour agitation periods are as follows:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO
1	24	0·075	0·035	53·3	0·27	2·70
2	48	0·075	0·035	53·3	0·75	3·33
3	24	0·075	0·03	60·0	0·57	2·93
4	48	0·075	0·03	60·0	0·99	3·53

The grinding is indicated by the following screen tests:

Mesh	Weight, per cent	
	Tests Nos. 1 and 2	Tests Nos. 3 and 4
+ 48	0·6	....
− 48+ 65	4·1	....
− 65+100	13·4	3·8
−100+150	17·7	12·2
−150+200	14·3	14·7
−200	49·9	69·3
	100·0	100·0

The following cyanidation tests were carried out for shorter periods of agitation in order to observe if the graphite was precipitating gold from solution.

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
5........	4	0·075	0·03	60·0	0·45	2·86
6........	8	0·075	0·025	66·7	0·57	3·04
7........	16	0·075	0·025	66·7	0·57	3·10

The grinding was finer than in the preceding tests, as shown by the following screen test:

Mesh	Weight, per cent
+100	0·4
−100+150	5·7
−150+200	12·6
−200	81·3
	100·0

The above tests are not conclusive as to the action of the graphite. They show a lower tailing with finer grinding, indicating that the gold is closely associated with the pyrite grains. Duplicate tests were carried out of the ore by grinding in water with a small amount of kerosene (1 c.c. per 1,000 grammes of ore), the object of the kerosene being to provide a film or coating on the particles of graphite, thereby rendering them inactive as precipitants of gold from solution. The ground pulp was dewatered and then repulped in cyanide solution and agitated for 8 hours. The grinding was the same as for Test No. 6.

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
8........	0·075	0·01	86·6	0·18	2·58	1·5 : 1
9........	0·075	0·015	80·0	0·24	2·61	1·5 : 1

In order to confirm these results a test was carried out similarly to the above without using kerosene. The results are as follows:

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
10.......	0·075	0·02	73·3	0·26	2·61	1·5 : 1

The results of Tests Nos. 8 and 9 indicate that in providing a protective film on the graphite a lower tailing is obtained, thus confirming the action of graphite in precipitating gold from solution.

The four tests are tabulated below to illustrate these conclusions:

Test No.	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Cyanide consumption, lb./ton	Remarks
6........	0·025	66·7	0·57	Grinding in cyanide.
10........	0·02	73·3	0·26	Grinding in water and filtered. No kerosene.
8........	0·01	86·6	0·18	Grinding in water and kerosene.
9........	0·015	80·0	0·24	Grinding in water and kerosene.

Test No. 11

The object was to make a separation of the graphite and sulphides by flotation. The results were not encouraging, as considerable pyrite came over in the graphite float and the pyrite concentrate contained considerable graphite. The tailing was low in gold, a fact indicating that the gold is largely associated with the pyrite.

The results are as follows:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Au, oz./ton	Per cent					
			C	S	Au	C	S	
Feed................	100·0	0·10	0·84	4·21	100·0	100·0	100·0	
Graphite concentrate.	2·1	0·54	21·2	19·88	11·3	52·9	9·9	47·6 : 1
Pyrite concentrate....	19·3	0·42	2·05	18·39	80·8	47·1	84·3	5·2 : 1
Tailing..............	78·6	0·01	....	0·31	7·9	0·0	5·8	

A screen test indicated that 77·4 per cent was minus 200 mesh.

Test No. 12

To confirm further the close association of the gold with the sulphides, a sample of ore was ground to have 82·7 per cent minus 200 mesh and a bulk concentrate floated off. This concentrate was tabled to obtain a comparatively clean sulphide product for cyanidation. The sulphide product assayed 0·44 ounce of gold per ton and after cyanidation the tailing assayed 0·16 ounce of gold per ton.

The flotation tailing carried 0·01 ounce of gold per ton and 1·39 per cent of sulphur. A sample of this product was panned on the Haultain superpanner and the clean sulphides and panner tailing were assayed. The results are tabulated below:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent
Feed (flotation tailing)..................................	100·00	0·01	100·0
Panner concentrate..	1·35	0·51	68·8
Panner tailing..	98·65	0·003	31·2

The results obtained indicate that the gold in the graphitic ore is largely associated in the dense sulphides (pyrite) and that it is extremely fine.

A series of cyanidation tests was carried out on ore from the main body mixed with different amounts of the graphitic ore to determine the effect of such additions of graphitic ore on extraction.

The mixtures were made in the following proportions, and the approximate analysis of graphite (carbon) and gold for each charge is as shown:

Test No.	Ore from main body, per cent	Graphitic ore, per cent	Carbon, per cent	Gold, oz./ton
13	100·0			0·20
14	87·5	12·5	0·105	0·18
15	75·0	25·0	0·21	0·17
16	62·5	37·5	0·305	0·15
17	50·0	50·0	0·42	0·14

The charges were ground in cyanide for the same time, and agitated for 24 hours. The results are tabulated below:

Test No.	Grinding per cent −200 mesh	Assay, Au, oz./ton		Extraction, of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
		Feed	Tailing		KCN	CaO	
13	74·3	0·20	0·01	95·0	0·81	2·95	1·5 : 1
14	75·6	0·18	0·01	94·4	0·85	3·52	1·5 : 1
15	75·6	0·17	0·01	94·1	0·90	3·05	1·5 : 1
16	76·1	0·15	0·015	90·0	1·05	3·05	1·5 : 1
17	76·0	0·14	0·01	92·8	0·96	3·11	1·5 : 1

Additions of graphitic ore up to 50 per cent therefore do not change appreciably the tailing value from that obtained by cyanidation of the main ore alone. The addition of graphitic ore, however, lowers the value of the feed and consequently the percentage extraction falls with increased proportions of graphitic ore.

CONCLUSIONS

When the graphitic ore is cyanided alone by standard methods the graphite tends to influence the extraction and to cause a high tailing. On mixing graphitic ore with ore from the main body this tendency is not apparent; the reason is not clear, but it may be due to one or more of the gangue constituents of the main ore providing a protective film on the graphite or rendering the grains inactive to the dissolved gold.

The results indicate that the graphitic ore can be mixed with the main ore without seriously interfering with the extraction.

Ore Dressing and Metallurgical Investigation No. 741

GOLD-SILVER ORE FROM THE BERENS RIVER MINES, LIMITED, AT FAVOURABLE LAKE, ONTARIO

Shipment. One shipment was received on June 23, 1937, containing 190 pounds of ore, and another on January 17, 1938, containing 335 pounds of ore. The samples were submitted by M. D. Banghart, Manager, Berens River Mines, Limited, 904 McArthur Building, Winnipeg, Manitoba.

Location of Property. This property is at Favourable Lake, Patricia District, Ontario, and is about 125 miles east and slightly north of Berens River post, on Lake Winnipeg at the mouth of Berens River.

Characteristics of the Ore. Six polished sections were prepared from each of the samples and were examined microscopically to determine the metallic minerals.

The sample received in June, 1937, is described as follows:

The *gangue* is fine-textured, grey, translucent quartz, with streaks of green chlorite.

The *metallic minerals* in the polished sections are: pyrite, pyrrhotite, sphalerite, chalcopyrite, magnetite, galena, and covellite. No native gold is visible.

Pyrite occurs as coarse to fine grains disseminated in the gangue, and in some areas is so abundant as to form small masses.

A small quantity of pyrrhotite occurs as coarse to fine irregular grains in the gangue, and more rarely, within pyrite masses.

Small quantities of sphalerite and chalcopyrite occur as moderately coarse to fine irregular grains in the gangue. Some of the chalcopyrite is associated with the sphalerite.

A very small quantity of galena occurs as irregular grains in the gangue. The grain sizes vary between 65 mesh and 1600 mesh, the estimated average size being about 200 mesh or a little smaller.

Magnetite is disseminated in the gangue, and a very small quantity of covellite occurs as tiny grains in the gangue and is a product of surface alteration that will not be present at depth.

The sample of ore received in January, 1938, is described as follows:

The *gangue* is fine-textured, grey, translucent quartz, with a very small quantity of finely divided carbonate, which occurs along fine fractures in the quartz.

The *metallic minerals* in the polished sections are, in their order of abundance: pyrite, sphalerite, galena, chalcopyrite, pyrrhotite, magnetite, tetrahedrite, and native gold. Pyrite is moderately abundant as coarse to fine disseminated grains and as masses. Sphalerite and galena, although locally abundant, are in only small quantities; they occur as irregular masses and disseminated grains. A small quantity of chalcopyrite occurs as irregular grains in the gangue, usually associated with pyrite, and pyrrhotite in very small quantity occurs in the same manner; both minerals also are present in the pyrite as tiny inclusions. Occasional disseminated grains of magnetite are present, and rare grains of tetrahedrite occur within the galena.

Only one grain of native gold is visible in the sections. This is a rounded particle, about 35 microns in diameter, enclosed within a grain of pyrite.

Alkalinity. A sample of the ore was ground with distilled water and a sample of the resulting solution was found to have a pH value of 8·4, indicating that the ore is naturally alkaline.

Previous examinations of ore from Favourable Lake, Ontario, have disclosed the presence of pyrite, sphalerite, galena, chalcopyrite, magnetite, covellite, and accessory arsenopyrite, but no gold has yet been observed in polished sections.

Sampling and Assaying. The shipments were dry-crushed to pass through a 14-mesh screen and were riffled down to a few pounds. The samples assayed as follows:

—		June, 1937 sample	January, 1938 sample
Gold	oz./ton	0·265	0·51
Silver	"	8·17	24·80
Copper	per cent	0·10	0·39
Lead	"	2·81	1·76
Zinc	"	4·34	3·91
Iron	"	7·30	7·23
Sulphur	"	8·33	7·97
Arsenic	"	Nil	Nil
Graphite	"	0·10	0·03
Pyrrhotite	"	0·116	0·32
Magnetite	"	0·26	0·55
Insoluble	"	70·94	71·90
Silica	"	65·97	66·15
Antimony	"	Nil	0·03
Cadmium	"	Nil	0·02
Carbon dioxide	"	0·75	0·88
Sulphate sulphur	"	0·10	0·05
Manganese	"	0·24	0·28

Experimental Tests

Investigation No. 692, published in 1936, deals with a previous shipment of ore from this property. The present shipments, the last one in particular, having been taken from greater depth and therefore considered more representative of the ore to be treated in a mill, were sent in to see if they would respond to the same method of treatment as the first one. It has been found possible to treat them by straight cyanidation provided the lime is carefully controlled and kept very low. It will not be necessary, therefore, to grind in water and pre-aerate, although the ore could be so treated.

More than 96 per cent of the gold and about 75 per cent of the silver can be extracted by direct cyanidation when the lime is kept to the minimum. The latest sample was found to be naturally alkaline and best extractions were obtained when no lime was added to the grinding or agitation circuits. Lime added for settling should be used sparingly and precipitated from the thickener overflow by the addition of some salt such as ammonium sulphate.

The tests are described in detail as follows:

Part I—Tests Conducted on the Sample of Ore Received in June, 1937

STRAIGHT CYANIDATION

Tests Nos. 1 to 8

Samples of the ore were ground 67·4, 76·5, 83·9, and 90·3 per cent through 200 mesh in ball mills and agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for periods of 24 and 48 hours. The cyanide tailings were assayed for gold and silver.

Summary:

Test No.	Grinding, per cent −200 mesh	Agitation, hours	Tailing assay, oz./ton		Extraction, per cent		Reagents consumed, lb./ton	
			Au	Ag	Au	Ag	KCN	CaO
1......	67·4	24	0·085	4·95	67·92	39·41	1·80	5·60
2......	76·5	24	0·12	5·36	54·72	34·39	2·04	5·63
3......	83·9	24	0·085	5·44	67·92	33·41	2·10	5·90
4......	90·3	24	0·085	5·41	67·92	33·78	2·34	5·96
5......	67·4	48	0·085	5·13	67·92	37·21	2·45	5·75
6......	76·5	48	0·11	5·44	58·49	33·41	2·51	5·90
7......	83·9	48	0·07	5·34	73·58	34·64	2·61	5·96
8......	90·3	48	0·055	5·21	79·25	36·23	3·00	6·17

CYANIDATION WITH LEAD SALTS ADDED

Tests Nos. 9 to 12

Samples of the ore were ground as in Tests Nos. 1 to 8 and agitated in cyanide solution, 2·0 pounds of potassium cyanide per ton, for periods of 48 hours. Red lead was added in the proportion of 1·0 pound per ton of ore. The cyanide tailings were assayed for gold and silver.

Summary:

Test No.	Grinding, per cent −200 mesh	Agitation, hours	Tailing assay, oz./ton		Extraction, per cent		Reagents consumed, lb./ton	
			Au	Ag	Au	Ag	KCN	CaO
9......	67·4	48	0·07	5·35	73·58	34·52	2·54	4·2
10......	76·5	48	0·075	5·41	71·70	33·78	2·96	4·35
11......	83·9	48	0·07	6·21	73·58	23·99	2·99	4·45
12......	90·3	48	0·08	4·96	69·81	39·29	3·29	4·45

FLOTATION WITH CYANIDATION OF THE CONCENTRATE

Test No. 13

A sample of the ore was ground 90 per cent through 200 mesh and a bulk concentrate floated. The concentrate was reground practically all through 325 mesh, aerated in lime pulp for 20 hours, and agitated in cyanide solution, 5·0 pounds of potassium cyanide per ton, for 72 hours. The products were assayed for gold and silver.

Charge to Ball Mill:

Ore	2,000 grms. −14 mesh
Soda ash	6·0 lb./ton
Aerofloat No. 31	0·07 "

Reagents to Cell:

	Lb./ton
Potassium amyl xanthate	0·10
Pine oil	0·15
Copper sulphate	1·0
Sodium sulphide	3·0

Summary:

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent	
		Au	Ag	Au	Ag
Concentrate	23·33	1·06	34·20	94·16	96·30
Tailing	76·67	0·02	0·40	5·84	3·70
Feed (cal.)	100·00	0·263	8·29	100·00	100·00
Concentrate cyanided	....	0·09	24·25	....	....

Extraction by Cyanidation of Concentrate:

86·17 per cent total gold;
28·0 " " silver.

Reagents Consumed:

	Lb./ton ore
KCN	1·46
CaO	4·62

CYANIDATION WITH PRE-AERATION

Tests Nos. 14 to 21

Samples of the ore were ground 90·3 per cent through 200 mesh in ball mills, and were aerated in lime pulp for 20 hours. Lime was added to the aerator at the rate of 10 pounds per ton of ore in Tests Nos. 14 to 19 and at the rate of 4 pounds per ton of ore in Tests Nos. 20 and 21. The aerated pulps were filtered, washed, and repulped in fresh cyanide solution, 2·0 pounds of potassium cyanide per ton, and were agitated for 48 hours at 1·5 : 1 dilution. The amount of lime in the cyanide solution was varied from about 1·0 pound per ton of solution down to 0·10 pound per ton of solution to see what effect it would have on the tailing assays.

Summary:

Test No.	Tailing assay, oz./ton		Extraction, per cent		Reagents in solution, lb./ton		Reagents consumed, lb./ton	
	Au	Ag	Au	Ag	KCN	CaO	KCN	CaO
14	0·04	4·66	84·90	42·96	2·44	1·0	2·75	2·50
15	0·025	4·66	90·57	42·96	2·44	1·14	2·75	2·30
16	0·02	4·12	92·45	49·57	1·70	0·63	1·94	0·85
17	0·02	3·94	92·45	51·78	1·88	0·44	1·67	1·14
18	0·015	3·72	94·34	54·47	2·02	0·30	2·10	Nil
19	0·015	4·12	94·34	49·57	1·92	0·30	2·25	Nil
20	0·01	3·61	96·23	55·81	2·10	0·10	2·24	Nil
21	0·01	3·69	96·23	54·83	2·02	0·10	2·33	Nil

NOTE.—The lime added to the aerator in each case is not included in the above figures for lime consumed.

