New Building, Ore Dressing Laboratory, Bureau of Mines, Department of Mines and Resources, Ottawa.

CANADA

DEPARTMENT OF MINES AND RESOURCES

MINES AND GEOLOGY BRANCH

BUREAU OF MINES

INVESTIGATIONS IN ORE DRESSING AND METALLURGY

(Testing and Research Laboratories)

January to June, 1939

Price, 50 cents.

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CONTENTS

METALLIC MINERALS DIVISION INVESTIGATIONS IN ORE DRESSING AND METALLURGY, JANUARY TO JUNE, 1939

I

REVIEW OF INVESTIGATIONS

Chief, Division of Metallic Minerals

During the period under review, fifty-four investigations were completed and reports were issued to those interested; of this number, thirteen are now printed in full and appear in Section II, the remainder are listed by title only in Section III. Section IV deals with the activities of the chemical, mineralogical, physical testing, and heat-treating laboratories as well as the analyses and tests made, including miscellaneous items, tests, and sampling jobs. A review of the special research in progress and completed is given in Section V.

Summary of Investigations:

Investigations on which reports were prepared	54
Investigations completed, but on which no formal reports were issued	17
Gold-bearing ores	34
Mill problems	7
Iron ore containing nickel and chromium	1
Silver-lead-zinc ore	1
Copper-gold ores	2
Copper ore	1
Pyrites	1
Microscopic examinations (special)	13
Steel and alloy products	14
Number of ores investigated	39

Provincially, the above ores originated as follows: Ontario, 14; Quebec, 10; British Columbia, 7; Northwest Territories, 1; Manitoba, 3; Nova Scotia, 2; Saskatchewan, 2.

In addition to the listed activities, the staff was called on by the Department of National Defence to review a number of specifications and reports in connection with ferrous and non-ferrous materials entering into the manufacture of munitions and to carry out physical tests and examinations of aeroplane parts for the British Air Ministry, the Royal Canadian Air Force, and the Department of Transport.

Members of the staff were consulted frequently in connection with the manufacture of the Bren gun and of armour plate, as well as on other matters, and more than half the time of two technical officers specializing in such subjects was so occupied.

Staff:

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

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

All spectrographic work has been carried out by L. S. Macklin, assisted by J. A. Rivington.

All special investigational and research work was conducted under the supervision of R. J. Traill, senior engineer, with B. P. Coyne and various members of the staff assisting.

The metallurgical laboratory work on iron, steel, and alloys was supervised by G. S. Farnham, assisted by N. B. Brown.

The Chemical Laboratory was supervised by J. A. Fournier, Chief Chemist, with the following staff of chemists: R. A. Rogers, A. Sadler, T. T. Merrifield, R. W. Cornish, J. S. McCree, S. R. M. Badger, A. E. Larochelle, J. A. Rivington, and L. Lutes and C. H. Derry, assayers.

New Ore Dressing Building: A new building for the Ore Dressing Laboratories, which was under construction during 1938, was completed and equipped ready for occupation the first part of this year (1939).

This building (*See* Plate I, frontispiece) is 100 feet long by 60 feet wide and contains three main floors with 13-foot ceilings, and a basement, giving a total vertical clearance of 50 feet. The interior of the building is divided by a vertical cross-wall of brick. The front part is 25 by 60 feet and contains a complete crushing and sampling plant. The ground floor is used as the receiving floor. All dry crushing and sampling equipment is in this section and exhaust fans on each floor produce a slight vacuum, which prevents the dust from working into the main section of the mill containing the testing equipment.

The main part of the building containing the testing equipment is 75 by 60 feet. In the centre is a well extending from the ground floor to the monitor roof in which a 5-ton travelling crane facilitates the shifting of equipment from floor to floor in order to build up the proper arrangement for special flow-sheets.

The general layout of the equipment is shown in Figure 1. In addition to the equipment removed from the old mill building, a complete automatic sampling plant with a capacity of 3 tons an hour, and a complete, 100- to 250-pound an hour, continuous, self-contained cyanide plant have been provided.

In addition to equipping the new ore dressing laboratories, the following important scientific equipment has been acquired by the Division:

