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HON. T. A. CRERAR, MINISTER; CHARLES CAMSELL, DEPUTY MINISTER

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W. B. TIMM, CHIEF

INVESTIGATIONS IN ORE DRESSING AND METALLURGY

(Testing and Research Laboratories)

July to December, 1937

OTTAWA
J. O. PATENAUDE, I.S.O.
PRINTER TO THE KING'S MOST EXCELLENT MAJESTY
1938

No. 788

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INVESTIGATIONS IN ORE DRESSING AND METALLURGY, JULY TO DECEMBER, 1937

I

REVIEW OF INVESTIGATIONS

C. S. Parsons

Chief of Division of Metallic Minerals

This report embodies a description of investigations undertaken and reported on during the latter half of the year 1937, July to December inclusive, together with brief details of work on problems calling for special research, whether developing from the investigations undertaken in the laboratories or arising independently.

Seventy-six reports of investigation were issued to the companies concerned, but only fifteen are published in full. The rest, being of less general interest, are listed by title only in Part III.

Photomicrographs produced in the mineragraphic laboratory have greatly assisted the investigation and these are available to those interested.

The chemical laboratories made nearly 8,000 chemical and assay determinations in connection with the investigations, and in the mineragraphic laboratory over 400 polished sections of ores and products were prepared and examined microscopically.

Special investigations were made in connection with the productive capacity of certain industries, involving much travel and occupying a third of the time of one metallurgist. The amount of work coming from other Government departments and the other branches of this department, increases steadily, mostly in connection with the metallurgy of iron and steel and non-ferrous alloys. Much work of this nature is done for the Naval and Air Services, Department of National Defence. Numerous calls for test work and information also come from the Department of Transport, Department of Public Works, and the Royal Mint.

Although the work increased on other materials, gold still held first place, the number of gold ores investigated being thirty-two.

Among the investigations undertaken on the ores of base metals particular attention is directed to Investigations Nos. 725 and 731, the "Microscopic Examination of Ore and Concentrate from the Britannia Mining and Smelting Company", and the "Flotation Concentration of Ore from the Stirling Mine, Nova Scotia".

Special investigations were made to determine the cause of serious fouling in the operation of the Northern Empire and the Central Patricia mills and how to overcome this, the results of which are contained in Investigations Nos. 717 and 718.

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

A. Investigations:	
Examination of gold ores	37
Examination of base metal ores	3
Miscellaneous examinations	5
Special studies	10
Total	55
B. Spectrographic analyses	70
C. Polished sections prepared:	
For mineragraphic laboratory	436
For other divisions	4
For Bureau of Economic Geology	105
For others	40
Total	585

Chemical Laboratories. During the year, the staff was moved from the laboratories at the Bureau of Mines Building on Sussex Street, to the main laboratories on Booth Street.

During the period, July to December, 1937, 2,716 samples of ores, minerals, and metal products were analysed by the staff of the chemical laboratories, and a total of 7,958 chemical and assay determinations was made, involving 61 different mineral constituents.

The samples analysed consisted of:

Metallic ores and mill products	2,158
Metallurgical Laboratory	63
Industrial minerals and mill products	18
Products from Fuel Research Laboratories	19
Samples from Bureau of Geology and Topography	114
Custom assays	344
Total samples	2,716
Total determinations	7,958
Total gold assays	2,974
Total silver assays	749

Custom samples = 12·7 per cent of the total.

The work on ore dressing was carried out under the supervision of A. K. Anderson, W. R. McClelland, J. D. Johnston, H. L. Beer, and W. S. Jenkins, and the associated microscopic and spectrographic work was done by M. H. Haycock. All special research was under the general supervision of R. J. Traill, assisted by L. S. Macklin. 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 chemical laboratories in which the inorganic work is done for the whole Bureau of Mines, is under the supervision of J. A. Fournier, Chief Chemist. The following chemists and assayers are employed: R. A. Rogers, B. P. Coyne, A. Sadler, R. W. Cornish, J. S. McCree, S. R. M. Badger, A. E. LaRochelle, J. A. Rivington, L. Lutes, and C. H. Derry.

The building of a new laboratory to house all the ore-dressing equipment was begun late in the autumn of 1937. W. E. Ellis, one of the mill operators, was placed in charge of the mechanical draughting required in the design and layout of the equipment, and he was 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. 717

FLOTATION CONCENTRATE FROM THE NORTHERN EMPIRE MINES, LIMITED, EMPIRE, ONTARIO

Shipment. One can of flotation concentrate, weighing 50 pounds, was received on July 23, 1937, from W. S. Hargraft, Mill Superintendent, Northern Empire Mines, Limited, Empire, Ontario. Recently, it was found that during the agitation period the consumption of sodium cyanide had increased from 10 or 12 pounds to more than 20 pounds per ton.

Location of the Property. The property is in the Beardmore area, on the Canadian National Railway, 132 miles northeast of Port Arthur, Thunder Bay District, northern Ontario.

Sampling and Analysis. After cutting by standard methods a sample was obtained which assayed as follows:

Gold	10·58 oz./ton
Silver	1·665 "
Copper	0·40 per cent
Pyrrhotite	19·50 "

A screen test on the sample as received showed the grinding as follows:

Mesh	Weight, per cent
− 35+ 48	0·3
− 48+ 65	1·5
− 65+100	5·2
−100+150	9·6
−150+200	7·8
−200	75·6

EXPERIMENTAL TESTS

The test work included cyanidation with and without the use of lead salts, pre-aeration, and regrinding prior to agitation. The use of lead salts and pre-aeration was found to be beneficial, whereas regrinding was of no apparent benefit.

Tests Nos. 1 to 6

In all these tests the concentrate was agitated for periods of 24 or 48 hours, 10 pounds of potassium cyanide per ton of solution being added at the start and the solution kept at that strength by additional amounts of cyanide when necessary. Ten pounds of lime per ton of concentrate was added to maintain protective alkalinity.

In Test No. 1 the concentrate was agitated for 24 and 48 hours with a solution strength of 10 pounds of potassium cyanide per ton.

In Test No. 2, two pounds of litharge per ton was added.

In Test No. 3 the concentrate was reground in cyanide solution to pass 99 per cent through 200 mesh prior to agitation.

In Test No. 4 the concentrate was reground in cyanide solution and two pounds of lead nitrate per ton was added during the agitation period.

In Test No. 5 the concentrate was aerated in a lime pulp for 16 hours prior to agitation.

In Test No. 6 the concentrate was aerated 16 hours in a lime pulp and 2 pounds of lead nitrate per ton of concentrate added prior to agitation.

Results:

Test No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Lead salts added, lb./ton	Reagents consumed, lb./ton	
					KCN	CaO
1	24	0·40	96·2	Nil	28·40	7·90
1	48	0·28	97·4	Nil	46·00	7·00
2	24	0·19	98·2	2	8·76	7·00
2	48	0·16	98·5	2	13·30	7·20
3	24	0·75	92·9	Nil	25·79	6·25
4	24	0·28	97·4	2	12·98	7·84
5	24	0·23	97·8	Nil*	9·40	7·90
6	24	0·19	98·2	2*	6·40	8·00

*Aerated prior to agitation.

In order to illustrate the effect of the lead salt on the cyanide solution, a partial analysis was made of the solutions of Tests Nos. 3 and 4 after agitation. The solution of Test No. 3 contained no lead salt and that of Test No. 4 contained 2 pounds of lead nitrate per ton of concentrate.

Test No.	Reducing power, c.c. $\frac{N}{10}$ $KMnO_4$/litre	KCNS, grm./litre	Copper, grm./litre	Iron, grm./litre
3	1032	0·620	0·10	0·527
4	372	0·155	0·03	0·120

SUMMARY AND CONCLUSIONS

As shown in the different tests, the addition of a lead salt during the agitation period reduced the cyanide consumption to a satisfactory figure for this concentrate. Aeration in a lime pulp prior to agitation was also beneficial. A combination of the two methods produced the best results, as shown in Test No. 6, and this procedure should give the required results if followed in mill practice.

Ore Dressing and Metallurgical Investigation No. 718

GOLD ORE FROM CENTRAL PATRICIA GOLD MINES, LIMITED,
PICKLE CROW, ONTARIO

Shipment. A sample shipment, weighing 240 pounds, was received on July 28, 1937.

Purpose of Test Work. The purpose of the test work was to determine what minerals were interfering with cyanidation and what steps were necessary to bring about a more normal cyanide consumption and a higher extraction of the gold.

Sampling and Analysis. A sample was cut from the shipment for analysis, the results being as follows:—

Gold	0·50	oz./ton
Total iron	22·06	per cent
Ferrous iron	19·89	"
Total sulphur	5·07	"
Sulphur by evolution	3·33	"
Calcium oxide	5·97	"
Sulphuric anhydrite (SO_3)	0·13	"
Magnesia	2·63	"
Arsenic	1·50	"

Samples were taken and microscopic sections prepared and examined.

Pyrrhotite was found to be the most abundant mineral and there are smaller amounts of arsenopyrite and chalcopyrite. The gangue consists of quartz silicates and carbonate.

The chemical examination indicates the presence of ferrous iron other than that associated with pyrrhotite. Further tests, chemical and microscopical, confirm the presence of this ferrous iron as carbonate.

Carbonates of calcium and magnesium are also present.

Theoretical Considerations. The presence of pyrrhotite (ferrous sulphide) and ferrous carbonate is undoubtedly the reason for the extreme "fouling" of the cyanide solution in cyaniding and the low extraction obtained in plant operation.

The pyrrhotite appears to be a variety that is very readily attacked in even slightly alkaline solutions. A pulp of the ore and neutral water shows a pH value of around 8·4.

In grinding in water, it would appear that the pyrrhotite breaks up, the sulphur forming oxygen-sulphur compounds such as calcium sulphide, sulphite, thiosulphate, and other combinations. Samples of the water from the ground pulp showed appreciable reducing power when titrated with permanganate solution. In grinding in lime, the reactions are somewhat similar, except that a larger amount of sulphide is likely to be formed in comparison to sulphite or thiosulphate.

When grinding in cyanide (as practised at the Central Patricia plant), the calcium sulphide would react with the cyanide forming thiocyanate (KCNS), the ferrous iron from pyrrhotite and ferrous carbonate would react with cyanide forming ferrocyanide, and, in addition, oxygen-sulphur compounds of the nature described above would be present.

Low-strength cyanide solutions should, therefore, be expected to give low extraction as the fouling or consumption of the cyanide would be measurably rapid. High-strength cyanide solutions should possibly give higher extraction but at an increased rate of cyanide consumption.

The prevention of the formation of these reducing agents appears impossible in any sort of grinding, but the trouble of the fouling of the cyanide or of its excessive consumption can be corrected to some extent.

This would appear to be best accomplished by the system of aerating the ground pulp and preferably filtering before cyaniding.

As pointed out above, sulphide is likely to be first formed in the grinding circuit, the sulphide being gradually oxidized to sulphite and on to thiosulphate as the end product.

Thiosulphate is considered difficult to decompose in alkaline solution, nevertheless, it does oxidize slowly and would help to deplete the cyanide solution of necessary oxygen as the sulphide is formed. Ferrous salts are similarly built up as the process proceeds. Aeration oxidizes ferrous salts to ferric and in alkaline solutions, ferric hydrate would likely be formed and its effect on cyanide would be very slight.

Aeration, therefore, assists in oxidizing these chemical compounds and probably also assists in making the mineral more resistant to further reaction, whereby the cyanide is consumed. It does not, however, completely correct the formation of reducing matters in the cyanide solution as the amount of pyrrhotite and ferrous carbonate is too great for complete oxidation by means of aeration.

Tests show that by aeration, either in water or in lime circuit without cyanide, the subsequent consumption of potassium cyanide is materially reduced and the extraction maintained at a maximum of about 98 per cent.

Experimental Tests

Tests were conducted under varying conditions and a summary of the results obtained is shown in the table on page 8.

In Tests Nos. 1, 1A, 3, and 3A the ore was ground in cyanide and agitated for 24 hours. Tests Nos. 1 and 1A, using 1 pound of potassium cyanide, show low gold extraction and high fouling or reducing power, the latter caused by the formation of thiocyanate, ferrocyanide, and thiosulphate. Tests Nos. 3 and 3A with 3 pounds of potassium cyanide show high extraction, high cyanide consumption, and high fouling. The complex nature of the reducing matter in solution prohibits accurate determination of the constituents, but unquestionably it contains much thiocyanate, ferrocyanide, thiosulphate, and possibly other complex reducing salts.

In Tests Nos. 2, 4, 11, and 12 the ore was ground in water, aerated for 6 hours in a Denver laboratory agitator, filtered, and the ground ore repulped in fresh water with two pounds of potassium cyanide and lime.

The recovery in Test No. 4 is low for some reason, but Tests Nos. 2, 11, and 12 show extractions of 97 to 98 per cent of the gold with a potassium cyanide consumption of 3·7, 3·24, and 2·16 pounds respectively. The fouling of the solution in terms of reducing power is still high and the reducing material is of a complex nature. In Tests Nos. 5 and 6 the ore was ground with 5 pounds of lime, aerated 6 and 16 hours respectively, filtered, and agitated in fresh water with 2 pounds of potassium cyanide. The extraction is moderate, 93 to 94 per cent, the consumption of potassium cyanide slightly lower at 2·12 to 2·56 pounds. Lime consumption is about 7 pounds and the reducing power is still high, but lower in ferrocyanide.

In Tests Nos. 7 and 8 the ore was ground with 5 pounds of lime, aerated in alkaline pulp maintained at 0·3 pound of lime per ton of solution, 4 pounds being consumed in aeration. The aerated pulp was filtered and the ground ore repulped in fresh water with 2 pounds of potassium cyanide and 1 pound of lead oxide (PbO). The extraction was 97 to 98 per cent with 2 pounds of potassium cyanide and 11 pounds of lime.

The reducing power of the final cyanide solution is still somewhat high and consists mainly of thiocyanate and thiosulphate with very little ferrocyanide.

In Tests Nos. 9 and 10 the ore was ground in water without lime, filtered, and then aerated 6 hours. To the unfiltered pulp, 2 pounds of potassium cyanide was added with sufficient lime to keep the solution alkaline. The extraction drops off to 87 to 91 per cent, the consumption of potassium cyanide rises to about 4 pounds and the reducing power increases.

Tests Nos. 13 and 14 are duplicates of Tests Nos. 7 and 8, except that potassium permanganate was added during aeration in an attempt to oxidize reducing matter and to determine any possible effect on the sulphide mineral. No beneficial results were noted using this procedure, the extraction being similar to that of Tests Nos. 7 and 8. The consumption of potassium cyanide was slightly higher and the reducing power was also higher.

Tests Nos. 15 and 16 were conducted similarly to Tests Nos. 7 and 8, namely, grinding and aerating the ore in excess of lime. The cyanide consumption and reducing power are slightly lower and the extraction fairly satisfactory.

Tests Nos. 17 and 18, grinding in water, filtering, and repulping in fresh water and cyanide, showed high cyanide consumption.

CONCLUSIONS

From the small number of test results obtained to date, it appears that grinding in water with or without lime, and aerating for a period of six or more hours, followed by filtering and repulping the ground ore in fresh water and cyanide, results in satisfactory gold extraction and lowered consumption of chemicals. The test work indicated that grinding and aerating in alkaline circuit reduces the consumption of cyanide but at the expense of increased consumption of lime.

In any case, judging from present knowledge, the cyanide solutions resulting will still contain appreciable reducing matter, which will build up unless barren cyanide solution be systematically discarded.

Test No.	Grind 1,000 grms. 750 c.c. water, 45 min.	Aeration hours, 2 : 1 ratio	Remarks	Cyaniding, lb./ton 1·5 : 1, 24 hours	KCN consumption, lb./ton ore	CaO consumption, lb./ton	Tailing assay, oz./ton	Extraction, per cent	Reducing power, c.c. N/10 $KMnO_4$/litre	KCNS, grm./litre	Fe, grm./litre	Cu, grm./litre	Total S, grm./litre
1..........	In 1 lb. KCN	None	....	1	2·46	5·6 Final CaO, 0·25	0·095	81	....	....	....	....	....
1A.........	In 1 lb. KCN	None	....	1	1·88	Final CaO, 0·16	0·14	72	692	0·63	0·024	0·010	....
2..........	In water No CaO	6 No CaO	Filtered and repulped for cyaniding	2	3·7	3·62	0·015	97		Not determined			
3..........	In 3 lb. KCN	None	....	3	4·65	6·47	0·01	98	....	....	....	....	....
3A.........	In 3 lb. KCN	None	....	3	4·54	Final CaO, 0·18	0·015	97	920	Complex	0·067	0·012	....
4..........	In water No CaO	16 No CaO	Filtered and repulped 1·5 : 1	2	1·70	2·88 Final CaO, 0·08	0·035	93	628	1·0	0·018	0·012	....
5..........	In water 5 lb. CaO	6	Filtered and repulped 1·5 : 1	2	2·12	7 Final CaO, 0·14	0·03	94	492	0·81	0·0132	0·0024	....
6..........	In water 5 lb. CaO	16	Ditto	2	2·56	7 Final CaO, 0·08	0·035	93	500	0·78	0·0143	0·0032	....
7..........	In water 5 lb. CaO	6 4 lb. CaO added	Ditto	2 1 lb. PbO	2·0	11 Final CaO, 0·30	0·01	98	588	1·00	0·0003	0·0028	....
8..........	Ditto	Ditto	Ditto	Ditto	2·0	11 Final CaO, 0·30	0·015	97	440	0·90	0·0007	0·0016	....
9..........	2,000 grms. In water, 40 mins. No CaO Filtered	6 1·5 : 1 No CaO	Not filtered, cyanided direct	2	3·97	2·25	0·045	91	652	Complex	0·066	Not det.	0·4569
10.........					4·04	2·25 Final CaO, Tr.	0·065	87	580	Complex	0·069	Not det.	0·4599
11 Same as Tests Nos. 2 and 4	1,000 grms. In water	6 No CaO	Filtered and repulped	2	3·24	3 Final CaO, 0·025	0·01	98	448	Complex	0·054	Not det.	....
12.........	No CaO	16			2·16	2	0·01	98	528	Complex	0·055	Not det.	....
13.........	1,000 grms. In water	6 With added CaO—$KMnO_4$	Ditto	2	2·42	11 Final CaO, 0·10	0·01	98	764	1·10	0·011	Not det.	....
14.........	5 lb. CaO				2·46	11 Final CaO, 0·10	0·01	98	716	1·00	0·012	Not det.	....
15 Same as Tests Nos. 7 and 8	1,000 grms. In water	8 With CaO added 4 lb.	Ditto	2	1·62	11	0·015	97	560	0·80	0·003	Not det.	0·3847
16.........	5 lb. CaO				1·62	Final CaO, 0·26	0·02	96	560	0·75	0·004	Not det.	0·3882
17.........	1,000 grms. In water	Filtered and repulped 1·5 : 1 for cyaniding		2	7·26	8	0·130	74	1,760	Complex	0·110	Not det.	1·1844
18.........					7·32	Final CaO, 0·18	0·135	73	1,696	Complex	0·078	Not det.	1·0923
19.........	In water 45 mins. Aerated in CaO 48 hrs.	Filtered and repulped 1·5 : 1 for cyaniding		2	1·60	1·85 (0·1)	0·01	98	516	0·80	0·0067	Not det.	....
20.........					1·54	1·87 (0·08)	0·01	98	402	0·56	0·0039	Not det.	....

Ore Dressing and Metallurgical Investigation No. 719

GOLD ORE FROM THE COULSON CONSOLIDATED GOLD MINES, LIMITED, MATHESON, ONTARIO

Shipment. Eighteen bags of ore, total weight 2,359 pounds, were received on July 17, 1937, from Henry Reincke, Mine Manager, Coulson Consolidated Gold Mines, Limited, Matheson, Ontario.

Location of Property. The property of the Coulson Consolidated Gold Mines, Limited, from which the shipment was received, is located in Beatty and Coulson townships, Larder Lake mining division, Timiskaming District, northern Ontario.

Sampling and Analysis. The eighteen bags were crushed and sampled separately and the assay results forwarded to the mine management.

A composite sample was then made of the whole shipment. The research work was conducted on this composite sample. After cutting, crushing, and grinding by standard methods, a representative portion was obtained, which assayed as follows:

Gold	1·135 oz./ton
Silver	0·49 "
Arsenic	0·41 per cent
Iron	4·61 "
Sulphur	2·66 "
Copper	0·13 "
Pyrrhotite	0·62 "
Tellurium	Nil.

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

The *gangue* is white to greenish grey quartz crossed by numerous fine veinlets of carbonate.

The *metallic minerals* seen in the polished sections are, in their order of abundance: pyrite, chalcopyrite, arsenopyrite, sphalerite, pyrrhotite, and native gold.

Pyrite is moderately abundant as coarse to fine irregular grains disseminated in the gangue; it is abundant locally in the form of stringers of massive pyrite. A considerable quantity of the chalcopyrite occurs as coarse to fine irregular grains, usually associated with pyrite and arsenopyrite. The latter mineral is disseminated in considerable quantity as fine irregular grains and elongated crystals commonly associated with pyrite. A small quantity of sphalerite occurs as medium to fine rounded grains in gangue and in chalcopyrite; it contains numerous tiny dots of chalcopyrite. A small quantity of pyrrhotite is present as medium to small grains associated with pyrite, arsenopyrite, and chalcopyrite.

Twenty grains of native gold are visible in the sections. All occur within chalcopyrite, usually associated with sphalerite grains, and are very small, as shown in the following grain analysis:

Grain Analysis of Native Gold in the Chalcopyrite:

Mesh	Weight, per cent
+ 800	24
− 800+1100	14
−1100+1600	10
−1600+2300	27
−2300	25
	100

Experimental Tests

The research procedure on this ore consisted of cyanidation, flotation, table and blanket concentration, and amalgamation.

The flotation concentrate from the cyanide tailing was also roasted, reground, and agitated in cyanide solution. Amalgamation was tried on the hydraulic and blanket concentrates prior to agitation of the blanket tailing in cyanide solution. Straight cyanidation of the ore gave an extraction of 92 per cent of the gold content. This was raised somewhat by the concentration of the cyanide tailing and regrinding and re-agitation of the resulting sulphides. When roasting of the concentrates preceded cyanidation an additional 5·8 per cent of the gold was recovered.

Section A

AMALGAMATION AND CYANIDATION

Tests Nos. 1 and 2

The ore at minus 14 mesh was ground in a ball mill to pass 44·1 per cent through a 200-mesh screen in Test No. 1, and 73·4 per cent in Test No. 2. The pulps were amalgamated with mercury for 1 hour in a jar mill. The amalgam tailings were assayed for gold.

Results:

Test No.	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
1	1·135	0·475	58·1
2	1·135	0·36	68·3

The above tests were made to determine the total amounts of gold set free by these particular degrees of comminution, and the results are not comparable to the amount of gold that could be recovered by either plates or blankets.

Test No. 3

The ore at minus 14 mesh was ground in a ball mill to pass 64·2 per cent through a 200-mesh screen. The pulp was passed through a hydraulic classifier or trap and the coarse gold and heavier sulphides concentrated.

The trap tailing was passed over a corduroy blanket and a blanket concentrate recovered. The combined concentrates were barrel-amalgamated and the amalgam tailing added to the blanket tailing. This product was agitated in cyanide solution of 1 pound per ton strength for 24 hours. The various products were assayed for gold. A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65+100	2·8
−100+150	15·5
−150+200	17·5
−200	64·2
	100·0

Hydraulic Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	1·135	100·0	
Trap concentrate	0·91	45·23	36·3	110 : 1
Tailing	99·09	0·73	63·7	

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·73	100·0	
Blanket concentrate	2·35	11·33	36·5	42·5 : 1
Tailing	97·65	0·475	63·5	

The combined trap and blanket concentrates were barrel-amalgamated and the amalgam residue added to the blanket tailing. This product assayed 0·53 ounce of gold per ton.

Cyanidation of Blanket Tailing and Amalgam Residue:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·53	0·10	81·1	0·3	4·0

Summary:

	Per cent
Gold recovered in trap concentrate	36·3
" " in blanket concentrate	23·2
" " by amalgamation	53·3
" " by cyanidation	37·9
Overall recovery	91·2

Section B

CYANIDATION

Tests Nos. 1, 2, and 3

The ore at minus 14 mesh was ground to pass 80·5 per cent through 200 mesh. Portions of the pulp were agitated in cyanide solution of 1 pound per ton strength for 24- and 48-hour periods. Five pounds of lime per ton of ore was added. In Test No. 2 the pulp was aerated for 16 hours in lime solution. In Test No. 3, 1 pound of litharge was added to the aerated pulp prior to agitation.

A screen test showed the grind as follows:

Mesh	Weight, per cent
− 65+100	0·3
−100+150	5·2
−150+200	14·0
−200	80·5
	100·0

Results:

Feed: gold, 1·135 oz./ton

Test No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton		Remarks
				KCN	CaO	
1	24	0·105	90·75	0·44	3·90	
1	48	0·095	91·60	0·74	4·20	
2	24	0·095	91·60	1·24	3·70	Aerated 16 hours
2	48	0·085	92·50	2·16	4·00	Aerated 16 hours
3	24	0·08	92·95	0·66	3·60	1 lb./ton PbO added
3	48	0·085	92·50	0·86	4·10	1 lb./ton PbO added

Test No. 4

In order to determine the effect of exceedingly fine grinding on cyanide extraction, the ore at minus 14 mesh was ground in a ball mill to pass 98·4 per cent minus 200. The pulp was agitated in cyanide solution of 1 pound per ton strength for a 24-hour period.

Results:

Feed: gold, 1·135 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
24	0·09	92·1	3·40	5·0

Test No. 5

Cycle Tests

In order to determine whether the cyanide solution fouled during continued use in the mill circuit, a series of cycle tests was made. The ore at minus 14 mesh was ground in cyanide solution at a strength of 1 pound of potassium cyanide per ton with 5 pounds of lime per ton of ore added, to pass 80·5 per cent minus 200 mesh. The pulp was agitated in cyanide solution of the same strength for a 24-hour period, and was filtered, and the filtrate used in grinding and agitating a fresh batch of ore. The process was repeated for four cycles.

Results:

Feed: gold, 1·135 oz./ton

Cycle No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
1	24	0·100	91·2	1·30	4·4
2	24	0·102	91·1	0·90	4·4
3	24	0·100	91·2	0·85	4·5
4	24	0·100	91·2	0·50	4·5

An analysis of the final cyanide solution resulted as follows:

Reducing power	384 mg. $\frac{N}{10}$ $KMnO_4$/litre
KCNS	0·535 grm./litre
Copper	0·14 "

Section C

CONCENTRATION AND CYANIDATION

Test No. 1

The ore at minus 14 mesh was ground to pass 80·5 per cent through a 200-mesh screen and the pulp was agitated in cyanide solution at a strength of 1 pound of potassium cyanide per ton for a 24-hour period. The cyanide tailing was washed and conditioned with 4 pounds of soda ash per ton, reactivated with 1·0 pound of copper sulphate, and floated with 0·15 pound of amyl xanthate, 0·07 pound of Aerofloat No. 31, and 0·05 pound of pine oil per ton. A flotation concentrate was removed. The flotation tailing was passed over a corduroy blanket and a blanket concentrate recovered. The flotation and blanket concentrates were combined, reground to pass 99·0 per cent minus 200 mesh, and agitated in cyanide solution of 3 pounds per ton strength for a 48-hour period.

Results:

Feed: gold, 1·135 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
24	0·09	92·1	0·50	4·30

Flotation of Cyanide Tailings:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·09	100·0	
Flotation concentrate	7·55	0·76	64·1	13·2 : 1
Tailing	92·45	0·035	35·9	

Blanket Concentration of Flotation Tailing:

Feed	100·00	0·035	100·0	
Blanket concentrate	1·00	1·03	29·4	100 : 1
Tailing	99·00	0·025	70·6	

Cyanidation of Flotation and Blanket Concentrates:

Feed: gold, 0·79 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
48	0·58	25·6	9·0	8·8

Summary:

	Per cent
Gold extracted by straight cyanidation	92·1
Gold extracted by cyanidation of concentrates	1·5
Overall recovery	93·6

Test No. 2

The ore was ground to pass 69·9 per cent through 200 mesh and the pulp was agitated in cyanide solution at a strength of 1 pound of potassium cyanide per ton for a 24-hour period. Five pounds of lime per ton of ore was added. The cyanide tailing was concentrated on a Wilfley table and the table tailing was passed over a corduroy blanket. The combined table and blanket concentrates were reground to pass 99 per cent through 200 mesh and agitated in cyanide solution at a strength of 3 pounds per ton for a 48-hour period. A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65+100	1·6
−100+150	10·3
−150+200	18·2
−200	69·9
	100·0

Results:

Feed: gold, 1·135 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
24	0·11	90·3	0·55	4·50

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·11	100·0	
Table concentrate	5·6	0·53	27·1	17·9 : 1
Tailing	94·4	0·085	72·9	

Blanket Concentration of Table Tailing:

Feed	100·00	0·085	100·0	
Blanket concentrate	1·66	1·27	24·8	75 : 1
Tailing	98·34	0·065	75·2	

Cyanidation of Combined Concentrates:

Feed: gold, 0·67 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
48	0·335	50·00	5·8	7·6

Summary:

	Per cent
Gold extracted by straight cyanidation	90·30
Gold extracted by cyanidation of concentrates	2·15
Overall recovery	92·45

Test No. 3

In order to determine the effect of roasting the flotation concentrates from the cyanide tailing, prior to agitation, the following procedure was adopted:

The ore was ground to pass 80·5 per cent through a 200-mesh screen and the cyanide tailing floated in a manner similar to Test No. 1, 0·1 pound of Aerofloat No. 31 being added. Conditions otherwise were the same. The flotation concentrate was dried and roasted in a muffle furnace to a temperature of 600° C. The temperature was held at 300° C. until the greater portion of the arsenical fumes had passed. The calcine was reground and agitated in cyanide solution of 3 pounds per ton strength.

Results:

Feed: 1·135 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
24	0·11	90·3	0·75	4·60

Flotation of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·11	100·0	
Flotation concentrates	11·56	0·76*	80·0	8·7 : 1
Tailing	88·44	0·025	20·0	

*Calculated.

After the roasting of the flotation concentrate, this product assayed 0·71 ounce of gold per ton.

Cyanidation of Calcine:

Feed: gold, 0·71 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
48	0·18	74·6	24·21	14·06

Summary:

	Per cent
Gold extracted by straight cyanidation	90·3
Gold extracted by cyanidation of calcine	5·8
Overall recovery	96·1

Section D

SETTLING

Tests Nos. 1 and 2

These tests were carried out in a tall glass tube having an inside diameter of 2 inches. The ore was ground in cyanide solution to pass 80·5 per cent through 200 mesh, 5 pounds of lime per ton of ore being added. The pulp was transferred to the glass tube, and the level of solids in decimals of feet read every five minutes. Readings were made for a one-hour period. At the end of the tests the solutions were titrated for alkalinity.