The results show rather conclusively that high lime in the cyanide circuit reduces extraction of both gold and silver. When the lime is kept as low as in Tests Nos. 20 and 21, however, settling becomes very slow but this might be overcome by the use of alkaline starch.

GRINDING IN CYANIDE SOLUTION WITH LOW LIME

Test No. 22

The encouraging results obtained in Tests Nos. 14 to 21 suggested that grinding in cyanide solution might be possible if the lime were controlled closely enough. This was tried in the following test.

A sample of the ore was ground 90 per cent through 200 mesh in a ball mill with cyanide solution. Lime was added to the charge at the rate of 2 pounds per ton of ore and this was sufficient to maintain the lime at 0·25 pound per ton of solution throughout the 48-hour period of agitation. The strength of the cyanide solution was kept at the equivalent of 2·0 pounds of potassium cyanide per ton. The tailing was assayed for gold and silver.

Summary of Results:

Feed sample	0·265 Au, oz./ton 8·17 Ag, "
Cyanide tailing	0·015 Au, oz./ton 4·05 Ag, "
Extraction: Au	94·34 per cent
Ag	50·43 "

Reagents	KCN	CaO
Final titration, lb./ton solution	1·60	0·26
Consumed, lb./ton ore	2·47	1·60

CYCLE TEST

Test No. 23

The ore was ground in cyanide solution with a small quantity of lime. The pregnant solution from the first test was precipitated with zinc dust and the barren solution was used to grind and agitate a second batch of ore. Lead acetate was not used in precipitation of the solutions. A very small quantity of lime was added to the grinding circuit in each subsequent test. This procedure was continued until five lots of ore had been so treated. The tailings were assayed for gold and the final solution was examined for the presence of fouling matter. Agitation was continued for 48 hours.

Summary:

Test No.	Lime added, lb./ton ore	Tailing assay, oz./ton		Extraction, per cent		Final titration, lb./ton solution	
		Au	Ag	Au	Ag	KCN	CaO
Cycle 1.....	2·00	0·02	4·08	92·45	50·06	1·68	0·40
" 2.....	0·50	0·01	3·69	96·23	54·83	1·80	0·17
" 3.....	0·60	0·01	3·80	96·23	53·49	1·88	0·18
" 4.....	0·80	0·01	3·87	96·23	52·63	1·86	0·15
" 5.....	1·00	0·01	3·64	96·23	55·45	2·06	0·14

The final solution showed the following on analysis:

Reducing power....................................	876 c.c. $\frac{N}{10}$ $KMnO_4$/litre
KCNS..	1·088 grm./litre
Iron..	0·045 "
Ferrous iron......................................	Present
Copper..	0·03 grm./litre
Zinc..	0·70 "

CYANIDATION FOLLOWED BY FLOTATION

Test No. 24

A sample of the ore was ground in cyanide solution for 45 minutes and agitated for 48 hours. During the agitation the solution was kept at the equivalent of 2·0 pounds of potassium cyanide per ton and about 0·40 pound of lime per ton. The dilution ratio was 3 : 1 and the grinding was about 90 per cent through 200 mesh.

Cyanidation Results:	Gold	Silver
Feed sample assay, oz./ton................................	0·265	8·17
Cyanide tailing assay.....................................	0·02	4·20
Extraction, per cent......................................	92·5	48·6

Reagents:	KCN	CaO
Final titration, lb./ton solution.........................	2·00	0·40

The cyanide tailing, after being filtered, washed and sampled for assay, was reconditioned and floated selectively to yield lead, zinc and pyrite concentrates. The concentrates were not recleaned.

Reagents:

To Lead Cell	Lb./ton
Soda ash..	3·0
Zinc sulphate...	1·0
Sodium cyanide..	0·10
Butyl xanthate..	0·05
Cresylic acid...	0·10

To Zinc Cell	Lb./ton
Lime..	3·0
Copper sulphate...	0·5
Amyl xanthate...	0·05
Pine oil..	0·05

To Pyrite Cell	Lb./ton
Sulphuric acid..	1·8
Amyl xanthate...	0·10
Pine oil..	0·05

Summary of Test No. 24—Flotation

Product	Weight, per cent	Assay				Distribution, per cent			
		Oz./ton		Per cent					
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Lead concentrate..	7·35	0·056	32·21	30·80	2·39	20·60	56·32	73·37	4·22
Zinc concentrate...	12·51	0·071	8·71	4·50	30·50	44·40	25·93	18·25	91·58
Pyrite concentrate.	8·11	0·042	5·21	2·30	1·27	17·00	10·04	6·05	2·47
Flotation tailing...	72·03	0·005	0·45	0·10	0·10	18·00	7·71	2·33	1·73
Feed (cal.).........	100·00	0·02	4·20	3·08	4·17	100·00	100·00	100·00	100·00

The lead concentrate is rather low in grade but this would be improved by a cleaning cell. It carries a considerable quantity of silver and with local conditions favourable it could be handled at a profit.

All further work was done on a more recent shipment of ore, the present one being all used up.

Part II—Tests Conducted on the Sample of Ore Received in January, 1938

CYANIDATION

Tests Nos. 1 to 20

This series of tests was conducted with varying quantities of lime and cyanide in the solutions and with different periods of agitation. In each case the ore was batch ground in cyanide solution and the lime was added as detailed in the following table. Lime to primary thickener means lime added at the beginning of the agitation period, and lime to secondary thickener means lime added at the end of the agitation period. The dilution ratio was 1·5 : 1 in all cases.

A comparison of the results of Tests Nos. 5 to 8 with those of Tests Nos. 1 to 4 and 9 to 12 shows the benefit derived from using a smaller quantity of lime and at the same time keeping it out of the grinding circuit. The solutions were kept at about 2·0 pounds of potassium cyanide per ton in all these tests. In Tests Nos. 17 and 18 the same amount of lime was used as in Tests Nos. 5 to 8 but it was divided between the grinding circuit and the agitators, and with somewhat coarser grinding the tailing assays are just slightly higher. Tests Nos. 13 and 16 show the advantage of 2·0 pound cyanide solution over 1·0 pound cyanide solution, other things being the same. It also appears that a 2·0 pound cyanide solution will remain active under cycling much better than a 1·0 pound solution, although these preliminary tests indicate that with 48 hours of agitation a 1·0 pound solution will give as good extraction as a 2·0 pound solution will give in 24 hours. (*See* Test No. 30B).

Summary of Results:

Test No.	Grind, per cent −200 mesh	Agitation, hours	Solution strength, KCN lb./ton	Total lime added, lb./ton ore			Tailing assay, oz./ton		Extraction per cent		Reagents consumed, lb./ton ore	
				To grinding circuit	To primary thickener	To secondary thickener	Au	Ag	Au	Ag	KCN	CaO
1	69·4	24	1·0	1·0			0·04	7·89	92·16	68·19	2·76	0·55
2	78·5	24	2·0	1·0			0·03	8·68	94·12	65·00	2·82	0·50
3	82·3	24	2·0	1·0			0·03	7·75	94·12	68·75	2·84	0·55
4	88·7	24	2·0	1·0			0·03	7·41	94·12	70·12	2·84	0·52
5	69·4	24	2·0		0·50		0·015	6·39	97·06	74·23	3·17	0·11
6	78·5	24	2·0		0·50		0·015	6·34	97·06	74·44	2·99	0·17
7	82·3	24	2·0		0·50		0·015	6·28	97·06	74·68	3·54	0·05
8	88·7	24	2·0		0·50		0·015	6·17	97·06	75·12	3·51	0·11
9	63·22	48	2·0	1·0			0·026	6·82	94·90	72·50	3·07	0·49
10	69·80	48	2·0	1·0			0·023	6·65	95·49	73·19	3·25	0·49
11	78·50	48	2·0	1·0			0·020	6·37	96·08	74·31	3·03	0·48
12	88·70	48	2·0	1·0			0·015	5·91	97·06	76·17	3·54	0·46
13	56·90	20	1·0			1·0	0·078	7·93	84·71	68·02	2·15	0·60
14	54·90	20	1·0			1·0	0·083	8·30	83·73	66·53	2·09	0·50
15	57·28	24	2·0			1·0	0·0175	6·13	96·57	75·28	2·91	0·37
16	57·14	24	2·0			1·0	0·0178	5·92	96·51	76·13	2·85	0·40
17	57·0	24	2·0	0·30	0·20		0·02	6·21	96·08	74·96	2·82	0·11
18	57·0	24	2·0	0·30	0·20		0·02	6·46	96·08	73·95	2·70	0·11
19	57·0	48	1·0			1·0	0·02	5·90	96·08	76·21	2·51	0·58
20	57·0	72	1·0			1·0	0·02	6·27	96·08	74·72	2·86	0·52

Screen Analysis, Cyanide Tailing—Test No. 13:

Mesh	Weight, per cent	Assay, oz./ton		Distribution of gold and silver			
				Per cent content		Per cent total	
		Au	Ag	Au	Ag	Au	Ag
+100	6·14	0·31	8·52	24·41	6·59	3·74	2·11
−100+150	13·10	0·08	5·37	13·44	8·87	2·05	2·84
−150+200	23·88	0·06	6·13	18·38	18·45	2·81	5·90
−200	56·88	0·06	9·22	43·77	66·09	6·69	21·13
Average tailing	100·00	0·078	7·93	100·00	100·00	15·29	31·98

Extraction.. 84·71 per cent gold
68·02 per cent silver

Screen Analysis, Cyanide Tailing—Test No. 15:

Mesh	Weight, per cent	Assay, oz./ton		Distribution of gold and silver			
				Per cent content		Per cent total	
		Au	Ag	Au	Ag	Au	Ag
+100	2·70	0·08	4·35	12·31	1·91	0·42	0·47
−100+150	16·50	0·03	3·83	28·22	10·30	0·97	2·55
−150+200	23·52	0·02	4·51	26·82	17·29	0·92	4·27
−200	57·28	0·01	7·55	32·65	70·50	1·12	17·43
Average tailing	100·00	0·0175	6·13	100·00	100·00	3·43	24·72

Extraction.. 96·51 per cent gold
75·28 per cent silver

The high silver assays in the minus 200-mesh fractions are probably due to silver being associated with slimed galena.

Having determined the importance of lime control with respect to extraction and keeping in mind the part it plays in settling, a series of cycle tests was conducted to see how much lime could be permitted in the return solution to the grinding circuit.

CYCLE TESTS

Nos. 21 to 25

The first batch of ore was ground 58 per cent through 200 mesh in cyanide solution, 2·0 pounds of potassium cyanide per ton, without lime and then made up to a dilution of 1·5 : 1 and agitated for 24 hours. Hydrated lime was added at the beginning of the agitation period in the proportion of 0·50 pound per ton of ore which was sufficient to keep the solution at 0·25 pound of lime per ton. At the end of the agitation more lime was needed for settling at this dilution ratio and a total of 0·50 pound of hydrated lime per ton of ore was added in successive small amounts and the pulp finally settled well. At this point the solution was found to contain 0·36 pound of lime per ton. The solution was filtered off and precipitated with lead acetate and zinc dust, it was made up to volume with fresh water and to strength with fresh cyanide. Without being re-aerated the solution was used to treat a second batch of ore and this process was repeated until five lots of ore had been treated. The solutions were not de-aerated before precipitation.

Results:

Feed sample: gold, 0·51 oz./ton; silver, 24·80 oz./ton

Test No.	Alkalinity (CaO + NaOH) Lb./ton solution		Tailing assay, oz./ton		Extraction, per cent	
	Grinding circuit	Final solution	Au	Ag	Au	Ag
21.........	Nil	0·36	0·02	7·53	96·08	69·64
22.........	0·40	0·21	0·065	8·58	87·25	65·40
23.........	0·30	0·67	0·055	6·84	89·22	72·42
24.........	0·73	0·68	0·07	7·65	86·27	69·15
25.........	0·73	0·76	0·07	7·21	86·27	70·93

The final solution was examined and reported as follows:

Reducing power..................................	280 c.c. $\frac{N}{10}$ $KMnO_4$/litre
KCNS......................................	0·29 grm./litre
Copper......................................	0·007 "
Iron (total)................................	0·01 "

The cyanide consumption was estimated at 2 30 pounds KCN per ton of ore.

As the alkalinity in the grinding solution goes up the tailing assays also rise. Subsequent tests have shown that alkalinity in the form of sodium hydroxide in the grinding solution does not retard extraction even in fairly large amounts, but that alkalinity due to lime does. The sodium hydroxide generated in the precipitation process has an adverse effect on settling and therefore necessitates the addition of larger amounts of lime than would otherwise be needed for settling. The sodium hydroxide accumulates because the ore being alkaline in itself does not neutralize it. The harmful effect of sodium hydroxide in the solutions is therefore indirect rather than direct.

CYCLE TESTS—PRECIPITATION WITHOUT LEAD ACETATE

Tests Nos. 26 to 30B

To determine whether the lead acetate used in precipitation was in any way responsible for the high tailing assays obtained in Tests Nos. 21 to 25, this series of tests was conducted without using this salt. In the fifth cycle the barren solution was re-aerated before being used again and in the sixth cycle the pregnant solution was de-aerated under vacuum before precipitation and the barren solution re-aerated before it was used to treat the ore. An extra test was conducted in which the ore was ground about 80 per cent through 200 mesh and agitated 48 hours with the solution remaining at 1·0 pound of potassium cyanide per ton. The solution was de-aerated before precipitation and re-aerated afterwards in this test also. This test gave an unusually high tailing assay and the residue carried a considerable quantity of coarse native silver, which on assaying was found to contain about 10 per cent gold.

Results:

Feed sample: gold, 0·51 oz./ton; silver, 24·80 oz./ton

Test No.	Alkalinity (CaO + NaOH), lb./ton solution		Tailing assay, oz./ton		Extraction, per cent	
	Grinding circuit	Final solution	Au	Ag	Au	Ag
26.......	Nil	0·37	0·03	6·53	94·12	73·67
27.......	0·38	0·34	0·065	7·66	87·25	69·11
28.......	0·34	0·68	0·05	7·48	90·20	69·84
29.......	0·45	0·52	0·055	7·67	89·22	69·07
30.......	0·88	0·68	0·07	7·56	86·27	69·52
30A.....	0·90	0·84	0·035	5·99	93·14	75·85
30B.....	0·90	0·75	0·29	9·38	43·14	62·18

The final solution was examined and reported as follows:

Reducing power..................................	348 c.c. $\frac{N}{10}$ $KMnO_4$/litre
KCNS....................................	0·44 grm./litre
Copper....................................	0·03 "
Iron (total)................................	0·05 "

The cyanide consumption was e timated at 2·70 pounds of potassium cyanide per ton of ore.

The same quantities of lime were added in this series of tests as in Tests Nos. 21 to 25, and in Test No. 30A a sharp drop in the tailing assay is noted when the pregnant solution was de-aerated before precipitation and the barren solution re-aerated before it came in contact with the ore.

The lead acetate does not appear to be injurious in any way.

CYCLE TESTS WITHOUT PRECIPITATION OF SOLUTION TO PREVENT FORMATION OF SODIUM HYDROXIDE

Tests Nos. 31 to 35

In this series of tests with no sodium hydroxide in the solutions the lime was kept down to 0·20 pound per ton in the grinding circuit for the first three cycles and the tailing assays remained constant. In the fourth and fifth cycles it was purposely raised to slightly less than 0·40 pound per ton and the tailing assays immediately went up. The ore was ground 58 per cent through 200 mesh.