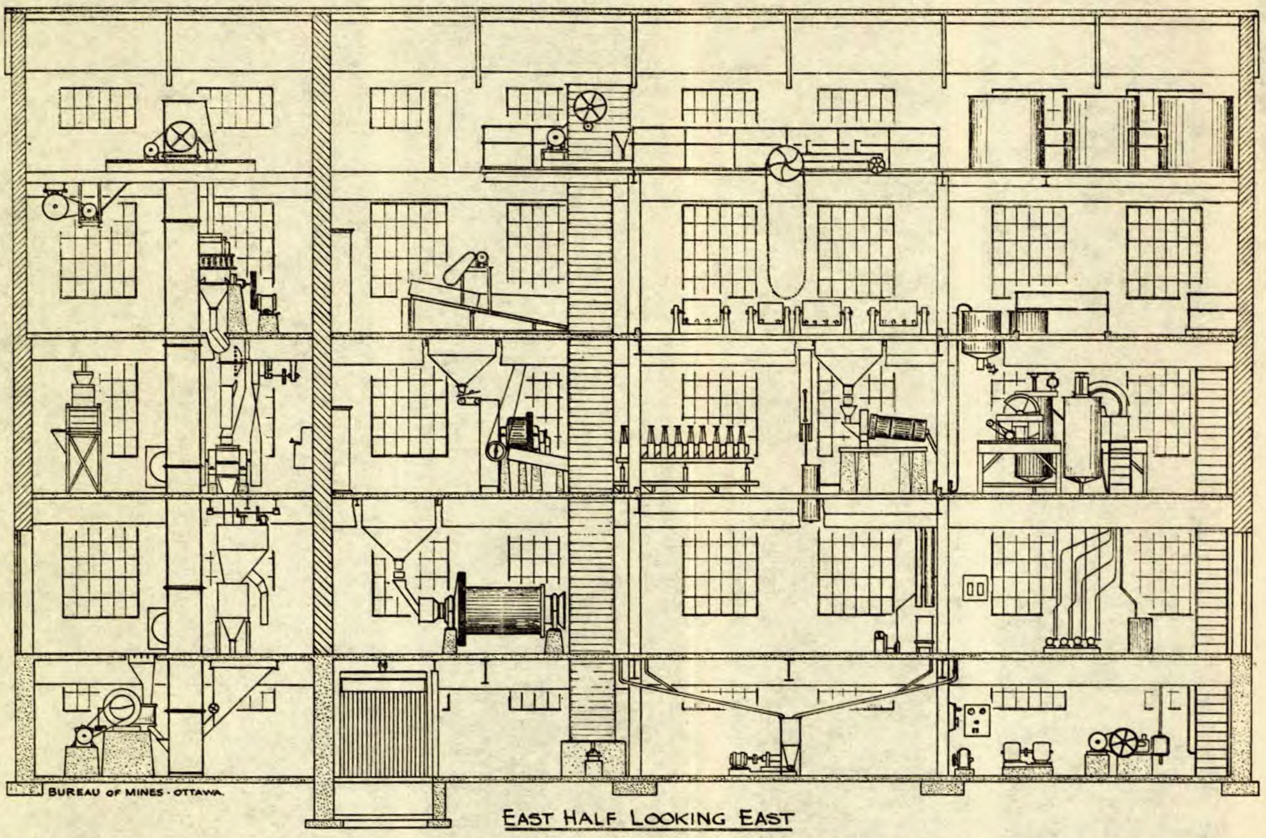
(a) A vacuum electric high-frequency induction furnace.

(b) A modified Föppl-Pertz damping capacity machine.

(c) A grating spectrograph with the accessory equipment for quantitative analyses.

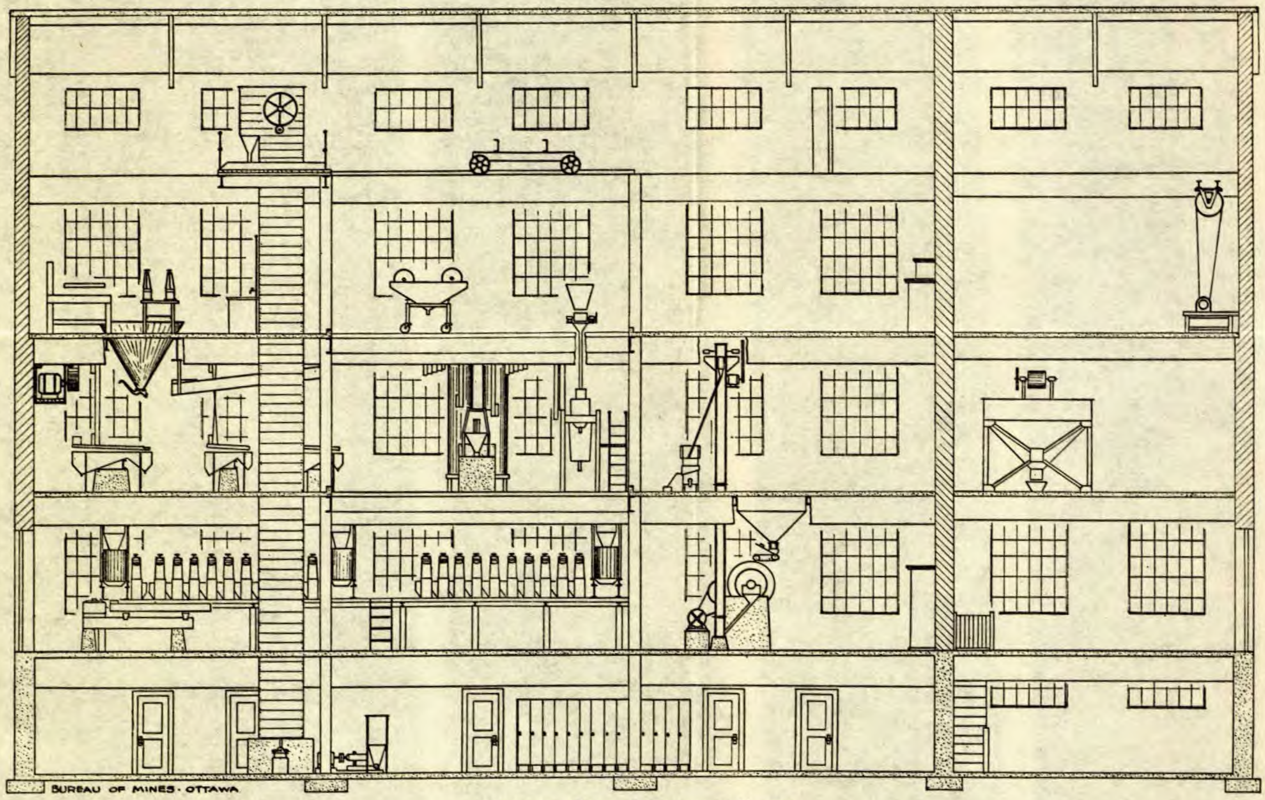
(d) An X-ray diffraction machine with four types of camera and three types of target, i.e. copper, cobalt, and molybdenum.