Results:

—	Test No. 1	Test No. 2
Ratio of solid to liquid	2 : 1	1·5 : 1
Lime added per ton of ore	5·0 lb.	5·0 lb.
Alkalinity of solution at end of test	0·86	0·90
Overflow solution	Clear	Clear
Rate of settling	0·94 ft./hr.	0·51 ft./hr.

The pulp settles faster than normally.

SUMMARY AND CONCLUSIONS

Ninety-two per cent of the gold can be recovered by straight cyanidation of the ore at a grind of 80·0 per cent minus 200 mesh. A further recovery of 5·8 per cent can be obtained by flotation of the cyanide tailing, roasting of the flotation concentrate, regrinding, and final cyanidation.

The mill recommended, based on the results of the research work performed on this shipment of ore, is a cyanide mill in which jigs, traps, or blankets are used to remove any coarse gold from the grinding circuit. If after the mill is in operation a higher recovery is deemed economical, a flotation plant can be added and the cyanide tailing concentrated, the concentrate being roasted and cyanided or shipped directly to a smelter.

Ore Dressing and Metallurgical Investigation No. 720

GOLD ORE FROM THE HALLIWELL GOLD MINES, LIMITED, ROUYN, QUEBEC

Shipment. Twenty-two sacks of ore, total weight 1,386 pounds, were received on August 4, 1937, from John J. Collins, resident geologist, Halliwell Gold Mines, Limited, Rouyn, Quebec.

Location of Property. The property of the Halliwell Gold Mines, Limited, from which this shipment was received is in Beauchastel Township, Quebec, 1 mile from Arntfield, Quebec.

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

Feed Sample Assays:

Gold	1·46 oz./ton
Silver	1·02 "
Copper	0·11 per cent
Iron	2·20 "
Sulphur	0·26 "
Bismuth	1·09 "
Lead	Nil
Tellurium	Nil
Silica	70·52 "
Acid insoluble	86·63 "

Analysis of Acid Insoluble:

Silica	81·41 per cent
Alumina	12·05 "
Lime	0·35 "
Magnesia	Trace

Characteristics of the Ore. A microscopic examination was carried out on three special samples for determining the nature of the ore and predicting, as far as possible, the method of treatment likely to be necessary. Eighteen polished sections were prepared and examined.

The *gangue* is composed of grey to greenish grey, mottled, and fine-textured quartz. The mottling effect appears to be due to "ghosts" of chloritic material. A considerable quantity of carbonate is finely disseminated in the quartz.

Metallic minerals are so finely divided and form such a small proportion of the ore that in the hand specimen they are recognizable by the naked eye only with difficulty. In polished sections they are seen to be very finely disseminated chalcopyrite, an unknown mineral, pyrite, and native gold. Tests for the unknown mineral are given below:

Colour: Galena white; with chalcopyrite shows slightly grey cast.
Hardness: C to D, about the same as chalcopyrite.
Crossed nicols: Strongly anisotropic, with two extinctions and showing light and dark greys.
Etch tests: HNO_3—most grains slowly tarnish iridescent to brown without effervescence; some negative and others very slightly affected.
HCl, KCN, $FeCl_3$, KOH, $HgCl_2$—all negative on numerous checks.
Microchemical tests: S..........Essential constituent.
Cu......... " "
Bi......... " "
As, Sb, Se, Te, Fe, Ni, Co, Pb, Zn, Ag, Au.......Nil.

The grain analysis of the metallic minerals is shown in the following table. The figures are rough approximations, but indicate the fineness and degree of combination.

Grain Analysis:

Mesh	Mineral X		Chalcopyrite	
	Free, per cent	Combined with chalcopyrite, per cent	Free, per cent	Combined with mineral X, per cent
+200	10·0	....	22	
−200+280	10·0	....	17	
−280+400	15·0	2·5	19	3
−400+560	35·0	2·5	22	3
−560	22·5	2·5	11	3
Totals	92·5	7·5	91	9
	100·0		100·0	

Occurrence of the Gold. Only three grains of native gold are visible in the eighteen polished sections. The grains of gold range in size from about 800 mesh to 1600 mesh (between 9 and 19 microns). Two are in contact with small grains of chalcopyrite and the third is free in the quartz. Chalcopyrite, the unknown mineral, and gold are all associated with carbonate.

Conclusions from Microscopic Examination. The dense character of the ore was evidenced by the fine-textured quartz gangue. Grinding, therefore, was likely to be difficult, although it might be slightly modified by the presence of considerable disseminated carbonate.

The fineness of the metallic mineral particles, particularly of the gold, indicated that extremely fine grinding would be necessary to free them. Obviously, straight cyanidation was indicated as the best possibility.

On the microscopic evidence it seemed probable that this was a very difficult ore to treat.

Experimental Tests

The research procedure consisted of cyanidation, and concentration of the cyanide tailing by means of flotation, tabling, and blanketing. The ore proved to be very refractory, a cyanide extraction of not over 80 per cent of the gold being obtained when the ore was ground as fine as economic procedure would dictate. Flotation, tabling and blanket concentrations were unsuccessful in segregating the gold. Of the bismuth contained in the ore, some 60 per cent was concentrated by flotation.

Section A—Cyanidation

The ore at minus 14 mesh was ground to different degrees of fineness in ball mills and agitated 24 to 48 hours in cyanide solutions of different strengths. Five pounds of lime per ton of ore was added to maintain protective alkalinity in all cases.

In Test No. 1 the ore was ground to pass 52·6 per cent through a 200-mesh screen and agitated in cyanide solution for 24 and 48 hours.

In Test No. 2 the ore was ground to pass 74·6 per cent.

In Test No. 3 the ore was ground to pass 92·7 per cent through 200 mesh.

In Test No. 4 the grind was 92·7 per cent through 200 mesh, 2 pounds of potassium cyanide per ton of solution being added.

In Test No. 5 the grind was 92·7 per cent through 200 mesh, and 1 pound of potassium cyanide and 1 pound of litharge per ton of ore were added.

In Test No. 6, one pound of lead nitrate replaced the litharge.

In Test No. 7, the ore was ground to pass 97·5 per cent through 200 mesh and 3 pounds of potassium cyanide per ton of solution was added prior to agitation.

In Test No. 8 the ore was ground in cyanide solution to pass 92·7 per cent through 200 mesh.

In Test No. 9 the ore was ground in a lime pulp to pass 97·5 per cent through 200 mesh and the pulp was aerated for 16 hours prior to agitation in cyanide solution of 2 pounds per ton strength.

A screen test showed the grindings as follows:

Screen Test:

Mesh	Weight, per cent			
	No. 1	No. 2	Nos. 3, 4, 5, 6, and 8	Nos. 7 and 9
− 35+ 48	0·1	....	....	....
− 48+ 65	1·4	....	....	....
− 65+100	9·9	1·1	....	....
−100+150	18·7	7·3	1·0	....
−150+200	17·0	16·9	6·3	2·4
−200	52·6	74·6	92·7	97·5

Cyanidation Results:

Test No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents added, lb./ton		Reagents consumed, lb./ton		Remarks
				KCN	CaO	KCN	CaO	
1.........	24	1·01	30·8	1	5	0·43	3·00	
1.........	48	0·93	36·3	1	5	0·43	3·00	
2.........	24	0·95	34·9	1	5	0·43	3·50	
2.........	48	0·74	49·3	1	5	0·63	3·60	
3.........	24	0·575	60·6	1	5	0·43	3·40	
3.........	48	0·555	61·9	1	5	0·63	3·60	
4.........	24	0·30	79·5	2	5	0·68	3·60	
4.........	48	0·32	78·1	2	5	0·68	3·90	
5.........	24	0·70	52·1	1	5	0·69	3·50	1 lb./ton PbO added
6.........	24	0·84	42·5	1	5	0·61	3·70	1 lb./ton $PbNO_3$ added.
7.........	24	0·355	75·7	3	4	1·80	2·80	
8.........	24	1·01	30·8	1	5	0·63	3·50	Cyanide grind.
9.........	24	0·51	65·1	2	5	0·47	4·20	Aerated 16 hours.

As a final test, in order to free as much as possible of the gold, the ore was ground to pass 99·0 per cent through a 325-mesh screen and the pulp agitated in cyanide solution of 2 pounds per ton strength for 24 hours. Six pounds of lime per ton of ore was added. The cyanide tailing was treated with hot aqua regia solution, washed, filtered, and assayed for gold.

Results of Cyanidation:

Feed: gold, 1·46 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
24	0·22	85·0	1·26	4·50

After the treatment of the cyanide tailing with aqua regia the tailing was assayed and showed 0·09 ounce of gold per ton.

In order to establish any relationship between the gold and bismuth contents of the ore, infrasizing tests were made on the feed sample and on the cyanide tailing. A sample of the ore was accordingly crushed to pass 97·5 per cent through a 200-mesh screen. A further sample was crushed to pass 92·0 per cent minus 200 mesh. This sample was agitated in cyanide solution of 2 pounds of potassium cyanide per ton strength for 24 hours. The cyanide tailing was washed, dried, and assayed. The feed sample assayed 1·47 ounces of gold per ton and the cyanide tailing 0·33 ounce of gold per ton. The two samples were run through the infrasizer and the resulting product weighed and assayed.

Results of Infrasizer Test:

Feed: gold, 1·47 oz./ton; bismuth, 1·09 per cent

Microns	Assay		Weight, per cent
	Au, oz./ton	Bi, per cent	
+56	1·485	1·19	0·72
+40	1·480	1·16	4·03
+28	1·29	1·14	11·26
+20	1·30	1·08	14·10
+14	1·345	1·00	13·52
+10	1·47	1·12	11·31
−10	1·565	0·80	45·06

Cyanide Tailing:

Feed: gold, 0·33 oz./ton; bismuth, 1·08 per cent

Microns	Assay		Weight, per cent
	Au, oz./ton	Bi, per cent	
+56	0·655	0·89	5·36
+40	0·57	0·79	10·10
+28	0·51	1·00	15·30
+20	0·445	1·12	13·36
+14	0·365	1·19	11·50
+10	0·30	1·09	9·12
−10	0·12	1·00	34·86

From the results of the above test no apparent relationship between the gold and bismuth contents of the ore can be discerned. The assays of the cyanide tailings show that the bulk of the gold remaining in the pulp after cyanidation is in the coarser-sized products.

Section B—Concentration

The tests comprised flotation, table and blanket concentration of the ore prior to cyanidation and also of the cyanide tailing.

FLOTATION

Test No. 1

The ore at minus 14 mesh was ground in a ball mill with 4 pounds of soda ash, 0·20 pound of Barrett No. 4 oil, and 0·1 pound of amyl xanthate per ton to pass 92·5 per cent through 200 mesh. The pulp was transferred to a flotation machine and 0·05 pound of pine oil per ton was added. A flotation concentrate was removed.

Results:

Product	Weight, per cent	Assay Au, oz./ton	Assay Bi, per cent	Distribution, per cent Au	Distribution, per cent Bi	Ratio of concentration
Feed	100·0	1·46	1·10	100·0	100·0	8·6 : 1
Flotation concentrate	11·6	6·84	5·86	54·3	60·2	
Tailing	88·4	0·755	0·47	45·7	39·8	

FLOTATION AND CYANIDATION

Test No. 2

The ore was ground in neutral solution to pass 97·0 per cent minus 200 mesh. The pulp was floated with 0·07 pound of cresylic acid and 0·05 pound of butyl xanthate per ton. A flotation concentrate was removed. The flotation tailing was washed and agitated in cyanide solution of 2 pounds of potassium cyanide per ton strength for 24 hours.

Flotation:

Product	Weight, per cent	Assay Au, oz./ton	Assay Bi, per cent	Distribution, per cent Au	Distribution, per cent Bi	Ratio of concentration
Feed	100·00	1·44*	1·00*	100·0	100·0	30·6 : 1
Flotation concentrate	3·26	16·82	13·47	38·1	43·9	
Tailing	96·74	0·92	0·58	61·9	56·1	

* Calculated.

Cyanidation of Flotation Tailing:

Agitation, hours	Assay, Au, oz./ton Feed	Assay, Au, oz./ton Tailing	Extraction, per cent	Reagents consumed, lb./ton KCN	Reagents consumed, lb./ton CaO
24	0·92	0·25	72·8	0·55	4·94

In the above test 38·1 per cent of the gold was recovered in the flotation concentrate and 45·1 per cent extracted by cyanidation of the flotation tailing.

A number of other flotation tests were conducted on the feed sample and also on the cyanide tailing. Caustic soda and lime replaced the soda ash, and different accelerators and collective agents were tried. The results did not differ greatly from Tests Nos. 1 and 2, showing that flotation is not successful in concentrating the gold.

CYANIDATION—TABLE AND BLANKET CONCENTRATION

Test No. 3

The ore at minus 14 mesh was ground to pass 95·2 per cent through a 200-mesh screen and the pulp agitated in cyanide solution at a strength of 2 pounds per ton for 24 hours. The cyanide tailing was washed and assayed for gold. A Wilfley table concentrate was made of this product and the table tailing passed over a corduroy blanket. The different products were assayed for gold and bismuth.

Cyanidation:

Feed: gold, 1·46 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
24	0·33	78·4	0·75	3·0

Table Concentration:

Feed: gold, 0·33 oz./ton; bismuth, 0·91 per cent

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Au, oz./ton	Bi, per cent	Au	Bi	
Feed	100·0	0·32*	0·92*	100·0	100·0	
Table concentrate	8·6	0·67	1·23	18·1	11·5	11·6 : 1
Tailing	91·4	0·285	0·89	81·9	88·5	

* Calculated.

Blanket Concentration:

Feed: gold, 0·29 oz/ton; bismuth, 0·89 per cent

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Au, oz./ton	Bi, per cent	Au	Bi	
Feed	100·0	0·29*	0·95*	100·0	100·0	
Blanket concentrate	1·7	0·755	3·13	4·4	5·6	59 : 1
Final tailing	98·3	0·28	0·91	95·6	94·4	

* Calculated.

Summary:

—	Gold, per cent	Bismuth, per cent
Extracted by cyanidation	78·4	
Recovered in table concentrate	3·9	11·5
Recovered in blanket concentrate	0·8	5·0

Test No. 4

The ore was agitated in cyanide solution in the same way as in Test No. 3. The cyanide tailing was passed over a corduroy blanket and a blanket concentrate recovered.

The cyanide tailing assayed 0·33 ounce of gold per ton. The blanket concentration of this product resulted as follows:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Au, oz./ton	Bi, per cent	Au	Bi	
Feed	100·0	0·33	0·97*	100·0	100·0	
Blanket concentrate	4·0	0·80	3·16	9·7	13·0	25 : 1
Tailing	96·0	0·30	0·88	90·3	87·0	

* Calculated.

Summary:

—	Gold, per cent	Bismuth, per cent
Extracted by cyanidation	78·4	
Recovered in blanket concentrate	2·1	13·0

Test No. 5

Table and blanket concentration preceded cyanidation of the blanket tailing. The ore at minus 14 mesh was ground in a ball mill to pass 52·2 per cent through 200 mesh and the pulp was passed over a Wilfley table. The table tailing was then passed over a corduroy blanket. The blanket tailing was then reground to pass 87·4 per cent minus 200 mesh and agitated for 24 and 48 hours in cyanide solution.

Table Concentration:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of distribution
		Au, oz./ton	Bi, per cent	Au	Bi	
Feed	100·00	1·46	0·75*	100·0	100·0	
Table concentrate	5·02	3·92	2·59	13·5	17·4	19·9 : 1
Tailing	94·98	1·33	0·65	86·5	82·6	

Blanket Concentration:

Feed	100·00	1·33	0·65	100·0	100·0	
Blanket concentrate	2·32	3·71	2·46	6·0	8·8	43 : 1
Tailing	97·68	1·28	0·61	94·0	91·2	

* Calculated.

Cyanidation:

Feed: gold, 1·28 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
24	0·32	75·0	0·3	2·3
48	0·315	75·4	0·3	2·9

Summary:

—	Gold, per cent	Bismuth, per cent
Recovered in table concentrate	13·5	17·4
Recovered in blanket concentrate	5·2	7·3

Extracted by cyanidation (24 hours)—61·0 per cent
(48 hours)—61·3 "

By this method the bismuth in the ore slimed easily and was not recovered to any extent on the table or blanket.

Section C—Amalgamation

Tests No. 1 and 2

The ore at minus 14 mesh was ground in a ball mill to pass 44·8 per cent through a 200-mesh screen in Test No. 1 and 65·4 per cent in Test No. 2. The pulps were amalgamated with mercury in a jar mill and the amalgam tailings assayed for gold.

Results:

Test No.	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
1	1·46	1·37	6·2
2	1·46	1·37	6·2

These tests show that there is no appreciable amount of free gold in the ore.

Section D—Settling

Tests Nos. 1 and 2

These were carried out in a tall glass tube having an inside diameter of 2 inches. The ore was ground in cyanide solution to 90·5 per cent through a 200-mesh screen, 4 pounds of lime per ton of ore being added. The pulp was transferred to the glass tube and the level of solids in decimals of feet read every 5 minutes. Readings were made for a 1-hour period. At the end of the tests the solutions were titrated for alkalinity.

Results:

—	Test No. 1	Test No. 2
Ratio of solid to liquid	1·5 : 1	2 : 1
Lime added per ton of ore	4·0 pounds	4·0 pounds
Alkalinity of solution at end of test	0·66 lb./ton	0·62 lb./ton
Overflow solution	Slightly cloudy	Slightly cloudy
Rate of settling	0·32 ft./hr.	0·58 ft./hr.

Considering the fineness of the grind the rate of settling is only slightly slower than normal.

SUMMARY

To obtain the maximum economic extraction of approximately 80 per cent of the gold, the ore was ground in water to pass 95 per cent through a 200-mesh screen and agitated in cyanide solution at a strength of 2 pounds of potassium cyanide per ton for 24 hours. Grinding 99 per cent through 325 mesh with 24-hour agitation gave an extraction of 85 per cent of the gold. The remaining 15 per cent was treated with aqua regia and showed an aqua regia tailing of 6·2 per cent of the gold.

Concentration methods proved ineffective for the recovery of the gold, both before and after cyanidation. Flotation concentration proved successful in recovering 60 per cent of the bismuth.

Ore Dressing and Metallurgical Investigation No. 721

GOLD ORE FROM THE DARWIN GOLD MINES, LIMITED, GOLD PARK, ONTARIO

Shipment. A shipment of gold ore, weight 1,040 pounds, was received on August 17, 1937, from the Darwin Gold Mines, Limited, Gold Park, via Sault Ste. Marie, Ontario. The sample was stated to be representative of the present ore and consisted roughly of 80 per cent from the new vein and 20 per cent from the Grace vein. The shipment was submitted by Mr. M. H. Frohberg, Mine Manager.

Characteristics of the Ore. Six polished sections were prepared and examined microscopically and show that the gangue consists largely of transparent to translucent vein quartz with a green chloritic rock that contains considerable finely disseminated carbonate.

The metallic minerals seen in the sections are as follows: pyrite, arsenopyrite, pyrrhotite, chalcopyrite, sphalerite, and native gold. The first two occur in major amount.

Pyrite occurs as coarse to fine, irregular grains disseminated in the gangue; locally it contains numerous inclusions of gangue, and is somewhat fractured with chalcopyrite in the fractures. Arsenopyrite occurs as medium to fine, disseminated crystals, occasionally associated with pyrite. Pyrrhotite is disseminated as medium to fine grains, and is commonly associated with chalcopyrite; the mineral also occurs in very small quantities as tiny inclusions in pyrite and sphalerite. Sphalerite is extremely rare as small, irregular grains associated with pyrite; as stated above, it contains tiny dots of pyrrhotite.

Five grains of native gold are visible in the sections. All are associated with pyrite, four occurring along the borders of this mineral with gangue and one occurring in dense pyrite. From evidence obtained on the specimens examined about 45 per cent of the gold observed occurs within pyrite, and 55 per cent is found along grain boundaries of pyrite. The grain size varies between 400 and 1100 mesh (Tyler).

Sampling and Assaying. The ore was crushed, mixed, and sampled by standard methods. The assay results are as follows:

Gold	1·215 oz./ton
Silver	0·06 "
Arsenic	0·03 per cent

A determination for pyrrhotite indicated 0·10 per cent of this mineral present in the ore.

Purposes of Investigation. In 1935 and 1936 investigations were carried out on ore from the Grace vein of the Darwin Gold Mines, Limited, the results of which are covered in Reports Nos. 637 and 668, Investigations in Ore Dressing and Metallurgy, Mines Branch Publications Nos. 771 and 774, respectively.

The present shipment was submitted for the investigation of the possibilities of improved extraction by table concentration of the sulphides in the cyanide tailing, followed by their regrinding and re-cyanidation in a separate circuit.

Results. The results of the above method showed an improvement in extraction.

The tests follow in detail:

Experimental Tests

Test No. 1

A sample of ore was ground in water to have 69 per cent minus 200 mesh. The pulp was passed over a hydraulic classifier to remove free gold. The overflow was filtered and repulped before feeding to a laboratory Wilfley table. A sulphide concentrate representing 4·69 per cent of the weight of the feed was recovered and reground to have 93 per cent minus 325 mesh. This was cyanided for 48 hours. Samples of the table tailing were cyanided separately for 24 and 48 hours respectively. Although the results showed no improvement over direct cyanidation of the ore, the fact that 98·5 per cent of the gold in the table concentrate was extracted indicates that fine grinding will render the gold in the sulphides amenable to cyanidation.

The results of the test follow:

HYDRAULIC CLASSIFICATION

Gold in feed	1·215 oz./ton
Gold in overflow	0·54 "
Free gold recovered in classifier underflow	55·56 per cent

Table Concentration of Classifier Overflow:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·56	100·00	
Concentrate	4·69	6·80	56·27	21·32 : 1
Tailing	95·31	0·26	43·73	

Cyanidation of Table Products:

Product	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Ratio of concentration
		Feed	Tailing		KCN	CaO	
Table concentrate	48	6·80	0·105	98·45	0·66	3·05	3 : 1
Table tailing	24	0·26	0·035	86·54	0·24	3·65	1·5 : 1
Table tailing	48	0·26	0·045	82·69	0·375	3·92	1·5 : 1

Summary:

	Per cent
Gold recovered in hydraulic classifier	55·56
Gold extraction by cyanidation of table concentrate	24·62
Gold extraction by cyanidation of table tailing	16·81
Overall recovery	96·99

Test No. 2

In this and the four tests following, the ore samples were ground in cyanide to approximately 70 per cent minus 200 mesh and the pulp was transferred to bottles for agitation in cyanide solution. The cyanide tailing was filtered and washed and after repulping was fed to a laboratory Wilfley table. A rough sulphide concentrate was taken off. This concentrate was reground in water and agitated in cyanide for 48 hours, the solution strength being equivalent to 3 pounds of potassium cyanide per ton.

The results of the test are as follows:

Grinding of Ore in Cyanide:

Pulp dilution	0·75 : 1
Strength of solution	2 lb. KCN/ton
Consumption of cyanide	0·75 lb. KCN/ton of ore
Lime added	5 lb./ton

Cyanidation—24 Hours:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing	1·215	0·04	96·7	1·31	3·63	1·5 : 1

The percentage extraction includes gold dissolved during grinding. The consumptions of potassium cyanide and lime shown in the table represent the reagents consumed during both grinding and agitation.

Tabling of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·06*	100·00	
Concentrate	13·24	0·36	68·73	7·55 : 1
Tailing	86·76	0·025	31·27	

* Calculated.

CYANIDATION OF TABLE CONCENTRATE

Lead salt in the form of lead nitrate was added to the pulp (1 pound $PbNO_3$ per ton of ore).

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton of conc.		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing	0·36	0·06	83·33	0·30	1·25	3 : 1

A screen test of the reground sulphide concentrate indicated a grinding as follows:

Mesh	Weight, per cent
+325	44·0
−325	56·0
	100·0

Summary:

	Per cent
Gold extraction, 24 hours' agitation	96·70
Additional gold extraction from sulphides (table concentrate)	1·89
Overall recovery	98·59

Test No. 3

This test was a duplicate of Test No. 2.

Cyanidation—24 Hours:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing	1·215	0·04	96·7	1·20	3·62	1·5 : 1

Extraction and reagent consumption shown in the table are total for both grinding and agitation periods.

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution per cent	Ratio of concentration
Feed	100·00	0·04	100·00	
Concentrate	19·79	0·14	58·01	5·05 : 1
Table tailing	80·21	0·025	41·99	

Cyanidation of Table Concentrate:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing..................	0·14	0·055	60·71	0·3	1·10	3 : 1

Summary:

	Per cent
Gold extraction, 24 hours' agitation....................................	96·70
Additional gold extraction from sulphides (table concentrate)............	1·16
Overall recovery..	97·86

Test No. 4

In this test the ore was agitated for a period of 48 hours after grinding in cyanide. In other respects the same procedure was followed as in Tests Nos. 2 and 3.

Cyanidation—24 Hours:

Lime added for protective alkalinity, 2 pounds per ton in grinding circuit.

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing..................	1·215	0·03	97·53	1·14	1·64	1·5 : 1

Extraction and reagent consumption shown in the table are total for both grinding and agitation periods.

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed..	100·00	0·03	100·00	
Concentrate.................................	16·99	0·12	55·12	5·89 : 1
Table tailing................................	83·01	0·02	44·88	

Cyanidation of Table Concentrate:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing..................	0·12	0·05	58·33	0·30	1·25	3 : 1

Screen tests on the table concentrate showed the results of regrinding to be as follows:

Mesh	Weight, per cent	Mesh	Weight, per cent
+200	15·0	+325	27·0
−200	85·0	−325	73·0
	100·0		100·0

Summary:

	Per cent
Gold extraction, 48 hours' agitation	97·53
Additional gold extraction from sulphides (table concentrate)	0·79
Overall recovery	98·32

Test No. 5

This test is a duplicate of Test No. 4.

Cyanidation—48 Hours:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing	1·215	0·03	97·53	1·07	1·70	1·5 : 1

Extraction and reagent consumption are total of both grinding and agitation.

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·03	100·00	
Concentrate	11·81	0·11	42·41	8·47 : 1
Table tailing	88·19	0·02	57·59	

Cyanidation of Table Concentrate:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing	0·11	0·06	45·45	0·30	1·25	3 : 1

A screen test indicated the primary grinding to be as follows:

Mesh	Weight, per cent
+ 65	0·2
− 65 +100	1·2
−100 +150	13·2
−150 +200	18·2
−200	67·2
	100·0

The regrinding of the sulphides (table concentrate) was to approximately the same fineness as in Test No. 4.

Summary:

	Per cent
Gold extraction, 48 hours' agitation	97·53
Additional extraction from sulphides (table concentrate)	0·48
Overall recovery	98·01

Test No. 6

This test was run with the object of making a higher ratio of concentration on the table. The general method of the test was the same as in Test No. 5. The primary grind was a little finer, as indicated by the screen test shown below:

Mesh	Weight, per cent
+150	10·3
−150 +200	16·9
−200	72·8
	100·0

The pulp dilution during grinding was 0·75 : 1 and the cyanide strength equivalent to 2 pounds of potassium cyanide per ton. The cyanide consumed during grinding was 0·90 pound of potassium cyanide per ton of ore. Two pounds of lime per ton was added to the grinding and no further additions were necessary for the agitation.

The results of 48 hours' cyanidation are as follows:

Cyanidation—48 Hours:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing	1·215	0·03	97·53	0·99	1·67	1·5 : 1

The results are those of grinding and agitation.

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·03	100·00	
Table concentrate	6·16	0·29	48·77	16·23 : 1
Table tailing	93·84	0·02	51·23	

CYANIDATION OF TABLE CONCENTRATE

The table concentrate was reground in water to 84 per cent minus 325 mesh and cyanided for 48 hours in solutions of strength equivalent to 3 pounds of potassium cyanide per ton. One pound of lead nitrate ($PbNO_3$) per ton was added as a precaution against undue cyanide consumption, and an addition of 2 pounds of lime per ton was made to provide protective alkalinity.

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing	0·29	0·07	75·86	0·66	1·18	3·3 : 1

Summary:

	Per cent
Gold extraction, 48 hours' agitation	97·53
Additional extraction from sulphides (table concentrate)	0·91
Overall recovery	98·44

CONCLUSIONS

The results obtained from investigating the introduction of table concentration in treating this ore indicate that an improved extraction can be effected.

The research shows that about 50 per cent of the gold remaining in the tailing after cyanide treatment is associated with the sulphides, and fine grinding is necessary to expose this gold to the action of cyanide. A good ratio of concentration was effected on the table, thus rendering the bulk of material for regrinding and recyanidation in a separate circuit reasonably small.

From the laboratory tests, and calculating on the basis of a 50-ton mill feed, the tables would produce about 3 tons of concentrate for re-treatment per day. It is probable that on standard size tables a concentrate would be made cleaner and of less bulk than that produced on the small laboratory tables.

Ore Dressing and Metallurgical Investigation No. 722

GOLD ORE FROM THE GRANGE CONSOLIDATED MINES, LIMITED, AT KELLY CREEK, NEAR PAVILION, BRITISH COLUMBIA

Shipment. The shipment, consisting of one bag of ore, net weight 114 pounds, was received on July 27, 1937. The ore was submitted by John Bennett, Secretary, Grange Consolidated Mines, Limited, 612 Standard Bank Building, Vancouver, British Columbia.

The ore was taken from the property of the Grange Consolidated Mines, Limited, situated on the east bank of the Fraser River, near the foot of Pavilion Mountain, in the Lillooet mining district of British Columbia. The sample was said to be representative of all ore intersected in the sixth and seventh levels, and to be actually the surplus ore from some fifty channel samples.

The property was formerly known as the Big Slide mine, of the Grange Mines, Limited, at Kelly Creek, near Pavilion, B.C., and a sample of this ore was investigated in 1934.

Characteristics of the Ore. Six polished sections were prepared containing 31 pieces of the crushed ore.

The *gangue* is fine-textured white quartz with some carbonate.

The *metallic minerals* present in the sections are as follows:

Pyrite	Abundant
Pyrrhotite	Abundant
Arsenopyrite	Moderately abundant
Chalcopyrite	Small quantity
Marcasite	Small quantity
"Limonite"	Small quantity
Native gold	

Pyrite and pyrrhotite occur as coarse masses and grains, associated in places with arsenopyrite. Both pyrite and arsenopyrite are considerably fractured and veined by chalcopyrite. The latter mineral, in addition to veining arsenopyrite and pyrite, occurs as coarse to fine grains in the gangue. Locally, small masses of marcasite appear to have resulted from the alteration of the pyrite. "Limonite" occurs as patches and stains, but is present in only a few of the pieces examined.