Results:

Feed sample: gold, 0·51 oz./ton; silver, 24·80 oz./ton

Test No.	Alkalinity (CaO only), lb./ton solution		Tailing assay, oz./ton		Extraction, per cent	
	Grinding circuit	Final solution	Au	Ag	Au	Ag
31.......	Nil	0·38	0·03	6·87	94·12	72·30
32.......	0·20	0·24	0·03	6·30	94·12	74·60
33.......	0·20	0·58	0·03	6·59	94·12	73·43
34.......	0·38	0·52	0·065	7·47	87·25	69·88
35.......	0·38	0·51	0·07	7·80	86·27	68·55

The final solution was examined and reported as follows:

Reducing power	360 c.c. $\frac{N}{10}$ $KMnO_4$/litre
KCNS	0·58 grm./litre
Copper	0·03 "
Iron (total)	0·03 "

The cyanide consumption was estimated at 2·80 pounds KCN per ton of ore.

An aqua regia test on No. 34 tailing brought it down to 0·025 ounce per ton gold and 0·13 ounce per ton silver.

GRINDING IN FRESH SOLUTION WITH SODIUM HYDROXIDE AS PROTECTIVE ALKALI

Tests Nos. 36 and 37

Samples of the ore were ground 58 per cent through 200 mesh in cyanide solution, 2·0 pounds of potassium cyanide per ton, with sodium hydroxide added to see if alkali in this form in the grinding circuit would reduce extraction. It was found that the sodium hydroxide had no harmful effect on extraction even when added in quantities up to 1·0 pound per ton of solution.

Results:

Test No.	NaOH, lb./ton solution	Tailing assay, oz./ton		Extraction, per cent	
		Au	Ag	Au	Ag
36	0·55	0·025	7·25	95·10	70·77
37	1·20	0·025	6·31	95·10	74·56

These and the foregoing tests seem to show that alkali in the form of lime is what causes the falling-off in extraction. It also appears that de-aeration of the pregnant solution followed by re-aeration of the barren solution is of the utmost importance (*See* Test No. 30A). The working solutions should at all times be kept saturated with "*dissolved*" oxygen. In practice this could best be done by using deep agitation tanks in which the air would be drawn down a central vortex and released at the bottom of the tank where the hydrostatic pressure would help to force oxygen into solution.

CYCLE TESTS USING STRAIGHT CYANIDATION WITH EMPHASIS ON DE-AERATION OF PREGNANT SOLUTION AND RE-AERATION OF BARREN SOLUTION

Tests Nos. 38 to 42

The ore was ground 58 per cent through 200 mesh in cyanide solution as before. The first batch of ore was ground and agitated without lime. At the end of the period of agitation pulp dilution was increased from 1·5 : 1 to 3 : 1 to correspond to the dilution in a washing thickener. At this point lime was added for settling. The pulp was then filtered, washed, and the cake assayed for gold and silver. Half of the filtrate was used to grind and agitate the next batch of ore while the other half was de-aerated, precipitated with lead acetate and zinc dust, and, after re-aeration, was used to dilute the agitated pulp at the end of the second cycle. This cycle was repeated till five lots of ore had been treated. The fifth test of the cycle was agitated in a sealed steel cylinder into which air was forced under pressure, it being otherwise treated the same as the other four tests of this cycle. The drop in the tailing assay of the fifth cycle as compared with the fourth is interesting and is no doubt due to the extra oxygen dissolved in the solution by reason of the air pressure within the jar. This condition would approach those conditions found in practice if deep agitator tanks be used and air released at the bottom level of the pulp.

Results:

Feed sample: gold, 0·51 oz./ton; silver, 24·80 oz./ton

Test No.	Alkalinity (CaO + NaOH), lb./ton solution		Tailing assay, oz./ton		Extraction, per cent	
	Grinding circuit	Final solution	Au	Ag	Au	Ag
38.......	Nil	0·28	0·02	6·28	96·08	74·68
39.......	0·175	0·50	0·025	6·34	95·10	74·44
40.......	0·325	0·48	0·03	6·49	94·12	73·83
41.......	0·375	0·64	0·05	7·21	90·20	70·93
42.......	0·50	0·68	0·03	7·13	94·12	71·25

Tests Nos. 43 to 47

As a further aid to cyanidation and for the specific purpose of precipitating part of the lime from the grinding solution coming back from the washing thickeners, ammonium sulphate was used in a series of tests.

The conditions were the same as those in Tests Nos. 38 to 42 except that ammonium sulphate was added to the solution in which the ore was ground and agitated in the proportion of 0·50 pound per ton of ore, or 0·33 pound per ton of solution. The steel pressure jar was not used to agitate any of them.

Results:

Feed sample: gold, 0·51 oz./ton; silver, 24·80 oz./ton

Test No.	Alkalinity (CaO + NaOH), lb./ton solution		Tailing assay, oz./ton		Extraction, per cent	
	Grinding circuit	Final solution	Au	Ag	Au	Ag
43.......	Nil	0·16	0·025	6·19	95·10	75·04
44.......	0·175	0·38	0·025	6·06	95·10	75·56
45.......	0·30	0·56	0·03	5·83	94·12	76·49
46.......	0·40	0·72	0·025	6·12	95·10	75·32
47.......	0·50	0·76	0·025	6·33	95·10	74·48

The final solution was examined and reported as follows:

Reducing power	360 c.c. $\frac{N}{10}$ $KMnO_4$/litre
KCNS	0·46 grm./litre
Copper	0·017 "
Iron (total)	0·03 "

The cyanide consumption was estimated at 2·90 pounds KCN per ton of ore.

Ammonium sulphate in the quantities used in these tests is apparently beneficial and if thorough aeration, as used in Tests Nos. 38 to 42, should in itself prove inadequate in practice this salt could be used as a further aid in the treatment of the ore.

Too much of the salt may be injurious, as some additional tests were conducted using the solution from Cycle Tests Nos. 43 to 47 in which this seemed to be the case.

In the first of the extra tests 1·0 gramme of the salt was added where previously the additions were limited to 0·25 gramme per cycle and the tailing assayed 0·040 ounce per ton in gold. In the second extra test 0·25 gramme was added and the tailing was still up to 0·048 ounce per ton in gold.

In the third extra test, however, still adding 0·25 gramme of the salt, the tailing assay was back to 0·023 ounce per ton in gold. This was the eighth cycle for this solution and it appears that this method of treatment will be successful if the additions of ammonium sulphate are kept within the proper limits.

Soda ash was tried unsuccessfully as a lime precipitant. When it was used with a recycled solution containing some lime, extraction fell off to about 75 per cent and the pulp would not settle with any amount of lime added. This test was conducted with the pulp heated to 120 degrees Fahrenheit.

CONCENTRATION AND CYANIDATION

Test No. 48

Bulk flotation and blanket concentration with cyanidation of the concentrates was also tried unsuccessfully, extraction falling off to less than 60 per cent. The superpanner showed that all sulphides and metallic gold were recovered in the concentrating operation, but the blanket tailing assayed 0·037 ounce per ton in gold and 1·46 ounces per ton in silver. The precious metals in the blanket tailing are therefore associated with the gangue, and an infrasizer analysis showed the following:

Infrasizer Analysis:

Size, in microns	Weight, per cent	Assay, oz./ton		Distribution, per cent content	
		Au	Ag	Au	Ag
+74	17·00	0·08	2·91	36·43	33·94
−74+56	1·25	0·10	2·45	3·35	2·10
−56+40	21·00	0·04	1·87	22·50	26·94
−40+28	19·46	0·03	1·21	15·64	16·16
−28+20	13·67	0·03	0·84	10·99	7·88
−20+14	8·97	0·015	0·67	3·60	4·12
−14+10	6·04	0·015	0·53	2·43	2·20
−10	12·61	0·015	0·77	5·06	6·66
Blanket tailing	100·00	0·037	1·46	100·00	100·00

This tailing sample was reduced to 0·025 ounce per ton in gold and 0·06 ounce per ton in silver by aqua regia.

Results:

Feed sample: gold, 0·51 oz./ton; silver, 24·80 oz/ton.

Product	Weight, per cent	Assay, oz./ton		Extraction, per cent total	
		Au	Ag	Au	Ag
Cyanide tailing from combined concentrates	26·24	0·69	32·06		
Blanket tailing	73·76	0·237	1·46		
Average tailing	100·00	0·208	9·49	59·21	61·73

Extraction:

	Per cent
Gold	59·22
Silver	61·73

Reagents:

Charge to Ball Mill

Ore	2,000 grms.—14 mesh
Water	1,500 c.c.
Soda ash	1·0 lb./ton
Barrett No. 4 oil	0·26 "

Reagents to Cell

Potassium amyl xanthate	0·20 lb./ton
Pine oil	0·10 "
Copper sulphate	1·0 "

The concentrate was agitated for 48 hours without regrinding in a recycled cyanide solution containing 2·0 pounds of potassium cyanide per ton. Alkalinity in the agitation circuit was 0·35 pound per ton.

Evidently the troublesome ingredient in the ore is a metallic mineral rather than a gangue constituent and it, therefore, passes into the flotation concentrate. Its activity perhaps increased with its concentration, thereby explaining the very low extraction obtained in this test.

The gold in the blanket tailing is evidently all enclosed in gangue, for the superpanner showed no trace of free gold or free sulphide minerals in it. The infrasizer analysis shows the grinding that would be necessary to liberate it.

CYANIDATION FOLLOWED BY FLOTATION TO RECOVER LEAD AND SILVER

Test No. 49

A sample of the ore was ground 78 per cent through 200 mesh in a recycled cyanide solution containing ammonium sulphate. The pulp was agitated 24 hours at 1·5 : 1 dilution and at the end of the agitation period dilution was raised to 3 : 1 and lime was added for settling. The pulp was filtered and the cake well washed and repulped in soda ash. A lead concentrate was floated off and cleaned once. Potassium ferrocyanide was used to depress the zinc. Perhaps any other iron salt would do just as well in the presence of free cyanide.

Reagents:	Lb./ton tailing
Soda ash	2·0
Zinc sulphate	1·50
Aerofloat No. 31	0·14
Potassium ferrocyanide	1·0
Sodium cyanide	0·10
Potassium ethyl xanthate	0·10
Cresylic acid	0·35

Results:

Feed sample: gold, 0·51 oz./ton; silver, 24·80 oz./ton

Product	Weight, per cent	Assay			Distribution, per cent content		
		Oz./ton		Pb, per cent			
		Au	Ag		Au	Ag	Pb
Concentrate	1·07	0·22	202·54	58·30	10·06	36·83	38·88
Cleaner tailing	1·15	0·13	85·18	29·14	6·39	16·65	20·89
Flotation tailing	97·78	0·02	2·80	0·66	83·55	46·52	40·23
Cyanide tailing (cal.)	100·00	0·023	5·88	1·60	100·00	100·00	100·00

The concentrate also carries about 2·0 per cent of copper.

Extraction by Cyanidation:	Per cent
Gold	95·49
Silver	76·29

By flotation of the cyanide tailing about 60 per cent of lead and 53 per cent of the contained silver are recovered in the final concentrate and cleaner tailing. In practice it would be reasonable to expect to recover at least 50 per cent of each in the final concentrate and perhaps bring the grade up a little higher than was done here.

Table concentration of the galena was tried and although a high-grade product can be produced recovery falls off to about 20 per cent owing to the fact that the galena slimes very readily and the greater part of it is too fine to be caught on a table. The microscopic examination of the June 1937 sample reports the galena grains as ranging from 65 to 1600 mesh in size, the average being 200 mesh or finer.

CYANIDATION WITH STRONG SOLUTION TO OBSERVE EFFECT ON SILVER EXTRACTION

Test No. 50

A sample of the ore was ground 78 per cent through 200 mesh in a recycled cyanide solution containing 0·52 pound of alkalinity per ton (CaO + NaOH) and the equivalent of 5·0 pounds of potassium cyanide per ton. Agitation was continued for 24 hours, after which the tailing was filtered, washed, and assayed for gold.

Results:

Feed Sample:		
Gold	0·51	oz./ton
Silver	24·80	"
Cyanide Tailing:		
Gold	0·023	oz./ton
Silver	6·25	"
Extraction:		
Gold	95·49	per cent
Silver	74·79	"
Reagents Consumed:		
KCN	4·45	lb./ton ore
CaO	0·25	" "

Evidently the stronger cyanide solution does not increase the silver extraction. This silver is probably enclosed in galena and immune to attack by cyanide at this grinding. The greater part of it can be extracted by aqua regia.

CONCLUSIONS

The two shipments of ore are similar in most respects. The later shipment is higher grade in gold and silver and contains more copper and pyrrhotite, and also small amounts of antimony and cadmium, which were not found in the earlier. Both need carefully controlled lime in small amounts for successful treatment by cyanidation.

The point of difference, however, that has an important bearing on the treatment of the two ores by cyanidation is that the earlier is naturally sufficiently acid to consume a certain amount of alkali and to permit careful control of the lime during cyanidation, whereas the later is distinctly alkaline and allows the alkali to accumulate, particularly the sodium hydroxide generated in precipitation of the gold by zinc dust. The sodium hydroxide hinders settling in the thickeners and calls for increased additions of lime to effect settling.

Alkali in the form of sodium hydroxide has no direct injurious effect on extraction of the gold by cyanide solution, but lime in anything more than a very small amount is decidedly injurious.

The exact reason for the reaction of the ore to lime is not definitely known but is thought to be due to the presence of galena in the ore or some impurity associated with it.

Thorough aeration of the pulp in the agitators is another matter of prime importance. The working solution should at all times be kept saturated with "dissolved" oxygen. To ensure this it would be wise to use deep agitator tanks and to draw the air down a central vortex to the bottom of the tank, where it would be released at the bottom level of the pulp and the hydrostatic pressure of the column of pulp would help to force the oxygen into solution. The importance of this is brought out in Cycle Test No. 42 in which the pulp was agitated in a steel jar under pressure.

Coming back to the subject of lime control, it will be difficult to measure the amount of lime in solution as such, in the presence of the sodium hydroxide that will build up in the solution. If the ordinary overflow type of thickener be used, the pulp will settle reasonably well with from 0·20 to 0·30 pound of lime per ton at 3 : 1 dilution when sodium hydroxide is absent or is present only in small amounts. As the sodium hydroxide builds up, settling becomes poor unless the lime be increased and then there is too much of it to permit use of the solution in the grinding circuit.

In this case ammonium sulphate can be added to the washing thickener overflow to convert part of the lime to calcium sulphate and so prepare the solution for use in the grinding circuit.

Alkaline starch might be used to advantage in place of lime for settling, particularly in the primary thickener, where a clear overflow is desirable. Another alternative would be to use a Genter thickener at this point instead of the overflow type. Any resultant de-aeration of the solution at this point would be all to the good, because this solution is going to precipitation anyway and at the same time the lime could be kept down.

In the washing thickeners a thick underflow will be required for filtering and the clarity of the overflow will be a matter of secondary importance as the overflow will go back to the grinding circuit and to dilute the classifier overflow. This solution should remain saturated with dissolved oxygen and for this reason the overflow type of thickener might be better at this point in the flow-sheet. The filtrate from the vacuum filters will be depleted in oxygen and it would be well to re-aerate it together with the washing thickener overflow before it is used in the grinding circuit.

From a purely metallurgical point of view, filter presses instead of thickeners, vacuum filters and re-aeration tanks might be best of all at this end of the circuit for this particular kind of ore, as it might mean the complete avoidance of lime from the mill solution and highest extraction is obtained in its absence.

The ore should be ground 75 or 80 per cent through 200 mesh and at the same time all of the classifier overflow should be finer than 100 mesh. This is brought out in the screen analysis on the cyanide tailing from Test No. 15. Some fine gold is enclosed in the gangue, as shown by the infrasizer analysis of a blanket tailing from which all free gold and free sulphides have been removed. (*See* Test No. 48.)

Lead and silver can be recovered by flotation from the cyanide tailing better than by table concentration or jigs, because the lead is so fine that most of it will escape over a table or jig. A table will make a higher grade product than flotation but will only recover about 20 per cent of the galena whereas flotation will recover 50 per cent or more with reasonably good grade.

The success of flotation of galena from the cyanide tailing will vary with the amount of lime maintained in the cyanide plant.