For details of this equipment see Section IV.



EAST HALF LOOKING EAST

FLOOR	EQUIPMENT READING FROM LEFT TO RIGHT
BASEMENT	JAW CRUSHER, DRY ORE ELEVATOR, STORAGE BIN, PASSENGER & FREIGHT ELEVATOR, WET ORE ELEVATOR, WILFLEY SAND PUMP, D.C. GENERATORS (2), AIR COMPRESSOR.
GROUND	DRY ORE ELEVATOR, VEZIN SAMPLER, CRUSHED ORE BIN, MARCY ROLLER MILL, WET ORE ELEVATOR, BARREN SOLUTION PUMP, MEASURING PIPES, WESCO PUMPS, WATER SEAL
FIRST	MAGNETIC SEPARATOR, DRY ORE ELEVATOR, CRUSHING ROLLS, SNYDER SAMPLER, CRUSHED ORE BIN, BALL MILL, WET ORE ELEVATOR, FLOTATION CELLS, CROWE VACUUM, CLARIFYING TANK, CRUSHED ORE BIN, HARDINGE MILL, GENTER THICKENER, OLIVER FILTER, AGITATOR TANKS, OLIVER FILTER.
SECOND	DRIVE BENCH CRUSHING UNIT, DRY ORE ELEVATOR, SYMONS CONE CRUSHER, DORR CLASSIFIER, WET ORE ELEVATOR, BOX AGITATORS, GENTER THICKENERS.
MONITOR	DRIVE DRY ORE ELEVATOR, DRIVE WET ORE ELEVATOR, CRANE, CYANIDE SOLUTION TANKS.



WEST HALF LOOKING WEST

FLOOR	EQUIPMENT READING FROM LEFT TO RIGHT
BASEMENT	DOOR ELECTRICAL ROOM, DOOR STORE ROOM, WET ORE ELEVATOR, WILFLEY SAND PUMP, DOOR DINING ROOM, LOCKERS, DOOR WASHROOM, DOOR TOILETS
GROUND	WILFLEY CONCENTRATING TABLE, DENVER CONDITIONERS AND FLOTATION CELLS, WET ORE ELEVATOR, WET ORE ELEVATOR, BALL MILL, ORE BIN, CUPBOARDS
FIRST	MOTOR TABLES DRIVE, WILFLEY CONCENTRATING TABLE, CALLOW SETTLING TANK, RICHARDS CLASSIFIER, BUTCHART CONCENTRATING TABLE, WET ORE ELEVATOR, PLAT-O CONCENTRATING TABLE, JAMES JIGS WITH WATER TANKS, ROCHE MAGNETIC SEPARATOR, DORR CLASSIFIER, WET ORE ELEVATOR, STORAGE BIN, HEAD OF BELT CONVEYOR
SECOND	LAB. CONCENTRATING TABLE, LAB. FLOTATION CELLS, SINK, WET ORE ELEVATOR, CALLOW SCREEN, DRY ORE FEEDER, CUPBOARDS, BENCH CRUSHING UNIT
MONITOR	DRIVE WET ORE ELEVATOR, CRANE.

Figure 1

II

INVESTIGATIONS THE RESULTS OF WHICH ARE RECORDED IN DETAIL

Ore Dressing and Metallurgical Investigation No. 762

GOLD ORE AND MILL PRODUCTS FROM DELNITE MINES, LIMITED, TIMMINS, ONTARIO

Shipments. Ten bags of ore, total weight 620 pounds, were received on May 11, 1938, and 11·5 pounds of classifier fine, 9·5 pounds of coarse classifier product, and 2·5 pounds of thickener froth were received on June 20, 1938. An additional shipment, 400 pounds, consisting of a low-grade tailing sample with its corresponding mill feed sample, and a higher grade tailing sample also with its corresponding mill feed sample, was received on October 7, 1938. The shipments were from K. C. Gray, Mine Manager, Delnite Mines, Limited, Timmins, Ontario.