Native gold is visible only in dense pyrite. It occurs as irregular to rounded particles, which are finely divided:

Grain Size of the Visible Gold:

Mesh	Approximate per cent
+1100	27
−1100+1600	22
−1600+2300	25
−2300	26*
Total	100

*Some of the grains in this size measure as small as 1 micron.

Purpose of the Investigation. The shipment was sent for determining the response of the ore to straight cyanidation, as the company proposes to erect a cyanide plant to avoid the costly transportation of concentrate.

Sampling and Analysis. The ore was crushed and sampled by standard methods and was found to contain the following values:

Gold	0·605 oz./ton
Silver	2·085 "
Copper	0·30 per cent
Zinc	Nil
Arsenic	0·30 per cent

Experimental Tests

The work undertaken consisted of straight cyanidation, concentration of copper minerals by flotation prior to cyanidation, aeration of flotation tailing prior to cyanidation, blanket concentration prior to cyanidation, selective separation of pyrite and regrinding and cyaniding it, and cyanidation of the pyrite flotation tailing.

STRAIGHT CYANIDATION

Tests Nos. 1 and 2

Procedure. Two samples of minus 14-mesh ore were ground in ball mills at a dilution of 4 parts of ore to 3 parts of water (dilution 4 : 3).

The ground ore was diluted to 2 parts of water to 1 part of ore (dilution 2 : 1) and cyanide was added at the rate of 1·0 pound of potassium cyanide per ton of solution. Lime was added to give protective alkalinity to the solution. The pulps were agitated for 24 hours, with additions of reagents as required. The cyanide tailings were assayed for gold and silver. A screen test was made on each to determine the degree of grinding.

Results:

Test No.	Assay, oz./ton				Extraction, per cent		Reagents consumed, lb./ton	
	Feed		Tailing					
	Au	Ag	Au	Ag	Au	Ag	KCN	CaO
1	0·605	2·085	0·33	0·88	45·45	57·80	3·06	9·36
2	0·605	2·085	0·255	0·71	57·85	65·95	3·54	9·42

Screen Test:

Mesh	Weight, per cent	
	Test No. 1	Test No. 2
+ 65	0·3	....
+100	4·7	1·0
+150	17·8	8·4
+200	15·2	19·5
−200	62·0	71·1
	100·0	100·0

The extraction by straight cyanidation is low, accompanied by a heavy consumption of reagents.

FLOTATION FOLLOWED BY CYANIDATION

Test No. 3

The high consumption of cyanide was attributed to copper in the ore. Therefore, this test was made to determine the extraction by cyanidation when the copper-bearing minerals had been removed by flotation, and the grade of copper concentrate that could be made.

A sample of minus 14-mesh ore was ground at a dilution of 4 : 3 to give a product 75 per cent minus 200 mesh. Lime was added to the ball mill at the rate of 5·0 pounds per ton of ore to depress pyrite.

The pulp was transferred to a flotation cell and conditioned for 3 minutes with 0·03 pound of butyl xanthate per ton of ore. Pine oil at the rate of 0·05 pound per ton was added and a concentrate was removed. The concentrate was cleaned. The tailing from the cleaner cell was added to the first tailing and filtered.

Samples were cut from the filter cake, repulped in a 1·0 pound per ton cyanide solution at a dilution of 2 : 1, and agitated for 24 and 48 hours.

Results:

Flotation:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Au, oz./ton	Cu, per cent	Au	Cu	
Feed	100·00	0·60	0·253	100·00	100·00	
Flotation concentrate	2·80	15·36	7·65	71·65	84·63	36 : 1
Flotation tailing	97·20	0·175	0·04	28·35	15·37	

Cyanidation of Flotation Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO
3A	24	0·175	0·035	80·00	1·21	5·20
3B	48	0·175	0·025	85·71	1·79	7·20

Summary:

—	24 hours, per cent	48 hours, per cent
Recovery of gold by flotation	71·65	71·65
Extraction by cyanidation in 24 hours, 80 × 28·35	22·68	....
Extraction by cyanidation in 48 hours, 85·71 × 28·35	....	24·30
Totals	94·33	95·95

FLOTATION FOLLOWED BY CYANIDATION

Test No. 4

The object of the test was to increase the grade of the copper concentrate by finer grinding of the ore.

A sample of ore was ground with lime, 4·0 pounds per ton, to 79 per cent minus 200 mesh.

The pulp was conditioned with 0·03 pound of butyl xanthate per ton of ore in a flotation cell for 3 minutes. Pine oil was added at the rate of 0·05 pound per ton and a concentrate was removed. The concentrate was recleaned. The tailing from the cleaner cell was added to the first tailing and filtered.

Samples were cut from the filter cake, repulped 1 : 2 in a 1·0 pound per ton cyanide solution, and agitated for 24 and 48 hours.

Results:

Flotation:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Au, oz./ton	Cu, per cent	Au	Cu	
Feed	100·00	0·60	0·31	100·00	100·00	
Flotation concentrate	1·90	21·25	10·40	67·30	64·68	53 : 1
Flotation tailing	98·10	0·20	0·11	32·70	35·32	

Cyanidation of Flotation Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO
4A	24	0·20	0·075	62·50	1·99	6·58
4B	48	0·20	0·05	75·00	1·96	9·52

Summary:

—	24 hours, per cent	48 hours, per cent
Recovery of gold by flotation	67·3	67·3
Extraction by cyanidation in 24 hours, 62·5 × 32·7	20·4	....
Extraction by cyanidation in 48 hours, 75·0 × 32·7	....	24·5
Totals	87·7	91·8

FLOTATION FOLLOWED BY CYANIDATION

Test No. 5

In this test, still finer grinding was used.

A sample of ore was ground with lime, 5·0 pounds per ton of ore, to 89 per cent minus 200 mesh.

The pulp was conditioned with 0·015 pound of butyl xanthate per ton for 3 minutes in a flotation cell, 0·05 pound of pine oil per ton was added, and a concentrate was removed. A second addition of 0·015 pound of butyl xanthate per ton was made and flotation was continued. The concentrate was cleaned and the tailing was filtered.

Samples of the tailing were cyanided for 24 and 48 hours at a dilution of 2 : 1 in a solution containing 1·0 pound of potassium cyanide per ton, as in the preceding tests.

Results:

Flotation:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Au, oz./ton	Cu, per cent	Au	Cu	
Feed	100·00	0·58	0·28	100·00	100·00	
Flotation concentrate	1·35	30·76	14·30	71·84	68·50	74 : 1
Flotation tailing	98·65	0·165	0·09	28·16	31·50	

Cyanidation of Flotation Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
5A	24	0·165	0·06	63·64	1·49	4·35
5B	48	0·165	0·03	81·82	2·37	6·25

Summary:

—	24 hours, per cent	48 hours, per cent
Recovery of gold by flotation	71·84	71·84
Extraction by cyanidation in 24 hours, 63·64 × 28·16	17·92	....
Extraction by cyanidation in 48 hours, 81·82 × 28·16	....	23·04
Totals	89·76	94·88

FLOTATION FOLLOWED BY AERATION OF FLOTATION TAILING

Test No. 6

In this test the effect of finer grinding on the recovery of copper was studied, also the cyanidation of the flotation tailing for different periods of time.

A sample of ore was ground with lime, 5 pounds per ton of ore, to 91 per cent minus 200 mesh.

The pulp was conditioned with 0·02 pound of butyl xanthate per ton for 3 minutes, 0·05 pound of pine oil per ton was added, and a concentrate was removed and cleaned with lime.

The cleaner tailing and flotation tailing were filtered and repulped at a dilution of 2·5 : 1 in a Wallace super-agitator and agitated for 20 hours with 9·92 pounds of lime per ton.

The pulp was filtered and washed. Samples from the cake were repulped in a solution containing 1·0 pound of potassium cyanide per ton, at a dilution of 2 : 1, and agitated for periods of 8, 16, 24, 30, 48, and 72 hours.

Results:

Flotation:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton		Per cent				
		Au	Ag	Cu	Au	Ag	Cu	
Feed	100·00	0·605	2·085	0·24	100·00	100·00	100·00	
Flotation concentrate	1·17	37·77	135·13	17·92	73·04	75·83	87·63	85·5 : 1
Flotation tailing	98·83	0·165	0·51	0·03	26·96	24·17	12·37	

Cyanidation of Flotation Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
6A	8	0·165	0·055	66·67	1·40	2·68
6B	16	0·165	0·025	84·85	1·05	2·69
6C	24	0·165	0·025	84·85	1·41	2·48
6D	30	0·165	0·020	87·88	1·32	2·60
6E	48	0·165	0·015	90·91	1·65	2·69
6F	72	0·165	0·025	84·85	2·58	4·89

The test shows that a maximum of extraction was obtained in 48 hours' agitation. (Note higher tailing in Test No. 6F, 72-hour agitation.)

Summary:

(48-hour agitation)		Per cent
Recovery of gold in copper concentrate		73·04
Percentage of gold left in cyanide feed	26·96 per cent	
Extraction of gold from cyanidation Test No. 6E, 26·96 × 90·91		24·51
Total gold recovered		97·55

The following table shows the total recoveries for different periods of agitation:

Test No.	Gold in Cu concentrate, per cent	Gold extracted by cyanidation, per cent	Total recovery of gold, in per cent of feed
6A	73·04	17·97	91·01
6B	73·04	22·88	95·92
6C	73·04	22·88	95·92
6D	73·04	23·69	96·73
6E	73·04	24·51	97·55
6F	73·04	22·88	95·92

Extraction of Silver by Cyanidation of Flotation Tailing in Test No. 6E:

Feed to cyanidation.....Ag		0·51 oz./ton
Cyanide tailing.....Ag		0·05 "
Extraction	90·20 per cent	
Recovery of silver in the copper concentrate		75·83 per cent
Extraction by cyanidation, 90·2 × 24·17		21·80 "
Total silver recovered		97·63 "

FLOTATION FOLLOWED BY CYANIDATION

Test No. 7

In this test the flotation middling was not added to the flotation tailing.

A sample of ore was ground with lime, 5·0 pounds per ton of ore, to 80 per cent minus 200 mesh.

The pulp was floated with 0·03 pound of butyl xanthate per ton in additions of 0·015 pound per ton at a time, and pine oil at the rate of 0·05 pound per ton. The concentrate was cleaned. The cleaner tailing was filtered and assayed for copper and gold (designated as middling in the table). The flotation tailing was filtered and samples of the tailing were cyanided for 24 and 48 hours at a dilution of 2 : 1 in a solution containing 1·0 pound of potassium cyanide per ton.

Results:

Flotation:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton		Cu, per cent				
		Au	Ag		Au	Ag	Cu	
Feed	100·00	0·57	2·085	0·30	100·00	100·00	100·00	
Flotation concentrate	2·27	19·26	72·08	10·03	76·45	78·47	83·84	44 : 1
Flotation middling	2·72	1·98	6·37	1·09	9·42	8·31	10·90	37 : 1
Flotation tailing	95·01	0·085	0·29	0·015	14·13	13·22	5·26	

Cyanidation:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
7A	24	0·085	0·03	64·71	0·44	4·35
7B	48	0·085	0·025	70·59	0·81	4·35

Summary:

—	24 hours, per cent	48 hours, per cent
Recovery of gold by flotation	76·45	76·45
Extraction by cyanidation in 24 hours, 14·13 × 64·71	9·14	....
Extraction by cyanidation in 48 hours, 14·13 × 70·59	....	9·97
Totals	85·59	86·42

It is apparent that the introduction of flotation middling into the cyanide feed is responsible for an appreciable consumption of cyanide.

FLOTATION AND BLANKET CONCENTRATION FOLLOWED BY CYANIDATION

Test No. 8

A sample of ore was ground with lime, 5·0 pounds per ton of ore, to 80 per cent minus 200 mesh.

The pulp was conditioned for 3 minutes with 0·02 pound of butyl xanthate per ton and after adding 0·05 pound of pine oil per ton a concentrate was removed. The concentrate was cleaned and the cleaner tailing was added to the flotation tailing, filtered, and sampled.

The tailing was repulped in water and passed over a corduroy blanket, sloping 2·5 inches in 12 inches. The blanket concentrate was panned to remove gangue. The blanket tailing was filtered and samples of the tailing were cyanided for 24 and 48 hours at a dilution of 2 : 1 in a solution containing 1·0 pound of potassium cyanide per ton.

Results:

Flotation:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Au, oz/ton	Cu, per cent	Au	Cu	
Feed	100·00	0·605	0·30	100·00	100·00	
Flotation concentrate	1·45	28·47	14·56	68·23	81·07	69 : 1
Flotation tailing	98·55	0·195	0·05	31·77	18·93	

Flotation concentrate.................................. 0·88 per cent arsenic

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·195	100·00	
Blanket concentrate	0·84	6·69	28·82	119 : 1
Blanket tailing	99·16	0·14	71·18	

Cyanidation of Blanket Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton of ore	
		Feed	Tailing		KCN	CaO
8A	24	0·14	0·045	67·86	1·17	7·03
8B	48	0·14	0·025	82·14	1·86	7·13

Summary:

	Per cent
Distribution of gold in flotation concentrate	68·23
Distribution of gold in blanket concentrate, 31·77 × 28·82	9·16
	77·39
Per cent gold in cyanide feed, 31·77 × 71·18	22·61

—	24 hours, per cent	48 hours per cent,
Gold extracted in 24 hours, 22·61 × 67·86	15·34	....
Gold recovered in both concentrates	77·39	....
Gold extracted in 48 hours, 22·61 × 82·14	....	18·57
Gold recovered in both concentrates	....	77·39
Totals	92·73	95·96

The removal of a concentrate from the flotation tailing by blankets did not increase the overall recovery. No lower tailing was obtained from the cyanidation of the blanket tailing than was obtained when the flotation tailing was cyanided directly.

Test No. 9

In this test, after removing a copper concentrate, the flotation tailing was aerated and cyanided. The sulphides in the cyanide tailing were then floated.

A sample of ore was ground with lime, 5 pounds per ton of ore, to 80 per cent −200 mesh.

The pulp was floated with 0·02 pound of butyl xanthate and 0·05 pound of pine oil per ton. The concentrate was not cleaned.

A sample of the flotation tailing was repulped at a dilution of 3 : 1 and agitated with lime for 23 hours in a Wallace super-agitator. Lime was consumed at the rate of 10·15 pounds per ton of ore.

The pulp was filtered, washed, and repulped in cyanide solution, 1·0 pound of potassium cyanide per ton, at a dilution of 2 : 1. The agitation was discontinued after 24 hours and the tailing was filtered, washed, and repulped in a flotation cell. The reagents added to the cell were: soda ash, 4·0 pounds per ton; amyl xanthate, 0·2 pound per ton; and pine oil, 0·1 pound per ton. A concentrate was removed.

Results:

Copper Flotation:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton		Cu, per cent				
		Au	Ag		Au	Ag	Cu	
Feed	100·00	0·62	1·30	0·32	100·00	100·00	100·00	
Flotation concentrate	3·69	13·36	27·16	6·70	79·75	77·02	76·25	27 : 1
Flotation tailing	96·31	0·13	0·32	0·08	20·25	22·98	23·75	

Cyanidation of Aerated Flotation Tailing:

Feed assay, oz./ton		Tailing assay, oz./ton		Extraction, per cent		Reagents consumed, lb./ton	
Au	Ag	Au	Ag	Au	Ag	KCN	CaO
0·13	0·32	0·03	0·12	76·92	62·50	0·94	3·63

Flotation of Cyanide Tailing:

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Feed	100·00	0·03	0·12	100·00	100·00	
Flotation concentrate	13·72	0·18	0·66	74·17	77·77	7·3 : 1
Flotation tailing	86·28	0·01	0·03	25·83	22·23	

Flotation concentrate.................................... 0·20 per cent, Cu

Summary:

		Per cent
Distribution of gold in copper concentrate		79·75
Per cent gold in cyanide feed	20·25 per cent	
Extraction by cyanidation of flotation tailing, 20·25 × 76·92		15·58
Overall recovery of gold		95·33
Per cent gold in cyanide tailing, 20·25 − 15·58	4·67 per cent	
Distribution of gold in flotation of cyanide tailing:		
In pyrite concentrate, 74·17 × 4·67	3·46 per cent	
In flotation tailing, 25·83 × 4·67	1·21 "	
	4·67 per cent	
Loss of gold in cyanide tailing		4·67
		100·00
Distribution of silver in copper concentrate		77·02
Distribution of silver in the cyanide feed	22·98 per cent	
Extraction of silver by cyanidation of flotation tailing, 22·98 × 62·5		14·36
Overall recovery of silver		91·38
Distribution of silver in the cyanide tailing, 22·98–14·36	8·62 per cent	
Distribution of silver in flotation of cyanide tailing:		
Silver in pyrite concentrate, 77·77 × 8·62	6·70 per cent	
Silver in pyrite flotation tailing, 22·23 × 8·62	1·92 "	
	8·62 per cent	
Loss of silver in cyanide tailing		8·62
		100·00

Aeration of the cyanide feed in a lime pulp has a tendency to reduce the consumption of cyanide with no great improvement in extraction of the gold. Flotation of the cyanide tailing shows that much of the gold unattacked by cyanide is associated with the iron pyrite.

Test No. 10

In this test, a selective flotation of the copper sulphides and iron sulphides was made. The iron pyrite concentrate was reground and cyanided. Cyanidation tests were also made on the flotation tailing.

A sample of ore was ground to 95·6 per cent minus 200 mesh with 7·0 pounds of soda ash per ton and 0·1 pound of sodium cyanide per ton. The pulp was conditioned with 0·02 pound of butyl xanthate per ton for 3 minutes and floated with 0·05 pound of pine oil per ton. The concentrate was cleaned without further reagents and the cleaner tailing was returned to the flotation cell.

The pyrite was reactivated by the addition of 1·0 pound of copper sulphate per ton and conditioned for 10 minutes with 0·1 pound of amyl xanthate per ton. Pine oil was added at the rate of 0·125 pound per ton.

To obtain sufficient material, the pyrite concentrate was mixed with that from Test No. 11. A sample was cut out and the remainder was ground in cyanide solution, 3·0 pounds of potassium cyanide per ton at a dilution of 4 : 3, to 99·8 per cent minus 325 mesh. After grinding, the pulp was filtered, divided into three charges, and agitated in cyanide solution, which was made up from the solution from the grinding operation, for periods of 24, 48, and 72 hours. The strength of the solution was 2·0 pounds of potassium cyanide per ton and the dilution was 3 : 1. Frequent additions of reagents were required to maintain the solutions at strength.

Samples from the flotation tailing were agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, dilution 2 :1, for 24 and 48 hours.

Results:

Flotation:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concen- concen-
		Au, oz./ton	Cu, per cent	Au	Cu	
Feed	100·00	0·61	0·30	100·00	100·00	
Copper concentrate	2·06	22·24	9·70	74·68	71·28	48·5 : 1
Pyrite concentrate	16·67	0·81	0·41	22·01	24·37	6 : 1
Flotation tailing	81·27	0·025	0·015	3·31	4·35	

Cyanidation of Flotation Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
10A	24	0·025	0·01	60·0	1·84	5·00
10B	48	0·025	0·0075	70·0	2·32	7·70

Cyanidation of Pyrite Concentrate (From Tests Nos. 10 and 11):

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·81	0·145	82·10	15·41	18·10
48	0·81	0·105	87·04	19·46	17·47
72	0·81	0·07	91·36	24·85	19·61

Summary:

24-Hour Agitation:

Product	Weight, per cent	Assay, Au, oz./ton	Units, assay × per cent weight	Distribution, per cent
Feed	100·00	0·61	61·349	100·00
Copper concentrate	2·06	22·24	45·814	74·68
Cyanide tailing from pyrite concentrate	16·67	0·145	2·417	3·94
Extraction by cyanidation of pyrite concentrate			11·086	18·07
Cyanide tailing from flotation tailing	81·27	0·01	0·813	1·32
Extraction by cyanidation of flotation tailing			1·219	1·99
Overall recovery of gold				94·74
Combined cyanide tailings		0·033		

(Cyanide consumed: 4·10 pounds potassium cyanide per ton of feed.)

48-Hour Agitation:

Product	Weight, per cent	Assay, Au, oz./ton	Units, assay × per cent weight	Distribution, per cent
Feed	100·00	0·61	61·349	100·00
Copper concentrate	2·06	22·24	45·814	74·68
Cyanide tailing from pyrite concentrate	16·67	0·105	1·750	2·85
Extraction by cyanidation of pyrite concentrate			11·752	19·16
Cyanide tailing from flotation tailing	81·27	0·0075	0·610	0·99
Extraction by cyanidation of flotation tailing			1·422	2·32
Overall recovery of gold				96·16
Combined cyanide tailings		0·024		

(Cyanide consumed: 5·17 pounds of potassium cyanide per ton of feed.)

72-Hour Agitation of Pyrite Concentrate and 48-Hour Flotation Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Units, assay × per cent weight	Distribution, per cent
Feed	100·00	0·61	61·349	100·00
Copper concentrate	2·06	22·24	45·814	74·68
Cyanide tailing from pyrite concentrate	16·67	0·07	1·167	1·90
Extraction by cyanidation of pyrite concentrate			12·336	20·11
Cyanide tailing from flotation tailing	81·27	0·0075	0·610	0·99
Extraction by cyanidation of flotation tailing			1·422	2·32
Overall recovery of gold				97·11
Combined cyanide tailings		0·018		

(Cyanide consumed: 6·07 pounds of potassium cyanide per ton of feed.)

A 72-hour agitation of the combined pyrite concentrate and 24-hour flotation tailing gave a combined cyanide tailing of 0·020 ounce of gold per ton. The cyanide consumed was 5·67 pounds of potassium cyanide per ton of feed.

Test No. 11

This test was a duplicate of Test No. 10, except that 0·08 pound of cresylic acid per ton was used instead of pine oil in the copper flotation. This reagent produced a higher grade concentrate.

Results:

Flotation:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Au, oz./ton	Cu, per cent	Au	Cu	
Feed	100·00	0·61	0·30	100·00	100·00	
Copper concentrate	1·36	31·94	13·10	71·08	66·62	73·5 : 1
Pyrite concentrate	17·83	0·81	0·41	23·63	27·33	5·6 : 1
Flotation tailing	80·81	0·04	0·02	5·29	6·05	

Cyanidation of Flotation Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
11A	24	0·04	0·01	75·0	2·52	5·07
11B	48	0·04	0·01	75·0	2·53	8·23

Summary:

24-Hour Agitation:

Product	Weight, per cent	Assay, Au, oz./ton	Units, assay × per cent weight	Distribution, per cent
Feed	100·00	0·61	61·112	100·00
Copper concentrate	1·36	31·94	43·438	71·08
Cyanide tailing from pyrite concentrate	17·83	0·145	2·585	4·23
Extraction by cyanidation of pyrite concentrate			11·857	19·40
Cyanide tailing from flotation tailing	80·81	0·01	0·808	1·32
Extraction by cyanidation of flotation tailing			2·424	3·97
Overall recovery of gold				94·45
Combined cyanide tailings		0·034		

(Cyanide consumed: 4·67 pounds of potassium cyanide per ton of feed.)

48-Hour Agitation:

Product	Weight, per cent	Assay, Au, oz./ton	Units, assay × per cent weight	Distribution, per cent
Feed	100·00	0·61	61·112	100·00
Copper concentrate	1·36	31·94	43·438	71·08
Cyanide tailing from pyrite concentrate	17·83	0·105	1·872	3·06
Extraction by cyanidation of pyrite concentrate	....	....	12·570	20·57
Cyanide tailing from flotation tailing	80·81	0·01	0·808	1·32
Extraction by cyanidation of flotation tailing	....	....	2·424	3·97
Overall recovery of gold	....	....	....	95·62
Combined cyanide tailings	....	0·027	....	....

(Cyanide consumed: 5·35 pounds of potassium cyanide per ton of feed.)

72-Hour Agitation of Pyrite Concentrate and 24- and 48-Hour Agitation of Flotation Tailings:

Product	Weight, per cent	Assay, Au, oz./ton	Units, assay × per cent weight	Distribution, per cent
Feed	100·00	0·61	61·112	100·00
Copper concentrate	1·36	31·94	43·438	71·08
Cyanide tailing from pyrite concentrate	17·83	0·07	1·248	2·04
Extraction by cyanidation of pyrite concentrate	....	....	13·194	21·59
Cyanide tailing from flotation tailing	80·81	0·01	0·808	1·32
Extraction by cyanidation of flotation tailing	....	....	2·424	3·97
Overall recovery of gold	....	....	....	96·64
Combined cyanide tailings	....	0·021	....	....

(Cyanide consumed: 6·25 pounds of potassium cyanide per ton of feed.)

SUMMARY

Straight cyanidation at grinds of 62 and 71 per cent minus 200 mesh gave extractions of 45 and 58 per cent of the gold and 58 and 66 per cent of the silver, respectively.

Several flotation tests show that over 70 per cent of the gold and silver can be recovered in a copper concentrate containing 18 per cent of copper at a ratio of concentration of 85 to 1. These results required grinding to at least 90 per cent minus 200 mesh. Coarser grinding gave lower grade of copper concentrate but did not materially affect the recovery of gold.

Cyanidation of the flotation tailing at 75 per cent minus 200 mesh gave final tailings of 0·035 ounce per ton in 24 hours and 0·025 ounce per ton in 48 hours. Increasing the grind to 90 per cent minus 200 mesh gave erratic tailings.

Aeration of the flotation tailing shows the maximum extraction by cyanidation in 48 hours, at a grind of 91 per cent minus 200 mesh, and the highest overall recovery of gold obtained, 97·6 per cent.

Separation of the pyrite followed by regrinding and cyaniding for 72 hours and cyaniding the flotation tailing separately did not increase the overall recovery of gold but greatly increased the consumption of reagents.

CONCLUSIONS

The experimental tests show that straight cyanidation without removing the copper minerals is unsuitable, owing to low extraction and high reagent consumption.

Over 70 per cent of the gold and silver can be recovered in a copper concentrate at 85 : 1 ratio of concentration and containing 18 per cent copper, 100 ounces of silver, and over 30 ounces of gold per ton. The flotation tailing when aerated with lime and cyanided for 48 hours gave a maximum overall recovery of 97·6 per cent of the gold and silver. Cyanidation for 16 hours gave an overall recovery of 96 per cent of the gold.

No increase in recovery was obtained by separating, regrinding, and cyaniding the pyrite.

It is not practicable entirely to avoid the shipment of concentrate. Copper minerals in the ore prove to be strong cyanicides that cause high loss of reagents when straight cyanidation is attempted. When these copper minerals have been removed, the residue may be cyanided to extract the remaining precious metals.

The presence of considerable pyrrhotite makes it necessary to aerate, thicken, and preferably to filter the flotation tailing before entering the cyanide plant. If fouling of solution then occurs, a percentage of the barren cyanide solution should be run continuously to waste.

Ore Dressing and Metallurgical Investigation No. 723

GOLD ORE FROM THE QUESNELLE QUARTZ MINE,
HIXON CREEK, CARIBOO, BRITISH COLUMBIA

Shipment. A shipment of gold ore, net weight 120 pounds, was received on August 24, 1937, from the property of the Quesnelle Quartz Mining Company, Limited, located on Hixon Creek, Cariboo, midway between Quesnel and Prince George.

The shipment was submitted by N. J. Ker, President, Quesnelle Quartz Mining Company, Limited, 1000 Hall Building, 789 Pender Street West, Vancouver, B.C.

Characteristics of the Ore. The sample was already crushed to minus ½ inch when received. Fifty of these small pieces were mounted and polished and examined microscopically.

The *gangue* consists of fine-textured, grey to greenish grey, chloritic rock with a considerable quantity of finely disseminated carbonate.

Pyrite is the only abundant metallic mineral. It occurs as coarsely crystalline masses and coarse to medium grains containing small inclusions of pyrrhotite, chalcopyrite, and galena. In a few places pyrrhotite and chalcopyrite occur as narrow veinlets along fractures in the pyrite.

Native gold is visible only in pyrite, and appears to be largely in the dense mineral. Commonly the gold is not associated with inclusions of other minerals, but in a few cases it occurs with inclusions of galena in pyrite. The grain size of the gold seen ranges from 400 mesh (approximately 35 microns) to less than 2300 mesh (6 microns).

Sampling and Analysis. The sample was crushed and sampled by standard methods and was found to contain:

Gold	0·345	oz./ton
Silver	0·44	"
Copper	0·10	per cent
Arsenic	0·18	"
Sulphur	19·70	"
Pyrrhotite	0·30	"

Experimental Tests

Tests consisted of amalgamation followed by cyanidation, straight cyanidation, trap concentration followed by cyanidation and blanket concentration, amalgamation of blanket concentrate and cyanidation of the blanket tailing together with the residue from amalgamation, cyanide cycle tests and flotation of a cyanide tailing, and settling tests on a cyanide tailing.

AMALGAMATION

Tests Nos. 1 and 2

Samples of minus 14-mesh ore were crushed dry to pass through 48- and 100-mesh screens. These were amalgamated by barrel amalgamation at a dilution of 1 : 1 with water and 10 per cent of the dry weight of the ore of mercury.

The amalgamation tailing was sampled and assayed. A screen test on the ground ore shows the degree of grinding.

Screen Test:

Mesh	Weight, per cent	
	−48 mesh	−100 mesh
− 48+ 65	7·3	....
− 65+100	17·7	....
−100+150	19·0	13·4
−150+200	10·8	18·4
−200	45·2	68·2
	100·0	100·0

Results:

Test No.	Mesh	Assay, Au, oz./ton		Extraction, per cent
		Feed	Tailing	
1	− 48	0·345	0·205	40·58
2	−100	0·345	0·195	43·48

CYANIDATION OF AMALGAMATION TAILING

Samples from the amalgamation tailing were repulped in cyanide solution (1·0 pound of potassium cyanide per ton) at a ratio of dilution of 2 parts of solution to 1 part of ore, and agitated for periods of 16, 24, and 48 hours. Lime was used to supply protective alkalinity.