The successful treatment of the ore is therefore largely a matter of solving the settling problem with low lime in the circuit and of keeping the solutions saturated with "*dissolved*" oxygen.

On examination of the solutions from the cycle tests the reducing powers were not found to be too high but if they should become so after a longer period of use some barren solution will need to be bled out of the circuit daily and replaced with fresh.

Ore Dressing and Metallurgical Investigation No. 742

GOLD-SILVER ORE FROM THE NEGUS MINES, LIMITED, YELLOWKNIFE RIVER AREA, NORTHWEST TERRITORIES

Shipment. Nine sacks, weight 587 pounds, of high-grade gold ore were received on March 3, 1938, from the Negus Mines, Limited, Yellowknife area, Northwest Territories. The shipment was submitted by Hon. Charles McCrea, President, Negus Mines, Limited, 410 Royal Bank Building, Toronto, Ontario.

Characteristics of the Ore. Six polished sections were prepared and examined microscopically for determining the character of the ore.

The *gangue* is greyish white quartz with stringers of carbonate, which contains a small quantity of iron. The gangue is stained brown by iron oxides in some places.

The *metallic minerals* present in the sections are: pyrite, arsenopyrite, sphalerite, tetrahedrite, jamesonite, chalcopyrite, "limonite", covellite, and native gold.

Moderate quantities of pyrite and arsenopyrite occur as coarse to medium disseminated grains and crystals; it is possible that some of the grains identified as arsenopyrite may belong to the cobalt-mineral group, but of the several tested microchemically none contained cobalt. Small quantities of sphalerite, tetrahedrite, and jamesonite ($4PbS \cdot FeS \cdot 3Sb_2S_3$) occur as irregular grains and as stringers in the quartz. A lesser quantity of chalcopyrite occurs in the same manner. "Limonite" is present chiefly as stains in the gangue, but also is seen along fine veinlets in the sulphides; usually where these veinlets cut across chalcopyrite there is some alteration of the latter to covellite.

Native gold is well exhibited by the sections. The most common mode of occurrence of the metal is as coarse irregular grains in the gangue, sometimes associated with jamesonite, pyrite, and arsenopyrite. Where associated with these minerals it usually occurs around the borders of the grains or along fractures in the pyrite and arsenopyrite, with the exception of the occurrence of occasional small grains within jamesonite. As will be seen from the table of grain size, the percentage of finely divided gold is relatively small.

Grain Size of the Gold. The following table shows the grain size of the gold in the sections:

(Percentages by volume)

Mesh	Free in gangue	Bordering and veining pyrite and arsenopyrite	Bordering and within jamesonite	Associated with chalco-pyrite	Totals
+ 65	38·8	....	8·5	....	47·3
− 65+ 100	9·0	....	3·3	....	12·3
− 100+ 150	6·4	....	4·0	....	10·4
− 150+ 200	5·7	....	0·7	0·7	7·1
− 200+ 280	5·3	0·5	0·5	1·0	7·3
− 280+ 400	4·6	....	0·7	....	5·3
− 400+ 560	3·7	0·3	0·8	....	4·8
− 560+ 800	2·6	0·2	0·2	....	3·0
− 800+1100	1·1	....	0·4	....	1·5
−1100+1600	0·7	....	0·2	....	0·9
−1600	0·1	....	....	....	0·1
Totals	78·0	1·0	19·3	1·7	100·0

The following camera lucida drawing illustrates the association of the gold in the ore:

Figure 1. Drawing of polished section of gold ore from Negus Gold Mines, Limited, Yellowknife, N.W.T. Gold—black; jamesonite—dotted; arsenopyrite—crossed; quartz—plain white. Magnification: X 20 approximately.

Sampling and Assaying. The ore was crushed and carefully sampled by standard methods. This was assayed and gave the following results:

Gold	4·09 oz./ton (average of 6 assays)
Silver	4·38 oz./ton
Copper	0·12 per cent
Arsenic	0·10 "
Iron	2·10 "
Antimony	0·19 "
Lead	Trace
Zinc	0·41 per cent
Sulphur	0·65 "

RESULTS OF EXPERIMENTAL RESEARCH

Treating the ore by jig and blanket and amalgamation of the resultant concentrates shows a gold recovery of 85·86 per cent. Cyanidation of the blanket and amalgamation tailings accounts for a further gold recovery of 13·15 per cent, an overall gold recovery of 99·01 per cent. Flotation of the blanket tailing does not give so high an overall recovery of the gold, but has the advantage of recovering around 78 per cent of the silver. The silver recovery by cyanidation is very low. Amalgamation followed by flotation indicated an overall recovery of gold around 96·58 per cent. The ratio of concentration in flotation is 50·5 : 1, and the grade of concentrate had a precious metal content of gold 35·84 ounces per ton and silver 160·64 ounces per ton.

The data and results of the investigation follow in detail:

Test No. 1

A 50-pound sample of minus 14-mesh ore was fed to a laboratory Denver jig. The bed consisted of two layers of steel shot and 2 inches of coarse magnetite. The depth of magnetite was found to be too great and the charge was rerun with the magnetite bed reduced to ½ inch. The latter depth proved satisfactory. Considerable coarse free gold was recovered in the hutch product.

The jig overflow was filtered, dried, well mixed, and cut by means of a Jones riffle into eight portions of about equal amount. Each of these portions was ground for different periods, from 5 to 40 minutes, in a small ball mill and the pulp from each grind was fed to the jig, the overflow passing over a corduroy blanket set at a slope of 2½ inches in the foot.

The blanket tailings were filtered, dried, weighed, and assayed separately and retained for cyanidation and flotation tests.

The jig concentrates and blanket concentrates were combined and barrel-amalgamated.

The summary of the test is shown in the following table:

Summary of Jigging and Blankets:

Products	Weight, per cent	Grinding, per cent −200 mesh	Assay, oz./ton		Ratio of concentration
			Gold	Silver	
Feed	100·00	....	4·66*	4·38	
Jig and blanket concentrate	1·32	....	277·563	92·54	75·76 : 1
Blanket tailing A	11·98	19·9	1·55	3·00	
" " B	10·78	28·4	1·21	3·29	
" " C	12·31	40·7	1·06	3·12	
" " D	12·53	47·0	0·98	3·21	
" " E	12·52	58·8	0·95	3·24	
" " F	12·72	65·1	0·87	3·23	
" " G	13·11	70·3	0·80	3·24	
" " H	12·73	77·6	0·725	3·275	

* The gold in the feed is calculated from the assays of the blanket tailings and jig and blanket concentrates.

BARREL AMALGAMATION OF CONCENTRATES

The combined concentrates were ground to a fineness of 85·5 per cent minus 200 mesh and barrel-amalgamated with mercury and lime for one hour. The gold from the amalgam was recovered and weighed 2·7244 grammes. This weight plus the assay of the amalgam tailing gave the gold content of the combined concentrates as 277·563 ounces per ton.

Gold in amalgam tailing	3·62 oz./ton
Gold recovery	98·7 per cent
Silver in combined concentrates	92·54 oz./ton
Silver in amalgam tailing	48·38 "
Silver recovery	47·72 per cent

The gold recovery based on the blanket tailing at the finest grinding (H) shows a gold recovery in the concentrates of 84·65 per cent, or an overall recovery by amalgamation of 98·7 per cent of 84·65 per cent, = 83·55 per cent.

A sample of blanket tailing B (28·4 per cent minus 200 mesh) and one of blanket tailing F (65·1 per cent minus 200 mesh) were panned separately on the Haultain superpanner to determine if any free gold still remained in these products. Fine free gold was visible to the eye in the peak of the pan. This indicated that fine free gold was carried over the blanket. In order to determine if this gold could be recovered by further blanketing, a composite sample of blanket tailings, C, D, E, G, and H was repulped and run over a blanket at a slope of 1½ inches in the foot.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, of gold, per cent	Ratio of concentration
Feed	100·00	0·92	100·0	
Concentrate	0·32	45·92	15·9	312·5 : 1
Tailing	99·68	0·78	84·1	

By incorporating this increased blanket recovery in the original calculations, the overall gold recovery is found to be 85·86 per cent.

CYANIDATION OF BLANKET TAILINGS

Samples from four of the blanket tailings were agitated in cyanide of strength equivalent to 1 pound of potassium cyanide per ton and 2 pounds of lime per ton of ore was added as protective alkalinity. Periods of agitation were 24 and 48 hours.

Results:

Blanket tailing	Agitation, hours	Assay, feed, oz./ton		Assay, tailing, oz./ton		Extraction, per cent		Reagents consumed, lb./ton		Pulp dilution
		Au	Ag	Au	Ag	Au	Ag	KCN	CaO	
B	24	1·21	3·29	0·22	2·61	81·8	20·7	0·40	1·63	1·5 : 1
	48	1·21	3·29	0·145	2·45	88·0	25·5	1·15	2·63	1·5 : 1
D	24	0·98	3·21	0·11	2·86	88·8	10·9	1·08	1·66	1·35 : 1
	48	0·98	3·21	0·095	2·36	90·3	26·5	1·19	1·65	1·40 : 1
F	24	0·87	3·23	0·07	2·225	91·95	31·1	0·86	2·58	1·40 : 1
	48	0·87	3·23	0·06	1·94	93·1	39·9	0·97	2·68	1·27 : 1
H	24	0·725	3·275	0·05	2·30	93·1	29·8	1·45	1·55	1·5 : 1
	48	0·725	3·275	0·075	2·43	89·6	25·8	1·65	2·63	1·5 : 1

CYANIDATION OF AMALGAMATION TAILING

Strength of solution equivalent to 2 pounds of potassium cyanide per ton. Five pounds of lime per ton as protective alkalinity. Pulp dilution, 3 : 1.

Results:

Agitation, hours	Assay, feed, oz./ton		Assay, tailing, oz./ton		Extraction, per cent		Reagent consumption, lb./ton	
	Au	Ag	Au	Ag	Au	Ag	KCN	CaO
48	3·62	48·38	0·47	43·70	87·0	9·6	7·77	3·35

The presence of antimony in the ore would account for the moderately high consumption of cyanide.

The gold recovery by cyanidation of blanket and amalgamation tailings is as follows:

Blanket tailing—93·1 per cent of 12·91 per cent..................	12·02
Amalgamation tailing—87·0 per cent of 1·3 per cent............	1·13
Total recovery by cyanidation............................	13·15 per cent
Overall recovery by amalgamation and cyanidation, 85·86 + 13·15..	99·01 per cent

FLOTATION OF BLANKET TAILING

The following tests show the results obtained from flotation of the blanket tailings. Feed for Test 2A comprised equal amounts of tailings A and B; for Test 2B, of tailings C and D; for Test 2C, of tailings E and F; and for Test 2D, of tailings G and H.

Test No., grinding	Product	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
			Au	Ag	Au	Ag	
2A, 22·6 per cent −200 mesh.	Feed..........	100·00	1·44	2·91	100·0	100·0	19·5 : 1
	Concentrate...	5·14	18·18	46·56	65·0	82·1	
	Tailing.......	94·86	0·53	0·55	35·0	17·9	
2B, 41·8 per cent −200 mesh.	Feed..........	100·00	1·09	3·19	100·0	100·0	20·2 : 1
	Concentrate...	4·96	17·32	56·48	79·0	87·8	
	Tailing.......	95·04	0·24	0·41	21·0	12·2	
2C, 62 per cent −200 mesh	Feed..........	100·00	0·92	3·19	100·0	100·0	15·2 : 1
	Concentrate...	6·56	11·74	44·98	83·3	92·4	
	Tailing.......	93·44	0·165	0·26	16·7	7·6	
2D, 75·3 per cent −200 mesh.	Feed..........	100·00	0·80	3·16	100·0	100·0	37·2 : 1
	Concentrate...	2·69	24·00	105·02	80·6	89·5	
	Tailing.......	97·31	0·16	0·34	19·4	10·5	

Reagents used (lb./ton):

Reagent	Test No. 2A	Test No. 2B	Test No. 2C	Test No. 2D
Soda ash..................................	2·0	2·0	2·0	2·0
Aerofloat No. 31..........................	0·21	0·14	0·14	0·07
Amyl xanthate Z-5.........................	0·20	0·20	0·20	0·20
Pine oil..................................	....	0·05	0·05	0·10
Cresylic acid.............................	0·088	....	....	....

The ratio of concentration in the above tests is somewhat low, owing to considerable gangue coming over in the concentrate. This was largely due to the fact that the blanket tailings had dried and caused some surface oxidation on the gangue particles.

In order to overcome this condition a sample of fresh ore was carried through the stages of primary jigging, regrinding of overflow, secondary jigging, blankets and flotation in direct sequence. The flotation results are shown below and indicate that a high ratio of concentration is possible.

Products	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Feed	100·00	0·89	3·55	100·0	100·0	
Concentrate	1·98	35·84	160·64	80·1	89·5	50·51 : 1
Tailing	98·02	0·18	0·38	19·9	10·5	

The flotation tailing was panned on the superpanner and the amount of sulphide in the concentrate was very small. No free gold was visible to the eye, but two fine rounded grains were observed under the binocular microscope.

The gold recovery by amalgamation and flotation is as follows:

	Per cent
Recovery by amalgamation	85·86
Recovery by flotation, 83 per cent of 12·91 per cent	10·72
Overall recovery	96·58

The recovery of silver by amalgamation is around 13·3 per cent.

The overall recovery by amalgamation and cyanidation is 42 per cent and by amalgamation and flotation 78 per cent.

SUMMARY AND CONCLUSIONS

The gold is freed at moderately fine grinding and indicates a recovery of 85·86 per cent by amalgamation, employing jigs and blankets. Combined with cyanidation the overall recovery is 99·0 per cent; and combined with flotation it is somewhat lower, being around 96·6 per cent.

The sample investigated is very high-grade, but the method of treatment outlined should be applicable to ore of lower grade.

Ore Dressing and Metallurgical Investigation No. 743

CONCENTRATE FROM THE MONTAGUE GOLD MINES, LIMITED, HALIFAX COUNTY, NOVA SCOTIA

Shipment. The sample of concentrate from treatment tests of gold ore from the Montague Gold Mines, Limited, was received on May 20, 1938, from Professor A. E. Flynn, of the Nova Scotia Technical College, Halifax, for a microscopic determination of the condition of the gold. Professor Flynn had been able, by very fine grinding (to 0·008 mm. or 8 microns) and amalgamation, to reduce the grade of the concentrate from 0·8 ounce per ton to 0·3 ounce per ton.

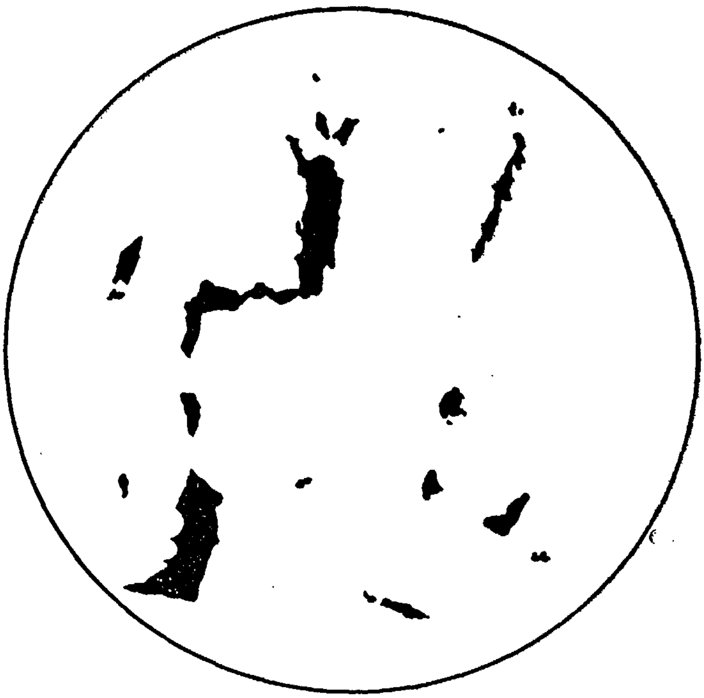


Figure 1. Drawing of polished section of gold ore from Montague Gold Mines, Ltd., Halifax County, N.S. Gold—black; arsenopyrite—plain white. Magnification: X 500.