Previously, a shipment of 2,500 pounds of ore had been received on January 23, 1936, and the tests are described in Investigation No. 678, Bureau of Mines Report No. 774.

Location of the Property. The property of the Delnite Mines, Limited from which the present shipments were received is situated in Deloro Township, Cochrane mining district, Porcupine area, northern Ontario.

Sampling and Analysis. After cutting, crushing, and grinding by standard methods, samples of the ores and mill products were obtained, which assayed as follows:

Analyses of Products:

Product	Assay, oz./ton		S, per cent	Graphitic C, per cent	As, per cent	$FeCO_3$, per cent	Cu, per cent	CaO, per cent	MgO, per cent
	Au	Ag							
Ore shipment of May 11...	0·27	0·045	1·42	0·05	0·19	....	Trace	......	......
Coarse classifier...........	0·91	....	2·11	0·27	0·28	....	......	......	......
Fine classifier.............	0·81	....	1·48	0·88	0·23	10·66	......	6·88	2·98
Thickener froth...........	0·26	....	0·40	1·23	0·33	....	......	......	......
Lower grade ore...........	0·23	0·035	1·42	0·11	0·27	2·79	0·02	6·27	6·26
Lower grade tailing........	0·02	0·015	1·38	0·11	0·27	2·16	0·02	6·35	6·23
Higher grade ore..........	0·275	0·055	1·43	0·11	0·38	2·50	0·03	5·94	6·11
Higher grade tailing.......	0·03	0·02	1·40	0·09	0·35	2·40	0·01	5·99	6·03

On the mill products, screen tests were made on the individual samples, resulting as follows:

Mesh	Weight, per cent				
	Coarse classifier	Fine classifier	Thickener froth	Lower grade tailing	Higher grade tailing
+ 20	1·2	....	....	....	....
− 20 + 28	1·8	....	....	....	....
− 28 + 35	4·0	....	....	....	....
− 35 + 48	4·5	4·3	....	....	....
− 48 + 65	9·1	3·6	2·4	....	....
− 65 + 100	13·8	5·4	14·8	0·3	0·4
−100 + 150	18·3	8·2	16·0	8·4	8·8
−150 + 200	12·3	3·5	3·0	10·0	9·4
−200	35·0	75·0	63·8	81·3	81·4
Totals	100·0	100·0	100·0	100·0	100·0

Characteristics of the Ore:

Six polished sections of the ore sample received on May 11, 1938 were examined microscopically to determine the character of the ore.

The *gangue* is an aggregate of dark grey rock minerals with a small quantity of milky-white quartz and some disseminated carbonate. Some hand specimens are distinctly greenish grey in colour, which may be due to chlorite.

Pyrite and arsenopyrite are the only two *metallic minerals* of appreciable amount in the ore, the former being the more abundant. They occur intimately mixed and separately in the gangue as coarse to fine grains and crystals showing mutual relationships. Both contain inclusions of gangue and both are somewhat fractured and veined with chalcopyrite, which mineral is present also in gangue as occasional, small, irregular grains.

Some twenty-four grains of gold are visible in the sections, all occurring in arsenopyrite and pyrite—59 per cent in the former mineral and 41 per cent in the latter mineral. They are all extremely small, the greater number being less than 6 microns (−2300 mesh) in size, the largest being 50 microns (+400 mesh). All the gold visible in pyrite is in the dense mineral, whereas most of that in arsenopyrite is along tiny fractures, both alone and with chalcopyrite. The latter association points to the fact that the gold and chalcopyrite were deposited contemporaneously, and both later than the host minerals.

Six polished sections of each ore sample received on October 7, 1938 were examined microscopically to determine the mode of occurrence of the gold.

The two samples do not show essential differences in their microscopic character, and appear to be similar to those received on May 11, 1938. The *gangue* is dark greenish grey rock with rather abundant disseminated iron-bearing carbonate and milky-white vein quartz. The *metallic minerals* are: pyrite, arsenopyrite, magnetite, and chalcopyrite. Pyrite predominates, and magnetite and chalcopyrite are present in only small quantities. Native gold is visible in both samples, but is more abundant in sections from the "high-grade" sample.

Mode of Occurrence of the Gold:

"Low-grade". Two grains of native gold, 24- and 4-micron size, occur in gangue, in contact with pyrite. An irregular veinlet of gold, with a maximum width of about 15 microns, occurs in arsenopyrite.