Results:

Cyanidation of Minus 48-Mesh Amalgamation Tailing from Test No. 1

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
1	16	0·205	0·045	78·05	0·53	3·63
1	24	0·205	0·03	85·37	0·73	4·38
1	48	0·205	0·03	85·37	0·84	4·50

Summary:

	Per cent
Total extraction of gold by amalgamation..............................	40·58
Gold left in amalgamation tailing, 100 − 40·58...... 59·42 per cent	
Feed to cyanidation contains 59·42 per cent of gold in the ore	
Extraction by cyanidation, 59·42 × 78·05 (in 16-hour period)............	46·38
Total recovery from this test....................................	86·96

Cyanidation of Minus 100-Mesh Amalgamation Tailing from Test No. 2

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
2..................	16	0·195	0·02	89·74	0·50	4·08
2..................	24	0·195	0·02	89·74	0·84	4·74
2..................	48	0·195	0·015	92·31	1·06	4·76

Summary of Tests Nos. 1 and 2:

Test No.	Agitation, hours	Recovery by amalgamation, per cent	Extraction by cyanidation, per cent	Total recovery of gold, per cent
1..................................	16	40·58	46·38	86·96
1..................................	24	40·58	50·73	91·31
1..................................	48	40·58	50·73	91·31
2..................................	16	43·48	50·72	94·20
2..................................	24	43·48	50·72	94·20
2..................................	48	43·48	52·17	95·65

STRAIGHT CYANIDATION

Test No. 3

Samples of minus 14-mesh ore were crushed dry to pass through 48-, 100-, 150-, and 200-mesh screens and agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 24 and 48 hours, at a dilution of 2 : 1. Lime was used to give a protective alkalinity to the solution.

Results:

Mesh	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
− 48...............	24	0·345	0·045	86·96	2·05	4·61
− 48...............	48	0·345	0·035	89·86	2·06	7·00
−100...............	24	0·345	0·02	94·20	2·47	6·43
−100...............	48	0·345	0·015	95·65	2·44	8·80
−150...............	24	0·345	0·015	95·65	2·91	7·68
−150...............	48	0·345	0·01	97·10	3·08	11·95
−200...............	24	0·345	0·02	94·20	3·96	9·79
−200...............	48	0·345	0·01	97·10	4·40	14·10

These results indicate that when the ore is dry ground to minus 150 mesh, the gold is readily soluble in cyanide solution.

GRINDING IN CYANIDE SOLUTION

Test No. 4

Samples of minus 14-mesh ore were ground in ball mills at a dilution of 4 parts of ore to 3 parts of cyanide solution, 1·0 pound of potassium cyanide per ton, to approximately 85 per cent minus 200 mesh. Lime was added to the mill to give protective alkalinity to the solution.

After grinding, three portions of the pulp were diluted to 1 part of ore to 1·5 parts of cyanide solution, 1·0 pound of potassium cyanide per ton, and agitated for 24 hours. Reagents were added to the solution as required to maintain it at strength.

Results:

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
4-A	0·345	0·015	95·65	1·40	9·75
4-B	0·345	0·02	94·20	1·47	9·70
4-C	0·345	0·015	95·65	1·52	9·85

The results of the test show that cyanicides in the ore cause an abnormal consumption of cyanide.

CONCENTRATION BY HYDRAULIC TRAP AND CYANIDATION OF TRAP TAILINGS

Test No. 5

A sample of minus 14-mesh ore was ground in a ball mill at a dilution of 4 parts of ore to 3 parts of water to give a product 78 per cent minus 200 mesh. The ground ore was passed through a hydraulic trap. The trap concentrate was panned but no free gold was observed in the pan. The concentrate was dried, weighed, and the gold content was calculated.

The trap tailing was filtered and sampled. From the remainder, two samples were repulped and agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, at a dilution of 2:1 for periods of 24 and 48 hours. Lime was used to give protective alkalinity.

Results:

Trap Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·345	100·00	
Trap concentrate	0·13	15·69	5·91	770 : 1
Trap tailing	99·87	0·325	94·09	

Cyanidation of Trap Tailing:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·325	0·015	95·38	0·95	4·43
48	0·325	0·015	95·38	1·28	4·76

Summary:

		Per cent
Gold in trap concentrate		5·91
Gold left in trap tailing, 100 − 5·91	94·09 per cent	
Extraction by cyanidation, 94·09 × 95·38		89·74
		95·65
Loss in trap tailing		4·35
		100·00

No advantage was obtained by agitating more than 24 hours. No benefit is derived by trapping gold from the ball mill discharge. Coarse gold is not present.

BLANKET CONCENTRATION, AMALGAMATION OF BLANKET CONCENTRATE, AND CYANIDATION OF AMALGAMATION RESIDUE TOGETHER WITH THE BLANKET TAILING

Test No. 6

A sample of minus 14-mesh ore was ground in a ball mill, dilution 4 : 3, to give a product 90 per cent minus 200 mesh.

The ground ore was concentrated on a corduroy blanket sloping 2·5 inches in 12 inches.

The blanket concentrate was amalgamated with mercury by grinding it in an iron mortar with mercury. After separating the mercury and amalgam the amalgamation residue was mixed with the blanket tailing and filtered.

After sampling for assay, two samples were repulped in cyanide solution, 1·0 pound of potassium cyanide per ton, at a dilution of 2 : 1 and agitated for 24 and 48 hours. Lime was used for protective alkalinity.

Results:

Blanket Concentration and Amalgamation:

Assay, Au, oz./ton		Extraction of gold, per cent
Feed	Amalgamation and blanket tailings	
0·345	0·245	29·0

Cyanidation:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·245	0·015	93·88	0·75	14·10
48	0·245	0·015	93·88	1·00	15·35

Summary:

	Per cent
Extraction by amalgamation	29·00
Extraction by cyanidation, 71·0 × 93·88	66·65
Overall extraction of gold	95·65
Loss in cyanide tailing	4·35
	100·00

The cyanidation tests show that washing out cyanicides lowers the consumption of cyanide.

STRAIGHT CYANIDATION CYCLE TEST

Test No. 7

Previous tests indicated the presence of cyanicides in the ore and this test was made to determine their effect in a cycle test. A cyanide tailing was floated to determine the distribution of gold in the products.

Samples of minus 14-mesh ore were ground in cyanide solution, 1·0 pound of potassium cyanide per ton, dilution 4 : 3, to approximately 90 per cent minus 200 mesh, with lime as a protective alkali.

The pulp was filtered and the solution from the grind was used for agitating the ground ore without any addition of fresh water. In order to do this, five charges of 1·5 : 1 dilution were used for the first cycle and agitated for 24 hours. At the conclusion of the agitation period, the pulps were filtered and the solution was precipitated with zinc dust. The amount of solution recovered from the first cycle was sufficient for four similar charges in the second cycle. Similarly, the third cycle used three charges and this procedure was carried out for five cycles. Cyanide and lime were added as required to maintain the strength of the solution.

The pulp from the fifth cycle was filtered, washed, and concentrated by flotation.

The pulp was conditioned for 10 minutes in a flotation cell with 2 pounds of soda ash per ton and 1·0 pound of copper sulphate per ton. After this, 0·2 pound of amyl xanthate per ton was added; followed by conditioning for 5 minutes. Then 0·1 pound of cresylic acid per ton was added and a concentrate was removed.

The results show that 87 per cent of the gold in the cyanide tailing reports in the flotation concentrate. It is believed that the gold was locked in the sulphide particles and thus was not in contact with the cyanide solution.

Results:

Cycle Test:

Cycle No.	Assay, Au, oz./ton		Extraction of gold, per cent	Total reagents consumed, lb./ton ore	
	Feed	Tailing		KCN	CaO
1	0·345	0·015	95·65	2·13	13·40
2	0·345	0·015	95·65	2·48	14·47
3	0·345	0·010	97·10	1·84	13·11
4	0·345	0·015	95·65	2·60	14·16
5	0·345	0·020	94·20	2·88	14·00

Cycle No. 5—Flotation of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·02	100·00	
Flotation concentrate	48·91	0·035	87·04	2·04 : 1
Flotation tailing	51·09	0·005	12·96	

The test shows that fouling is taking place in the fifth cycle.

The flotation test shows the distribution of gold between sulphide and gangue minerals in the cyanide tailing.

SETTLING TEST ON CYANIDE TAILING

Test No. 8

In this test the rate of settling of a cyanide tailing was determined for ratios of dilution of 1·5 : 1 and 2 : 1. The cyanide tailing was then concentrated by flotation to check the results obtained in Test No. 7, Cycle No. 5.

A sample of minus 14-mesh ore was ground in a ball mill to 90 per cent minus 200 mesh, using a solution of cyanide 1·0 pound of potassium cyanide per ton, dilution 4 : 3, with lime as the protective alkali. The pulp was diluted to 1·5 : 1 and agitated for 24 hours.

After agitation, the pulp was placed in a glass cylinder and the settling rate determined, by readings at 5-minute intervals for 1 hour. The pulp was then diluted to 2 : 1 and the alkalinity was adjusted to the same value as in the 1·5 : 1 dilution test. The readings were repeated for 1 hour.

After filtering and washing, the cyanide tailing was floated similarly to that in Test No. 7, using the following reagents:

Conditioning for 10 minutes with 2·0 pounds of soda ash and 1·0 pound of copper sulphate per ton, and for 5 minutes with 0·2 pound of amyl xanthate per ton and the addition of 0·1 pound of pine oil per ton. A concentrate was removed.

Results:

Settling Tests:

Time interval, 5 minutes	Dilution 1·5 : 1. Amount of settling, in feet	Rate, feet/hour	Dilution 2 : 1. Amount of settling, in feet	Rate, feet/hour
Start............	0·000	Rate for 1 to 15 minutes, 0·64 feet per hour.	0·000	Rate for 1 to 20 minutes, 1·26 feet per hour.
1...............	0·055		0·120	
2...............	0·060		0·100	
3...............	0·045		0·100	
4...............	0·050		0·100	
5...............	0·050		0·095	
6...............	0·050	Rate for next 45 minutes, 0·60 feet per hour.	0·090	Rate for next 40 minutes, 1·02 feet per hour.
7...............	0·050		0·090	
8...............	0·050		0·075	
9...............	0·050		0·085	
10...............	0·050		0·090	
11...............	0·045		0·085	
12...............	0·050		0·070	
Titration: KCN..................1·0 lb./ton sol. CaO....................0·2 "			Titration: KCN.............0·60 lb./ton sol. CaO...............0·20 "	

The extraction of gold by cyanidation was 95·65 per cent. The consumption of potassium cyanide was 1·67 pounds per ton of ore; that of lime was 7·7 pounds per ton of ore.

Flotation of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, of gold, per cent	Ratio of concentration
Feed..	100·00	0·015	100·00	
Flotation concentrate.........................	38·68	0·03	79·07	2·6 : 1
Flotation tailing.............................	61·32	0·005	20·93	

The results of the flotation test show that 79 per cent of the gold remaining in the cyanide tailing was in the sulphide portion of the ore. These results are similar to those obtained in Test No. 7.

SUMMARY

From the microscopic examination of polished sections of the ore it appears that the gold occurs mostly in the pyrite grains and that part of the observed gold is very fine and completely locked up within the grains of dense pyrite.

A flotation test of the tailing from one of the cyanide tests showed that practically all of the gold remaining in the tailing was associated with or contained in the sulphides. (See latter part of Test No. 7.)

The analysis of the ore shows the presence of 0·10 per cent of copper as chalcopyrite, a cyanicide, and the presence of 0·3 per cent of pyrrhotite, also a cyanicide.

Amalgamation tests of the ore ground to various degrees of fineness showed recoveries varying from 29 per cent to as high as 43 per cent. Tests with hydraulic traps showed that very little gold can be recovered by such apparatus, only about 5 per cent of the total gold reporting in the trap concentrate. Tests made by passing the ground ore (90 per cent minus 200 mesh) over corduroy blankets indicate that only about 29 per cent of the total gold can be recovered by amalgamating this product.

No free pieces of gold could be observed in either the trap or blanket concentrates.

The cyanide tests show that good extractions of the gold can be obtained. At a grind of approximately 68 per cent minus 200 mesh about 89 per cent of the gold can be extracted or an 0·03 to 0·02 ounce per ton tailing obtained; finer grinding gave extractions up to 97 per cent.

Tests Nos. 5 and 6 indicate that traps or blankets to remove a concentrate for amalgamation will not reduce the tailing loss.

By grinding the ore in cyanide solution to about 85 per cent through 200 mesh, extractions up to 95·5 per cent were obtained. Finer grinding, to 90 per cent through 200 mesh, increased the extraction to 97 per cent. Increased time of agitation over 24 hours gave a small increase in extraction.

All cyanide tests show high cyanide consumption and the cycle tests show that the solution will foul and affect the extraction.

The fouling is no doubt due to the presence of the copper and pyrrhotite.

The ore can be classed as moderately hard to grind.

The pulp settled at the rate of 0·64 feet per hour at 1·5 : 1 dilution and 1·26 feet per hour at 2 : 1 dilution, with an alkalinity of 0·2 pound of lime per ton of solution.

CONCLUSIONS

The gold can be extracted by cyanidation provided the ore is ground fine enough to expose it to the action of the cyanide solution and the solutions are prevented from becoming foul. The tests tend to indicate that the ore should be ground to approximately 85 to 90 per cent through 200 mesh and that a classification system should be provided that will allow the sulphides to be retained in the grinding circuit and be given a differential grind.

In order to prevent fouling of the solution, which will result in a lower extraction, systematic bleeding should be practised and periodic analysis of the solution made for KCNS, copper, and reducing power.

On an ore such as this our experience has shown that the control of the alkalinity of the solution is important. Low alkalinity will reduce the tendency of the solutions to become foul. The settling rate of the ore will, of course, be affected by the alkalinity of the solution and it may be found more expedient to add lime to the thickeners than to maintain this required alkalinity all through the circuit.

Ore Dressing and Metallurgical Investigation No. 724

GOLD ORE FROM THE MONETA PORCUPINE MINE AT TIMMINS, ONTARIO

Shipment. A carload shipment of 20 tons of ore was received on August 4, 1937. The shipment was submitted by W. E. Segsworth, President, Moneta Porcupine Mines, Limited, 67 Yonge Street, Toronto, Ontario.

This property is located in Tisdale Township, Timiskaming County, Ontario, and adjoins the western boundary line of Hollinger Consolidated Gold Mines, Limited.

Characteristics of the Ore. A description of the ore will be found in the report of Investigation No. 714, covering preliminary test work on another shipment of ore from the property.

Sampling and Assaying. The ore was crushed to approximately ¼ inch and a sample cut from it by a Vezin sampler. The sample assayed as follows:

Gold	0·454	oz./ton
Silver	0·046	"
Iron	12·45	per cent
Sulphur	6·16	"
Insoluble	48·20	"

Experimental Tests

The work was confined to mill runs in which the ore was treated by two distinct methods:

(1) The ore was ground in cyanide solution and agitated. Then it was thickened, filtered, washed with water and the cyanide residue repulped and floated.

(2) The ore was ground in water with suitable reagents and floated, the concentrate to be reground and treated by cyanidation.

Maximum extraction obtained by cyanidation was 86·78 per cent of the gold with a cyanide tailing assaying 0·065 ounce of gold per ton. An additional 8·63 per cent was recovered in the form of concentrate by floating the cyanide tailing. By flotation of the ore, 96·29 per cent of the gold was recovered in the form of a concentrate.

A description of the flow-sheets used and results obtained by the two methods of treatment follows.

In the first test run the ore was ground in cyanide solution, approximately 1·0 pound of potassium cyanide per ton, and the lime was kept down to not more than 0·20 pound per ton of solution in the grinding circuit.

The classifier overflow at about 5 : 1 dilution was sent to a Genter thickener, the filtrate from which was sent to precipitation. The Genter underflow at 1 : 1 dilution went to two agitators in series and from there to an Oliver filter where it was washed with water only. The filter cake was repulped in water and sent to the flotation circuit. The barren solution was returned to the grinding circuit.

After four days of operation the classifier overflow was sent direct to the agitators at 5 : 1 dilution after which it was thickened and filtered, the thickener filtrate going to precipitation as before. Soda ash and copper sulphate were added to the repulper and Barrett No. 4 oil, xanthate, and pine oil to the conditioning tank and flotation cells. Alkaline starch solution was added to the agitator discharge as an aid to settling and filtration.

Samples for assay were taken daily at regular intervals throughout the circuit, each day's samples being assayed separately.

A summary of the results obtained follows:

65726—5½

Test Run No. 1

Summary of Results—Cyanidation:

Date	Feed per hour	Dilution classifier overflow, per cent	Assay, Au, oz./ton: Feed	Assay, Au, oz./ton: Classifier overflow	Assay, Au, oz./ton: Genter underflow	Assay, Au, oz./ton: Agitation discharge	Assay, Au, oz./ton: Cyanide residue	Assay, Au, oz./ton: Pregnant solution	Assay, Au, oz./ton: Barren solution	Extraction	Titrations: Classifier overflow: KCN	Titrations: Classifier overflow: CaO	Titrations: Agitators: KCN	Titrations: Agitators: CaO	Residue: KCN	Residue: CAO
August																
11........	150	16 to 18	0·454	0·190	0·26?	0·11	0·09	0·359	0·345	80·17	0·9 to 2·0	0·12 to 0·3	0·50	0·05	Tr.	0·05
12........	150	16 to 18	0·454	0·275	0·295?	0·14	0·115	0·481	0·409	74·66	1·0 to 2·3	0·30 to 0·4	0·50	0·05	Tr.	0·05
13........	117	15 to 20	0·454	0·225	0·220	0·215	0·127	0·314	0·002	72·03	1·0 to 2·0	0·50 to 0·6	0·80	0·10 to 0·05	....	....
14........	117	20 to 25	0·454	0·240	0·180	0·170	0·132	0·234	0·109	70·92	1·3 to 2·0	0·50 to 1·25	0·80	0·10 to 0·20	....	0·10
16*........	117	20 to 22	0·454	0·135	0·130	0·145	0·115	0·265	0·135	74·66	0·8 to 1·4	0·20 to 0·075	0·90	0·10 to 0·075	....	0·075
17........	117	20 to 22	0·410	0·170	0·140	0·145	0·090	0·262	0·114	78·05	1·0 to 1·5	0·10 to 0·15	0·80	0·075 to 0·05	....	0·05
18........	117	19 to 20	0·485	0·150	0·120	0·120	0·075	0·156	0·065	86·59	1·0 to 0·9	0·10 to 0·075	0·90	0·075 to 0·05	....	0·05
19........	117	20 to 23	0·454	0·175	0·130	0·135	0·065	0·192	0·060	86·78	0·9 to 1·7	0·10 to 0·075	0·90	0·075 to 0·05	....	0·05

* Change in flow-sheet (see text).

Summary of Results—Flotation:

Date	** Feed per hour	Dilution flotation feed, per cent solids	Reagents, lb./ton: $CuSO_4$	Reagents, lb./ton: Soda ash	Reagents, lb./ton: Barrett No. 4	Reagents, lb./ton: Sodium xanthate	Reagents, lb./ton: Pine oil	Assay, oz./ton: Feed	Assay, oz./ton: Concentrate	Assay, oz./ton: Tailing	Ratio	Extraction, per cent	Total extraction, cyanidation and flotation	Recovery by flotation of total gold in feed, per cent
August														
11................	100	25 to 27	1·6	2·00	Nil	0·16	0·10	0·090	0·56	0·03	8·83	70·44	94·13	13·96
12................	100	25 to 31	0·88	1·00	Nil	0·26	0·20	0·115	0·67	0·025	7·16	81·21	95·23	20·57
13................	100	22 to 24	0·88	1·65	0·11	0·22	0·10	0·127	0·47	0·015	4·06	91·15	98·50	25·48
14................	100	31 to 39	1·32	1·60	0·11	0·44	0·15	0·132	No flot.	....	....	....	....	....
16................	100	....	....	....	....	....	....	0·115	"	....	....	....	....	....
17................	100	22 to 25	0·66	1·00	0·11	0·11	0·15	0·090	0·52	0·025	7·61	75·92	94·70	16·65
18................	100	22 to 33	0·60	1·10	0·05	0·275	0·10	0·075	0·46	0·020	8·00	76·66	96·87	10·28
19................	100	10 to 24	0·66	1·00	0·11	0·260	0·10	0·065	0·44	0·025	10·38	65·25	95·41	8·63

**Estimated.

Screen Tests:

Date	Classifier overflow				Cyanide residue				Flotation concentrate				Flotation tailing			
	+65	+100	+200	−200	+65	+100	+200	−200	+65	+100	+200	−200	+65	+100	+200	−200
August																
11	0·20	1·20	11·5	86·70	Nil	0·40	12·6	86·70	....	....	....	....	....	....	....	....
12	0·80	2·30	13·8	82·90	0·10	1·40	11·3	26·90	....	....	....	....	....	....	....	....
13	0·40	1·90	11·9	85·60	....	....	....	....	Nil	0·20	4·3	94·8	Nil	1·5	11·2	86·9
14	1·10	9·00	7·2	82·70	0·90	3·00	13·9	82·20	....	....	....	....	....	....	....	....
16	....	....	3·0	96·90	0·90	2·20	14·6	82·30	....	....	....	....	....	....	....	....
17	0·50	1·60	11·4	86·50	....	0·90	8·9	89·90	....	....	....	....	0·30	2·20	15·0	82·5
18	0·20	0·60	7·6	91·50	....	....	....	....	....	....	1·1	98·7	....	0·50	7·3	92·2
19	....	0·20	5·7	94·10	....	....	....	....	Nil	0·10	1·2	98·6	....	0·60	9·4	90·0

A sample of the concentrate produced from the cyanide tailing on August 11, 1937, was sent to A. L. Blomfield, Managing Director, Lake Shore Mines, Limited, Kirkland Lake, Ontario, who investigated how to recover as much gold as possible from the concentrate. It contained 13·96 per cent of the gold originally in the ore and by cyanidation without further grinding 32·0 per cent was extracted, bringing the total extraction up to 84·63 per cent. The concentrate as received was ground 45·6 per cent through 10 microns and by regrinding to 88·7 per cent through 10 microns total extraction was raised to 86·89 per cent of the total gold in the ore, a figure that checks closely the results obtained at the Ore Dressing Laboratories when finer grinding of the ore was practised.

Roasting tests followed by cyanidation of the calcine showed that approximately 85 per cent of the gold in the concentrate could be extracted bringing the total extraction on the basis of the original ore up to 92·03 per cent.

Assays of clean pyrite produced by panning infrasized fractions of the concentrate show that the gold content of the pyrite falls rapidly with the fineness of the grind, as set forth in the following table:

Size	Pyrite distribution, per cent	Assay, gold, $/ton
+56 microns	4·8	$43.23
−56+40 "	11·6	23.56
−40+28 "	14·4	17.61
−28+20 "	17·9	13.22
−20+14 "	14·6	11.02
−14+10 "	10·3	10.35
−10 "	26·4	9.00
	100·0	$14.76

This sudden drop in the pyrite assay value is no doubt due to the incipient fractures in the pyrite described fully in Investigation No. 714, from which two excerpts are reproduced:

> Pyrite is the only metallic mineral in appreciable quantity. It occurs as medium to small cubic crystals and irregular grains disseminated abundantly in the gangue. In polished sections the mineral appears to be dense in character, with only slight fracturing. When it is etched with nitric acid, however, numerous tiny incipient fractures and inter-grain boundaries are brought out; this indicates that many of the grains are potentially composed of a mosaic of grains which appear only on etching.
>
> The particles (of gold) appear to occur within dense pyrite, but when the sample is etched with nitric acid the incipient fractures and inter-grain boundaries already referred to are seen to be in contact with the gold particles in almost all cases. The effect of such lines of weakness in the pyrite will be to lessen the difficulty of grinding to expose the gold particles.

In the second test run the ore was ground in water in a ball mill-classifier grinding circuit at the rate of 480 pounds per hour. The classifier overflow at about 30 per cent solids and 73 to 76 per cent through 200 mesh flowed into a conditioning tank and from there to the second cell in a battery of ten. From Cells Nos. 2 and 3 a concentrate was taken off and returned to Cell No. 1 for cleaning. The tailings from the cleaning cell and the first roughers went on to Cells Nos. 4 to 10 where a second rougher concentrate was taken off and returned to Cell No. 2 along with the original feed. On the second day of operation seven additional cells were used as roughers.

The following reagents were used the first day:

	Lb./ton
Soda ash	1·0
Barrett No. 4 oil	0·15
Sodium xanthate	0·20
Pine oil	0·15
Copper sulphate	0·30

Toward the end of the day potassium amyl xanthate was substituted for sodium xanthate.

On the second day the reagents were as follows:

	Lb./ton
Soda ash	0·80
Aerofloat No. 25	0·10
Potassium amyl xanthate	0·40
Copper sulphate	0·30
Pine oil	0·15

The soda ash was fed to the ball mill while the pine oil and Aerofloat were divided between the ball mill and the cells. The xanthate was divided between the conditioning tank and the cells and the copper sulphate was fed to the rougher cells.

Summary of Flotation Tests on the Ore:

Date	Feed rate, lb./hour	Per cent solids, classifier overflow	Assay, Au, oz./ton				Ratio of concentration	Extraction, per cent total gold
			Feed	Classifier overflow	Concentrate	Tailing		
Aug. 20	480	29 to 35	0·435	0·350	3·02	0·020	7·22 : 1	96·15
Aug. 21	480	27 to 31	0·460	0·395	2·88	0·020	6·50 : 1	96·29

CONCLUSIONS

The results obtained from large-scale test runs are much the same as those obtained by previous small-scale tests of the ore. When the ore is ground 90 per cent or more through 200 mesh in a closed grinding circuit about 85 or 86 per cent of the gold can be extracted by cyanidation and a further 8 or 10 per cent recovered in the form of a pyrite concentrate by floating the cyanide tailing. In order to float the cyanide residue, however, lime in the grinding and agitation circuits will have to be kept very low, about 0·05 pound per ton of solution.

The figure for extraction by cyanidation is a little higher than that obtained in small-scale tests in which the ore was ground in batches and agitated directly. This higher extraction is, no doubt, due to selective finer grinding of sulphides in the mill-classifier circuit than was obtained in batch grinding.

The intimate association of the gold with pyrite as well as the existence of numerous fractures in the pyrite grains has been established by microscopic examination of the ore (see Investigation No. 714) and the high sulphide content noted in the classifier return during the test runs seems to agree with this. Further evidence of the effect of fine grinding on the gold content of the pyrite in the concentrate is furnished by the infrasizing and panning test conducted at Lake Shore Mines. The very rapid drop in the gold content of the pyrite is no doubt due in some measure to fractures occurring naturally in this mineral.

In practice, therefore, with efficient classification and resultant fine grinding of the pyrite, such as might be obtained by the use of a hydro separator, it may be possible to bring extraction by cyanidation up to 90 per cent and in this case flotation of the cyanide tailing would be a decidedly marginal proposition.

In any case a concentrate floated from a cyanide tailing would be comparatively low grade because the ore contains much pyrite and this means a low ratio of concentration. Depending on the extraction obtained in the cyanidation circuit, the concentrate would assay anywhere from an upper limit of 0·60 to 0·70 ounce per ton in gold down to perhaps less than 0·40 ounce per ton.

Another possible method of treating this ore would be to concentrate it by flotation, then regrind and cyanide the concentrate. This is not to be recommended, however, as the advantage of grinding the ore in cyanide solution would be lost and extraction of the gold would become a secondary in place of the primary operation it should be when possible. From the results of previous small-scale test work it seems that about the same amount of gold can be extracted from the ore by direct cyanidation as by concentration and then cyanidation of the concentrate, but the first of these two methods offers the possibility of recovering a further 8 or 10 per cent of the gold from the cyanide tailing in the form of a low-grade pyrite concentrate which might be profitably treated.

Ore Dressing and Metallurgical Investigation No. 725

MICROSCOPIC EXAMINATION OF ORE AND CONCENTRATE FROM THE BRITANNIA MINING AND SMELTING COMPANY, LIMITED, BRITANNIA BEACH, BRITISH COLUMBIA

This investigation was undertaken at the request of C. V. Brennan, Assistant General Manager, Britannia Mining and Smelting Company, Limited, who, in a letter dated November 5, 1937, states the problem as follows:

> The Britannia Mining and Smelting Company, Limited, is now developing a new ore-body of considerable size, wherein part of the copper and most of the zinc minerals occur in a different form when compared to the copper-zinc minerals in the ore this company has mined and milled to date.
>
> In the normal ore practically all of the copper occurs as chalcopyrite, zinc as sphalerite, and iron as pyrite, in which case a good recovery of the copper-zinc-iron sulphides can be made in a bulk float (grinding 25 to 30 per cent on 65 mesh), which can be reground and treated to make a three-mineral separation.
>
> But, in the new ore, all of the minerals are finely disseminated, calling for fine grinding. Most of the zinc mineral appears to occur as bluish-black sulphide, contaminated with copper (covellite?), to the extent that this form of zinc mineral persists in floating along with the copper mineral, regardless of any depressing agents used to make a separation.

The samples received by the Division of Metallic Minerals on November 18, 1937, were:

No. 1—"A" Samples.	Cleaned concentrate +100 mesh	(2)
No. 2—"A" "	" " −100+200 mesh	(2)
No. 3—"A" "	" " −200 mesh	(2)
No. 4—"B" Sample.	Ore ground and panned	(2)
No. 5—"A" Samples.	Dry feed as used in tests	(2)
No. 6—"A" "	" " " " " "	(2)
No. 7—"B" Samples.	Grab sample of ore from mine	(3)
No. 8—"B" "	" " " " " " "	(3)
	Polished sections	(18)

Eighteen polished sections were prepared, distributed as shown by the figures in brackets in the list of samples.

Procedure. The investigation was limited to examination of the polished sections under the reflecting microscope. In the ore itself, the minerals were identified, their modes of occurrence noted, and an approximate grain analysis made by measuring a large number of grains. The various products were studied for mineral content, and grain analyses made in a similar manner. The figures given in the grain analyses are approximate only, probably showing considerable departure from the average percentages in the ore, for several reasons: (1) It is improbable that a few polished sections could represent truly the ore as a whole, and therefore the results are subject to errors dependent upon sampling in a very acute form; (2) Some of the minerals such as covellite and chalcocite are, as will be apparent, very difficult to measure, owing to their form; (3) All observations are subject to a personal factor.

Owing to the exigency of the problem, it was inadvisable to spend much time on an exhaustive study, for the figures indicate the true conditions if viewed *relatively*.

Characteristics of the Ore. The ore consists of a highly siliceous gangue through which the metallic minerals are disseminated as irregular grains. The *metallic minerals* present are:

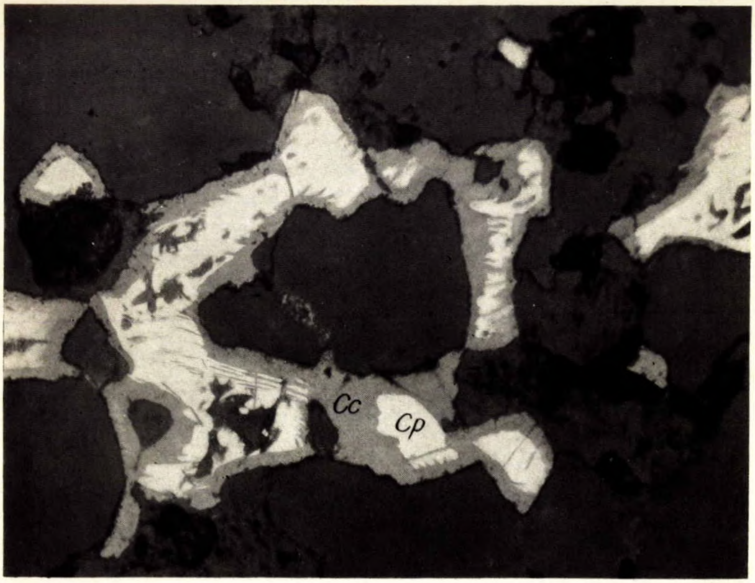
Pyrite	FeS_2
Chalcopyrite	$CuFeS_2$
Sphalerite	ZnS
Chalcocite	Cu_2S
Covellite	CuS
Pyrrhotite	Fe_xS_y
Native copper.	