Character of the Concentrate. The concentrate consists largely of arsenopyrite and pyrite, the former predominating. Minor quantities of pyrrhotite, chalcopyrite, and sphalerite, and occasional grains of gangue, are present.

Mode of Occurrence of the Gold. Microscopic examination of six polished sections of the concentrate confirmed the presence of native gold, the particles being too small to see with the unaided eye. Over 60 per cent of the visible gold occurs in dense arsenopyrite (*see* table and Figure 1) and a little less than 40 per cent as free particles. That in the arsenopyrite is extremely finely divided, some particles being so small as to be barely detectable under high powers of magnification, or well under 1 micron in size. The following table, which shows the distribution and size of the visible gold in the polished sections, indicates that 22·1 per cent of the gold, all in dense arsenopyrite, is less than 6 microns in size.

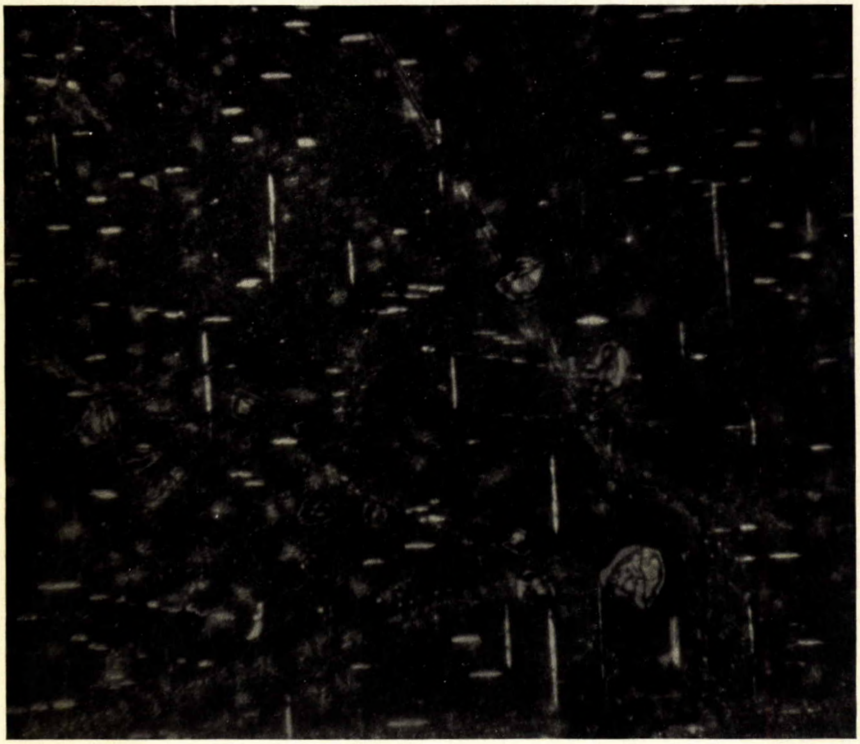
Distribution and Size of Visible Gold:

Size in microns	Approximate Tyler mesh	Gold in dense arsenopyrite, per cent	Free gold, per cent	Totals, per cent
+26	+ 560	18·4	23·7	42·1
−26+13	− 560+1100	7·4	10·5	17·9
−13+ 6	−1100+2300	13·7	4·2	17·9
− 6	−2300	22·1	....	22·1
Totals	....	61·6	38·4	100·0

CONCLUSIONS

It may be inferred from the microscopic work that with very fine grinding, such as was carried out by Professor Flynn, most of the gold larger than 6 microns would be freed, or nearly 80 per cent. It is highly improbable, however, that under normal conditions of amalgamation, particles finer than around 1100 mesh, or 13 microns, would break the surface tension of the mercury and thus be recovered. About 60 per cent of the gold would therefore be recovered, and this checks closely with Professor Flynn's results. Probably the larger factor in accounting for the gold remaining in the amalgamation tailing is its presence as tiny particles in the arsenopyrite, which are not freed in grinding, and a minor factor is the inability to amalgamate extremely finely divided gold even when freed.

PLATE I



Photomicrograph of polished section of galena from the Anglo-Huronian property in Mayo area, Yukon. The surface has been etched with HNO_3 (1 : 1), revealing the presence of dots and rods of a silver-bearing mineral, probably argentite. (*See* page 117) Magnification: X 240.

Ore Dressing and Metallurgical Investigation No. 744

SILVER-LEAD ORE FROM THE ANGLO-HURONIAN PROPERTY, MAYO AREA, YUKON

Shipment. A small sample of silver-lead ore from the Anglo-Huronian property in the Mayo area, Yukon, was received on May 16, 1938, from the General Engineering Company of Canada, Limited, Toronto, Ontario. Six polished sections were prepared, largely from pieces of galena, and examined microscopically.

Characteristics of the Ore. The sample consists chiefly of coarse crystals and cleavage fragments of galena. A small quantity of grey rock material is present, and some weathering to "limonite", and possible anglesite ($PbSO_4$), has taken place. Under the microscope polished sections show only one metallic mineral, namely, coarsely crystalline galena. A pure sample of galena, drilled from the section under the microscope, gave a strong microchemical test for silver. When the galena is etched with HNO_3 1 : 1, the presence of numerous tiny lenticular or rod-like inclusions, lying along the cleavages of the galena, is revealed (see Plate I). Positive identification of this mineral was impossible, but it is similar to the silver sulphide (Ag_2S—argentite) inclusions described by Schneiderhöhn* and may be regarded here as the same. Schneiderhöhn also states that a maximum of about 0·1 per cent Ag_2S (about 25 ounces of silver to the ton) can be carried in solid solution in galena. As the pure galena in this sample would assay far more than that, it seems certain that some of the silver, probably by far the greater proportion, occurs in the inclusions.

* Schneiderhöhn, Hans, and Ramdohr, Paul—"Lehrbuch der Erzmikroskopie"; p. 252.

Ore Dressing and Metallurgical Investigation No. 745

GOLD ORE FROM ST. ANTHONY GOLD MINES, LIMITED, STURGEON LAKE, ONTARIO

Shipment. Three samples of gold ore from St. Anthony Gold Mines Limited, Sturgeon Lake, Ontario, were received on May 21, 1938. B. D. Elderkin, of Toronto, Ontario, who submitted the samples on behalf of St. Anthony Gold Mines, Limited, requested the following information:

1. Genesis of deposition
2. Sequence of disturbances
3. Extent and magnitude of disturbance just prior to deposition of the gold
4. Mineral associations.

Six polished sections were prepared, two from each sample, and a microscopic examination was carried out with particular regard for any evidence bearing on the above points.

Characteristics of the Ore. The ore, as shown by the small samples available, consists largely of white quartz with a small quantity of green schistose wall rock. Minor quantities of pyrite, sphalerite, galena, pyrrhotite, and chalcopyrite, and native gold are present. The gold and sulphides occur most commonly along sinuous fissures in the quartz, a small proportion of the gold being associated with the sulphides.

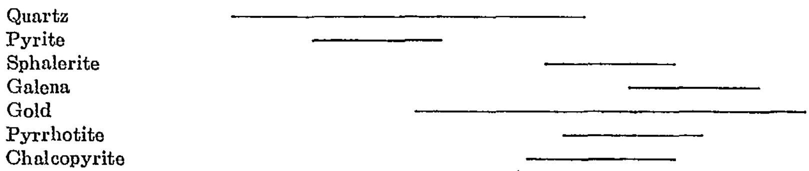
Relationships of the Minerals. The character of the fissures in the quartz does not indicate any major disturbance prior to, or contemporaneous with, mineralization. The fine sinuous character of the fissures, which follow curved courses rather than angular ones, indicates that their loci were determined by conditions of emplacement and solidification of the quartz gel and not by deformation. Gold and the sulphides, sphalerite and galena, occur along such fissures, which in places expand into "lakes" of gold or sulphides that appear to be interstitial to the quartz individuals. Gold does not replace the quartz, but the sulphides appear to have been deposited at the expense of the quartz, which they have slightly replaced in some places.

Pyrite is common in the wall rock, rare in the quartz. It is disseminated in both, and has been included in, and in some cases replaced by, sphalerite. Sphalerite and galena occur along the fissures as mentioned earlier. The sphalerite has been somewhat altered to, or replaced by, an undetermined non-metallic mineral, and some gold and galena appears to have accompanied this. Where relationships are shown, galena is apparently later than the sphalerite. A small quantity of pyrrhotite and rare chalcopyrite occurs in the sphalerite as rods following the crystallographic structure of this mineral.

A small portion of the gold occurs as rounded blebs in the quartz, without apparent connection with the fissures. A very large portion occurs along the fissures, as narrow veinlets and small "lakes" in the quartz. And a small portion is associated with galena, probably contemporaneously deposited. The rare grains of gold that are associated with sphalerite are probably younger than this sulphide, although the evidence is not conclusive.

Lastly, a small amount of fracturing appears to have taken place subsequent to the mineralization, but the fissures so developed contain no metallic minerals. This fracturing may have taken place during the mounting of the sample, and in any case has no significance on genesis.

Paragenesis. The paragenesis of the minerals, in so far as can be determined from the samples examined, may be represented as follows:



CONCLUSION

The fact that the present examination was carried out on only three small specimens of ore must be kept in mind when interpreting the data presented in this report. It seems certain, however, that no major disturbances have been recorded, and that the mineralization belongs to a single period starting with quartz and ending with gold deposition, the latter being most intense after the deposition of the other minerals had ceased. The disturbance which is responsible for the emplacement of the quartz must have been a major one, but its record must be sought in the relationships of the country rock to the vein zones.

Ore Dressing and Metallurgical Investigation No. 746

GOLD-SILVER ORE FROM THE PRIVATEER MINE, ZEBALLOS RIVER AREA, BRITISH COLUMBIA

Shipment. Seventeen sacks of ore, weighing 1,560 pounds, were received on March 23, 1938, from D. S. Tait, Secretary, Privateer Mine, Limited, 601 Bank of Toronto Building, Victoria, British Columbia.

Location of the Property. The property of the Privateer Mine, Limited from which the present shipment was received is situated on Spud Creek, Zeballos River area, Clayoquot Mining Division, Vancouver Island, British Columbia.

Sampling and Analysis. After cutting, crushing, and grinding by standard methods, a sample of the ore was obtained which assayed as follows:

Gold	4·70 oz./ton
Silver	2·33 "
Copper	0·17 per cent
Lead	0·86 "
Zinc	1·96 "
Iron	8·41 "
Arsenic	1·09 "
Sulphur	7·50 "

Characteristics of the Ore. Six polished sections of selected specimens from the shipment were prepared and examined in the mineragraphic laboratory. At a later date, several character samples taken from various parts of the vein were received. This microscopic report is based on a study of both shipments.

The *gangue* consists largely of white translucent vein quartz, locally mottled light grey by impurities. Some hand specimens contain a considerable quantity of calcite. The white quartz, mottled light grey quartz, and the carbonate quartz show a pronounced tendency to occur along parallel zones, as do also the sulphides, thus imparting to the ore a coarsely banded appearance. Locally coarsely-crystalline quartz surrounds open cavities, the inner surfaces of which are lined with well-formed prisms of quartz that has obviously grown in free space. These drusy surfaces also show certain sulphides, as will be noted later.

The *metallic minerals* present in the ore are, in their order of abundance: pyrite, arsenopyrite, sphalerite, galena, chalcopyrite, pyrrhotite, marcasite, and native gold.

Pyrite is moderately abundant as coarse grains and masses. It is considerably fractured, and is veined by sphalerite, galena, chalcopyrite, and native gold. A smaller quantity of arsenopyrite occurs as disseminated crystals, often associated with pyrite. The relationships between the pyrite and arsenopyrite are usually those of contemporaneous deposition, but in rare instances veinlets of gangue in pyrite are seen to carry arsenopyrite. Galena and sphalerite also vein arsenopyrite.

Sphalerite and galena are common, occurring as coarse grains and small masses. They are often seen together, where their relationships suggest their deposition at the same time, although in some places the galena appears to have attacked and enclosed remnants of sphalerite. The latter contains tiny dots and rods of chalcopyrite parallel to its crystallographic directions. Chalcopyrite, in relatively small quantity, occurs as noted under sphalerite, and as small masses and grains associated with the other sulphides. A small quantity of pyrrhotite is present as small masses and grains, usually rimmed by a thin layer of well crystallized pyrite; the structure would indicate that the pyrrhotite had filled a cavity that had previously been lined with pyrite. There is a slight alteration of the pyrrhotite, usually along cleavage fractures, to marcasite.

Native gold appears to be consistently associated with the sulphides, although it occurs to a large extent in the gangue in close proximity to these rather than within them. The table of gold occurrence and grain size best conveys this information.

TABLE I

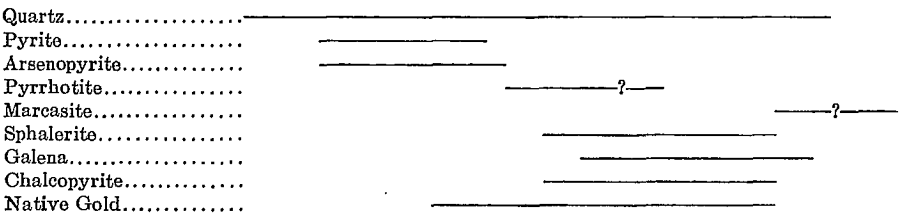
Mode of Occurrence and Grain Size of the Native Gold in Samples from the Privateer Mine, Limited:

Mesh	Gold in quartz, with sulphides but not enclosed by them, per cent	Gold in sulphides					Totals, per cent
		In pyrite		In dense arsenopyrite, per cent	In galena, per cent	In sphalerite, per cent	
		Along fractures, per cent	In dense pyrite, per cent				
+ 65..	26·1						26·1
− 65+ 100..	17·4	8·0			5·0	5·0	35·4
− 100+ 150..					6·3		6·3
− 150+ 200..	11·4	2·2					13·6
− 200+ 280..	4·7	1·7			3·2		9·6
− 280+ 400..	2·2						2·2
− 400+ 560..	2·6		0·9				3·5
− 560+ 800..	0·5				1·1		1·6
− 800+1100..			0·3	0·4			0·7
−1100........	0·3			0·5	0·2		1·0
Totals........	65·2	11·9	1·2	0·9	15·8	5·0	100·0
		13·1					
		34·8					

Paragenesis. Considerable information regarding the order of deposition of the metallic minerals is furnished by the microscopic examination and by the occurrence of certain sulphides among the quartz crystals that line the cavities in the massive quartz. Pyrite and arsenopyrite appear to have been deposited at about the same time; sphalerite, galena, and chalcopyrite are definitely later, and are also later than the last quartz crystals, which line the cavities, for they occur among them. Native gold has probably been deposited throughout much of the period of mineralization for it is partly contemporaneous with pyrite, arsenopyrite, and quartz, partly contemporaneous with the later sulphides. Probably the greater part of the gold deposition was post-pyrite. The following table represents graphically the deductions as to paragenesis of the minerals:

TABLE II

Paragenesis of the Minerals, Privateer Mine, Limited:



Conclusions from Microscopic Examination. The gold is largely moderately coarse. There is, however, a portion, probably about 1 to 2 per cent, which may prove refractory, and some of this is in dense pyrite and arsenopyrite. The mineralization of the gold is closely related to that of the sulphides, which latter should be a good indicator of gold in the mine.

Investigational Work. Treatment by concentration, amalgamation, and cyanidation constituted the research procedure on this ore. Recovery by means of jigs, traps, or blankets, followed by amalgamation of the resulting concentrates, gave an extraction of over 80 per cent of the gold in the ore. Flotation recovered over 18 per cent of the remaining gold and alternately cyanidation raised the overall recovery to 98·5 per cent, the final flotation and cyanide tailings assaying 0·02 and 0·04 ounce of gold per ton, respectively.