"High-grade". Seventeen grains of native gold are visible in sections from the "high-grade" sample. The following table shows its mode of occurrence and grain size:

Mesh	In dense pyrite, per cent	Along fractures in pyrite, per cent	In gangue in contact with pyrite, per cent	Totals, per cent
+ 560	....	....	14·5	14·5
− 560 + 800	8·5	10·2	....	18·7
− 800 + 1100	....	19·6	13·2	32·8
−1100 + 1600	5·1	10·2	4·2	19·5
−1600 + 2300	....	3·4	9·4	12·8
−2300	....	....	1·7	1·7
Totals	13·6	43·4	43·0	100·0

Experimental Tests

The mine manager reported that when an influx of graphitic ore occurs in the mill feed, the solution fouls rapidly and the cyanide residue is double the value of the normal tailing.

The work on the different shipments was undertaken in order to determine the cause of this condition and, if possible, to suggest a remedy.

The higher grade shipment of October 7, 1938, assaying 0·275 ounce gold per ton, showed some fouling of the cyanide solution, resulting in a higher tailing after several batches of this ore had been subjected to a cycle cyanidation test. This condition was corrected by using a high-lime circuit and aeration of the barren solution.

From the tailing shipments, additional agitation extracted over 50 per cent of the gold remaining in these cyanide residues.

The tests of the different mill products indicated that additional aeration and a high-lime circuit are beneficial in controlling fouling and increasing extraction slightly.

The prevalence of iron carbonate ($FeCO_3$) is probably the cause of the fouling of cyanide solution and resulting high residue. This condition is accentuated in the sample of classifier fine, which assayed 10·6 per cent of $FeCO_3$ against 2·5 per cent in the normal feed sample.

The test work is divided into two parts, A and B.

In *Part A* is given the work performed on the ore shipment received on May 11, 1938, and also on the mill products received on June 20, 1938.

In *Part B* are given the results obtained from the ore and mill tailing received on October 7, 1938.

Part A

CYANIDATION

Test No. 1

Samples of the ore at −14 mesh were ground in a ball mill in cyanide solution of a strength of 1 pound of potassium cyanide per ton, to pass 81 per cent −200 mesh. Five pounds of lime per ton of ore was added. The pulp was bottle-agitated for different periods of time. A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65 + 100	1·3
−100 + 150	6·7
−150 + 200	11·0
−200	81·0
Total	100·0

Results of Cyanidation:

Feed: gold, 0·27 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
1A	7	0·025	90·8	1·00	0·32	0·21	4·36
1B	15	0·025	90·8	0·88	0·32	0·24	4·36
1C	21	0·015	94·5	1·00	0·22	0·37	4·44
1D	28	0·015	94·5	0·92	0·22	0·41	4·56
1E	39	0·015	94·5	1·00	0·28	0·41	4·56

Maximum extraction was obtained after 21 hours of agitation.

CYCLE CYANIDATION

Test No. 2

This was to determine whether any noticeable fouling of cyanide solution occurred during grinding and agitation, as would be indicated by an increase in the gold content of the tailing. The ore at −14 mesh was ground in cyanide solution of a strength of 1 pound of potassium cyanide per ton, to pass 84·4 per cent −200 mesh. Five pounds of lime was added per ton of ore. The pulp was bottle-agitated for 24 hours, the tailing was filtered, washed, and assayed for gold. A fresh portion of the ore was ground in the same cyanide solution and the pulp was agitated for 24 hours. The process was repeated for five cycles of grinding and agitation.

Results of Cyanidation:

Feed: gold, 0·27 oz./ton.

Cycle 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
1	24	0·015	94·5	0·96	0·32	0·49	5·36
2	24	0·015	94·5	0·88	0·16	0·78	4·10
3	24	0·015	94·5	0·92	0·20	0·60	4·10
4	24	0·015	94·5	1·00	0·30	0·90	3·90
5	24	0·0187	93·1	0·92	0·20	0·78	4·10

The final cyanide solutions from the fifth cycle assayed:

Reducing power..180 ml. $\frac{N}{10}$ $KMnO_4$/litre.
KCNS..0·10 grm./litre.

It can be seen from the assay of the final cyanide residue and the reducing power of the final cyanide solution that the amount of fouling taking place was very slight.

Test No. 3 (A to G)

The ore at −14 mesh was ground in a ball mill to pass 91·1 per cent −200 mesh. The pulp was mixed with portions of the mill products, as noted below, and bottle-agitated in cyanide solution of a strength of 1 pound of potassium cyanide per ton for 24 hours. The cyanide residues were assayed for gold and the solutions were assayed for reducing power.

The proportions of the ore and the mill products in the different tests were as follows:

Test No.	Weight, per cent			
	Ore	Classifier coarse	Classifier fine	Thickener froth
3A	87·5	12·5	....	....
3B	75·0	25·0	....	....
3C	87·5	....	12·5	....
3D	75·0	....	25·0	....
3E	87·5	....	....	12·5
3F	75·0	....	....	25·0
3G	100·0	....	....	....