Pyrite is the most abundant sulphide. It occurs as coarse to very fine grains, and shows good cubic crystallization, particularly in the finer sizes. It is disseminated without relation (genetically) to the other sulphides, and may contain any of them as fine veinlets; in certain cases it also shows slight corrosion and replacement by other sulphides, and it is sometimes included in chalcopyrite and sphalerite.

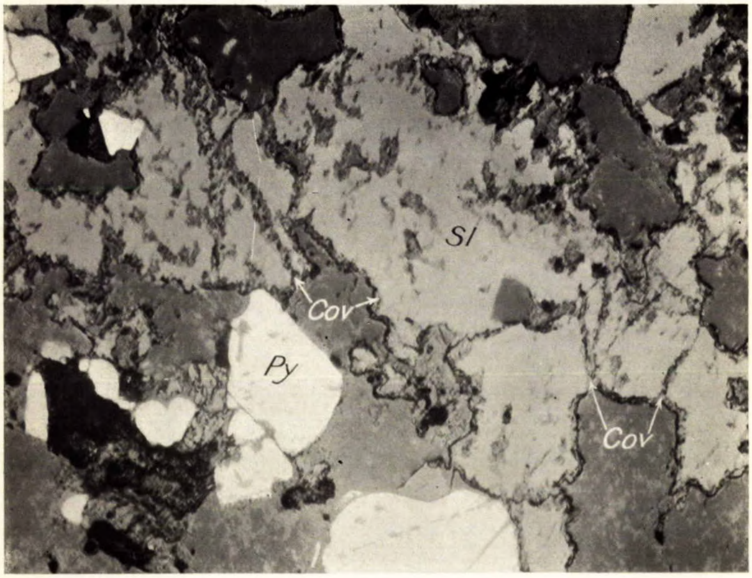
Chalcopyrite occurs as irregular grains, some of which are comparatively coarse, but most of which are medium to fine. Sphalerite occurs in the same manner but its grain size is somewhat coarser. There appears to be little admixing of chalcopyrite and sphalerite except where tiny dots of chalcopyrite occur in the latter. Both minerals have been replaced along their borders and along fractures by chalcocite and covellite, the result being effectively to armour the mineral grains with these secondary copper sulphides. The border zones, or shells (shown in Plates IA, IB, and IIA) vary from thin films to 30 or more microns in thickness. Covellite definitely favours the sphalerite, chalcocite favours the chalcopyrite. In rare cases both covellite and chalcocite occur around chalcopyrite grains, and here the relationships strongly suggest that the covellite has replaced the chalcocite. Covellite, and a third secondary alteration product, which is probably "limonite", also borders some of the pyrite grains; this condition is indicated in Table V. Only a very small proportion of the chalcopyrite is represented by the occurrence of that mineral as tiny dots in sphalerite, and as occasional small rounded grains in pyrite.

Pyrrhotite is rare. It occurs in one section as medium irregular grains, obviously later than and enclosing pyrite. One small grain of native copper (less than 200 mesh) was observed along a stringer of highly altered and porous gangue; probably it is very rare and confined to the highly oxidized parts of the ore.

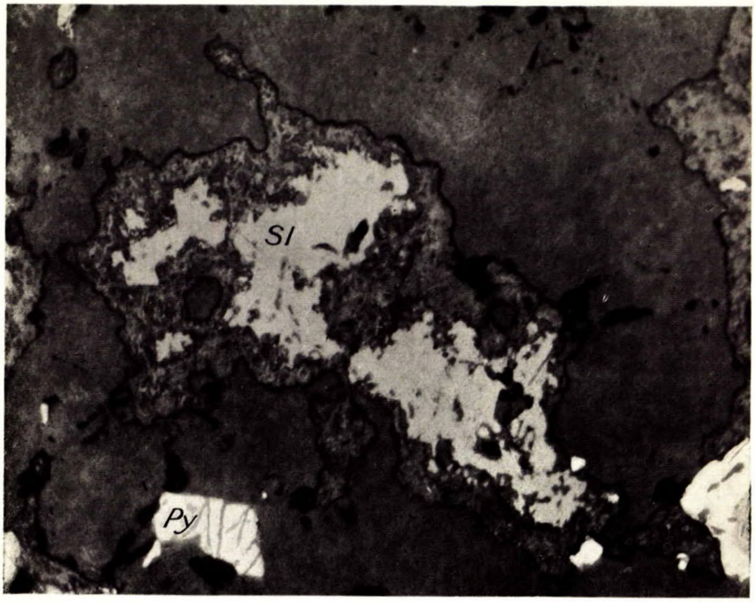
PLATE I



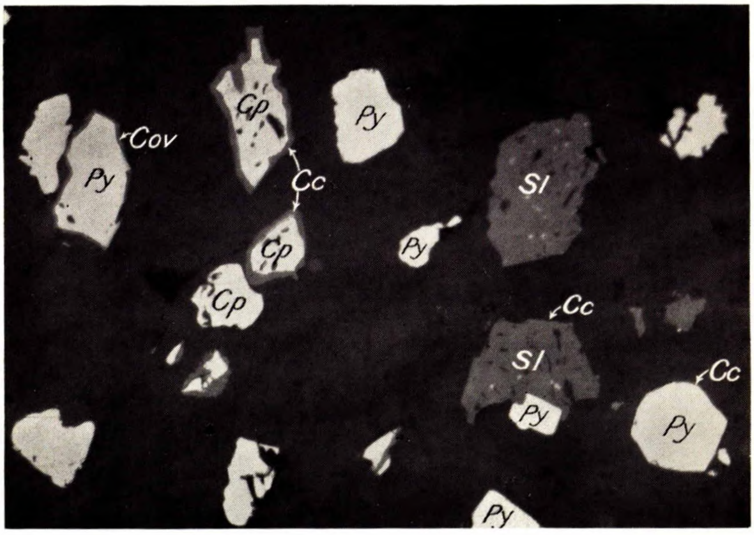
A. Border zones of chalcocite (Cc) (grey), replacing chalcopyrite (Cp) (white). Gangue is dark grey, pits are black. Pyrite (Py). Note tongues of chalcocite penetrating into the chalcopyrite. ×200.



B. Narrow border zones of covellite (Cov) (grey), replacing sphalerite (Sl) (light grey). Pyrite (Py). Note the replacement veinlets of covellite cutting across the sphalerite. ×200.

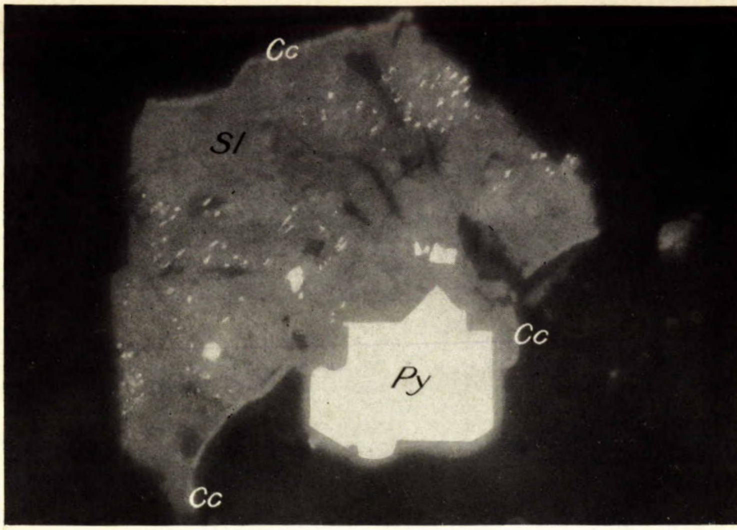


A. Advanced stage of replacement of sphalerite (Sl) (light grey) by covellite (dark grey, about the same as gangue but rough, indicating its softer character). Pyrite (Py). ×200.



B. Panned concentrate. Borders of chalcocite (Cc) around chalcopyrite (Cp) and borders of chalcocite, and covellite (Cov) around sphalerite (Sl). Pyrite (Py). ×100.

PLATE III



Grain of sphalerite (Sl) shown in Plate IIB, showing narrow border zones of chalcocite (Cc) around part of the grain. The bright dots in the sphalerite are chalcopyrite. Pyrite (Py). ×400.

The paragenesis of the ore minerals, as indicated by the above observations, may be outlined as follows:

	Primary mineralization	Secondary alteration
Pyrite......................	------	
Pyrrhotite..................	?------- ?	
Sphalerite..................	--------	
Chalcopyrite................	-----?--	
Chalcocite..................		----
Covellite...................		-----
"Limonite" (if present).....		-----
Native copper...............		-----

Grain Analysis of Ore. The microscopic grain analysis of the metallic minerals in the ore is shown in Table I. All sections of Samples Nos. 5, 6, 7, and 8 were used in this analysis. The figures are given by weight, the specific gravities assumed here and throughout the calculations being as follows: pyrite 5·0; chalcopyrite, 4·2; sphalerite, 4·0; chalcocite, 5·7; and covellite, 4·6.

TABLE I

Grain Analysis of the Metallic Minerals in the Ore:

Mesh	Chalcopyrite				Sphalerite			Chalcocite		Covellite		Pyrite, per cent	Total, per cent
	Free, per cent	Replaced by Cc, per cent	Replaced by Cov, per cent	In Sl, per cent	Free, per cent	Replaced by Cc, per cent	Replaced by Cov, per cent	In Cp, per cent	In Sl, per cent	In Cp, per cent	In Sl, per cent		
+150..........	9·4	4·3	....	....	2·8	....	16·9	....	....	....	....	25·2	58·6
−150+200..........	2·5	0·4	....	....	0·5	....	0·5	....	....	....	....	4·7	8·6
−200+280..........	2·0	0·6	0·2	....	0·2	0·2	0·1	....	....	....	....	5·7	9·0
—280+400..........	1·3	0·7	0·1	....	0·1	....	0·1	....	....	....	....	4·6	6·9
−400+560..........	0·5	0·6	0·2	....	0·3	....	....	....	....	....	....	3·4	5·0
−560..............	0·8	0·6	....	0·2	0·4	....	....	1·7	0·1	0·2	1·0	6·9	11·9
	16·5	7·2	0·5	0·2	4·3	0·2	17·6	1·7	0·1	0·2	1·0		
Totals........	24·4				22·1			1·8		1·2		50·5	100·0

NOTE 1: In all tables and figures the following designations are used for the minerals: Chalcopyrite—Cp; Sphalerite—Sl; Chalcocite—Cc; Covellite—Cov; Pyrite—Py.

NOTE 2: Owing to the difficulty in measuring the films of chalcocite and covellite, these minerals are placed in the minus 560-mesh size, although rightly a certain percentage should appear in the two larger mesh sizes.

TABLE II

Distribution of the Copper (in terms of 100 per cent copper content) in the Copper Minerals in the Ore:

Mesh	As Chalcopyrite				As Cc		As Cov		Total, per cent
	Free, per cent	Replaced by Cc, per cent	Replaced by Cov, per cent	In Sl, per cent	On Cp, per cent	On Sl, per cent	On Cp, per cent	On Sl, per cent	
+150.....	30·6	14·1	....	....	....	....	....	....	44·7
−150+200.....	8·2	1·3	....	....	....	....	....	....	9·5
−200+280.....	6·5	1·8	0·5	....	....	....	....	....	8·8
−280+400.....	3·9	2·0	0·4	....	....	....	....	....	6·3
−400+560.....	1·6	1·8	0·6	....	....	....	....	....	4·0
−560..........	2·7	2·0	0·2	0·8	12·8	0·6	1·0	6·6	26·7
	53·5	23·0	1·7	0·8	12·8	0·6	1·0	6·6	100·0
Totals.....	79·0				13·4		7·6		

TABLE III

Distribution of the Zinc (in terms of 100 per cent zinc content) in the Ore All as Sphalerite:

Mesh	Free, per cent	Replaced by Cc, per cent	Replaced by Cov, per cent	Total, per cent
+150..................................	12·3	....	76·7	89·0
−150+200..............................	2·0	....	2·0	4·0
−200+280..............................	0·8	0·8	0·7	2·3
−280+400..............................	0·6	....	0·5	1·1
−400+560..............................	1·2	....	....	1·2
−560..................................	2·1	0·3	....	2·4
Totals..........................	19·0	1·1	79·9	100·0
		81·0		

Sample No. 4

Ore Ground and Panned. The concentrate obtained by panning the ground ore was found to contain the minerals shown in Table IV.

TABLE IV

Distribution of the Minerals in the Panned Ore: (Percentages by Weight)

Mineral	Per cent
Pyrite..	75·3
Chalcopyrite..	11·8
Chalcocite..	5·7
Sphalerite..	4·6
Covellite...	2·6
	100·0

The concentration of the comparatively heavy pyrite and chalcopyrite, and particularly the chalcocite is to be noted in the panning product. Conversely much of the lighter sphalerite does not appear here, and it shows a comparatively low percentage. It is noteworthy that no native copper is present here, indicating that the quantity must be extremely small and probably very much localized in the ore.

The minerals show the same replacement, or "rimming" by the secondary products. Table V shows the percentage of free and combined grains for each mineral. Plates II B and III show this product.

TABLE V

Characteristics of the Metallic Mineral Grains in Concentrate Panned from Ore

		Per cent
Chalcopyrite:	Free	7·6
	With chalcocite rims	89·6
	With covellite rims	2·8
	In sphalerite	Trace
		100·0
Sphalerite:	Free	30·0
	With chalcocite rims	Trace
	With covellite rims	70·0
		100·0
Pyrite:	Free	88·5
	With films of covellite and probably a little "limonite"	11·5
		100·0
Chalcocite:	On chalcopyrite	100·0
	On sphalerite	Trace
		100·0
Covellite:	On sphalerite	43·0
	On pyrite	33·0
	On chalcopyrite	24·0
		100·0

Cleaned Concentrate, Screened. Three samples of screened concentrate were analysed to determine whether there is any improvement in freedom of the copper and zinc minerals with decreasing grain size. The complete grain analyses are shown in Table VI, which indeed shows a slight improvement in the finer sizes, but appreciable percentages remain combined in the minus 200-mesh fraction. Furthermore, as only sections of the grains are seen, many grains that appear to be free may still possess adhering fragments of the contaminating mineral sufficiently large to cause them to float with those that are wholly covered; the figures may thus be considered as minima for the combined material and maxima for the free material.

TABLE VI

Distribution of the Metallic Minerals in Three Products Screened from Cleaned Concentrate

(Percentages by Weight)

Mesh	Chalcopyrite					Sphalerite					Chalcocite		Covellite			Pyrite				Totals, per cent
	Free	Combined with				Free	Combined with				On Cp	On Sl	On Cp	On Sl	On Py	Free	Combined with			
		Cc	Cov	Sl	Py		Cc	Cov	Cp	Py							Cp	Sl	Cov	
+100........	1·9	3·9	0·3	0·8	0·3	7·5	1·0	5·5	1·5	4·8	1·5	0·1	0·1	0·4	0·3	59·2	0·5	6·3	4·1	100·0
−100+200........	2·8	5·5	0·3	0·3	0·2	8·3	0·3	3·7	0·6	4·6	2·6	0·5	0·3	0·6	0·4	62·0	0·4	2·8	3·8	100·0
−200.............	5·2	5·4	0·2	......	......	6·0	0·2	1·1	......	0·9	2·0	0·2	0·1	0·1	0·3	74·1	......	2·0	2·2	100·0

CONCLUSIONS

The zinc mineral is sphalerite. In the ore it occurs with narrow replaced border zones of covellite and, to a much lesser extent, of chalcocite. Roughly 80 per cent of the sphalerite is contaminated.

The fact that the grains are contaminated with covellite would lend them a bluish black colour, which is apparently what has been misleading.

Analyses of the products show that even in the ground state a large number of the sphalerite grains have a partial coating of covellite and/or chalcocite. These coatings would in any case make the sphalerite behave like the copper mineral chalcopyrite; but a considerable proportion of the chalcopyrite itself possesses coatings of chalcocite and, to a lesser extent, of covellite. It, therefore, would seem impossible to make a selective float of the copper and zinc minerals without first removing the coatings from the sphalerite sufficiently to cause it to be depressed. From the analyses shown in Table VI there seems to be some removal of these coatings in the finer sizes. Whether extremely fine grinding will remove them sufficiently to make an effective separation and whether this mechanical process can be assisted by chemical means are problems to be determined experimentally.

Ore Dressing and Metallurgical Investigation No. 726

GOLD ORE FROM THE FRANCOEUR GOLD MINES, LIMITED, ARNTFIELD, QUEBEC

Shipment. The shipment, net weight 225 pounds, was received on September 29, 1937. The ore was from the property of the Francoeur Gold Mines, Limited, Dasserat Township, near Arntfield, Quebec, and was submitted by John Knox, Jr., Manager.

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

The *gangue* is brownish grey to reddish grey in colour, and is highly siliceous. Traces of schistosity are conspicuous in some specimens, suggesting that the rock is a highly silicified schist.

The *metallic minerals* in the sections are: pyrite, chalcopyrite, and native gold. Pyrite is moderately abundant as coarse to fine disseminated grains, which show considerable fracturing. A very small quantity of chalcopyrite occurs as tiny grains in the pyrite, commonly along the fractures.

Native gold was observed only as irregular grains in the pyrite, and in all cases appeared to lie along fractures in the pyrite. The grain size of the gold is as follows:

Mesh	Weight, per cent
+ 280	16·2
− 280+ 400	21·6
− 400+ 560	25·7
− 560+ 800	23·0
− 800+1100	8·1
−1100	5·4
	100·0

Purpose of the Investigation. The shipment of ore was made for the determination of its response to straight cyanidation; for table concentration with regrinding of the concentrate followed by cyanidation; for amalgamation prior to cyanidation; and to note if average grinding capacity in the ball mills could be obtained.

Sampling and Analysis. The ore was crushed and sampled by standard methods and was found to contain the following values.

	Oz./ton
Gold	0·425
Silver	0·05

Experimental Tests

The work undertaken consisted of straight cyanidation of the ore at various grinds and periods of agitation without re-treating the tailings; straight cyanidation followed by concentrating the tailing on a Wilfley table, regrinding and cyaniding the concentrate; grinding and concentrating the ore on a Wilfley table, regrinding the concentrate and cyaniding the concentrate and table tailing separately; amalgamation of the ore prior to cyanidation; and a settling test on a cyanide tailing.

The results of the investigation show that at grinds of 60, 78, and 95 per cent through 200 mesh, extractions of 86, 92, and 95 per cent of the gold, respectively, were obtained in 24 hours' agitation. The same results were obtained from 48 hours' agitation.

By combined cyanidation and table concentration from ore ground 83 per cent through 200 mesh, 94·87 per cent extraction was obtained. The combined tailing was 0·02 ounce of gold per ton.

Amalgamation of the ore at a grind of 73·5 per cent through 200 mesh gave a recovery of 38·8 per cent of the gold. The amalgamation tailing was cyanided for 24 hours and gave a total extraction of 85·9 per cent of the gold.

Grinding tests of the ore show that it grinds easily and should give average grinding capacity in the ball mills.

STRAIGHT CYANIDATION

Tests Nos. 1 to 6

Samples of ore were ground in ball mills at a dilution of four parts of ore to three parts of cyanide solution, 1·0 pound of potassium cyanide per ton, dilution 4 : 3, for various periods of time. After grinding, the pulp was diluted to 1·5 parts of solution, 1·0 pound of potassium cyanide per ton, to one part of ore. Lime was used to give the solution protective alkalinity.

The periods of agitation were 24 and 48 hours. The tailings were assayed and screen tests were made to show the degree of grinding.

To check the results obtained in Test No. 3, two additional tests, Nos. 3-A and 3-B, were made.

Screen Tests:

Mesh No.	Weight, per cent							
	Test No. 1	Test No. 2	Test No. 3	Test No. 3-A	Test No. 3-B	Test No. 4	Test No. 5	Test No. 6
+ 48	1·8	....	....	....	....	5·9	....	....
− 48+ 65	6·4	0·2	....	....	....	12·2	0·2	....
− 65+100	12·0	3·3	....	....	0·1	14·7	2·7	....
−100+150	14·4	8·7	0·5	0·7	0·9	13·3	8·1	0·7
−150+200	5·3	10·0	0·9	3·0	3·7	1·0	7·6	1·4
−200	60·1	77·8	98·6	96·3	95·3	52·9	81·4	97·9
	100·0	100·0	100·0	100·0	100·0	100·0	100·0	100·0

Results of Cyanidation:

Test No.	Per cent −200 mesh	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
Agitation: 24 Hours.						
1........	60·1	0·425	0·06	85·88	0·10	3·10
2........	77·8	0·425	0·035	91·76	0·33	3·10
3........	98·6	0·425	0·025	94·12	0·60	4·90
3-A......	96·3	0·425	0·02	95·30	0·63	5·52
3-B......	95·3	0·425	0·02	95·30	0·69	5·58
Agitation: 48 Hours.						
4........	52·9	0·425	0·06	85·88	0·33	3·80
5........	81·4	0·425	0·035	91·76	0·46	3·78
6........	97·9	0·425	0·025	94·12	0·75	5·70

To determine where the residual gold is in the tailing, samples of the tailing from Tests Nos. 3-A and 3-B were digested with aqua regia. The insoluble portion showed only a trace of gold.

It is therefore apparent that the gold remaining in the cyanide tailing is associated with sulphides. This assumption is confirmed by the results of the microscopic examination of the polished sections, in which all visible gold was in the pyrite.

STRAIGHT CYANIDATION FOLLOWED BY TABLING, REGRINDING AND CYANIDATION

Test No. 7

Procedure. Samples of ore were ground in ball mills, dilution 4 : 3, in cyanide solution, 1·0 pound of potassium cyanide per ton, to give 83 per cent through 200 mesh. The pulps were diluted to 1·5 : 1 and agitated for 24 hours in a solution of 1·0 pound of potassium cyanide per ton. Lime was used to give protective alkalinity.

The pulps were filtered, washed, and sampled.

The cyanide tailings were concentrated on a Wilfley table. The concentrate was reground in the cyanide solution recovered from the initial cyanidation, brought up to 2 pounds of potassium cyanide per ton, and agitated at 3 : 1 dilution for 24 and 48 hours. The table tailing was assayed.

Results of Straight Cyanidation on Ore:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
Feed	Tailing		KCN	CaO
0·425	0·04	90·59	0·40	3·20

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·04	100·00	
Table concentrate	8·46	0·30	58·09	11·8 : 1
Table tailing	91·54	0·02	41·91	

Cyanidation of Table Concentrate:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton of concentrate	
		Feed	Tailing		KCN	CaO
7-A	24	0·30	0·065	78·33	3·10	16·86
7-B	48	0·30	0·07	76·67	3·54	20·15

Summary of the Test (Agitation, 24 Hours):

		Per cent
Gold extracted by cyanidation of the ore		90·59
Gold left in the cyanide tailing, 100 − 90·59	9·41	
Gold in table concentrate, 9·41 × 58·09	5·47	
Gold extracted from concentrate in 24 hours, 5·47 × 78·33		4·28
Total extraction		94·87
Loss of gold in table tailing		3·94
Loss of gold in table concentrate cyanide tailing		1·19
Total		100·00

Combined tailing assay	0·024 oz./ton
Cyanide consumed	0·66 lb./ton feed in 24-hour test
Lime consumed	4·43 lb./ton feed in 24-hour test

CYANIDATION OF TABLE CONCENTRATE AND TAILING SEPARATELY

Test No. 8

Procedure. Samples of ore were ground in ball mills, dilution 4 : 3, in water to give 69 per cent through 200 mesh.

The pulps were concentrated on a Wilfley table. The table concentrate was reground in cyanide solution and agitated for 48 hours at 3 : 1 dilution in a solution of 2 pounds of potassium cyanide per ton.

Samples of the table tailing were agitated for 24 hours at 1·5 : 1 dilution in solution of 1·0 pound of potassium cyanide per ton. Lime was used for protective alkalinity.

Results:

Table Concentration of Ore:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·41	100·00	10·7 : 1
Table concentrate	9·36	2·20	50·80	
Table tailing	90·64	0·22	49·20	

Cyanidation of Table Concentrate:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, of gold, per cent	Reagents consumed, lb./ton of concentrate	
		Feed	Tailing		KCN	CaO
8-A	48	2·20	0·08	96·36	2·78	31·52

Cyanidation of Table Tailing:

Test	Agitation, hours	Assay, Au, oz./ton		Extraction, of gold, per cent	Reagents consumed, lb./ton tailing	
		Feed	Tailing		KCN	CaO
8-B	24	0·22	0·02	90·91	0·24	2·94

Summary of Test No. 8:

—	Weight, per cent	Assay, Au, oz./ton	Per cent weight × assay	Distribution, per cent
Feed (calculated)	100·00	0·405	40·5328	100·00
Cyanide tailing from table concentrate	9·36	0·08	0·7488	1·85
Extraction by cyanidation of table concentrate			19·8432	48·96
Cyanide tailing from table tailing	90·64	0·02	1·8128	4·47
Extraction by cyanidation of table tailing			18·1280	44·72
Overall extraction of gold				93·68
Combined cyanide tailing		0·026		

Cyanide consumed	0·46 lb. KCN/ton feed
Lime consumed	5·42 lb. CaO/ton feed

AMALGAMATION FOLLOWED BY CYANIDATION

Test No. 9

Procedure. Samples of ore were ground in ball mills, dilution 4 : 3, in water to give 73·5 per cent through 200 mesh. The ground pulps were amalgamated by plate amalgamation.

Samples from the plate tailing were repulped in cyanide solution, 1·0 pound of potassium cyanide per ton, at a dilution of 1·5 : 1 and agitated for 24 and 48 hours. Lime was used for protective alkalinity.

Results:

Amalgamation:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
0·425	0·26	38·82

Cyanidation of Plate Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
9-A......	24	0·26	0·06	76·92	0·15	2·90
9-B......	48	0·26	0·07	73·08	0·48	2·95

Summary:

—	24-hour agitation	48-hour agitation
	Per cent	Per cent
Gold recovered by amalgamation....................................	38·82	38·82
Gold left in cyanide feed, 100 − 38·82....................61·18 per cent		
Gold extracted by cyanidation,		
24 hours, 76·92 × 61·18..	47·06	
48 hours, 73·08 × 61·18..	....	44·71
Total extraction of gold..................................	85·88	83·53
Loss in cyanide tailing..................................	14·12	16·47
Total..	100·00	100·00

SETTLING TESTS OF A CYANIDE TAILING

Test No. 10

In this test, the rate of settling of a cyanide tailing was determined for ratios of dilution of 1·5 : 1 and 2 : 1.

Procedure. A sample of ore was ground and prepared similarly to that in Test No. 7 (83 per cent through 200 mesh). The pulp was cyanided for 24 hours at 1·5 : 1 dilution in solution of 1·0 pound of potassium cyanide per ton, using lime to give protective alkalinity.

After the agitation, the pulp was placed in a glass cylinder and the rate of settling was determined by readings at 5-minute intervals for one hour. The pulp was then diluted to 2 : 1, lime was added to give about the same alkalinity as in the 1·5 : 1 dilution, and the readings were repeated for one hour.

Results of Settling Tests:

Time interval, 5 minutes	Dilution, 1·5 : 1		Dilution, 2 : 1	
	Amount of settling, in feet	Rate, in feet per hour	Amount of settling, in feet	Rate, in feet per hour
Start	0·000	Rate for 1 to 40 minutes, 0·52 foot per hour	0·000	Rate for 1 to 40 minutes, 1·22 feet per hour
1	0·050		0·140	
2	0·045		0·105	
3	0·040		0·110	
4	0·040		0·100	
5	0·040		0·095	
6	0·045		0·095	
7	0·040		0·080	
8	0·045		0·090	
9	0·040		0·075	
10	0·040		0·085	
11	0·040		0·075	
12	0·040		0·070	

Titration for 1·5 : 1 Pulp:

KCN	0·72 lb./ton
CaO	0·32 "

Titration for 2 : 1 Pulp:

KCN	0·52 lb./ton
CaO	0·26 "
Cyanide consumed	0·66 lb. KCN/ton feed
Lime consumed	3·50 lb. CaO/ton feed
Extraction of gold	90·6 per cent

SUMMARY AND CONCLUSIONS

The results show that 95·3 per cent of the gold can be extracted by straight cyanidation. This was at a grinding of 95 per cent through 200 mesh. The remainder of the gold was shown by the aqua regia tests to be locked within the sulphides.

Straight cyanidation at a grind of 83 per cent through 200 mesh followed by table concentration, with regrinding and cyaniding the concentrate, gave an overall recovery of 94·87 per cent of the gold in 24 hours' agitation. No advantage in increased extraction was obtained by agitating the cyanide pulps longer than 24 hours.

Amalgamation of the ore at 73·5 per cent through 200 mesh shows a recovery of 39 per cent of the gold, and when the amalgamation tailing was cyanided an overall recovery of 85·88 per cent of the gold was obtained. The same recovery was obtained in Test No. 1 by straight cyanidation at a grind of 60 per cent through 200 mesh.

About the same recovery is obtained by grinding to 95 per cent through 200 mesh followed by straight cyanidation as is obtained by grinding between 75 and 80 per cent through 200 mesh followed by cyanidation of the reground pyrite concentrate.

Ore Dressing and Metallurgical Investigation No. 727

GOLD ORE FROM THE B. R. X. CONSOLIDATED MINES, LIMITED, BRIDGE RIVER, BRITISH COLUMBIA

Shipment. Two bags of ore, weight 140 pounds, were received on October 25, 1937, from E. R. Shepherd, Mine Manager, B. R. X. Consolidated Mines, Limited, Bridge River, British Columbia.

Location of the Property. The property of the B. R. X. Consolidated Mines, Limited, from which this shipment was received, is situated in the Bridge River area, Lillooet Mining Division, British Columbia.

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

Gold	0·62 oz./ton
Silver	0·15 "
Copper	0·01 per cent
Sulphur	2·68 "
Iron	4·60 "
Arsenic	0·14 "

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

The *gangue* is white vein quartz and fine-textured grey country rock. A considerable quantity of carbonate occurs as veinlets and disseminated grains.