BARREL AMALGAMATION

Tests Nos. 1 and 2

In these tests the ore at minus 14 mesh was ground in a ball mill to pass 47·1 per cent through a 200-mesh screen in Test No. 1 and 79·1 per cent in Test No. 2. The pulps were then amalgamated with mercury for 1 hour in a jar mill. The amalgamation tailings were assayed for gold and silver.

Results:

Feed: gold, 4·70 oz./ton; silver, 2·33 oz./ton

Test No.	Tailing assay, oz./ton		Recovery, per cent	
	Au	Ag	Au	Ag
1	0·58	0·95	87·7	59·2
2	0·48	0·87	89·8	62·7

The above tests were for determining the total amounts of gold set free by these particular degrees of comminution, and the results are not comparable to the amounts of gold that could be recovered by either traps or blankets.

CONCENTRATION, AMALGAMATION, AND FLOTATION

Test No. 3

The ore at minus 14 mesh was ground in a ball mill to pass 79·7 per cent minus 200 mesh. The pulp was passed through a hydraulic classifier or trap and a trap concentrate obtained. The trap tailing was passed over a corduroy blanket, set at a slope of $2\frac{1}{2}$ inches per foot, and a blanket concentrate was recovered. The combined concentrates were amalgamated with mercury for 1 hour in a jar mill. The blanket tailing was conditioned with 3 pounds of soda ash per ton and floated with 0·10 pound of amyl xanthate and 0·05 pound of pine oil per ton. The different products were assayed for gold and silver.

A screen test showed the grinding as follows:

Screen Test:

Mesh	Weight, per cent
− 65+100	2·2
−100+150	3·6
−150+200	14·5
−200	79·7
	100·0

Results:

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Trap Concentration:						
Feed	100·00	4·70	2·33	100·0	100·0	
Trap concentrate	0·45	670·58	208·07	64·2	40·2	222 : 1
Trap tailing	99·55	1·69	1·40	35·8	59·8	
Blanket Concentration:						
Feed	100·00	1·69	1·40	100·0	100·0	
Blanket concentrate	1·54	61·80	30·16	56·3	33·2	65 : 1
Blanket tailing	98·46	0·75	0·95	43·7	66·8	
Flotation:						
Feed	100·00	0·86*	1·00*	100·0	100·0	
Flotation concentrate	11·51	6·46	7·70	86·0	88·5	8·7 : 1
Flotation middling	3·22	2·43	2·26	9·1	7·3	31 : 1
Final tailing	85·27	0·05	0·05	4·9	4·2	

* Calculated.

Amalgamation of Trap and Blanket Concentrates:

Feed assay, oz./ton		Tailing assay, oz./ton		Recovery, per cent	
Au	Ag	Au	Ag	Au	Ag
206·75	72·51	15·42	12·08	92·54	83·34

Summary:

—	Au, per cent	Ag, per cent
Recovered in trap concentrate	64·2	40·2
Recovered in blanket concentrate	20·2	19·8
Recovered in flotation concentrate	13·4	35·4
Overall recovery	97·8	95·4
Recovered by amalgamation of trap and blanket concentrates	78·1	50·0

CONCENTRATION, AMALGAMATION, AND FLOTATION

Test No. 4

The ore at minus 14 mesh was ground in a ball mill to pass 47·6 per cent minus 200 mesh. The pulp was passed over a small Denver jig and a jig concentrate obtained. The jig tailing was passed over a corduroy blanket and a blanket concentrate removed. The combined jig and blanket concentrates were reground and amalgamated. The blanket tailing was reground in a ball mill with the addition of 3 pounds of soda ash and 0·2 pound of Barrett No. 4 oil per ton and floated with 0·15 pound of amyl xanthate and 0·05 pound of pine oil per ton. A flotation concentrate was obtained.

Screen tests on the jig feed and the flotation feed resulted as follows:

Mesh	Weight, per cent	
	Jig feed	Flotation feed
− 48+ 65	4·2	....
− 65+100	13·5	0·1
−100+150	16·1	2·0
−150+200	18·4	7·0
−200	47·8	90·9
	100·0	100·0

Results:

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Jig Concentration:						
Feed	100·00	4·70	2·33	100·0	100·0	
Jig concentrate	6·85	54·88	17·97	80·0	52·8	14·6 : 1
Jig tailing	93·15	1·01	1·18	20·0	47·2	
Blanket Concentration:						
Feed	100·00	1·01	1·18	100·0	100·0	
Blanket concentrate	3·08	17·69	16·60	53·9	43·3	32·5 : 1
Blanket tailing	96·92	0·48	0·69	46·1	56·7	
Flotation:						
Feed	100·00	0·46*	0·62*	100·0	100·0	
Flotation concentrate	9·95	3·98	5·50	86·1	88·0	10 : 1
Flotation middling	4·83	0·54	0·49	5·6	3·8	20·7 : 1
Final tailing	85·22	0·045	0·06	8·3	8·2	

* Calculated.

The flotation concentrate assayed 4·80 per cent As.

Amalgamation of Jig and Blanket Concentrates:

Feed assay, oz./ton		Tailing assay, oz./ton		Recovery, per cent	
Au	Ag	Au	Ag	Au	Ag
44·57	17·60	3·61	5·40	91·9	69·3

Summary:

—	Au, per cent	Ag, per cent
Recovered in jig concentrate	80·0	52·8
Recovered in blanket concentrate	10·8	20·4
Recovered in flotation concentrate	7·9	23·6
Overall recovery	98·7	96·8
Recovered by amalgamation	83·4	50·7

Test No. 5

This was similar to Test No. 4. In the regrind of the blanket tailing, 0·07 pound of Aerofloat No. 31 replaced the Barrett No. 4 oil. The combined jig and blanket concentrates were amalgamated without regrinding. Conditions were otherwise the same.

Results:

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Jig Concentration:						
Feed	100·0	4·70	2·33	100·0	100·0	
Jig concentrate	5·2	73·79	26·21	81·6	58·5	19·2 : 1
Jig tailing	94·8	0·91	1·02	18·4	41·5	
Blanket Concentration:						
Feed	100·0	0·91	1·02	100·0	100·0	
Blanket concentrate	3·2	18·15	18·56	63·8	58·2	31·3 : 1
Blanket tailing	96·8	0·34	0·44	36·2	41·8	
Flotation:						
Feed	100·0	0·34	0·44	100·0	100·0	
Flotation concentrate	13·3	2·22	2·88	86·7	87·2	7·5 : 1
Flotation middling	5·6	0·52	0·79	8·5	10·0	17·9 : 1
Final tailing	81·1	0·02	0·015	4·8	2·8	

The flotation concentrate assayed 3·53 per cent As.

Amalgamation of Combined Jig and Blanket Concentrates:

Feed assay, oz./ton		Tailing assay, oz./ton		Recovery, per cent	
Au	Ag	Au	Ag	Au	Ag
53·98	23·48	12·62	7·76	76·6	67·0

Summary of Test No. 5:

—	Au, per cent	Ag, per cent
Recovered in jig concentrate	81·6	58·5
Recovered in blanket concentrate	11·7	24·1
Recovered in flotation concentrate	5·8	15·2
Overall recovery	99·1	97·8
Recovered by amalgamation	71·5	55·3

This test shows the necessity of regrinding the jig and blanket concentrates prior to amalgamation. In Test No. 4, where the concentrates were reground, 11·9 per cent additional gold was recovered. The use of Aerofloat No. 31 in place of Barrett No. 4 oil lowers the ratio of concentration and produces a low-grade flotation concentrate.

CYANIDATION

Test No. 6

The ore at minus 14 mesh was ground in a ball mill in cyanide solution of 1 pound per ton strength. In Tests A and B the ore was ground to pass 44·7 per cent minus 200 mesh and in Tests C and D, 76·6 per cent minus 200 mesh. The pulps were then agitated for 24- and 48-hour periods. Three pounds of lime per ton of ore was added to the grind. The cyanide tailings were assayed for gold and silver.

Screen tests showed the grinding as follows:

Screen Tests:

Mesh	Weight, per cent	
	Tests A and B	Tests C and D
− 48+ 65	2·9	....
− 65+100	13·3	0·7
−100+150	21·6	7·1
−150+200	17·5	15·6
−200	44·7	76·6
	100·0	100·0

Results:

Feed: gold, 4·70 oz./ton; silver, 2·33 oz./ton

Test No.	Agitation, hours	Tailing assay, oz./ton		Extraction, per cent		Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Au	Ag	Au	Ag	KCN	CaO	KCN	CaO
6A	24	0·10	0·51	97·9	78·1	0·72	0·26	1·38	3·40
6B	48	0·09	0·46	98·1	80·3	0·92	0·26	1·58	3·50
6C	24	0·08	0·45	98·3	80·7	0·72	0·28	1·38	3·40
6D	48	0·08	0·45	98·3	80·7	0·92	0·28	1·52	3·50

Test No. 7

In this test the ore at minus 14 mesh was ground in a ball mill and the pulp passed through a gold jig and the jig tailing passed over a corduroy blanket. The grinding and procedure were similar to those of Test No. 5. The combined concentrates were reground and amalgamated. The amalgam residue was added to the blanket tailing and this product was reground in cyanide solution of 1 pound per ton strength to pass 90·9 per cent minus 200 mesh. The pulp was agitated for 24- and 48-hour periods. The different products were assayed for gold and silver.

Results of Test No. 7:

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Jig Concentration:						
Feed	100·00	4·70	2·33	100·0	100·0	
Jig concentrate	4·20	88·64	27·65	79·2	49·8	23·8 : 1
Jig tailing	95·80	1·02	1·22	20·8	50·2	
Blanket Concentration:						
Feed	100·00	1·02	1·22	100·0	100·0	
Blanket concentrate	3·18	18·10	17·96	56·3	46·8	31·5 : 1
Blanket tailing	96·82	0·46	0·67	43·7	53·2	

The blanket tailing and amalgam residue assayed 0·85 ounce of gold and 1·23 ounces of silver per ton.

Cyanidation:

Feed: gold, 0·85 oz./ton; silver, 1·23 oz./ton

Agitation, hours	Tailing assay, oz./ton		Extraction, per cent		Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Au	Ag	Au	Ag	KCN	CaO	KCN	CaO
24	0·075	0·57	91·2	53·7	0·74	0·10	1·31	3·20
48	0·065	0·46	92·4	62·6	0·96	0·24	1·43	3·90

Summary:

—	Au, per cent	Ag, per cent
Recovered in jig concentrate	79·2	49·8
Recovered in blanket concentrate	11·7	23·5
Recovered by amalgamation	81·9	47·3
Extracted by cyanidation (24 hours)	16·5	28·3
Extracted by cyanidation (48 hours)	16·7	33·0
Overall recovery	98·6	80·3

Test No. 8

The ore at minus 14 mesh was ground in a ball mill to pass 79·7 per cent minus 200 mesh. The pulp was passed through a hydraulic classifier or trap and the trap tailing was passed over a corduroy blanket. The combined trap and blanket concentrates were reground and amalgamated. The amalgam residue was added to the blanket tailing and this product was agitated in cyanide solution for 24- and 48-hour periods.

A screen test on the cyanide tailing showed the grinding as follows:

Mesh	Weight, per cent
− 65+100	2·2
−100+150	3·6
−150+200	14·5
−200	79·7
	100·0

Results:

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Trap Concentration:						
Feed	100·0	4·70	2·33	100·0	100·0	
Trap concentrate	0·5	601·70	195·36	64·0	41·8	200 : 1
Trap tailing	99·5	1·70	1·36	36·0	58·2	
Blanket Concentration:						
Feed	100·00	1·70	1·36	100·0	100·0	
Blanket concentrate	3·32	31·70	16·20	61·9	39·6	30 : 1
Blanket tailing	96·68	0·67	0·85	38·1	60·4	

The combined trap and blanket concentrates were reground and amalgamated and the amalgam residue was added to the blanket tailing, this product assaying 0·85 ounce of gold and 1·09 ounces of silver per ton. The cyanidation resulted as follows:

Cyanidation:

Agitation, hours	Tailing assay, oz./ton		Extraction, per cent		Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Au	Ag	Au	Ag	KCN	CaO	KCN	CaO
24	0·07	0·53	91·8	51·4	0·84	0·18	1·10	3·20
48	0·06	0·46	93·0	57·8	0·88	0·18	1·38	3·60

Summary of Test No. 8:

—	Au, per cent	Ag, per cent
Recovered in trap concentrate	64·0	41·8
Recovered in blanket concentrate	22·3	23·0
Recovered by amalgamation	81·9	53·2
Extracted by cyanidation (24 hours)	16·6	24·0
Extracted by cyanidation (48 hours)	16·8	27·0
Overall recovery	98·7	80·2

CYANIDATION AND CONCENTRATION

Test No. 9

The ore at minus 14 mesh was ground in cyanide solution of 1 pound per ton strength to pass 79·7 per cent minus 200 mesh. The pulp was agitated for a 24-hour period. The cyanide residue was passed over a Wilfley table and a table concentrate was obtained. This concentrate was reground in cyanide solution of 3 pounds per ton strength to pass 99·0 per cent minus 325 mesh and the pulp was agitated for a 48-hour period. The different products were assayed for gold and silver.

Results:

Cyanidation:

Feed: gold, 4·7 oz./ton; silver, 2·33 oz./ton

Agitation, hours	Tailing assay, oz./ton		Extraction, per cent		Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Au	Ag	Au	Ag	KCN	CaO	KCN	CaO
24	0·10	0·60	97·9	74·3	0·92	0·26	1·14	3·6

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Feed	100·00	0·10	0·60	100·0	100·0	
Table concentrate	21·66	0·28	1·69	60·8	60·8	4·6 : 1
Table tailing	78·34	0·05	0·30	39·2	39·2	

Cyanidation of Table Concentrate:

Feed: gold, 0·28 oz./ton; silver, 1·69 oz./ton

Agitation, hours	Tailing assay, oz./ton		Extraction, per cent		Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Au	Ag	Au	Ag	KCN	CaO	KCN	CaO
48	0·145	1·18	48·2	30·2	2·9	0·8	5·6	13·0

Summary of Test No. 9:

—	Au, per cent	Ag, per cent
Extraction by straight cyanidation	97·9	74·3
Recovered in table concentrate	1·3	15·6
Extraction from table concentrate	0·6	4·7
Total extraction by cyanidation	98·5	79·0

CYANIDATION AND CONCENTRATION

Test No. 10

The ore at minus 14 mesh was ground in cyanide solution of 2 pounds per ton strength to pass 86·4 per cent minus 200 mesh. The pulp was agitated for a 48-hour period. The cyanide tailing was washed, conditioned with 3 pounds of soda ash and 1·0 pound of copper sulphate per ton and was floated with 0·15 pound of amyl xanthate, 0·07 pound of Barrett No. 4 oil, and 0·05 pound of pine oil per ton. The flotation concentrate was reground in cyanide solution of 3 pounds per ton strength to pass 99·0 per cent minus 325 mesh and the pulp agitated for a 48-hour period. A screen test on the primary cyanide tailing showed the grinding as follows:

Mesh	Weight, per cent
− 65+100	0·2
−100+150	3·4
−150+200	10·0
−200	86·4
	100·0

Results:

Cyanidation:

Feed: gold, 4·70 oz./ton; silver, 2·33 oz./ton

Agitation, hours	Tailing assay, oz./ton		Extraction, per cent		Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Au	Ag	Au	Ag	KCN	CaO	KCN	CaO
48	0·10	0·485	97·9	79·2	1·84	0·22	2·06	3·55

Flotation of Cyanide Tailing:

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Feed	100·0	0·10	0·485	100·0	100·0	
Flotation concentrate	18·0	0·49	2·28	87·7	84·8	5·6 : 1
Flotation tailing	82·0	0·015	0·09	12·3	15·2	

Cyanidation of Flotation Concentrate:

Feed: gold, 0·49 oz./ton; silver, 2·28 oz./ton

Agitation, hours	Tailing assay, oz./ton		Extraction, per cent		Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Au	Ag	Au	Ag	KCN	CaO	KCN	CaO
48.........	0·30	1·42	38·8	37·7	2·9	0·5	10·7	19·8

Summary:

—	Au, per cent	Ag, per cent
Extraction by straight cyanidation..................................	97·9	79·2
Recovered in flotation concentrate..................................	1·8	17·6
Extraction from flotation concentrate..................................	0·7	6·6
Total extraction by cyanidation..........................	98·6	85·8

CYCLE CYANIDATION TEST

Test No. 11

In order to determine whether any fouling takes place in the mill solution with consequent lowered extraction of the gold and silver, a series of cycle tests was run.