Results of Cyanidation:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reducing power, ml. $\frac{N}{10}$ $KMnO_4$/litre	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing			KCN	CaO	KCN	CaO
3A	24	0·35	0·05	85·7	68	1·0	0·25	0·69	5·5
3B	24	0·43	0·075	82·6	80	1·0	0·25	0·69	5·5
3C	24	0·35	0·045	87·2	76	1·0	0·3	0·44	6·4
3D	24	0·42	0·07	83·3	108	1·0	0·3	0·44	6·4
3E	24	0·27	0·025	90·7	92	1·0	0·25	0·69	6·5
3F	24	0·27	0·025	90·7	120	1·0	0·25	0·69	6·5
3G	24	0·27	0·015	94·5	60	1·0	0·25	0·44	5·5

It is evident that the reducing power of the cyanide solution increases with each addition of mill products, as, for example, from 60 in Test No. 3G, in which the feed was 100 per cent of ore, to 120 in Test No. 3F, in which the feed was 25 per cent of thickener froth.

Test No. 4 (A and B)

This test was made on portions of the thickener froth. In Test No. 4A the pulp was bottle-agitated in cyanide solution of a strength of 1 pound of potassium cyanide per ton without regrinding. In Test No. 4B the pulp was ground in water, filtered, washed, and repulped in cyanide solution, prior to agitation.

Results of Cyanidation:

Feed: gold, 0·26 oz./ton

Test No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reducing power, ml. $\frac{N}{10}$ KMnO₄/litre	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
					KCN	CaO	KCN	CaO
4A........	48	0·055	78·8	260	1·1	0·35	3·57	16·3
4B........	48	0·055	78·8	60	1·0	0·3	3·00	15·4

The effect of this water grind and subsequent washing, in lowering the reducing power from 260 to 60 and decreasing the cyanide consumption somewhat, is probably due to the discarding of the water soluble cyanicides after the grind. Owing to the depletion of the sample it was not possible to proceed further along these lines.

Test No. 5 (A to N)

This was made on samples of the classifier fine.

Portions of this mill product were bottle-agitated in cyanide solution of a strength of 1 pound of potassium cyanide per ton for 24 and 48 hours, as noted below:

Results of Cyanidation:

Feed: gold, 0·81 oz./ton

Test No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reducing power, ml. $\frac{N}{10}$ $KMnO_4$/litre	Titration, lb./ton solution		Reagents consumed, lb./ton ore		Additions to agitation, lb./ton
					KCN	CaO	KCN	CaO	
5A	48	0·24	70·4	312	0·9	0·55	2·30	8·9	8 lb. CaO
5B	48	0·25	69·2	264	0·9	0·55	2·30	12·9	12 lb. CaO
5C	48	0·245	69·7	280	0·8	0·6	2·50	9·8	0·5 lb. Pb acetate, 5 lb. CaO
5D	48	0·26	67·9	280	0·9	0·65	2·30	12·7	0·5 lb. Pb acetate and 5 lb. CaO
5E	48	0·20	75·3	172	0·9	0·55	3·33	18·9	Aeration in cyanide
5F	48	0·20	75·3	168	1·0	0·60	2·75	18·9	Aeration in cyanide and Pb acetate
5G	24	0·285	64·8	152	0·9	0·4	2·56	11·2	Washed prior to agitation
5H	48	0·225	72·2	208	1·0	0·4	3·0	13·3	
5I	24	0·225	72·2	156	1·0	0·35	0·64	7·3	Aerated in lime pulp
5J	48	0·20	75·3	164	0·9	0·3	0·80	7·4	
5K	24	0·19	76·5	172	0·9	0·35	2·94	14·3	Reground in water
5L	48	0·165	79·6	232	1·0	0·35	3·77	14·3	
5M	24	0·175	78·4	136	1·0	0·35	0·69	7·3	Reground and aerated in lime pulp
5N	48	0·175	78·4	160	1·0	0·3	0·69	7·3	

It is apparent that this material causes marked fouling of cyanide solution. High lime conditions, with or without addition of lead salt have no very marked effect on reducing power or cyanide consumption in bottle-agitation. (*See* Test No. 5 A, B, C, and D.) Aeration in cyanide prior to bottle-agitation increases extraction about 5 per cent over straight bottle-agitation and lowers the reducing power, but shows higher cyanide consumption. (*See* Test No. 5 E and F.) The higher cyanide consumption here may be due to oxidation of cyanide or formation of thiocyanate or ferrocyanide, preferentially to that of thiosulphate.

Washing prior to agitation in cyanide lowers reducing power slightly and increases extraction approximately 2 per cent, but is accompanied by high cyanide consumption. (*See* Test No. 5 G and H.) Regrinding in water and filtering, prior to agitation in cyanide, shows a 10 per cent increase in extraction but is also accompanied by high cyanide consumption. (Test No. 5 K and L.) The reason for this is not clear, but perhaps in washing or regrinding, fresh surfaces of pyrite or ferrous carbonate are exposed that are readily attacked by the cyanide, forming KCNS and ferrocyanide.