Pyrite is the most prominent metallic mineral. It occurs as coarse to fine disseminated grains, and shows considerable shattering. Arsenopyrite is present in small quantity as disseminated crystals, commonly associated with pyrite. Rare small grains of chalcopyrite occur in gangue and in pyrite.

Native gold is visible only in the pyrite in the polished sections, although panning tests indicated the presence of some coarse gold. It occurs as small grains in apparent dense pyrite in the sections, the grain size being shown in the following table:

Grain Size of Gold in Pyrite:

Mesh	Per cent, gold
− 800+1100	8·8
−1100+1600	21·4
−1600+2300	25·9
−2300	43·9
	100·0

The proportion of relatively coarse gold occurring in the gangue is not known.

Investigational Work

The research procedure consisted of amalgamation, flotation, and cyanidation. Straight cyanidation extracted 95 per cent of the gold in the ore at a grind of 84 per cent minus 200 mesh.

AMALGAMATION

Tests Nos. 1 and 2

The ore at minus 14 mesh was ground in a ball mill to pass 44·8 per cent through a 200-mesh screen in Test No. 1 and 75·7 per cent in Test No. 2. The pulps were then amalgamated with mercury in a jar mill. The amalgamation tailings were assayed for gold.

Results:

Test No.	Assay, Au, oz./ton		Recovery of gold, per cent
	Feed	Tailing	
1	0·62	0·115	81·5
2	0·62	0·11	82·3

The above tests were made to determine the approximate 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.

AMALGAMATION AND CYANIDATION

Test No. 3

In this test the ore at minus 14 mesh was ground in a ball mill to pass 65·9 per cent minus 200 mesh. The pulp was amalgamated with mercury for one hour in a jar mill. The amalgamation tailing was sampled and divided into three portions. No. 3A was agitated in cyanide solution of 1 pound per ton strength, with 5 pounds of lime per ton of ore added, for a 24-hour period. In No. 3B the lime was reduced to 4 pounds per ton, and in No. 3C the lime was 4 pounds per ton and 0·5 pound of litharge per ton of ore was added. The strength of solution was 1 pound of potassium cyanide per ton in all cases.

A screen test showed the grinding as follows:

Screen Test:

Mesh	Weight, per cent
− 65+100	3·8
−100+150	15·9
−150+200	13·7
−200	65·9

Amalgamation:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
0·62	0·095	84·7

Cyanidation:

Feed: gold, 0·095 oz./ton

Test No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton of solution		Reagents consumed, lb./ton ore	
				KCN	CaO	KCN	CaO
3A	24	0·035	63·2	1·0	0·4	0·3	4·2
3B	24	0·035	63·2	0·9	0·35	0·4	3·3
3C	24	0·035	63·2	1·0	0·4	0·3	3·2

Summary:

	Per cent
Gold recovered by amalgamation	84·7
Gold extracted by cyanidation	9·7

CONCENTRATION, AMALGAMATION, AND CYANIDATION

Test No. 4

The ore at minus 14 mesh was ground to pass 64·9 per cent through a 200-mesh screen. The pulp was passed through a hydraulic classifier or trap and the trap tailing passed over a corduroy blanket set at a slope of 2·5 inches per foot. The combined trap and blanket concentrates were amalgamated with mercury and the amalgam residue added to the blanket tailing. This product was agitated in cyanide solution of 1 pound of potassium cyanide per ton strength for 24- and 48-hour periods. Four pounds of lime per ton of tailing was added to maintain protective alkalinity.

A screen test showed the grinding as follows:

Screen Test:

Mesh	Weight, per cent
− 65+100	4·5
−100+150	15·8
−150+200	14·3
−200	64·9

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Hydraulic Concentration:				
Feed	100·0	0·62	100·0	
Trap concentrate	0·7	36·09	40·7	143 : 1
Trap tailing	99·3	0·37	59·3	
Blanket Concentration:				
Feed	100·00	0·37	100·0	
Blanket concentrate	1·66	14·00	62·8	60 : 1
Blanket tailing	98·34	0·14	37·2	

The combined trap and blanket concentrates were amalgamated and the amalgam residue added to the blanket tailing. This product assayed 0·16 ounce of gold per ton.

Cyanidation:

Feed: gold, 0·16 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
			KCN	CaO	KCN	CaO
24	0·045	71·9	1·0	0·3	0·43	3·4
48	0·035	78·1	0·9	0·3	0·63	3·4

Summary:

	Per cent
Gold recovered in trap concentrate	40·7
Gold recovered in blanket concentrate	37·2
Gold recovered by amalgamation	74·2
Gold extracted by cyanidation (24 hours)	18·5
Gold extracted by cyanidation (48 hours)	20·2
Overall recovery (48 hours)	94·4

CYANIDATION

Test No. 5

In this test the ore at minus 14 mesh was ground in water to pass 84·4 per cent minus 200 mesh. Portions of the pulp were agitated in cyanide solutions of different strength for 24- and 48-hour periods. The cyanide tailings were assayed for gold. A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65+100	0·2
−100+150	4·8
−150+200	10·2
−200	84·4

Additions of Reagents:

Tests A and B:	KCN	1	lb./ton solution
	CaO	5	" ore
Tests C and D:	KCN	1	lb./ton solution
	CaO	4	" ore
Tests E and F:	KCN	1	lb./ton solution
	CaO	5	" ore
	PbO	0·5	" "
Tests G and H:	KCN	2	lb./ton solution
	CaO	5	" ore

Results:

Feed: gold, 0·62 oz./ton

Test No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
				KCN	CaO	KCN	CaO
A	24	0·04	93·5	1·0	0·5	0·70	4·00
B	48	0·03	95·2	0·9	0·3	0·90	4·40
C	24	0·03	95·2	1·0	0·3	0·80	3·40
D	48	0·025	96·0	1·0	0·25	0·85	3·50
E	24	0·03	95·2	1·0	0·55	0·45	3·90
F	48	0·03	95·2	1·0	0·3	0·45	4·40
G	24	0·03	95·2	1·9	0·55	1·30	3·90
H	48	0·03	95·2	1·8	0·3	1·50	4·40

CYANIDATION

Test No. 6

In an effort to determine whether fine grinding would improve the extraction, the ore at minus 14 mesh was ground to pass 89 per cent through a 325-mesh screen. The pulp was agitated in cyanide solution of 1 pound per ton strength for 24- and 48-hour periods. Four pounds of lime per ton of ore was added to maintain protective alkalinity.

A screen test showed the grinding as follows:

Screen Test:

Mesh	Weight, per cent
−100+150	2·0
−150+200	5·3
−200+325	3·7
−325	89·0

Results:

Feed: gold, 0·62 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
			KCN	CaO	KCN	CaO
24	0·03	95·2	0·92	0·22	0·55	4·45
48	0·03	95·2	0·96	0·36	0·65	4·70

Test No. 7

In this test the ore at minus 14 mesh was ground to pass 84·4 per cent minus 200 mesh. The pulp was agitated in cyanide solution of a strength of 1 pound per ton for a 24-hour period. Four pounds of lime per ton of ore was added. The cyanide tailing was conditioned with 3 pounds of soda ash per ton; reactivated with 0·5 pound of copper sulphate per ton; and floated with 0·1 pound of amyl xanthate and 0·05 pound of pine oil per ton. The flotation concentrate was reground to pass 95 per cent minus 325 mesh and agitated in cyanide solution of 3 pounds of potassium cyanide per ton strength for a 24-hour period.

The different products were assayed for gold.

Results:

Cyanidation:

Feed: gold, 0·62 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
			KCN	CaO	KCN	CaO
24	0·035	94·3	0·92	0·25	0·37	3·50

Flotation of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·035	100·0	
Flotation concentrate	11·1	0·24	74·5	9 : 1
Flotation tailing	88·9	0·01	25·5	

Cyanidation of Flotation Concentrate:

Feed: gold, 0·24 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
			KCN	CaO	KCN	CaO
24	0·165	31·3	3·0	0·8	2·90	7·60

Summary:

	Per cent
Gold extracted by straight cyanidation	94·3
Gold extracted by cyanidation of flotation concentrate	1·3
Overall recovery	95·6

CYANIDATION AND TABLE CONCENTRATION

Test No. 8

In this test the ore was ground and agitated in cyanide solution similarly to Test No. 7. The cyanide tailing was passed over a Wilfley table and the table concentrate reground and agitated in cyanide solution.

Results:

Cyanidation:

Feed: gold, 0·625 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
			KCN	CaO	KCN	CaO
24	0·04	93·5	0·90	0·25	0·41	3·5

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·04	100·0	
Table concentrate	7·1	0·24	42·0	14·1 : 1
Table tailing	92·9	0·025	58·0	

Cyanidation of Table Concentrate:

Feed: gold, 0·24 oz./ton

Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
			KCN	CaO	KCN	CaO
24	0·085	64·6	2·7	0·65	2·8	8·0

Summary:

	Per cent
Gold extracted by straight cyanidation	93·5
Gold extracted by cyanidation of table concentrate	1·7
Overall recovery	95·2

FLOTATION

Test No. 9

In this test the ore at minus 14 mesh was ground with 3 pounds of soda ash per ton to pass 70·0 per cent through a 200-mesh screen. The pulp was conditioned with 0·1 pound of amyl xanthate per ton and floated with 0·05 pound of pine oil per ton.

A screen test showed the grinding as follows:

Screen Test:

Mesh	Weight, per cent
− 65+100	2·5
−100+150	12·9
−150+200	13·6
−200	70·0

Results:

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·48*	100·0	
Flotation concentrate	11·6	3·72	89·9	8·6 : 1
Flotation tailing	88·4	0·055	10·1	

*Calculated.

A Haultain panning test on the flotation tailing showed the presence of free gold and a small quantity of sulphides. The panner tailing assayed 0·03 ounce of gold per ton.

BLANKET CONCENTRATION AND FLOTATION

Test No. 10

The ore at minus 14 mesh was ground to pass 86·9 per cent minus 200 mesh and the pulp was passed over a corduroy blanket set at a slope of 2·5 inches per foot. The blanket concentrate was reground and amalgamated; and the blanket tailing was conditioned with 2 pounds of caustic soda per ton and floated with 0·1 pound of amyl xanthate and 0·05 pound of pine oil per ton.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Blanket Concentration:				
Feed	100·00	0·62	100·0	
Blanket concentrate	0·84	55·51	75·2	119 : 1
Blanket tailing	99·16	0·155	24·8	
Flotation of Blanket Tailing:				
Feed	100·0	0·155	100·0	
Flotation concentrate	7·2	1·41	66·1	13·9 : 1
Flotation middling	8·9	0·255	14·8	
Final tailing	83·9	0·035	19·1	

Amalgamation of Blanket Concentrate:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
55·51	0·92	98·3

Summary:

	Per cent
Gold recovered in blanket concentrate	75·2
Gold recovered in flotation concentrate	16·4
Overall recovery	91·6
Gold recovered by amalgamation of blanket concentrate	73·9

BLANKET CONCENTRATION AND FLOTATION

Test No. 11

The ore at minus 14 mesh was ground in a ball mill to pass 70·0 per cent minus 200 mesh. The pulp was treated similarly to Test No. 10. Three pounds of soda ash per ton replaced the two pounds of caustic soda per ton used in conditioning the blanket tailing.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Blanket Concentration:				
Feed	100·0	0·62	100·0	
Blanket concentrate	2·0	22·91	73·9	50 : 1
Blanket tailing	98·0	0·165	26·1	
Flotation of Blanket Tailing:				
Feed	100·00	0·165	100·0	
Flotation concentrate	11·55	1·17	83·9	8·7 : 1
Final tailing	88·45	0·03	16·1	

Amalgamation of Blanket Concentrate:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
22·91	1·78	92·2

Summary:

	Per cent
Gold recovered in blanket concentrate	73·9
Gold recovered in flotation concentrate	21·9
Overall recovery	95·8
Gold recovered by amalgamation of blanket concentrate	68·1

Test No. 12

In this test the ore at minus 14 mesh was ground to pass 87·8 per cent minus 200 mesh. The pulp was passed over a corduroy blanket and the blanket concentrate amalgamated. The blanket tailing was conditioned with 3 pounds of soda ash and 0·5 pound of copper sulphate per ton, and floated with 0·1 pound of amyl xanthate and 0·05 pound of pine oil per ton.

A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65+100	0·2
−100+150	2·9
−150+200	8·9
−200	87·8

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Blanket Concentration:				
Feed	100·0	0·62	100·0	
Blanket concentrate	1·0	47·15	76·0	100 : 1
Blanket tailing	99·0	0·15	24·0	
Flotation of Blanket Tailing:				
Feed	100·0	0·15	100·0	
Flotation concentrate	7·8	1·42	74·2	12·8 : 1
Flotation middling	6·9	0·31	14·4	
Final tailing	85·3	0·02	11·4	

Amalgamation of Blanket Concentrate:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
47·15	0·78	98·3

Summary:

	Per cent
Gold recovered in blanket concentrate	76·0
Gold recovered in flotation concentrate	17·8
Overall recovery	93·8
Gold recovered by amalgamation of blanket concentrate	74·7

In practice the overall recovery of the gold would be raised somewhat by the addition of a portion of the gold in the middling product. Owing to the comparatively large amount of sulphides in the ore, the ratio of concentration is low and results in a rather low-grade flotation concentrate.

SETTLING TESTS

Tests Nos. 13 and 14

These tests were carried out in a tall glass tube having an inside diameter of $2\frac{1}{8}$ inches. Two series of tests were made, one at a dilution of liquid to solid of 2 : 1 and the other at a dilution of 1·5 : 1. One pound of potassium cyanide per ton of solution was added. In both cases the addition of 4·0 pounds of lime per ton of ore was supplied. After grinding to 77·7 per cent minus 200 mesh the pulp was transferred to the glass tube and the level of solids in decimals of a foot read every 5 minutes. Readings were made for a one-hour period. At the end of this test the solution was titrated for alkalinity. The results of the tests are recorded in the following table:

Results:

—	Test No. 13	Test No. 14
Ratio of liquid to solid	2 : 1	1·5 : 1
Lime added, lb./ton solid	4·0	4·0
Potassium cyanide added, lb./ton solution	1·0	1·0
Alkalinity of solution at end of test	0·56	0·60
Overflow solution	Clear	Clear
Rate of settling, in feet per hour	0·49	0·36

A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65+100	0·5
−100+150	9·4
−150+200	11·9
−200	77·7

The rate of settling as shown in Tests Nos. 13 and 14 is slightly slower than normal.

SUMMARY AND CONCLUSIONS

Straight cyanidation of the ore recovers 94 to 95 per cent of the gold at a grind of 84 per cent minus 200 mesh. Concentration of the sulphides from the cyanide tailing, and regrinding and cyanidation of this product raised the overall recovery 1 to 2 per cent.

Although some 70 per cent of the gold can be recovered by amalgamation of trap and blanket concentrates, prior to agitation, the research work shows this procedure to be unnecessary as the gold is not sufficiently coarse to preclude a straight cyanidation treatment.

A slight increase in speed of solution of the gold can be obtained by adding the minimum amount of lime, about 4 pounds per ton of ore. The addition of litharge caused a slight decrease in the cyanide consumption.

The microscopic work showed that some of the gold is extremely finely divided, is enclosed in pyrite, and is shielded from the dissolving action of cyanide solution. Fine grinding and long agitation were unsuccessful in extracting the gold from this material, which carries the gold in amounts of 0·03 ounce per ton, and which remains in the cyanide tailing.

If the building of a cyanide plant is not feasible, a combination of traps, blankets, and flotation will recover over 95 per cent of the gold. A final flotation tailing of 0·02 ounce of gold per ton will be obtained at a grind of 87·8 per cent minus 200 mesh.

Up to 75 per cent of the gold can be recovered by barrel amalgamation of the trap and blanket concentrates. The remaining flotation concentrate together with the amalgam residue will assay about 1·5 ounces of gold per ton, this grade of concentrate being due to the comparatively large amount of free gold in the ore that is recovered prior to flotation; and also to the rather high percentage of pyrite that results in a low ratio of concentration.

Ore Dressing and Metallurgical Investigation No. 728

ANTIMONY ORE FROM THE TRIMBLE MINES, LIMITED, LILLOOET DISTRICT, BRITISH COLUMBIA

Shipment. One box, containing 155 pounds of antimony ore and several characteristic samples of high-grade ore, was received on September 28, 1937. These were shipped by A. R. Thompson for the Trimble Mines, Limited, Big Bar P.O., British Columbia. The property is on the north fork of Watson Bar Creek, about 70 miles north of Lillooet, British Columbia.

Characteristics of the Ore. Six polished specimens of the ore were prepared and examined microscopically to determine the antimony mineral, the presence of other minerals, and the character of the antimony mineralization.

The *gangue* appears to be a breccia of fine-textured and dense grey rock cemented with quartz. A very small quantity of carbonate is present, probably as calcite. Iron carbonate was not detected.

The only *metallic mineral* seen in the polished sections was stibnite (Sb_2S_3). It occurs as masses of considerable size, as coarse irregular stringers, and as coarse to fine needles and irregular grains in the gangue. It is largely coarse, only an exceedingly small fraction being finer than 200 mesh. An examination of products from one of the flotation tests containing from 7 to 11 per cent of arsenic showed occasional fine grains of arsenopyrite but much too little to account for the arsenic content.

A spectrographic analysis of a concentrate made from a flotation tailing by a Haultain superpanner showed the following metals:

Iron	Strong
Antimony	Moderate
Arsenic, silicon, zinc	Weak
Copper	Trace

This concentrate under a microscope showed a considerable number of canary-yellow coloured grains, the presence of realgar and/or orpiment is, therefore, a strong probability.

Purpose of Investigation. The shipment was made to determine if an economic method of concentration could be found for this class of ore.

Experimental Tests

The sample for investigation was crushed, ground, and sampled. Analysis showed it to contain:

Antimony	7·10 per cent
Arsenic	1·28 "
Copper	0·04 "
Lead	Nil
Zinc	0·05 per cent
Gold	Trace
Silver	Trace
Iron	1·82 per cent
Sulphur	2·67 "

Table concentration and flotation were studied, the results showing that table concentration was not satisfactory, owing to the large amount of fines produced during grinding. The coarser particles concentrated nicely but the slime was carried into the tailing.

Flotation was not particularly successful in making a high recovery. A concentrate containing 56 to 60 per cent antimony could be obtained with recoveries ranging from 61 to 88 per cent of the antimony. The arsenic content of the concentrate averaged about 10 per cent. None of the tests showed a possibility of removing arsenic from the stibnite concentrate.

TABLE CONCENTRATION

Samples of the ore ground minus 48 mesh and minus 65 mesh were concentrated on a Wilfley table with the following results.

Results:

Grind	Assay of concentrate, Sb, per cent	Assay of tailing, Sb, per cent	Distribution, per cent	
			Concentrate	Tailing
−48	27·66	6·39	37·8	62·2
−65	21·10	3·86	29·2	70·8

FLOTATION

The flotation investigation embraced flotation in natural circuit, in acid circuit, and in alkaline circuit.

Section No. I—Natural Circuit

Test No. 1

A sample of the ore was ground wet in a ball mill with 0·25 pound of fuel oil per ton and floated with 0·12 pound of pine oil per ton. The concentrate was cleaned once.

A screen test made on the flotation tailing showed the grind to be:

Mesh	Weight, per cent
+ 65	5·1
− 65+100	16·1
−100+150	20·7
−150+200	12·7
−200	45·4
Total	100·0

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	7·54	1·45	100·0	100·0
Concentrate	8·2	56·30	11·43	61·2	64·4
Middling	2·4	19·70	4·06	6·3	6·7
Tailing	89·4	2·75	0·47	32·5	28·9

Test No. 2

The quantity of fuel oil was increased to 1 pound per ton. All other conditions were the same as those in the preceding test.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	7·48	1·30	100·0	100·0
Concentrate	9·6	56·75	9·10	72·8	66·9
Middling	3·3	18·87	4·12	8·3	10·4
Tailing	87·1	1·62	0·34	18·9	22·7

The increase in fuel oil results in a larger quantity of concentrate of much the same grade as that in Test No. 1.

Test No. 3

To note the effect of fine grinding, a sample was ground to pass 88·6 per cent minus 200 mesh with 3·1 per cent plus 150 mesh, and floated as in Test No. 1.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	7·21	1·39	100·0	100·0
Concentrate	10·8	46·41	10·50	69·5	81·7
Middling	4·1	7·69	0·81	4·4	2·4
Tailing	85·1	2·21	0·26	26·1	15·9

Fine grinding apparently has a detrimental effect on recovery and grade of concentrate. In this test, 73·9 per cent of the antimony was floated in the rougher concentrate, whereas 81·1 per cent was recovered in the preceding test.

In several tests, of which no samples were taken, copper sulphate appeared to have a beneficial effect on the flotation of the stibnite. In some it was found that the xanthates alone produced very heavy matted froths, which did not respond to additions of pine oil.

Test No. 4

American Cyanamid reagent "Aerofloat No. 31" was used; 0·2 pound per ton being added to a sample of the ore, which was ground in a ball mill until 77·8 per cent passed 200 mesh; 1·0 pound of copper sulphate and 0·08 pound of potassium amyl xanthate per ton were added, together with 0·06 pound cresylic acid, and a concentrate was removed. This concentrate was cleaned once. The flotation conditions were very good, a fast-flowing froth being obtained. After 10 minutes, the froth was barren, containing no sulphides.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	8·43	1·26	100·0	100·0
Concentrate	11·3	56·91	6·25	76·3	55·8
Middling	4·3	8·60	7·11	4·4	24·2
Tailing	84·4	1·93	0·30	19·3	20·0

Test No. 5

The collector reagent was changed in order to determine if a higher recovery could be obtained.

In this test 0·3 pound of a mixture of water-gas tar, creosote, and cresylic acid in the proportions 65 : 25 : 10 was added, and ground with the sample. One pound of copper sulphate per ton also was added. After transferring the pulp to the flotation machine, 0·06 pound of cresylic acid was added, and a concentrate was removed. When sulphides had ceased to float a second addition of 1·0 pound of copper sulphate was made, together with 0·10 pound of potassium ethyl xanthate. More sulphides appeared in the froth and were taken off as Concentrate No. 2. These concentrates were not cleaned.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	7·29	1·23	100·0	100·0
Concentrate No. 1	8·5	44·39	7·00	51·7	48·2
Concentrate No. 2	6·1	40·25	6·70	33·7	33·1
Tailing	85·4	1·25	0·27	14·6	18·7

Although a lower tailing loss is recorded in this test the reagents used are not so efficient as those in preceding tests as flotation was forced. Copper sulphate has the effect of making the antimony more *floatable.* There is no indication of a selective flotation of either the antimony or arsenic sulphides.

Test No. 6

In preceding tests, no marked separation of antimony from arsenic was indicated.

In this test an attempt was made to obtain such a separation.

A sample of the ore was ground 66·4 per cent minus 200 mesh; 0·15 pound of cyanide, 1·0 pound of copper sulphate, 0·24 pound of Aerofloat No. 31 being added to the grinding mill. When the pulp was transferred to the flotation cell, 0·2 pound of potassium ethyl xanthate was added and a concentrate removed. A second addition of 1·0 pound of copper sulphate was made and changed the character of the froth to a gummy constituency. This concentrate contained some coarse particles of stibnite. The results of this test should be compared with those of Test No. 1, Section No. III.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	7·81	1·38	100·0	100·0
Concentrate No. 1	12·1	42·79	8·84	66·2	77·2
Concentrate No. 2	3·1	36·95	1·42	14·7	3·2
Tailing	84·8	1·76	0·32	19·1	19·6

Flotation with these reagents does not effect a selective separation of the arsenic and antimony minerals. The presence of cyanide seems to accelerate the flotability of the arsenic.

Section No. II—Acid Circuit

Flotation was carried out in solutions made acid with sulphuric acid added in quantities equivalent to 5·7 pounds per ton of ore. This gave the solutions a pH value of 6·0.

Test No. 1

A sample of the ore was ground with 0·25 pound of Aerofloat No. 31 to pass 78 per cent minus 200 mesh. The pulp was transferred to the flotation cell, where 5·7 pounds of sulphuric acid per ton was added. Following the addition of 0·04 pound of cresylic acid, a concentrate was removed. This concentrate was not cleaned.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	7·99	1·26	100·0	100·0
Concentrate	11·2	41·67	9·00	58·4	80·2
Tailing	88·8	3·75	0·28	41·6	19·8

Test No. 2

The Aerofloat No. 31 was replaced by Barrett No. 4 oil. Other conditions were the same as in the preceding test. The concentrate was cleaned.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	8·08	1·23	100·0	100·0
Concentrate	7·9	50·33	10·14	49·2	65·2
Middling	4·4	21·43	3·55	11·7	12·7
Tailing	87·7	3·61	0·31	39·1	22·1

Test No. 3

This test was made to note the effect of the addition of copper sulphate. In all other respects it parallels Test No. 1 with the exception that the rougher concentrate was cleaned.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	8·43	1·47	100·0	100·0
Concentrate	10·1	56·57	9·44	67·7	64·6
Middling	3·0	30·24	7·00	10·8	14·2
Tailing	86·9	2·09	0·36	21·5	21·2

The addition of copper sulphate resulted in 20·1 per cent more antimony being floated in the rougher concentrate than was recovered in Test No. 1.

Test No. 4

The effect of copper sulphate again was studied. This test is similar to Test No. 2, with the exception that 1·0 pound of copper sulphate per ton was added to the flotation cell.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	8·39	1·38	100·0	100·0
Concentrate	6·8	60·30	6·93	48·9	34·0
Middling	4·1	33·70	11·20	16·5	33·2
Tailing	89·1	3·26	0·51	34·6	32·8

Copper sulphate again slightly increases the recovery of the antimony.

A microscopic examination of the concentrate and middling showed only enough arsenopyrite to account for about 1 per cent arsenic.

Section No. III—Alkaline Circuit

In previous tests there was no indication of a separation between the antimony and arsenic minerals, either in natural or acid circuit. The following tests were made to note the behaviour of the ore when floated in alkaline solutions.

Test No. 1

This was run in a soda ash circuit. Sodium cyanide was added to depress some of the sulphides.

Soda ash equal to 6 pounds per ton and 0·3 pound of cyanide were added to the grinding mill where the sample was ground to pass 66 per cent through 200 mesh, 0·10 pound of butyl xanthate and 0·08 pound of pine oil per ton being added to the pulp in the flotation cell and a concentrate removed.

Copper sulphate, 1·0 pound per ton, was added, a second addition of butyl xanthate and pine oil was made, and a second concentrate was taken off.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	8·26	1·28	100·0	100·0
Concentrate No. 1	4·6	41·00	11·51	22·8	41·1
Concentrate No. 2	11·1	45·73	2·89	61·5	24·1
Tailing	84·3	1·54	0·53	15·7	34·8

The frothing condition in the test was poor. The concentrates tended toward a matted condition and the froth was sluggish.

It is apparent that the soda ash-cyanide combination does not effect a separation of the two minerals.

Test No. 2

The effect of adding sodium silicate to the ore was investigated.

A sample of the ore was ground 78 per cent minus 200 mesh; 1·0 pound of copper sulphate, 1·0 pound of sodium silicate, and 0·25 pound of Aerofloat No. 31 being added to the mill. After transferring to the flotation cell, 0·2 pound of potassium ethyl xanthate was added and a concentrate taken off. The concentrate was not cleaned.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	8·17	1·27	100·0	100·0
Concentrate	14·7	47·76	6·82	85·9	78·6
Tailing	85·3	1·35	0·32	14·1	21·4

Sodium silicate is apparently of some benefit as the rougher concentrate is cleaner than in preceding tests and a lower tailing loss is obtained.

Test No. 3

In Test No. 5, Section No. I, a tailing containing 1·25 per cent antimony was obtained by the use of a mixture of water-gas tar, creosote, and cresylic acid, in natural circuit.

In an endeavour to improve these results, this test was repeated with the addition of 1·0 pound of sodium silicate to the grinding mill.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	6·87	1·34	100·0	100·0
Concentrate No. 1	9·2	45·40	10·36	60·8	71·2
Concentrate No. 2	3·3	36·45	2·95	17·5	7·3
Tailing	87·5	1·70	0·33	21·7	21·5

The results show that the combination of reagents used is unsatisfactory.

Test No. 4

This test is similar to Test No. 3, with the exception that the grind was considerably finer, 88 per cent minus 200 mesh.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	6·65	1·56	100·0	100·0
Concentrate	11·7	43·05	10·52	75·8	79·0
Tailing	88·3	1·82	0·37	24·2	21·0

Fine grinding is not necessary. No improvement in results is noted.

Test No. 5

Sodium silicate was replaced by sodium hydroxide. A sample of the ore was ground with 2 pounds of caustic soda per ton, 0·25 pound of Aerofloat and 1·0 pound of copper sulphate being added to the grinding mill; 0·3 pound of potassium ethyl xanthate was added to the pulp in the flotation cell; 0·04 pound of pine oil produced a good froth. The concentrate was not cleaned.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	7·76	1·45	100·0	100·0
Concentrate	17·2	39·87	6·20	88·3	73·3
Tailing	82·8	1·10	0·47	11·7	26·7

Caustic soda has a marked effect on recovery, as the tailing from this test has the lowest antimony content yet recorded.

Part of the flotation tailing was concentrated on a Haultain super-panner. A small amount of sulphide concentrate was recovered, consisting of arsenopyrite and stibnite. Some of these grains were attached to gangue particles whereas others appeared to be coated with gangue slime. A concentrate of lower specific gravity than stibnite also was observed. This had, in the mass, a yellowish brown colour but under the microscope was seen to contain many canary-yellow coloured grains, free particles of stibnite of a dull lustre, and some partly coated grains.

The analysis of the tailing from this panning operation showed it to contain as much antimony as was in the feed.

Test No. 6

The quantity of caustic soda was increased to 3 pounds per ton and the Aerofloat No. 31 raised to 0·3 pound. Two concentrates were made. The first was made in the same way as that of Test No. 5, and cleaned. A second addition of copper sulphate and xanthate was made and a second concentrate taken off.

Results:

Product	Weight, per cent	Assay, per cent		Distribution, per cent	
		Sb	As	Sb	As
Feed (cal.)	100·0	7·02	1·44	100·0	100·0
Concentrate No. 1	5·9	30·30	2·32	25·4	9·5
Middling No. 1	5·8	10·81	4·88	8·9	19·6
Concentrate No. 2	7·6	48·70	9·85	52·7	51·9
Tailing	80·7	1·13	0·34	13·0	19·0

Apparently the increase in the quantity of caustic soda has a detrimental effect on the flotability of the stibnite.

A test was made on a sample in which lime was used as an alkaline reagent. The flotation of all sulphides was prevented. The supply of ore being exhausted, the investigation was discontinued.