The ore at minus 14 mesh was ground in a ball mill in cyanide solution of a strength of 1 pound of potassium cyanide per ton, to pass 81·0 per cent minus 200 mesh. The pulp was agitated for a 24-hour period. After agitation, the cyanide tailing was assayed for gold and silver and the filtered solution made up to strength and used in the grinding and agitation of a fresh batch of ore, this procedure being used for five cycles on five fresh batches of ore. The final solution was assayed for reducing power, KCNS, and copper.

A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65+100..	0·7
−100+150..	6·5
−150+200..	11·8
−200..	81·0
	100·0

Cyanidation:

Feed: gold, 4·70 oz./ton; silver, 2·33 oz./ton

Cycle No.	Agitation, hours	Tailing assay, oz./ton		Extraction, per cent		Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Au	Ag	Au	Ag	KCN	CaO	KCN	CaO
1	24	0·095	0·59	98·0	74·7	0·88	0·22	1·05	3·05
2	24	0·145	0·68	96·9	70·8	1·04	0·26	1·53	3·95
3	24	0·11	0·71	97·7	69·5	0·92	0·18	1·68	4·55
4	24	0·10	0·70	97·9	70·0	0·92	0·22	1·59	3·95
5	24	0·105	0·84	97·8	63·9	0·96	0·20	1·95	4·00

The analysis of the final solution resulted as follows:

Reducing power	368 mls. $\frac{N}{10}$ $KMnO_4$/litre
KCNS	0·53 grm./litre
Copper	0·08 "

SETTLING TEST

Test No. 12

A series of tests at different ratios of liquid to solid and varying amounts of lime was made. After grinding in cyanide, with the requisite amount of lime added, the pulp was transferred to a tall glass tube and the level of solids in decimals of feet read for a 1-hour period. At the end of the test the solution was titrated for alkalinity.

A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65+100	0·7
−100+150	6·5
−150+200	11·8
−200	81·0
	100·0

Results:

—	Test No.			
	A	B	C	D
Ratio of liquid to solid	1·5 : 1	2 : 1	1·5 : 1	2 : 1
Lime added per ton solid, pounds	3·5	3·5	6·0	6·0
Alkalinity of solution at end of test, lb./ton	0·10	0·08	0·40	0·24
Overflow	Clear	Clear	Clear	Clear
Rate of settling, ft./hour	0·28	0·38	0·36	0·56

The rate of settling is but slightly slower than normal.

SUMMARY AND CONCLUSIONS

The results of the work on this shipment of ore give two alternative methods for metallurgical treatment.

In the first method, concentration by means of traps, jigs and blankets, followed by amalgamation of these concentrates and flotation of the blanket tailing, gave an 80 per cent recovery of the gold by amalgamation and a shipping product representing 17 to 18 per cent of the remaining 20 per cent of the gold in the ore. Owing to the large percentage of sulphides in the ore, the ratio of concentration by flotation will necessarily be low, the work showing a probable 8 or 9 : 1 ratio. This concentrate would also contain considerable arsenic.

In the second method, straight cyanidation of the ore extracted 97 to 98 per cent of the gold at a grind of 80 per cent minus 200 mesh. When the cyanide tailing was concentrated, reground and agitated, an additional 0·05 ounce gold per ton was extracted. Jigs or traps should be used to catch the coarse gold and relieve the load on the agitators. As shown by the microscopic work, some 2 to 3 per cent of the gold is contained in dense pyrite or arsenopyrite and will require exceedingly fine grinding to free and make it soluble in cyanide solution.

The cycle tests showed some fouling of the solution; the reducing power at 368 millilitres N/10 $KMnO_4$ per litre was not exceptionally high, however, and cautious bleeding of the barren solutions or overwetting of the final filter cake with a barren wash preceded by a water wash, should correct this condition. The shipment assayed 4·70 ounces of gold per ton; if the regular run-of-mine ore assay less, the amount of sulphides in the ore should be lower and give less cause for fouling. In this connection, the microscopic work indicates that the gold is regularly associated with the sulphides.

Ore Dressing and Metallurgical Investigation No. 747

MILL PRODUCTS FROM THE BEATTIE GOLD MINES, LIMITED, DUPARQUET, QUEBEC

Early in 1938 the Beattie Gold Mines, Limited asked that certain of their mill products be examined microscopically, the objective being an increment in overall recovery through improvements in detailed plant operation. A knowledge of the distribution of the gold in the flotation tailing and cyanide residue was regarded as of major importance, and samples of these and various other products that might contribute information were received for study. Preliminary investigation indicated that the microscope alone could not be expected to provide conclusive evidence, and it was therefore decided to supplement the study of polished sections with infrasizer and superpanner tests as well as assays and chemical analyses.

Previous microscopic examination of the ore, in connection with treatment tests made in 1933, shows that it consists of a quartz-carbonate gangue throughout which pyrite and arsenopyrite are very finely disseminated. Native gold is extremely rare, and where visible occurs as tiny particles associated with arsenopyrite. The paucity of visible native gold and the behaviour of the ore under treatment prove that the gold is largely in sub-microscopic form in the sulphides, dictating the concentration and roasting of the sulphides in order to obtain satisfactory recovery and this in turn necessitated very fine grinding.

Shipment. The samples received from the Beattie Gold Mines, Limited are listed in Table I and those produced during the investigation are listed in Table II. In both cases the numbers of the corresponding polished sections are given.

TABLE I

Samples Received from the Beattie Gold Mines, Limited:

Sample No.	Polished section No.	Origin
1	1171	Classified (settled and decanted) slime from composite flotation tailing for January, 1938.
2	1200-A	Rougher flotation tailing
3	1200-B	Cyanide residue
4	1201-A	Air cell flotation tailing
5	1201-B	Rougher concentrate
6	1202-A	Cyanide residue
7	1202-B	Final concentrate
B-109	1211	No. 1 concentrate, of very high grade

TABLE II

Samples Prepared During Investigation:

Sample No.	Polished section No.	Origin
41	1203-A	Sample No. 4, screened through 200 mesh. This is +200-mesh fraction.
42	1203-B	Sample No. 4, infrasized. Products from cones 1 and 2 mixed.
43	1204-A	Sample No. 4, infrasized. Products from cones 3 and 4 mixed.
44	1204-B	Sample No. 4, infrasized. Products from cones 5 and 6 mixed.
45	1205-A	Sample No. 4, infrasized. Product from dust bag.
411	1205-B	Sample No. 41, +200-mesh screen fraction. Superpanner concentrate.
412	1206-A	Sample No. 41, +200-mesh screen fraction. Superpanner tailing.
31	1206-B	Sample No. 3, +200-mesh screen fraction.
32	1207-A	Sample No. 3, infrasized. Products from cones 1 and 2 mixed.
33	1207-B	Sample No. 3, infrasized. Products from cones 3 and 4 mixed.
34	1208-A	Sample No. 3, infrasized. Products from cones 5 and 6 mixed.
35	1208-B	Sample No. 3, infrasized. Product from dust bag.
41-L	None	Sample No. 41, +200-mesh screen fraction. Leached with 10 per cent HCl.
42-L	None	Sample No. 42, infrasizer product. Leached with 10 per cent HCl.
43-L	None	Sample No. 43, infrasizer product. Leached with 10 per cent HCl.
44-L	None	Sample No. 44, infrasizer product. Leached with 10 per cent HCl.
45-L	None	Sample No. 45, infrasizer product. Leached with 10 per cent HCl.

Sample No. 4

AIR CELL FLOTATION TAILINGS

Characteristics of the Ore. Microscopic examination of sections of this product (Section No. 1201-A) shows a considerable proportion of coarse (+200-mesh) material. The quantity of free sulphide particles is almost negligible, but almost all of the larger grains, and many of the smaller ones, contain an appreciable quantity of very finely divided pyrite and arsenopyrite.

Assay of Feed Sample. Assays of the feed sample of Sample No. 4, air cell flotation tailing, were run at the Beattie Gold Mines, Limited and checked in the laboratories of the Division of Metallic Minerals. Table III gives these data:

TABLE III

Assays and Analyses of Sample No. 4:

Element	Assays of Beattie Gold Mines, Limited	Assays of Division of Metallic Minerals, Ottawa
Gold	0·02 oz./ton	0·02 oz./ton
Iron	3·13 per cent	2·90 per cent
Arsenic	0·242 "	0·08 "
Sulphur	0·34 "	0·32 "

Infrasizer Results:

A portion of Sample No. 4 was first screened through 200 mesh. The minus 200-mesh fraction was used for four runs of 400 grammes each in the Haultain infrasizer. The results are shown in Table IV. As will be noted the analyses were carried out on combined fractions, the products from cones 1 and 2, 3 and 4, and 5 and 6, being mixed.

TABLE IV

Infrasized Products and Their Analyses—Sample No. 4:

Product	Weight, in grammes	Calculated, per cent	Au, oz./ton	Fe, per cent	As, per cent	S, per cent
+200 mesh (screened)..........	1124·9	41·3	0·03	2·64	0·16	0·50
Cone No. 1 (infrasized)..........	2·4	0·1	0·02	2·55	0·16	0·30
Cone No. 2 (infrasized)..........	184·5	6·8				
Cone No. 3 (infrasized)..........	211·0	7·7	0·015	2·29	0·06	0·21
Cone No. 4 (infrasized)..........	186·5	6·8				
Cone No. 5 (infrasized)..........	156·5	5·7	0·01	2·75	0·06	0·12
Cone No. 6 (infrasized)..........	144·0	5·3				
Dust bag (infrasized)..........	715·1	26·3	0·01	3·92	0·11	0·19
Totals..........	2724·9	100·0	....	....	....	....

Microscopic Character of Infrasized Products. Polished sections of the infrasized products from Sample No. 4 were examined microscopically.

Section No. 1203-A, Screened, +200 mesh; almost all of the sulphide, both pyrite and arsenopyrite, occurs as very finely divided grains enclosed within gangue.

Section No. 1203-B, Cones 1 and 2; almost all the sulphides occur as finely divided grains in gangue.

Section No. 1204-A, Cones 3 and 4; most of the sulphides occur as finely divided grains in gangue; tiny free particles are comparatively rare.

Section No. 1204-B, Cones 5 and 6; part of the sulphides occur as finely divided grains in gangue, part as tiny free particles; the latter are estimated to predominate over the former.

Section No. 1205-A, Dust bag; the sulphides occur largely as tiny free particles, and are only rarely present in gangue.

Superpanner Test:

The plus 200-mesh screened fraction of Sample No. 4 was panned in the Haultain superpanner. The concentrate obtained consisted of about one per cent of the feed, and under the microscope was found to be largely of coarse grains of gangue, which forms the matrix for abundant particles of sulphide. Many of the grains of sulphide were exposed at the surface of the gangue, and would be expected to float. No visible gold is present. Assays of the above products gave results that are obviously useless; the tailings assayed higher than the feeds, probably owing to the contamination of the collecting system by gold from previous runs on other ores.

Condition of the Iron:

Table IV shows that with increasing fineness of material in the infrasized products, the iron content increases; the sulphur and arsenic are insufficient to satisfy equations for pyrite and arsenopyrite. In order to ascertain the percentage of iron in the carbonates of the ore, the products were leached with a cold solution of 10 per cent hydrochloric acid for 4 days. The results are given in Table V.

TABLE V

Distribution of Sulphide and Carbonate Iron in Infrasized Products of Sample No. 4:

Product	Loss in weight, per cent of sample	Insoluble iron, per cent of sample	Soluble iron, per cent of sample	Calculated iron, as pyrite, per cent	Calculated iron, as arsenopyrite, per cent
+200 mesh (screened)............	0·48	2·63	0·01	0·38	0·12
Cones 1 and 2 (infrasized)........	18·60	1·26	1·29	0·20	0·12
Cones 3 and 4 (infrasized)........	20·43	0·90	1·39	0·16	0·04
Cones 5 and 6 (infrasized)........	25·97	0·87	1·88	0·08	0·04
Dust bag (infrasized)............	27·36	1·41	2·51	0·12	0·08

Discussion of Data on Sample No. 4:

The most significant characteristic of the air cell flotation tailing brought out by the above data is the relatively large proportion of plus 200-mesh size and the correspondingly large quantity of locked sulphide it carries into the tailing. Table IV indicates that the values diminish with diminishing grain size, and the microscope proves locked sulphides predominate in the larger sizes whereas extremely finely ground free sulphides predominate in the fine sizes.

Table V shows that a soluble mineral or minerals, probably largely carbonate, grinds more easily than the insoluble siliceous part of the ore, and that an appreciable proportion of the iron is contained in the carbonate. When the iron is calculated as pyrite, arsenopyrite, and as soluble iron, from somewhat less than 1 per cent to over $2\frac{1}{2}$ per cent remains unaccounted for; this is probably due to the presence of insoluble iron-bearing minerals such as magnetite and leucoxene.

Summarizing, loss of gold into the flotation tailing appears to be due primarily to sulphides locked in coarse gangue, secondarily to slimed sulphides that have not floated.

Sample No. 3

CYANIDE RESIDUE

Characteristics of the Ore. Polished sections of the cyanide residue show that it consists of iron oxides, gangue, and unoxidized sulphides. Iron oxide and gangue form the major part, the quantity of unoxidized sulphides being very small. The product is very finely ground, with only a small amount in the coarse sizes (plus 200 mesh). The larger gangue particles, however, contain tiny included particles of unoxidized sulphide, this being the most common occurrence of the latter; unoxidized sulphides also occur infrequently as free particles.

One tiny grain, tentatively identified as native gold, was observed with a grain of gangue associated with iron oxide. Its mode of occurrence would thus make it impregnable to cyanide solutions.

Assay of Cyanide Residue. Two samples of cyanide residue, Nos. 3 and 6, were assayed in the laboratories of the Division of Metallic Minerals. Sample No. 3 was also assayed at the Beattie Gold Mines, Limited. The results are shown in Table VI.

TABLE VI

Assays and Analyses of Sample No. 3:

Element	Sample No. 3 Assay of the Beattie Gold Mines, Limited	Sample No. 3 Assay of the Division of Metallic Minerals	Sample No. 6 Assay of the Division of Metallic Minerals
Gold	0·090 oz./ton	0·09 oz./ton	0·09 oz./ton
Iron	25·19 per cent	25·14 per cent	20·66 per cent
Arsenic	0·376 "	0·36 "	0·47 "
Sulphur	1·03 "	0·94 "	1·25 "

Infrasizer Results:

A part of Sample No. 3 was screened wet through 200 mesh; the minus 200-mesh fraction was infrasized and the fractions assayed, with the results shown in Table VII.