The best all-round results, namely those showing fair extraction, low consumption, and reduced fouling, were obtained by aeration in lime pulp (Test No. 5 I and J), or regrinding and aerating in lime pulp prior to cyanidation (Test No. 5 M and N).

Test No. 6

This was conducted on the different mill products as noted. The pulps were ground in cyanide solution of a strength of 1 pound of potassium cyanide per ton and bottle-agitated for 24- and 48-hour periods. The grind of both the classifier coarse and the classifier fine material was 99 per cent −200 mesh.

Results of Cyanidation:

Test No.	Agitation, hours	Mill product	Assay, Au, oz./ton		Extraction, per cent	Reducing power, ml. $\frac{N}{10}$ $KMnO_4$/litre	Titration lb./ton solution		Reagents consumed, lb./ton ore	
			Feed	Tailing			KCN	CaO	KCN	CaO
6A......	24	Coarse classifier	0·91	0·275	69·8	160	1·0	0·3	1·33	6·8
6B......	48	Coarse classifier	0·91	0·155	83·0	192	1·3	0·1	1·37	8·4
6C......	24	Fine classifier	0·81	0·195	75·9	200	0·5	0·2	2·72	9·6
6D.....	48	Fine classifier	0·81	0·165	80·3	264	1·3	0·15	3·04	11·7

REMARKS—Finer grinding tends to permit a slight increase in extraction.

AQUA REGIA TREATMENT OF CYANIDE RESIDUE

Test No. 7

In order to discover whether the gold remaining in the cyanide residues of Tests Nos. 6 (A, B, C, and D) and 4 (A and B) was in the sulphides or in the quartz gangue, the tailings were subjected to treatment by hot aqua regia solution and the aqua regia residues were assayed for gold.

Results:

Test No.	Tailing assay, Au, oz./ton	Aqua regia tailing, Au, oz./ton	Gold remaining in gangue, per cent
6A	0·275	0·025	9·1
6B	0·155	0·025	16·1
6C	0·195	0·055	28·2
6D	0·16	0·045	28·1
4A	0·055	0·035	63·6
4B	0·055	0·035	63·6

Remarks: In Test No. 6 (A, B, C, and D), it is evident that most of the gold is in the pyrite but an appreciable amount is held in the gangue at this grind (99 per cent −200 mesh).

In Test No. 4 (A and B) the proportion held in the gangue is higher.

Part B

This part of the report includes the work done on the ore and tailing shipments received on October 7, 1938.

CYANIDATION OF ORE SHIPMENTS

Test No. 1

The ores at −14 mesh were ground in a ball mill in cyanide solution of a strength of 1 pound of potassium cyanide per ton to pass 83·5 and 83·7 per cent −200 mesh. The pulps were bottle-agitated for 24 hours.

Screen tests showed the grinding as follows:

Mesh	Weight, per cent	
	Lower grade ore	Higher grade ore
− 65 + 100	1·3	1·4
−100 + 150	7·1	6·9
−150 + 200	8·1	8·0
−200	83·5	83·7
	100·0	100·0

Results of Cyanidation:

Product	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
Lower grade ore...........	24	0·23	0·015	93·5	0·92	0·34	0·37	5·3
Higher grade ore...........	24	0·275	0·015	94·5	0·92	0·36	0·31	5·3

JIG CONCENTRATION AND CYANIDATION OF ORE SHIPMENT

Test No. 2

The ores at −14 mesh were ground in a ball mill in cyanide solution of a strength of 1 pound of potassium cyanide per ton to pass 51·6 per cent −200 mesh. The pulps were passed through a Denver jig and a jig concentrate obtained. This jig concentrate and the jig tailing were separately reground in cyanide solution to pass 97·0 per cent −200 mesh and each bottle was agitated for 48 hours. After the primary grind the jig feeds assayed 0·18 ounce of gold per ton in the lower grade ore and 0·24 ounce of gold per ton in the higher grade.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Jig Concentration of Lower Grade Ore:				
Feed..	100·0	0·18	100·0	....
Jig concentrate..............................	6·7	1·71	63·7	14·9 : 1
Tailing..	93·3	0·07	36·3	....
Jig Concentration of Higher Grade Ore:				
Feed..	100·0	0·24	100·0	....
Jig concentrate..............................	8·2	1·86	63·7	12·2 : 1
Tailing..	91·8	0·095	36·3	....

Cyanidation of Reground Jig Products:

Product	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
Lower grade concentrate	48	1·71	0·04	97·7	2·7	0·15	2·90	24·0
Lower grade tailing....	48	0·07	0·01	85·7	1·0	0·25	0·95	6·5
Higher grade concentrate................	48	1·86	0·05	97·3	2·7	0·20	4·40	23·9
Higher grade tailing...	48	0·095	0·01	89·5	0·7	0·20	1·40	6·6

Summary of Results:

—	Per cent	
	Lower grade ore	Higher grade ore
Gold extracted in primary grind	21·8	12·7
Gold extracted from reground jig concentrate	48·6	54·1
Gold extracted from reground jig tailing	24·3	28·4
Overall extraction	94·7	95·2

TABLE CONCENTRATION AND CYANIDATION OF ORE SHIPMENTS

Test No. 3

A Wilfley table replaced the jig after the primary grind. Conditions otherwise were similar to Test No. 2.