SUMMARY AND CONCLUSIONS

Assuming that all the antimony is present as stibnite, a study of the analysis of the feed sample shows a deficiency of sulphur. Also, assuming the absence of all sulphides but stibnite, Sb_2S_3, calculations based on the sulphur content show a theoretical antimony content of 6·42 per cent. This leaves 0·68 per cent of antimony in a form other than stibnite.

The presence of arsenic, zinc, and copper indicates the combination of some of the sulphur with these metals, thus reducing still further the calculated quantity of stibnite. The extra antimony may exist as other compounds, possibly in an oxide form. These compounds probably are the cause of the high antimony content of the flotation tailing.

Microscopic examination of the products of Test No. 4, Section No. II, shows that although arsenopyrite was present it did not account for all the arsenic in the sample. There was 11·20 per cent of arsenic, of which approximately 1 per cent was as arsenopyrite. The major amount of the arsenic, therefore, is either combined with the stibnite or is in such a form that it cannot be distinguished from it microscopically. Concentrates panned from flotation tailing contained many canary-yellow coloured grains, which suggests the presence of realgar or orpiment.

The investigation shows that table concentration is not well adapted to this ore. The stibnite slimes easily and high tailing losses are recorded. Any gravity water concentration system should include jigs to take out coarse mineral as soon as freed.

The results of flotation show that the acid circuit is the least productive of high recoveries. This may be due to the deposition of sulphate coatings on the stibnite. This condition apparently is partly corrected by copper sulphate, as indicated in Tests Nos. 2 and 3, Section No. II. In natural and alkaline circuits, flotation results are also improved by the use of this reagent.

Soda ash apparently is not well suited to the flotation of this ore. Tests in which this reagent was used gave poor froth conditions.

Sodium silicate and caustic soda, as shown by the results in Section No. III, gave much the best results.

The reagent combinations best suited appear to be "Aerofloat" with a small addition of xanthate, or fuel oil with pine oil used as a frother. Xanthates by themselves produce poor froths that are difficult to remove.

Lime has a marked depressing action on stibnite.

Fine grinding is unnecessary. Poorer results were obtained on samples ground 88 per cent minus 200 mesh than on a 45 per cent minus 200-mesh product, as indicated in Tests Nos. 1 and 3, Section No. I, and in Tests Nos. 3 and 4, Section No. III.

The arsenic and antimony minerals do not respond to selective flotation. Tests such as No. 6, Section No. I, and No. 1, Section No. III, in which cyanide was used, show no depression of arsenic minerals. If anything the cyanide tends to increase the flotability of the arsenic compounds.

Microscopic examinations point to an intimate association of the two metals and to the presence of compounds of antimony and arsenic other than stibnite and arsenopyrite.

Recoveries of from 85 to 88 per cent of the antimony may be expected by flotation. These concentrates will contain about 10 per cent of arsenic and from 55 to 60 per cent of antimony.

Ore Dressing and Metallurgical Investigation No. 729

GOLD ORE FROM A CARBONATE ZONE, DOME MINES, LIMITED, SOUTH PORCUPINE, ONTARIO

Shipment. A sample shipment of gold ore, weight 155 pounds, was received on September 25, 1937, from the Dome Mines, Limited, South Porcupine, Ontario.

The sample was stated to come from a new development in which the ore is associated with carbonate dykes, and was submitted by C. W. Dowsett, Consulting Metallurgist, 25 King Street West, Toronto, Ontario.

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

The *gangue* consists of light grey to white vein quartz and a grey, somewhat schistose rock containing considerable finely disseminated carbonate, which is probably largely calcite.

The *metallic minerals* in the sections are as follows:

Pyrite	Moderately abundant
Chalcopyrite, Sphalerite	Small quantity
Galena, Arsenopyrite, Pyrrhotite	Traces

Pyrite occurs as moderately coarse to fine grains disseminated in both kinds of gangue. Small quantities of chalcopyrite and sphalerite occur as irregular grains in gangue, and occasional tiny grains of chalcopyrite are present in the pyrite. Rare small grains of galena are associated with sphalerite, and occasional small crystals of arsenopyrite are associated with the pyrite. Very rare tiny grains of pyrrhotite occur in the pyrite, and in some sections medium to fine irregular grains of the mineral occur in the gangue; the quantity of pyrrhotite is small.

Native gold is common in the sections. It occurs almost solely in the quartz and in the impure grey carbonate rock, often in close proximity to sphalerite and galena. The particles usually take the form of very irregular grains and fine veinlets, and as such are very difficult to measure for a grain analysis. Some of the grains are larger than 14 mesh, whereas a few are as small as 2300 mesh. By far the largest proportion are around 200 mesh or a little smaller. One grain, 6 microns (approximately 2300 mesh) in size, occurs with chalcopyrite in pyrite. With this exception all the gold seen occurs in the gangue.

Sampling and Analysis. The ore was crushed and sampled by standard methods and a feed sample cut out for analysis, the results of which are as follows:

Gold	0·45	oz./ton
Silver	0·13	"
Sulphur	2·41	per cent
Carbon dioxide	14·75	"
Pyrrhotite	0·23	"
Ferrous iron	4·79	"

It is probable that a certain small proportion of the ferrous iron is metallic iron taken up during the crushing of the ore.

Purpose of Investigation. This ore is found to be more refractory to extraction by cyanide than the normal milling ore of the Dome Mines, Limited, and the present research was carried out to determine the cause of this condition.

It is shown that 95·56 per cent of the gold can be recovered by amalgamation and cyanidation with comparatively fine grinding (90 per cent minus 200 mesh). The remaining 4·44 per cent contained in the tailing is refractory and a study of this fine locked-up gold has led to certain conclusions regarding its size and distribution in the ore.

Extremely fine grinding (94 per cent minus 325 mesh) fails to free this gold and render it amenable to attack by cyanide.

Concentration of the gold in the cyanide tailing by flotation, and also flotation of the raw ore, were carried out with a small measure of success. Difficulty was encountered in making a clean concentrate, owing to the tendency of the carbonate constituent of the gangue to float along with the sulphides.

The report is divided into Parts I and II, the former dealing with a general investigation of the ore and the latter dealing with a particular study of the refractory gold.

Part I

Preliminary investigations consisted of treating the ore approximately by the standard Dome practice of blankets, amalgamation, and cyanidation. Analyses were made of all solutions to determine any tendency to fouling, and comparative tests were run with and without pre-aeration. The results show a minimum tailing containing 0·02 ounce of gold per ton and no serious tendency towards fouling of the solution.

The results of these tests follow in detail:

CONCENTRATION, AMALGAMATION, CYANIDATION

Test No. 1

A sample of ore, 2,000 grammes, was ground in a water pulp (0·75 : 1 dilution) in a small ball mill for 15 minutes, and the pulp was passed over a corduroy blanket.

The blanket concentrate was amalgamated. The amalgamation tailing was combined with the dewatered blanket tailing and reground with 5 pounds of lime per ton for a period of 10 minutes. The reground pulp was transferred to a Denver super-agitator and aerated for 15 hours.

The aerated pulp was filtered and two portions of the dewatered pulp were repulped in cyanide solution of strength equivalent to 1 pound of potassium cyanide per ton and agitated for 24 and 48 hours respectively.

Amalgamation Results:

Feed to blankets—Au	0·45 oz./ton
Gold in combined amalgamation and blanket tailings	0·24 "
Recovery	46·7 per cent

Cyanidation Results:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
24	0·24	0·04	83·33	0·32	2·18	1·6 : 1
48	0·24	0·025	89·58	0·70	2·36	1·6 : 1

Overall Recovery—	Per cent
24-hour agitation	91·14
48-hour agitation	94·45

The fineness of the ore after the secondary grinding is indicated by the following screen test:

Screen Test:

Mesh	Weight, per cent
+100	0·4
−100+150	4·2
−150+200	9·8
−200	85·6
	100·0

Analysis of Solutions:

—	Primary grinding solution	Aerated solution
Reducing power, c.c. $\frac{N}{10}$ $KMnO_4$/litre	1·2	4·6
Ferrous iron, grm./litre	Nil	Nil
pH	7·6	10·2

Test No. 2

This test was carried out similarly to Test No. 1. The primary grinding was for 30 minutes and the secondary for a 10-minute period. The amalgamation results indicated a recovery of 50·0 per cent of the gold.

Cyanidation Results:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
24	0·225	0·025	88·9	0·37	2·20	1·6 : 1
48	0·225	0·02	91·1	0·74	2·49	1·6 : 1

Overall Recovery—	Per cent
24-hour agitation	94·45
48-hour agitation	95·56

The fineness of the ore after the secondary grinding is indicated by the following screen test:

Screen Test:

Mesh	Weight, per cent
+150	0·8
−150+200	3·8
−200	95·4
	100·0

Analysis of Solutions:

—	Primary grinding solution	Aerated solution	Pregnant cyanide solution
Reducing power, c.c. $\frac{N}{10}$ $KMnO_4$/litre	0·8	9·6	90·0
KCNS, grm./litre	....	....	0·116
Ferrous iron, grm./litre	Nil	Nil	Nil
Ferric iron, grm./litre	Nil	Nil	....
pH	8·0	9·1	....

As a further check of the cyanide consumption and the fouling of the solution, two comparative cyanidation tests were carried out. In the first the ore was ground in water, filtered, and repulped in cyanide solution; and in the second the ground pulp was aerated for 7 hours before agitation in cyanide. The results indicate that pre-aeration lowers appreciably the cyanide consumption. The data are given below:

Test No. 5—Without Pre-aeration

Cyanidation Results:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
24	0·45	0·035	92·22	0·26	2·48	1·3 : 1
48	0·45	0·03	93·33	0·33	2·48	1·3 : 1

Analysis of Solutions:

—	Grinding solution	Pregnant cyanide solution	
		24 hours	48 hours
Reducing power, c.c. $\frac{N}{10}$ $KMnO_4$/litre	4·0	51·0	94·0
KCNS, grm./litre	....	0·077	0·12
Copper, grm./litre	....	0·02	0·03
Ferrous iron, grm./litre	....	Nil	Nil

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

Mesh	Weight, per cent
+ 65	0·9
− 65+100	4·1
−100+150	10·7
−150+200	10·3
−200	74·0
	100·0

Test No. 4—With 7 Hours' Pre-aeration

The ore was ground for the same period as in Test No. 3 in a lime pulp of 5 pounds of lime per ton and aerated for 7 hours in a pulp dilution of 2 : 1. The pulp was filtered and the ore repulped for cyanidation.

Cyanidation Results:

Agitation hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
24	0·45	0·045	90·00	0·15	1·72	1·5 : 1
48	0·45	0·025	94·44	0·15	2·02	1·5 : 1

Analysis of Solutions:

—	Pre-aerated solution	Pregnant cyanide solution	
		24 hours	48 hours
Reducing power, c.c. $\frac{N}{10}$ $KMnO_4$/litre	11·0	40·0	76·0
KCNS, grm./litre	....	0·064	0·107
Copper, grm./litre	....	0·016	0·022
Ferrous iron, grm./litre	....	Nil	Nil

Although aeration appears to increase the reducing power of the solution in a water pulp, it apparently reduces the tendency to fouling during cyanidation.

The tests show that pre-aeration has a beneficial effect on cyanide consumption, but has no influence on the gold extraction.

Test No. 5

This test was to determine if very fine grinding of the ore would increase the extraction. A sample of ore was ground in a water pulp to a fineness of 94 per cent minus 325 mesh. The pulp was filtered and repulped in cyanide solution of strength equivalent to 1 pound of potassium cyanide per ton and agitated for 48 hours. The tailing assayed 0·02 ounce of gold per ton, indicating an extraction of 95·56 per cent. The result shows that extremely fine grinding does not increase the extraction.

It is apparent from the preceding tests that this 0·02 ounce of gold per ton represents refractory gold in the ore.

Part II of this report deals with an investigation of the refractory gold.

SETTLING TESTS

The rate of settling of the solids in finely ground pulp was tested by observing the fall of the pulp level of the material in a glass tube 4 feet long and 2 inches inside diameter.

Readings were taken at 5-minute intervals over a period of 1 hour.

Test A

Grinding	93·00 per cent—200 mesh
Pulp dilution	1·5 : 1
Titrations after 1 hour—	
KCN	0·92 lb./ton
CaO	0·30 "
Rate of settling	0·29 ft./hour
Conditions of settling	Uniform
Overflow solution	Trace of cloudiness

Test B

Grinding	93·00 per cent—200 mesh
Pulp dilution	2 : 1
Titrations after 1 hour—	
KCN	0·64 lb./ton
CaO	0·20 "
Rate of settling	0·43 ft./hour
Conditions of settling	Slightly slower during final 20 minutes
Overflow solution	Trace of cloudiness

FLOTATION

Flotation tests were carried out both on cyanide tailing and the raw ore. The production of a clean concentrate is made very difficult by the high carbonate content of the gangue. The use of caustic soda in place of sodium carbonate as a modifying agent and additions of sodium silicate as a dispersant improve somewhat the grade of the concentrate.

Investigations of the raw ore were limited owing to the depletion of the ore sample.

The results of flotation are discussed as follows:

Test A

A cyanide tailing was conditioned with 2 pounds of sodium hydroxide per ton, 2 pounds of copper sulphate per ton, and 0·2 pound of amyl xanthate per ton; 0·025 pound of pine oil per ton being used as frother.

Results:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	S	CO_2	Au	S	CO_2	
Feed	100·00	0·026	2·19	13·78	100·0	100·0	100·0	
Concentrate	20·35	0·07	9·10	21·44	54·4	84·7	31·7	4·91 : 1
Tailing	79·65	0·015	0·42	11·82	45·6	15·3	68·3	

The concentrate is poor grade and the ratio of concentration is low.

Test B

In this test soda ash, 2 pounds per ton, was used as a modifying agent. The concentrate assayed 0·07 ounce of gold per ton, indicating a recovery of only 37·0 per cent of the gold. The tailing carried 0·018 ounce of gold per ton.

Test C

A cyanide tailing was floated using the following reagents:

	Lb./ton
Sodium hydroxide	2
Sodium silicate	3
Copper sulphate	1
Amyl xanthate	0·2
Cresylic acid	0·35
Pine oil	0·05

The concentrate was cleaner and the ratio of concentration improved.

Results:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	S	CO_2	Au	S	CO_2	
Feed	100·00	0·03	2·08	14·23	100·0	100·0	100·0	9·12 : 1
Concentrate	10·96	0·215	16·70	19·51	72·6	88·0	15·0	
Tailing	89·04	0·01	0·28	13·58	27·4	12·0	85·0	

Test D

Feed—Cyanide tailing

Reagents:

	Lb./ton
Sodium hydroxide	3
Sodium silicate	4
Copper sulphate	2
Amyl xanthate	0·2
Potassium xanthate	0·1
Pine oil	0·05
Cresylic acid	0·79

Results:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	S	CO_2	Au	S	CO_2	
Feed	100·00	0·026	2·40	14·13	100·0	100·0	100·0	11 : 1
Concentrate	9·09	0·19	19·96	19·90	65·5	75·7	12·8	
Tailing	90·91	0·01	0·64	13·35	34·5	24·3	87·2	

Test E

Feed—Cyanide tailing

Reagents:

	Lb./ton
Sodium hydroxide	2
Sodium silicate	4
Copper sulphate	2
Potassium xanthate	0·4
Cresylic acid	0·528
Pine oil	0·10

Results:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	S	CO_2	Au	S	CO_2	
Feed	100·00	0·027	2·43	14·23	100·0	100·0	100·0	
Concentrate	7·05	0·19	25·04	17·20	49·0	72·5	8·5	14·2 : 1
Tailing	92·95	0·015	0·72	14·00	51·0	27·5	91·5	

The results of these tests show a cleaner concentrate and an improved ratio of concentration. The grade of the concentrate is still low.

Test F

In this an attempt was made to improve the grade of concentrate by cleaning. Three concentrates were combined and re-floated using the reagents as used in the above flotation tests. The results were not encouraging.

Results:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	S	CO_2	Au	S	CO_2	
Feed	100·00	0·19	20·13	17·76	100·0	100·0	100·0	
Cleaner concentrate	58·54	0·24	26·93	13·79	75·5	78·3	45·4	1·7 : 1
Cleaner middling	41·46	0·11	10·53	23·37	24·5	21·7	54·6	

FLOTATION OF RAW ORE

Owing to depletion of the ore sample only two flotation tests were made.

Test G

A sample of ore, weight 2,000 grammes, was ground in a water pulp to approximately 90 per cent minus 200 mesh. The pulp was passed over a corduroy blanket to remove coarse free gold and the blanket tailing was thickened and floated.

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·42	100·0	
Concentrate	0·18	63·54	27·0	555 : 1
Tailing	99·82	0·31	73·0	

Flotation of Blanket Tailing:

Reagents:

	Lb./ton
Sodium hydroxide	2
Sodium silicate	3
Copper sulphate	1·5
Amyl xanthate	0·2
Cresylic acid	0·41
Pine oil	0·05

Results:

Product	Weight, per cent	Assay				Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent						
		Au	S	CO_2	Cu	Au	S	CO_2	
Feed	100·00	0·31	2·40	14·09	....	100·0	100·0	100·0	
Concentrate	8·39	3·14	22·64	19·40	0·26	85·2	79·0	11·6	11·9 : 1
Tailing	0·05	0·05	0·55	13·60	....	14·8	21·0	88·4	

Overall recovery.. 89·2 per cent

Test H

In this the ground pulp was first run over a corduroy blanket as in *Test G* and the tailing was floated. The sodium hydroxide and sodium silicate were reduced to 1 pound per ton and 0·088 pound of Barrett No. 4 oil per ton was added along with amyl xanthate, and a concentrate was taken off using 0·176 pound of cresylic acid per ton and 0·025 pound of pine oil per ton. Before the end of the run 0·5 pound of copper sulphate per ton and an additional 0·1 pound of xanthate per ton were added to the cell. The reason for adding the copper sulphate at the end of the flotation test was to minimize any tendency of the copper sulphate to depress fine free gold.

The flotation results are as follows:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	S	CO_2	Au	S	CO_2	
Feed	100·00	0·26	2·36	13·87	100·0	100·0	100·0	
Concentrate	9·48	2·30	21·00	20·25	84·3	84·3	13·8	10·5 : 1
Tailing	90·52	0·045	0·41	13·20	15·7	15·7	86·2	

Overall recovery.. 89·4 per cent

The making of a clean sulphide concentrate is rendered difficult by the carbonate in the gangue. The sodium silicate and sodium hydroxide used serve as partial depressants for the carbonates, but the results obtained show that a considerable proportion of the carbonates persist in floating with the sulphides.

The flotation tests on the raw ore were limited owing to depletion of the sample and no comments on the results other than the above are made.

Part II

This part of the report covers a study of the refractory gold, and deals with results obtained by chemical means and by infrasizing and superpanning the minimum tailing obtained from cyanidation, in order to determine its association with the sulphides and gangue constituents of the ore.

A part of a cyanide tailing carrying 0·025 ounce of gold per ton was floated, giving a flotation tailing of 0·015 ounce of gold per ton. Two parts of this tailing of one assay ton each were treated with aqua regia and the leached residues were assayed. The residues carried 0·005 ounce of gold per ton. This indicates that about 33 per cent of the refractory gold in this tailing is in the silica gangue, and the remaining 67 per cent is in either the sulphides or carbonates. The gold contained in the aqua regia residues represents, at this grinding size, the gold that cannot be recovered by either cyanide or flotation, this part of the refractory gold being locked up in grains of quartz.

As the microscopic examination of the ore shows that gold occurs in the carbonate, a test was carried out to determine if any of the refractory gold was locked up in the carbonate minerals.

A sample of cyanide tailing, assaying in gold 0·025 ounce per ton, was treated with dilute acid to dissociate the carbonates, and after thorough washing the residue was treated with cyanide for 24 hours.

A blank cyanide test of the original cyanide tailing was run at the same time. The extraction in the blank test was nil, whereas the acid-leached residue showed an extraction of 16·6 per cent. This indicates that possibly around 16 per cent of the refractory gold is contained in the carbonates. During the weak acid-leach hydrogen sulphide was evolved, indicating that the pyrrhotite was attacked, and there is the possibility that some of the gold freed is contained in this mineral. The amount of pyrrhotite, however, in the ore is small in comparison, being less than one-quarter of 1 per cent. Subsequent tests by infrasizing and superpanning deal more fully with the relationship of the gold and sulphur.

SUPERPANNING AND INFRASIZING

Superpanning was carried out on several low-gold unsized tailings, primarily to determine the grade of the sulphides obtained and to provide a product for microscopic examination.

In the first test a cyanide tailing carrying 0·02 ounce of gold per ton was floated and gave a tailing of 0·018 ounce of gold per ton. This flotation tailing was panned on a Haultain superpanner with the following results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	0·018	100·00
Sulphides	0·56	0·375	11·58
Panner middling	3·22	0·05	8·87
Panner slime	96·22	0·015	79·55

Sulphur in panner slime.. 1·04 per cent

The results indicate an appreciable concentration of gold in the sulphides. The loss of gold in the slime is high and extremely fine sulphides, no doubt, account for a certain proportion of this gold.

Another panner test was made to obtain free sulphides for a microscopic examination. The product used was a cyanide tailing that assayed 0·02 ounce of gold per ton and had a fineness of grinding of 94 per cent minus 325 mesh. The panned products were assayed for gold and sulphur and the results are given below:

Product	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S
Feed	100·00	0·025	2·37	100·00	100·00
Sulphides	1·15	0·78	52·00	35·56	25·22
Panner middling	28·57	0·02	2·69	22·65	32·41
Panner slime	70·28	0·015	1·43	41·79	42·37

The ratio of the gold to sulphur as shown by the assays is as follows:

In feed...1 : 94·8
In sulphides..1 : 66·7
In middling..1 : 134·5
In slime...1 : 95·3

A portion of the sulphide concentrate was mounted in bakelite and polished. A microscopic examination was made and at the highest possible magnification no gold was visible, so if the gold is in the sulphides as free metallic gold the grain size is less than 0·1 micron. This is conclusive evidence that the gold tied up in the sulphides cannot be freed by any economic method of grinding.

INFRASIZING TESTS

The information already gained has shown that the refractory gold occurs both in the sulphides (largely pyrite) and the gangue.

In order to determine its relative occurrence at different grain sizes a cyanide tailing was infrasized in a Haultain infrasizer.

The cyanide tailing, assaying 0·02 ounce of gold per ton, was dried and thoroughly mixed by running through a Jones riffle sampler. Two samples of 400 grammes each were screened on a 200-mesh screen and the minus 200-mesh portions infrasized.

Each fraction was panned separately on a Haultain superpanner into a clean sulphide concentrate and a gangue tailing.

The products were assayed for gold and sulphur and the gold and sulphur assays of the combined sulphide concentrates were calculated. Assuming these assays to represent the relationship between the gold and sulphur the ounces per ton of gold for each 1 per cent of sulphur were found. Using this figure the relationship between the gold, sulphides, and gangue in each fraction was obtained.

The results are shown in the following table:

The gold and sulphur contents of the combined superpanner concentrates of infrasizer fractions are as follows:

Gold..	0·314 oz./ton
Sulphur..	40·6 per cent

Assuming that the gold in the sulphides is represented by the ratio 0·314:40·6, each 1·0 per cent of sulphur will be associated with 0·0077 ounce of gold per ton.

Infrasizer Results:

Grain size, microns	Weight, per cent	Assay		Distribution, per cent		Gold in sulphides, oz./ton	Gold in gangue, by difference
		Au, oz./ton	S, per cent	Gold	Sulphur		oz./ton
above 56..	5·3*	0·059	5·67	12·8	12·1	0·044	0·015
56 to 40..	25·8	0·023	1·70	28·4	17·6	0·013	0·010
40 to 28..	20·4	0·020	2·21	16·7	18·1	0·017	0·003
28 to 20..	12·0	0·026	3·09	12·8	14·8	0·024	0·002
20 to 14..	8·5	0·026	3·37	9·1	11·5	0·026	0·000
14 to 10..	7·5	0·022	2·52	6·8	7·6	0·019	0·003
below 10..	20·5	0·016	2·23	13·4	18·3	0·017	0·000
Totals....	100·0	0·022	2·49	100·0	100·0	per cent 82·6	per cent 17·4

* Includes the plus 200-mesh product recovered by screenings.

The results obtained indicate very clearly that the gold in the fractions is largely in the sulphides. Below 40 microns the refractory gold in the gangue has apparently been appreciably liberated.

The relationship of the gold and pyrite to grain size bears out the evidence previously obtained from the microscopic examination of the sulphides, which indicated that the gold was too fine to be observed at the highest degree of magnification.

DISCUSSION OF RESULTS

By amalgamation and cyanidation with moderately fine grinding 95·56 per cent of the gold in the ore can be extracted. The remaining 4·44 per cent is refractory to cyanide even at a fineness of grinding of 94 per cent minus 325 mesh (44 microns).

The cyanide consumption is slightly higher than the normal. This is due to the small amounts of pyrrhotite, chalcopyrite, and possibly some of the carbonates, which promote a fouling condition in the solution. Pre-aeration of the pulp reduces the tendency for increased consumption of cyanide to a reasonably low figure.

The infrasizer tests carried out on cyanide tailing (see table on infrasizer results) show that 87·2 per cent of the refractory gold occurs in material minus 56 microns in size; and 13·4 per cent occurs in the minus 10-micron fraction. It is evident that no economic degree of grinding will liberate this gold.

The results indicate that 82·6 per cent of the refractory gold is in the sulphides and 17·4 per cent in the gangue. They show clearly the relationship of the gold to the sulphide and gangue and its distribution in grain size of the ore.

The concentration of the sulphides in the cyanide tailing by flotation showed that although a concentrate could be made its grade was probably too low for successful roasting. The difficulty in improving this condition lay largely in the carbonates. The refractory gold in the gangue cannot be recovered by flotation.

The present investigation has determined the nature of the gold in the ore, especially the refractory part and has shown conclusively that the lowest tailing obtainable by cyanidation is 0·02 ounce of gold per ton.

Further investigation by flotation and roasting was not possible because of depletion of the ore sample.

Ore Dressing and Metallurgical Investigation No. 730

MILL TAILING FROM THE LEITCH GOLD MINES, LIMITED, BEARDMORE, ONTARIO

Shipment. A 6-pound sample of mill tailing, received on October 9, 1937, from W. R. Dennis, Mill Superintendent, Leitch Gold Mines, Limited, Beardmore, Ontario, was said to be a composite of the current mill tailing for the month of September, 1937. The shipment was made so that the sample might be subjected to separation by the Haultain infrasizer and superpanner, and further information gained concerning the mill practice.

The sample was assayed and found to contain:

Gold	0·022	oz./ton
Arsenic	0·16	per cent
Sulphur	0·65	"

Two 300-gramme representative portions of the sample were screened on 200 mesh and the minus fractions run through the infrasizer, which divided them as follows:

Size	Weight, per cent	
	Sample No. 1	Sample No. 2
+200 mesh	23·37	22·77
above 56 microns	0·67	0·77
56 to 40 "	9·73	9·90
40 to 28 "	10·63	10·93
28 to 20 "	8·73	8·83
20 to 14 "	7·40	7·37
14 to 10 "	6·23	6·17
below 10	33·24	33·26
Total	100·00	100·00

The plus 200-mesh and minus 200-mesh plus 56-micron portions were combined, and then the different sizes were panned on the superpanner and practically all the sulphides removed as a clean concentrate. No free gold was seen in any of the concentrates. Because of the small weights of the concentrates, all seven were combined for analysis. Each tailing was also analysed for gold, arsenic, and sulphur.

The ratio of gold to sulphur in the concentrate was calculated, and this ratio was applied to the sulphur analysis of the tailing. This gave the weight of gold associated with the sulphides in these products. This figure subtracted from the gold assays left the weight of gold in the gangue.

Results:

Product	Weight, per cent	Assay			Distribution, per cent			Distribution of gold, per cent	
		Oz./ton	Per cent						
		Au	As	S	Au	As	S	In sulphides	In gangue
Concentrates	1·1	0·825	9·72	0·46	41·8	66·4	50·7	41·8	Nil
Tailing above 56 microns	24·2*	0·015	0·03	0·18	16·7	4·5	6·6	5·6	11·1
" 56 to 40 "	9·7	0·015	0·03	0·50	6·7	1·8	7·3	6·3	0·4
" 40 to 28 "	10·7	0·01	0·06	0·40	4·9	4·0	6·5	4·9	Nil
" 28 to 20 "	8·8	0·018	Nil	0·28	7·3	....	3·8	3·2	4·1
" 20 to 14 "	7·2	0·015	0·03	0·23	5·0	1·3	2·5	2·0	3·0
" 14 to 10 "	6·0	0·01	0·05	0·28	2·8	1·9	2·5	2·2	0·6
" below 10	32·3	0·01	0·10	0·41	14·8	20·1	20·1	14·8	Nil
Totals	100·0	0·022	0·16	0·66	100·0	100·0	100·0	80·8	19·2

* 23·5 per cent of the total weight is plus 200 mesh.

CYANIDATION

The distribution of the gold in the sample points to the advisability of finer grinding of the plus 200-mesh portion of the gangue and concentration and regrinding of the sulphides.

As the proportion of sulphides in the ore is low, and the amount of the sample on hand was limited, no concentration test was made. Tests were made to determine if the gold content could be reduced by cyanidation.

One portion of the sample was reground in cyanide solution until 99 per cent passed through 200 mesh and was agitated for 24 hours.

A second portion was agitated for 24 hours without further grinding.

The tailing from both these tests had a gold content of 0·02 ounce per ton, showing an additional extraction of 0·002 ounce per ton.

As the supply of tailing was exhausted no further tests were made.

SUMMARY AND CONCLUSIONS

It is apparent that the larger part of the gold in the tailing is associated with the sulphides. A total of 80·8 per cent is found with the sulphides; 41·8 per cent in the clean sulphide concentrate; and 11·1 per cent of the gold in the gangue portion of the plus 56-micron material. This points to the advisability of finer grinding, and elimination of the plus 200-mesh material from the classifier overflow.

The cyanide tests did not indicate any benefit from regrinding the whole sample. However, as the major amount of the gold is associated with the sulphides, selective grinding of these should be practised. A system of classification whereby the sulphides are returned to the mill for further reduction in size should result in a lower tailing. This coupled with the elimination of plus 200-mesh gangue from the agitator feed should improve the gold recovery.

Ore Dressing and Metallurgical Investigation No. 731

COPPER-LEAD-ZINC ORE FROM THE STIRLING MINE, NOVA SCOTIA

Shipment. One carload of ore, containing some 49 tons, was received on August 13, 1937, from the Stirling mine, Stirling, Nova Scotia. It was shipped by the British Metal Corporation, Limited, Dominion Square Building, Montreal, Quebec. The ore was similar in character to that investigated in 1934, covered in Investigation No. 565, Mines Branch Report No. 747, Ore Dressing and Metallurgy, January to June, 1934.