TABLE VII

Infrasized Products and Their Analyses—Sample No. 3:

Product	Weight, in grammes	Calculated per cent	Au, oz./ton	Fe, per cent	As, per cent	S, per cent
+200 mesh (screened)	14·2	1·8	0·055	12·96	0·12	0·23
Cone No. 1 (infrasized)	2·0	0·3	Insufficient material			
Cone No. 2 (infrasized)	64·5	8·1	0·075	18·30	0·20	1·26
Cone No. 3 (infrasized)	105·0	13·2	0·07	20·82	0·22	1·30
Cone No. 4 (infrasized)	105·5	13·2	0·09	23·24	0·27	0·94
Cone No. 5 (infrasized)	83·0	10·4	0·085	24·64	0·28	0·66
Cone No. 6 (infrasized)	73·0	9·2	0·09	26·96	0·32	0·63
Dust bag (infrasized)	349·0	43·8	0·105	29·78	0·62	0·91
Totals	796·2	100·0				

Discussion of Data on Sample No. 3:

Table VII shows that decrease in grain size goes hand in hand with an increase in content of both gold and arsenic, whereas there appears to be little relationship between the gold content and the sulphur content. This suggests that the presence of locked gold particles in the coarse sizes is of little significance, and that the major reason for the losses may be the inability of the cyanide solution to attack the gold in the condition in which it is after roasting. This in turn makes it necessary to postulate the formation of compounds that render gold inactive in cyanide solution, such as iron-arsenic compounds or gold-arsenic eutectic, or both.

Sample No. 2

ROUGHER FLOTATION TAILING

Under the microscope pyrite and arsenopyrite are seen to predominate in this product. In general the sulphides are rather finely divided, those grains that have been freed in grinding predominating somewhat over those still enclosed in larger grains of gangue.

Analyses of the product are shown in Table VIII.

TABLE VIII

Analyses of Sample No. 2: Rougher Tailings:

Element	Analyses made at the Beattie Gold Mines, Limited	Analyses made at the Division of Metallic Minerals
Gold	0·055 oz./ton	0·0525 oz./ton
Iron	4·06 per cent	4·33 per cent
Arsenic	0·634 "	0·61 "
Sulphur	1·25 "	0·97 "

Sample No. 5

ROUGHER FLOTATION CONCENTRATE

The rougher flotation concentrate consists largely of freed sulphides and gangue. A small amount of the sulphides is locked in gangue. The analyses are given in Table IX.

TABLE IX

Analyses of Sample No. 5, Rougher Concentrate:

Element	Analyses made at Beattie Gold Mines, Limited	Analyses made at Division of Metallic Minerals
Gold	Not available	0·14 oz./ton
Iron	" "	5·45 per cent
Arsenic	" "	0·91 "
Sulphur	" "	2·42 "

Sample No. 7

FINAL CONCENTRATE

The product is composed of coarse to fine pyrite and arsenopyrite with minor gangue. The coarse grains of gangue contain sulphides. Rare chalcopyrite is present. The analysis is given in Table X.

TABLE X

Analysis of Sample No. 7, Final Concentrate:

Element	Quantity
Gold	0·48 oz./ton
Iron	20·46 per cent
Arsenic	3·10 "
Sulphur	19·68 "

Sample No. 1

CLASSIFIED (DECANTED) SLIME

Sample No. 1 was prepared by the Beattie Gold Mines, Limited from the composite flotation tailing for January, 1938, which was settled for 30 minutes and the slime decanted to form the sample designated as minus 400 mesh. The assay as given by the company is 0·02 ounce of gold per ton.

Under the microscope the product is seen to consist largely of gangue particles, with a minor quantity of sulphide as very tiny free grains seldom larger than a few microns in diameter. A microscopic grain analysis of the product, without regard for the character of the individual particles, is shown in Table XI. Most of the sulphide would fall within the −4 micron size ranges.

TABLE XI

Microscopic Grain Analysis of Sample No. 1:

Size in microns	Per cent
11	2·0
10	6·0
9	7·3
8	10·0
7	10·5
6	12·7
5	16·8
4	15·5
3	13·2
2	5·0
1	1·0
Total	100·0

No free gold was identified in this product, and it seems probable that the gold content is due to the presence of very finely divided (or slimed) sulphides.

Sample No. B-109

No. 1 CONCENTRATE

The assay of this sample, as given by the Beattie Gold Mines, Limited, is 50·5 ounces of gold per ton. The minerals identified under the microscope are, in their descending order of abundance, as follows:

Pyrite: Abundant;
Arsenopyrite: Abundant, but much less so than pyrite;
Chalcopyrite: Common, but small quantity;
"Limonite" (possibly somewhat confused with leucoxene): Small quantity;
Covellite: Rare; and
Native gold: Rare.

The native gold is largely free, rarely associated with arsenopyrite, all the particles being smaller than 20 microns in diameter. No quantitative results are available on the amount of native gold, but it is estimated that far less is visible in the sections than should be the case with a sample of this grade. This suggests a large proportion of sub-microscopic gold, probably largely in the sulphides.

CONCLUSIONS

The losses in treatment of ore at the Beattie Gold Mines, Limited are due primarily to losses into the final flotation tailing (Sample No. 4), secondarily to losses into the final residue (Sample No. 3). The work carried out indicates the following:

1. Loss of gold into the flotation tailing is due largely to sulphides locked in the coarser particles of gangue, but to a lesser extent to very finely divided free sulphides that have slimed and failed to float. Better control of grinding by more thorough classification, if economically practical, should result in better recovery by flotation.

2. Loss of gold into the cyanide residue appears to be due to the formation of compounds that inhibit the solution of the gold rather than to lack of grinding.

III

INVESTIGATIONS THE DETAILS OF WHICH ARE NOT PUBLISHED

Ore or Product	Source of Shipment	Address
Gold-lead-zinc	Cariboo-Hudson Gold Mines, Limited.	Cunningham Creek, Barkerville District, B.C.
Blanket tailing	San Antonio Mine	Bissett, Manitoba.
Gold	Wingold Mines, Ltd.	Bissett, Manitoba.
Gold	Ashloo Gold Mines, Limited	Squamish, B.C.
Gold	Arcadia Gold Mines, Limited	South Porcupine, Ont.
Gold	Hecla and Bareto Claims	Westbridge, B.C.
Copper	British Columbia Mine	Summit Camp, near Eholt, B.C.
Gold	Cole Gold Mines, Limited	Cole, Ontario.
Gold-silver	Bowhill Mine	Heron Bay, Ontario.
Gold	Alpine Mine	Nelson, B.C.
Gold	Madsen Red Lake Gold Mines, Limited.	Red Lake, Ontario.
Arsenical gold	Manitoba and Eastern Mine	Timagami, Ontario.
Gold-silver-lead-zinc	Calumet Island	Bryson, Quebec.
Gold	Halliwell Gold Mines, Limited	Rouyn, Quebec.
Arsenical gold concentrate.	Montague Gold Mines, Limited	Montague, Nova Scotia.
Gold	Senator-Rouyn, Ltd.	Rouyn, Quebec.
Arsenical gold	Gold Cup Mining Company, Limted.	Rossland, B.C.
Gold	Moneta-Porcupine Mines, Limited	Timmins, Ontario.
Gold	Orelia Mines, Ltd.	Mine Centre, Ont.
Graphite	C. H. Piggott	Griffith Township, Renfrew County Ont.
Copper-gold-silver	Copper King Mine	Kamloops, B.C.
Gold	St. Jude Gold Mines, Limited	Duprat Township, Que.
Arsenical gold	Nugold Mining Corp., Limited	Blockhouse, N.S.
Gold	Hutchison Lake Gold Mines, Limited.	Geraldton, Ont.
Gold	Cournor Mining Co., Limited	Perron, Quebec.
Gold	Chan Yellowknife Gold Property	Yellowknife District, N.W.T.
Chalcopyrite-molybdenite.	Regnery Metals	Hawk Junction, Ont.
Gold	South Vermillion Gold Mines, Limited.	Mine Centre, Ont.
Gold	Gurney Gold Mines, Limited	Gurney Siding, Manitoba.
Copper-gold	Brooklyn Mine	Greenwood, B.C.

Report of Laboratory Investigation re the Production of Bright Red Oxide Pigment from Bog Iron Ore, Labelle County, Quebec. (John MacFarlane & Son, Ltd.)

Tests on sample of steel rod. (Department of National Defence.)

The electric smelting of nickel-chromium magnetite concentrate obtained from mine tailings. (Canadian Johns-Manville Company.)

Tensile and impact tests on steel. (Atlas Steels, Limited.)

An examination of a failed austenitic manganese steel crusher jaw plate. (Sorel Steel Foundries, Limited.)

An examination of a failed austenitic manganese steel ball mill liner (Joliette Steel, Limited.)

An examination of a failed master rod from Tiger aircraft engine. (R.C.A.F., Department of National Defence.)

An examination of two chromium molybdenum steel end liners. (Sorel Steel Foundries, Limited.)

An investigation of the failure of a wing attachment aircraft fitting. (Department of Transport.)
Tensile testing of four aircraft bolts. (Department of National Defence, R.C.A.F.)
An examination of three aluminium-silicon die castings. (Department of National Defence.)
Historical identification of metal silver found near Waubaushene, Ontario. (Geological Survey.)
Magnetic concentration tests on treated Helen mine siderite. (A. T. Stewart.)
An investigation of two carburizing steels. (Canada Cycle and Motor Company, Limited.)
Examination of aluminium in defective gas tank. (Department of National Defence.)
Carburizing five landing gear parts. (Department of National Defence.)
Heat treatment of four steel bars. (Department of National Defence.)
Examination of manganese steel bullet proof hat. (Department of National Defence.)
Tensile and hardness tests on steel. (Department of National Defence.)
Impact test. (Dominion Engineering Company, Limited, Montreal.)
Hardness tests. (Hull Iron and Steel Foundries, Limited, Hull, Que.)
An examination of two galvanized iron sheets. (G. W. Benoit, St. Hyacinthe, Que.)
Tensile test. (Paxton Cooperage Company, Montreal.)
Investigation of the corrosion of piping installed at Banff Springs, Alta. (Parks Branch.)
Cleaning of 350 brass shells. (Dominion Archives.)
Microscopic grain analysis of four products from Aldermac Copper Corporation, Arntfield, Quebec.
Microscopic examination of two samples from Macassa Mines, Limited, Kirkland Lake, Ont.
Microscopic examination of sample from the Nicholson Mine property on the north shore of Lake Athabaska, Saskatchewan.
Microscopic examination of gold ore from the No. 230 vein, Hard Rock Gold Mines, Limited, Geraldton, Ont.

IV

A RÉSUMÉ OF SPECIAL INVESTIGATIONS AND RESEARCH COMPLETED, IN PROGRESS, OR UNDER CONSIDERATION

The activities of the Research Section of the Metallic Minerals Division show clearly the ever-widening range of the work of the Division.

Problems arise in connection with individual investigations of ores submitted that require increasing attention, as many of such ores are of a definitely refractory character. A combination of infrasizing, panning, chemical analysis and microscopic examination, employed in an effort to determine the association and mode of occurrence of the refractory gold, has been found to lead to a fairly accurate conclusion on the possibilities of recovery.

The amount of such work that can be done is limited, being governed to a large degree by the demand for investigations of ore samples submitted, and the capacity of the existing organization.

The metallurgical laboratory receives numerous requests for special examination of metal parts as fabricated, or of castings that have broken in use.

The utilization of certain natural resources or products for the production of alloys such as chromium steel and uranium steel is engaging attention. Progress is limited, however, by the size of the staff and the demand made upon it for general work.

The following is a résumé of research and special investigational work completed, in progress, or under consideration during the period January to June 1938:

Special investigations in connection with the treatment of gold ores were carried out. These ore samples were received from the following companies:

Cole Gold Mines, Ltd., Cole, Ontario.
Halliwell Gold Mines, Ltd., Rouyn, Quebec.
Tyranite Mines, Ltd., Gowganda, Ontario.
Gold Cup Mining Company, Ltd., Rossland, B.C.
Berens River Mines, Ltd., Empire, Ontario.
Cariboo-Hudson Gold Mines, Ltd., Wells, B.C.
Dome Mines, Ltd., South Porcupine, Ontario.

Each ore presented problems in gold extraction requiring special study and investigation. Some of the data thus obtained will be found in reports of the investigation of the particular ores.

Gold ores like these are being submitted with increasing frequency and it is becoming more and more evident that a thorough study of the part played by accessory minerals in their milling and cyanidation is very necessary if the maximum of extraction is to be obtained. Coupled with this should be a study of the mode of occurrence of gold in Canadian ores and minerals with relation to the problems of grinding.

Information on these subjects is accumulating from the investigative work on samples of ores submitted for testing, and will be supplemented by special studies as occasion arises. Thus, data have been obtained on the behaviour of pyrite and pyrrhotite in grinding and cyanide circuits, and on the chemical behaviour of stibnite in water, alkaline, and cyanide circuits, and a study of galena under similar conditions will follow.

A rapid and accurate method for the determination of oxygen in pulps has been sought by the industry for better control of milling practice. Three methods have been examined in the laboratory, namely, the evacuation method of Tromp and Schilz, the pyrogallic acid method of White, and the hydrosulphite method of Bowen and Weinig. For accuracy, particularly in cases of fouled solution, the evacuation method appears most suited, but about 2 hours of very careful manipulation are needed to obtain accurate results. The pyrogallic acid method has not always given satisfactory results; it is apparently affected by the presence of reducing or oxidizing constituents in the solution. Under carefully standardized conditions, however, it can be used satisfactorily for control of a given ore.

The hydrosulphite method also appears to be affected by the presence of copper and other interfering elements, which occur somewhat widely in Canadian ores.

At the laboratories of several gold mines trouble has been experienced in the determination of gold in pregnant and barren cyanide solutions, and a study was made of the methods employed or suggested, the most accurate results being obtained by a modification of the copper sulphate-sodium sulphite method suggested by S. B. Christy, or by means of the well known litharge evaporation method. In extremely fouled solutions the copper sulphate-sodium sulphite method requires special technique. The litharge method is decidedly the more accurate but takes much more time because of the necessary evaporation.

An investigation, in conjunction with the Industrial Minerals Division, was carried out on the utilization of a deposit of glauberite-anhydrite from Nova Scotia for the recovery of sodium sulphate. Leaching tests, involving percolation, agitation, and pressure leaching, were conducted; the agitation system gave the best results. Further work on the possible utilization of this material is under consideration.

The investigation into the production of red oxide pigment from bog iron ore, Labelle County, Quebec, started late last year, was completed, and a product satisfactory to trade requirements was produced. A procedure was suggested for pilot plant investigation.

In the study of the effect of molybdenum on the impact strength of cast iron, three series of cast irons containing respectively $2 - 2\frac{1}{2} - 3$ per cent of carbon have been made. Each series represented four irons, one molybdenum free, and three molybdenum-irons containing respectively $\frac{1}{4}$, $\frac{1}{2}$, and 1 per cent of molybdenum. The tensile strength, transverse strength, resilience, resistance to impact, and hardness of all irons have been determined. Preliminary tests indicate that the damping properties of the irons vary considerably, and it is intended to correlate these with the physical properties, and to repeat the work from this standpoint rather than as an investigation of the impact strength of the molybdenum cast iron.

In the investigation respecting uranium as an alloying element, high and medium carbon ferro-uranium has been made. Two uranium steel ingots have been cast and others will be made. All ingots will either be rolled or forged into one-inch bars for testing. The difficulty in the manufacture of uranium steel and ferro-uranium lies in the rapid oxidation of the uranium. The building of a vacuum- or atmosphere-controlled melting-furnace has been considered in order that this difficulty may be overcome.

The magnets originally supplied with the machine for studying damping properties were not properly designed and the work was delayed for their replacement, but several tests have now been made. To test the utility of the equipment the damping properties of various cast irons will be determined prior to beginning work on drill steel.

The following projects for various reasons remain in abeyance:

Rapid methods of desulphurizing and dephosphorizing iron and steel. This problem is of particular interest to the eastern iron and steel industry.

For such projects as the study of boiler plate brittleness and the ageing of steel, the age-hardening properties of copper-cast iron and the susceptibility to corrosion of aged materials, and the effect of hydrogen at high pressures on metallic materials, X-ray equipment would be essential if the work is to be done in a satisfactory manner.

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