This shipment was made to check the results obtained in the previous investigation and to determine, if possible, the reasons why these results were not obtained in the Stirling mill.

Investigative Work. As in previous research work, and in the current mill practice at the property, a talc concentrate was removed, followed by the selective concentration of the copper-lead sulphides. A zinc concentrate was obtained from the copper-lead tailing.

In each run the ore was fed continuously for about 8 hours at the rate of 500 pounds per hour to a ball mill in closed circuit with a classifier. The classifier overflow passed to a conditioning tank, and from there flowed to Cell No. 1 of a 10-cell flotation unit. The concentrate from the first five cells constituted the talc concentrate. That produced by the last five cells was returned with the feed to Cell No. 1. The talc concentrate was passed over a "Plato" concentrating table where sulphides carried by the talc were recovered. In the first two days' runs this concentrate was returned to the classifier overflow. Later on, it was sent to the copper-lead regrind mill.

The tailing from the talc cells passed to conditioning tank No. 2, where air was introduced at the bottom of the tank. The pulp discharged into Cell No. 4 of a 10-cell unit. The concentrate from Cells Nos. 4 and 5 was cleaned in Cells Nos. 2 and 3, and recleaned in Cell No. 1. This product constituted the copper-lead concentrate. The concentrate from Cells Nos. 5 to 10 was returned with the feed to Cell No. 4. The cleaner tailing from Cell No. 3 was pumped to a Genter thickener, which discharged into a small regrind mill. This reground middling product was refloated with the feed to the copper-lead circuit.

The tailing from this circuit was pumped to conditioning tank No. 3 and from there was fed to Cell No. 4 of a 10-cell zinc flotation unit. The concentrate from Cells Nos. 4 to 10 was sent to Cells Nos. 2 and 3 for cleaning. This concentrate was recleaned in Cell No. 1, producing the finished zinc product.

For the first eight runs the reagents were unchanged, with the exception that sodium silicate was added after the second day's run. Variations in the operation of the cells were confined to the amounts of talc concentrate and lead-copper middling that were taken off. The degree of grinding as shown by screen tests on the classifier overflow varied from 69 per cent to 78 per cent minus 200 mesh. The pulp density to the talc cells was held to about 35 per cent solids. The entire shipment was not sampled, but in each day's run samples of the feed and of the classifier overflow were taken.

The following reagents were added in all runs, unless otherwise stated. This reagent combination is that in use at the mine.

	Lb./ton feed
To Ball Mill:	
Soda ash	1·6
Zinc sulphate	1·6
Cyanide	0·06
To Talc Float:	
To conditioning tank, cresylic acid	0·115
To Cell No. 5, " "	0·069
To Cell No. 7, " "	0·023
To Copper-lead Float:	
To conditioning tank—	
Cyanide	0·016
Butyl xanthate	0·081
Sodium silicate	0·53
To Cell No. 7, butyl xanthate	0·081
To Zinc Float:	
To copper-lead tailing, copper sulphate	1·06
To conditioning tank, lime	1·1
To zinc cell feed,—	
Potassium ethyl xanthate	0·08
Pine oil	0·03
To Lead-copper Middling Regrind:	
Zinc sulphate	0·53
Cyanide	0·016

Mill Run No. 1

The first day's run was devoted to tuning up the circuit. No results are reported.

Mill Run No. 2

In this test no sodium silicate was used. The grind was 71·9 per cent minus 200 mesh.

Assays:

—	Copper, per cent	Lead, per cent	Zinc, per cent	Insoluble, per cent
Feed	0·70	1·51	6·16	....
Classifier overflow	0·58	1·59	5·46	....
Talc concentrate	0·18	0·56	2·22	....
Talc tailing	0·75	1·61	5·86	....
Wilfley table concentrate	0·51	1·68	7·80	....
Wilfley table tailing	0·17	0·72	1·52	....
Pb-Cu concentrate	9·24	22·90	12·54	9·24
Pb-Cu tailing	0·17	0·25	6·10	....
Pb-Cu middling to regrind	1·80	4·24	17·00	....
Zn concentrate	0·31	0·79	54·75	5·28
Zn tailing	0·10	0·15	0·50	....

Results:

Product	Weight, per cent	Assays, per cent			Distribution, per cent		
		Cu	Pb	Zn	Cu	Pb	Zn
Pb-Cu concentrate................	4·52	9·24	22·90	12·54	76·2	74·6	11·2
Zn concentrate....................	6·87	0·31	0·79	54·75	3·9	3·9	74·2
Zn tailing........................	59·66	0·10	0·15	0·50	10·9	6·5	5·9
Wilfley table tailing..............	28·95	0·17	0·72	1·52	9·0	15·0	8·7

Mill Run No. 3

In this run, sodium silicate was added to the conditioning tank of the copper-lead circuit. The grind was 77·5 per cent minus 200 mesh.

Assays:

—	Copper, per cent	Lead, per cent	Zinc, per cent	Insoluble, per cent
Classifier overflow............................	0·69	1·55	6·07	....
Talc concentrate...............................	0·22	0·81	2·22	....
Talc tailing.....................................	0·80	1·62	6·67	....
Wilfley table concentrate.....................	0·49	1·52	7·38	....
Wilfley table tailing...........................	0·22	0·89	1·77	....
Pb-Cu concentrate............................	15·78	28·40	4·25	7·11
Pb-Cu tailing..................................	0·21	0·33	7·00	....
Pb-Cu middling to regrind....................	3·73	11·00	14·67	....
Zn concentrate................................	0·50	1·04	51·51	5·32
Zn tailing......................................	0·21	0·25	0·58	....

Results:

Product	Weight, per cent	Assays, per cent			Distribution, per cent		
		Cu	Pb	Zn	Cu	Pb	Zn
Classifier overflow................	100·00	0·69	1·55	6·07	100·0	100·0	100·0
Pb-Cu concentrate................	3·09	15·78	28·40	4·25	67·6	66·8	2·2
Zn concentrate....................	9·95	0·50	1·04	51·51	6·9	7·9	85·8
Zn tailing........................	68·96	0·21	0·25	0·58	20·1	13·1	6·7
Wilfley table tailing..............	18·00	0·22	0·89	1·77	5·4	12·2	5·3

Mill Run No. 4

In this run, a heavier concentrate was taken off the copper-lead cells. The grind was 70·3 per cent minus 200 mesh.

Assays:

—	Copper, per cent	Lead, per cent	Zinc, per cent	Insoluble, per cent
Classifier overflow	0·77	1·62	6·67	....
Talc concentrate	0·27	0·76	2·25	....
Talc tailing	0·79	1·67	6·90	....
Wilfley table concentrate	0·50	1·40	7·20	....
Wilfley table tailing	0·24	0·74	1·92	....
Cu-Pb concentrate	13·68	26·80	5·56	10·28
Cu-Pb middling to regrind	3·16	11·76	16·60	....
Cu-Pb tailing	0·28	0·38	6·90	....
Zn concentrate	0·48	0·89	57·90	3·03
Zn tailing	0·22	0·25	0·83	....

Results:

Product	Weight, per cent	Assays, per cent			Distribution, per cent		
		Cu	Pb	Zn	Cu	Pb	Zn
Classifier overflow	100·00	0·77	1·62	6·67	100·0	100·0	100·0
Pb-Cu concentrate	3·59	13·68	26·80	5·56	68·2	74·7	3·1
Zn concentrate	9·55	0·48	0·89	57·90	6·3	6·6	85·0
Zn tailing	81·85	0·22	0·25	0·83	23·8	15·9	10·4
Wilfley table tailing	5·01	0·24	0·74	1·92	1·7	2·8	1·5

Mill Run No. 5

More talc concentrate was taken off in this test than was produced in the previous runs. The grind was 78 per cent minus 200 mesh.

Assays:

—	Copper, per cent	Lead, per cent	Zinc, per cent	Insoluble, per cent
Classifier overflow	0·71	1·62	6·46	....
Talc concentrate	0·18	0·76	2·17	....
Talc tailing	0·77	1·67	7·10	....
Wilfley table concentrate	0·39	1·42	7·00	....
Wilfley table tailing	0·15	0·76	1·87	....
Cu-Pb concentrate	10·98	24·74	8·50	10·74
Cu-Pb middling to regrind	2·57	8·51	16·20	....
Cu-Pb tailing	0·17	0·33	7·00	....
Zn concentrate	0·50	1·04	52·32	5·08
Zn tailing	0·15	0·23	0·66	....

Results:

Product	Weight, per cent	Assays, per cent			Distribution, per cent		
		Cu	Pb	Zn	Cu	Pb	Zn
Classifier overflow	100·00	0·71	1·62	6·46	100·0	100·0	100·0
Cu-Pb concentrate	4·82	10·98	24·74	8·50	74·8	76·5	6·4
Zn concentrate	10·17	0·50	1·04	52·32	7·2	6·8	82·6
Zn tailing	72·74	0·15	0·23	0·66	15·4	10·7	7·5
Wilfley table tailing	12·27	0·15	0·76	1·87	2·6	6·0	3·5

Mill Run No. 6

This run was continuous for 24 hours with each 8-hour shift sampled separately. The copper-lead cells were pulled quite strongly.

Results:

Product	Weight, per cent	Assays, per cent			Distribution, per cent		
		Cu	Pb	Zn	Cu	Pb	Zn
Shift No. 1: Grind—67·2 per cent minus 200 mesh							
Classifier overflow	100·00	0·67	1·52	6·37	100·0	100·0	100·0
Cu-Pb concentrate	5·29	11·02	22·50	10·42	77·1	73·6	8·1
Zn concentrate	10·69	0·36	0·84	51·60	5·1	5·6	80·7
Zn tailing	76·90	0·15	0·33	0·80	15·3	15·7	9·0
Wilfley table tailing	7·12	0·27	1·17	2·10	2·5	5·1	2·2
Shift No. 2: Grind—81·1 per cent minus 200 mesh							
Classifier overflow	100·00	0·76	1·52	6·28	100·0	100·0	100·0
Cu-Pb concentrate	4·77	10·90	24·23	11·54	72·4	75·4	7·9
Zn concentrate	11·17	0·56	1·04	49·79	8·7	7·6	80·1
Zn tailing	67·37	0·14	0·20	0·68	13·1	8·8	6·6
Wilfley table tailing	16·69	0·25	0·76	2·23	5·8	8·2	5·4
Shift No. 3: Grind—81·2 per cent minus 200 mesh							
Classifier overflow	100·00	0·76	1·65	6·74	100·0	100·0	100·0
Cu-Pb concentrate	6·13	8·00	19·90	18·70	78·6	77·1	17·2
Zn concentrate	9·66	0·40	0·79	51·40	6·2	4·8	74·6
Zn tailing	70·59	0·09	0·25	0·40	10·2	11·1	4·2
Wilfley table tailing	13·62	0·23	0·81	1·94	5·0	7·0	4·0

In these runs, the quantity of xanthate fed to the copper-lead float was a little too much. This resulted in a larger quantity of concentrate being produced containing more zinc and insoluble than in previous runs.

Mill Run No. 7

This run is a continuation of the preceding one, the mill having been idle from Saturday noon to Monday morning. The grind was 69·3 per cent minus 200 mesh.

Assays:

—	Copper, per cent	Lead, per cent	Zinc, per cent	Insoluble, per cent
Classifier overflow	0·67	1·32	6·84	....
Talc concentrate	0·28	1·32	3·07	....
Talc tailing	0·76	1·42	7·05	....
Wilfley table concentrate	0·65	1·82	10·52	....
Wilfley table tailing	0·33	1·22	2·76	....
Pb-Cu concentrate	8·34	17·16	15·84	7·22
Pb-Cu middling	1·10	2·18	10·42	....
Pb-Cu tailing	0·14	0·18	6·74	....
Zn concentrate	0·34	0·45	53·35	3·96
Zn tailing	0·07	0·20	0·56	....

Results:

Product	Weight, per cent	Assays, per cent			Distribution, per cent		
		Cu	Pb	Zn	Cu	Pb	Zn
Classifier overflow	100·00	0·67	1·32	6·84	100·0	100·0	100·0
Cu-Pb concentrate	6·91	8·34	17·16	15·84	84·3	81·6	15·3
Zn concentrate	10·26	0·34	0·45	53·35	5·2	3·2	76·5
Zn tailing	77·40	0·07	0·20	0·56	7·9	10·7	6·1
Wilfley table tailing	5·43	0·33	1·22	2·76	2·6	4·5	2·1

Mill Run No. 8

In Runs Nos. 8 to 12* an attempt was made to produce the same flotation conditions as those in the Stirling mill.

In this run, the sodium silicate to the copper-lead circuit was omitted, also the air to the conditioning tank. The butyl xanthate in this circuit was reduced by 0·027 pound per ton. The speed of the impellers in the flotation cells was reduced and the ball mill discharge thickened to 80 per cent solids.

A screen test on the classifier overflow showed 78·1 per cent minus 200 mesh with 1·7 per cent plus 100 mesh.

*Mr. Galloway, superintendent of the Stirling mill, was present during these runs.

Assays:

—	Copper, per cent	Lead, per cent	Zinc, per cent	Insoluble, per cent
Classifier overflow	0·72	1·62	6·64	....
Talc concentrate	0·27	0·96	2·71	....
Talc tailing	0·81	1·72	7·36	....
Wilfley table concentrate	0·58	2·33	9·40	....
Wilfley table tailing	0·30	1·09	2·40	....
Cu-Pb concentrate	12·42	23·32	8·17	6·53
Cu-Pb middling	1·70	5·22	17·37	....
Cu-Pb tailing	0·18	0·28	6·95	....
Zn concentrate	0·34	0·58	57·64	2·94
Zn tailing	0·22	0·18	0·82	....

Results:

Product	Weight, per cent	Assays, per cent			Distribution, per cent		
		Cu	Pb	Zn	Cu	Pb	Zn
Classifier overflow	100·00	0·72	1·62	6·64	100·0	100·0	100·0
Cu-Pb concentrate	5·34	12·42	23·32	8·17	74·2	78·7	6·9
Zn concentrate	8·64	0·34	0·58	57·64	3·3	3·2	78·4
Zn tailing	71·45	0·22	0·18	0·82	17·6	8·1	9·2
Wilfley table tailing	14·57	0·30	1·09	2·40	4·9	10·0	5·5

Samples taken over a period of time during the run showed that for every 100 tons of feed, 1·1 tons of lead-copper middling and 0·6 ton of Wilfley table concentrate were sent to the regrind mill.

Mill Run No. 9

During this run, observations were made on the response of the ore to various changes. No samples were taken.

Mill Run No. 10

This test was run under the same conditions as was Test No. 8. The cresylic acid added to the talc float was reduced by 0·06 pound per ton, the butyl xanthate to the lead-copper circuit was again reduced by 0·027 pound, and 0·023 pound of cresylic acid added to loosen up the froth; 0·075 pound of sodium silicate was added to the copper-lead conditioner.

Assays:

—	Copper, per cent	Lead, per cent	Zinc, per cent	Insoluble, per cent
Classifier overflow	0·70	1·52	6·23	....
Talc concentrate	0·53	1·43	5·11	....
Talc tailing	0·72	1·56	6·95	....
Wilfley table concentrate	0·43	3·17	5·11	....
Wilfley table tailing	0·55	1·32	4·96	....
Cu-Pb concentrate	11·10	27·37	7·87	6·01
Cu-Pb middling	1·90	10·29	15·33	....
Cu-Pb tailing	0·18	0·43	6·85	....
Zn concentrate	0·58	1·43	49·46	....
Zn tailing	0·19	0·31	0·76	....

Results:

Product	Weight, per cent	Assays, per cent			Distribution, per cent		
		Cu	Pb	Zn	Cu	Pb	Zn
Classifier overflow	100·00	0·70	1·52	6·23	100·0	100·0	100·0
Cu-Pb concentrate	3·78	11·10	27·37	7·87	61·5	66·4	4·4
Zn concentrate	11·04	0·58	1·43	49·46	9·4	10·2	80·0
Zn tailing	75·10	0·19	0·31	0·76	20·9	14·9	8·3
Wilfley table tailing	10·08	0·55	1·32	4·96	8·2	8·5	7·3

Mill Run No. 11

The reagents in this run were practically the same as those for Runs Nos. 2 to 7 with the exception of an increase of 0·05 pound of cresylic acid to the talc float and a decrease of 0·02 pound of butyl xanthate to the copper lead float. The grind was 77·7 per cent minus 200 mesh with 1·2 per cent plus 100 mesh.

Assays:

—	Copper, per cent	Lead, per cent	Zinc, per cent	Insoluble, per cent
Classifier overflow	0·72	1·62	6·33	....
Talc concentrate	0·21	0·86	2·66	....
Talc tailing	0·75	1·92	7·26	....
Wilfley table concentrate	0·47	1·76	8·28	....
Wilfley table tailing	0·22	0·86	1·94	....
Cu-Pb concentrate	12·34	26·56	5·82	8·68
Cu-Pb middling	2·33	12·67	15·02	....
Cu-Pb tailing	0·25	0·48	7·15	....
Zn concentrate	0·51	1·12	55·59	....
Zn tailing	0·21	0·28	0·74	....

Results:

Product	Weight, per cent	Assays, per cent			Distribution, per cent		
		Cu	Pb	Zn	Cu	Pb	Zn
Classifier overflow	100·00	0·72	1·62	6·33	100·0	100·0	100·0
Cu-Pb concentrate	4·52	12·34	26·56	5·82	70·8	72·9	4·3
Zn concentrate	9·08	0·51	1·12	55·59	5·9	6·2	81·9
Zn tailing	68·67	0·21	0·28	0·74	18·3	11·7	8·2
Wilfley table tailing	17·73	0·22	0·86	1·94	5·0	9·2	5·6

Mill Run No. 12

This run was made in a caustic soda circuit, 1·0 pound per ton of sodium hydroxide replacing the soda ash added to the ball mill. All other reagents were the same as in the preceding day's run. The grind was 74·9 per cent minus 200 mesh.

Assays:

—	Copper, per cent	Lead, per cent	Zinc, per cent	Insoluble, per cent
Classifier overflow	0·72	1·62	6·33	....
Talc concentrate	0·22	0·92	2·66	....
Talc tailing	0·76	1·84	7·36	....
Wilfley table concentrate	0·64	2·18	9·10	....
Wilfley table tailing	0·22	0·86	1·99	....
Cu-Pb concentrate	9·78	24·73	8·18	5·10
Cu-Pb middling	1·82	6·60	16·24	....
Cu-Pb tailing	0·30	0·48	7·16	....
Zn concentrate	0·60	1·22	52·32	....
Zn tailing	0·29	0·43	0·84	....

Results:

Product	Weight, per cent	Assays, per cent			Distribution, per cent		
		Cu	Pb	Zn	Cu	Pb	Zn
Classifier overflow	100·00	0·72	1·62	6·33	100·0	100·0	100·0
Cu-Pb concentrate	4·95	9·78	24·73	8·18	61·8	69·4	6·1
Zn concentrate	10·23	0·60	1·22	52·32	7·8	7·1	81·0
Zn tailing	73·01	0·29	0·43	0·84	27·1	17·8	9·3
Wilfley table tailing	11·81	0·22	0·86	1·99	3·3	5·7	3·6

SUMMARY AND CONCLUSIONS

This shipment of Stirling ore responds to flotation in much the same way as did the shipment investigated and reported on in 1934. The chalcopyrite, however, is not so sensitive to depression by cyanide as was that in the earlier consignment.

The general flotation conditions of the tests from Mill Run No. 1 to Mill Run No. 7 were entirely different from those at the mine. Good recoveries of lead, copper, and zinc are recorded in this investigation. These results were obtained on a grind of about 70 per cent minus 200 mesh.

In the attempt to produce in the test unit the type of froth, etc., met with at the mine, considerable difficulty was encountered. By increasing the density of the pulp in the ball mill, reducing the time of conditioning, and eliminating air conditioning, the type of froth sought was finally obtained. Under these conditions the flotation results went completely off, the zinc would not float and the lead dropped out into the tailing, leaving only a copper concentrate in the copper-lead circuit. This was done with no change in reagents.

The results point to the desirability of applying air conditioning to the treatment of Stirling ore. By regrinding a middling product, the ore should not require grinding finer than 70 per cent minus 200 mesh, at which recoveries of over 70 per cent of the copper and lead and 85 per cent of the zinc should be realized.

III

INVESTIGATIONS THE DETAILS OF WHICH ARE NOT PUBLISHED

Ore or Product	Source of Shipment	Address
Gold	Payrock Gold Syndicate	Barrie, Ontario.
Gold-copper	Wendigo Gold Mines, Limited	Kenora, Ontario.
Gold	Powell Rouyn Gold Mines, Limited.	Noranda, Que.
Gold	Olive Gold Mines, Limited	Olive, Ontario.
Drill cores	Minaki Mining and Development Co. Ltd.	Kenora-Rainy River District, Ontario.
Iron-pyrite	Matachewan Hub Pioneer Mines, Limited.	Elk Lake, Ontario.
Copper	Mining and Finance Corp., Ltd.	Copper Lake, Antigonish County, N.S.
Gold-silver-copper-lead-zinc	Tyee Consolidated Mining Co., Ltd.	Westholme, B.C.
Gold	Lapa Cadillac Gold Mines, Limited.	Heva River, Cadillac Township, Quebec.
Gold	Lake Rose Mines, Limited	Rose Lake *via* Senneterre, Que.
Copper	Jay Copper Gold Mines, Limited	Amos, Quebec.
Silver-zinc	Quebec Manitou Mines, Limited	Val D'Or, Quebec.
Red mud	Aluminum Company of Canada, Limited.	Arvida, Quebec.
Gold	Thurlow Gold Mines, Limited	Shoal Bay, Thurlow, B.C.
Gold	Golden Gate Mining Co., Limited	Swastika, Ontario.
Gold	Clark Gold Mines, Limited	Dyment, Ontario.
Gold	Placer material	"Aranka" Area, British Guiana.
Refinery material	Royal Canadian Mint	Ottawa, Ontario.
Gold	Claverny Gold Mines, Limited	Amos, Quebec.
Gold	Arcadia Gold Mines, Limited	South Porcupine, Ontario.
Gold	Oremond Gold Mines, Limited	Jellicoe, Ontario.
Chrome	Chromium Mining and Smelting Corporation.	Collins, Ontario.
Gold-tungsten	Ince Syndicate	Birch Lake, Patricia District, Ontario.
Gold	Horwood Lake Gold Mining Co., Limited.	Horwood Lake, Ont.
Gold	Kenricia Gold Mines, Limited	Kenora, Ontario.
Lead-zinc	Lake Geneva mine	Hess Township, Sudbury District, Ontario.
Gold	Central Duverny Gold Mines, Limited.	Amos, Quebec.
Gold	Big Thing Property	Yukon Territory.
Gold-silver-lead-zinc	Calumet Mines, Ltd	Calumet Island, Bryson, Quebec.
Gold	Packsack Mines, Ltd	Bissett, Manitoba.
Gold	Payore Gold Mines, Limited	Val D'Or, Quebec.

An examination of several samples taken from steel piling used at Rimouski, Quebec.

An examination of two austenitic manganese steels (Sorel Steel Foundries, Limited).

The setting and examination of a mild steel (Department of National Defence).

Tensile tests on alclad sheet (Department of National Defence).

An examination of two welded austenitic manganese steel dipper teeth (Sorel Steel Foundries, Limited).

The physical testing of a medium carbon steel (Naval Stores).

An examination of the aluminium alloy in Junkers plane CF-AMX (Department of Transport).

An examination of an austenitic manganese steel (Manitoba Steel Foundries, Ltd., Selkirk, Man.)

An examination of a failed austenitic manganese steel plate (Sorel Steel Foundries, Limited).

An examination of a worn austenitic manganese steel ball mill liner (Sorel Steel Foundries, Limited).

An examination of the steel used in engraved printing plates (British American Bank Note Company).

Physical tests on air-hardened nickel-chrome steel (Canadian Atlas Steels, Limited).

Steel examined and four landing gear parts carburized for R.C.A.F.

Bolt tested in tension for R.C.A.F.

Brass protecting tube of heating element examined for Dr. J. S. G. Shotwell.

Section of rail polished and examined for Mr. J. G. Sutherland of C.P.R. Offices, Toronto.

A microscopic examination made of a white iron grinding ball for Hull Iron and Steel Foundries, Limited.

Five hardness tests made on five white iron balls for Hull Iron and Steel Foundries, Limited.

A cast iron test piece tested in tension for Modern Machine Company, Ottawa.

Magnetic separation of one hundred pounds of high speed steel grindings made for Canadian Atlas Steels, Limited.

Eighteen nickel chromium heat resisting trays cast for Royal Mint.

Photomicrographs obtained from the under surfaces of two types of light bulbs.

An examination of a failed trailer coupling.

The effect of temperature on the grain size of three carburizing steels.

The impact testing of nine tool steels.

Physical testing of a steel cable.

Examination of sample of heavy metallic ore from Eldorado Gold Mines, Limited, Port Hope, Ont.

Microscopic examination of sample of gold ore from Beaver Lake, Saskatchewan (J. F. Wright, Winnipeg).

Microscopic examination of sections of drill cores from the Stirling Mine of the British Metal Corporation, Stirling, N.S.

Determination of minerals in samples of high-grade ore from MacLeod-Cockshutt Gold Mines, Limited, Little Long Lac, Ontario.

IV

A RÉSUMÉ OF SPECIAL RESEARCH COMPLETED, IN PROGRESS, OR UNDER CONSIDERATION

The immediate investigation of an ore or a metallurgical product may lead to research of a more general character, but at times work is undertaken by the Metallic Minerals Division independently of any such origin. The capacity of the existing organization for research on general problems is determined by the demands of its investigational duties. The staff of the Division has recently been strengthened by the appointment of a physical chemist.

The following is a résumé of research work completed and also of work in progress or under consideration during the period, July to December, 1937.

Gold ore from Central Patricia Gold Mines, Limited, Pickle Crow, Ontario. In connection with the test work on this ore it became necessary to determine what minerals were interfering with cyanidation and what steps were necessary to bring about a more normal cyanide consumption and a higher extraction of gold. (See Investigation No. 718.)

Silver-pitchblende concentrate from Eldorado Gold Mines, Limited, Great Bear Lake, N.W.T. and Port Hope, Ontario. No written report was made of this research, the purpose of which was to determine if the present roasting practice at Port Hope could be improved by using a rotary kiln in place of the Herreshoff furnace now in use for oxidizing roasting, and a reverberatory furnace for chloridizing. The possibility of combining the oxidizing roast and the chloridizing in one operation was also investigated. It was found that the present practice was the more efficient and controllable. Mr. Quinlan, Chemist of Port Hope, co-operated in this work and Mr. Pochon, Plant Manager at Port Hope, visited the laboratory during the investigation.

The chemical reactions occurring in ball mill grinding of ores, particularly with respect to cyanidation practice. The chemical reactions that take place in grinding an ore in water, alkaline water, and alkaline cyanide solution were studied comparatively. Ores containing pyrrhotite and pyrite were used and the solutions from the grinding were analysed for the products of chemical reaction. The investigation is still in progress.

The production of red oxide pigment from bog iron ore, Labelle County, Quebec. The samples were received late in the year and the work is in progress.

Effect of molybdenum on the impact strength of cast iron. Improvements in cast iron have led to its use under conditions in which considerable impact strength may be required, yet little information has been published on the impact strengths of molybdenum cast irons. This is being investigated, together with the effect of molybdenum on the high damping property of cast iron. Molybdenum cast iron test bars containing $\frac{1}{4}$, $\frac{1}{2}$, and 1 per cent molybdenum were prepared, three test bars containing respectively 2, $2\frac{1}{2}$, and 3 per cent carbon being made for each molybdenum content. Transverse test bars were machined from each iron so that information about the impact properties can be obtained from transverse test data. The hardnesses of all materials were determined and a metallographic examination was made of all irons. Tensile, damping and impact test specimens will be prepared and the results from the four different tests correlated.

Projected Research Problems

In selecting research problems to be undertaken, preference is given to those affecting the utilization of natural resources, their conservation by improved processing and utilization of by-products.

It is intended to undertake research upon the following subjects as time permits:

Uranium as an Alloying Element. The production of radium at Port Hope is to be increased and this entails an increase in the uranium products. With a view to the development of a market for this otherwise surplus material an investigation of the alloying properties of uranium appears to be advisable.

In the periodic table, uranium is grouped with chromium, molybdenum, and tungsten, all of which impart valuable properties to steel. The effect of these other three elements when alloyed with steel will be used as a guide to which field might be usefully investigated. The effect of uranium additions to certain non-ferrous materials will also be determined.

The steel making and physical testing equipment of the division are excellent and particularly suited to such investigational work.

Properties of Sponge Iron Steel. The problem of manufacturing sponge iron has been adequately investigated, and the difficulties now appear to be economic rather than technical. It has been claimed, however, that steel made from sponge iron has properties superior to those made from the raw materials generally used and this claim deserves investigation. A supply of sponge iron has been made from Texada Island iron ore and in order that the work may be done on a sufficiently large scale, the co-operation of a Canadian steel company will probably be sought. This company will be able to produce two steels, one from sponge iron and one from scrap, other manufacturing and fabricating variables being held constant. The physical properties of the two steels, especially damping and fatigue, could be tested and the results correlated with company service tests and with the behaviour of the steel in actual service.

An Exhaustive Study of the Mode of Occurrence of Gold in Canadian Ores. This is to incorporate the results of work already carried out on Canadian gold ores, with such supplementary studies as might be found advisable. The treatment of the subject will be to relate gold occurrence to (1) geological investigation and (2) milling.

A Study of the Mode of Occurrence of Minerals, with Relation to the Problems of Grinding. Tentatively, this study will seek to relate the information obtained through the use of the microscope to the problems concerned with grinding; it is not primarily concerned with any specific ore, but will be a general study, embracing the ores of gold, base metal, chrome, etc.

Attention is also being given to the following problems.:

With the purchase of the damping machine, a co-operative research may be undertaken with a Canadian steel company on the effect that melting, fabricating, and heat treatment has on the damping properties of special purpose steels, and the results will be correlated with the behaviour of the steels in service.

The elongation and reduction of area of tensile-test bars of acid melted steel vary considerably for no apparent reason. This may be due to the gas content of the steel prior to deoxidation and this supposition may be checked experimentally.

Rapid methods for desulphurizing and dephosphorizing iron and steel may be investigated.

An investigation of the brittleness of boiler plate and the ageing embrittlement of steel may be undertaken.

The progress being made in the production of magnesium metal is being closely followed. The electrothermic reduction is the most promising method of production, and is particularly well suited to Canadian conditions, where low-cost power and suitable magnesium ores are available. If staff were available, an investigation should be undertaken because the electrical furnace equipment of the Bureau of Mines is probably the most complete in the country.

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