After the primary grind in cyanide solution, the table feeds assayed 0·21 ounce of gold per ton in the lower grade ore and 0·205 ounce of gold per ton in the higher grade.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Table Concentration of Lower Grade Ore:				
Feed	100·0	0·21	100·0	....
Table concentrate	9·3	1·72	76·2	10·7 : 1
Tailing	90·7	0·055	23·8	....
Table Concentration of Higher Grade Ore:				
Feed	100·0	0·205	100·0	....
Table concentrate	12·1	1·48	87·1	8·3 : 1
Tailing	87·9	0·03	12·9	....

Cyanidation of Reground Table Products:

Product	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
Lower grade concentrate	48	1·72	0·065	96·2	3·0	0·25	3·8	21·5
Lower grade tailing	48	0·055	0·01	81·8	1·0	0·20	0·4	5·6
Higher grade concentrate	48	1·48	0·075	94·9	2·7	0·25	3·5	19·5
Higher grade tailing	48	0·03	0·005	83·3	0·8	0·20	0·4	5·3

Summary of Results:

—	Per cent	
	Lower grade ore	Higher grade ore
Gold extracted in primary grind	8·7	25·5
Gold extracted from reground table concentrate	67·0	61·6
Gold extracted from reground table tailing	18·1	8·0
Overall extraction	93·8	95·1

CYANIDATION AND FLOTATION OF CYANIDE RESIDUE

Test No. 4

The cyanide residues of Test No. 1 were washed and conditioned with 1·5 pounds of soda ash and 1·0 pound of copper sulphate per ton and floated by the addition of 0·10 pound amyl xanthate and 0·07 pound of pine oil per ton. The flotation concentrates were washed and reground in cyanide solutions of a strength of 3 pounds of potassium cyanide per ton to pass 97 per cent −325 mesh and the pulps were agitated for 48 hours.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Flotation of Lower Grade Cyanide Residue:				
Feed	100·0	0·015	100·0	....
Flotation concentrate	7·4	0·15	75·3	13·5 : 1
Tailing	92·6	0·004	24·7	....
Flotation of Higher Grade Cyanide Residue:				
Feed	100·0	0·015	100·0	....
Flotation concentrate	8·3	0·14	75·3	12·1 : 1
Tailing	91·7	0·004	24·7	....

Cyanidation of Flotation Concentrates:

Product	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
Lower grade concentrate	48	0·15	0·085	43·3	3·0	0·45	6·9	25·1
Higher grade concentrate	48	0·14	0·09	35·7	2·8	0·35	6·5	24·1

Summary of Results:

—	Per cent	
	Lower grade ore	Higher grade ore
Gold extracted by straight cyanidation	93·5	94·5
Gold extracted from flotation concentrate	1·8	1·5
Overall extraction	95·3	96·0

CYCLE CYANIDATION

Test No. 5

These tests were conducted on the ore samples to ascertain whether continued grinding and agitation resulted in a fouling of the cyanide solution, as indicated mainly by an increase in the amount of gold in the cyanide residues.

The ores at −14 mesh were ground in cyanide solutions of a strength of 1 pound of potassium cyanide per ton to pass 81 per cent −200 mesh, and the pulps were agitated for 24 hours. Sufficient lime was added to the grind and agitation to keep the solutions at a titration of 0·25 to 0·35 pound per ton. After agitation the pulps were filtered, the cyanide residues washed and assayed, and the same cyanide solutions used in the grinding and agitation of fresh portions of the ores. This procedure was carried out for five cycles of grinding and agitation. After the last cycle the solutions were assayed for reducing power and KCNS.

Results:

Product	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
Lower Grade Ore:								
1	24	0·23	0·015	93·5	0·84	0·28	0·50	4·40
2	24	0·23	0·015	93·5	1·00	0·26	0·94	4·22
3	24	0·23	0·015	93·5	1·00	0·34	0·74	3·80
4	24	0·23	0·015	93·5	1·00	0·30	0·82	4·00
5	24	0·23	0·015	93·5	1·00	0·26	0·61	4·10
Higher Grade Ore:								
1	24	0·275	0·015	94·6	0·96	0·35	0·50	4·60
2	24	0·275	0·015	94·6	0·96	0·35	0·82	3·80
3	24	0·275	0·016	94·2	1·00	0·32	0·74	4·00
4	24	0·275	0·020	92·7	1·00	0·32	0·62	4·00
5	24	0·275	0·022	92·0	0·96	0·32	0·70	3·90

An assay of the final cyanide solutions showed:

—	Per cent	
	Lower grade ore	Higher grade ore
Reducing power, in ml. $\frac{N}{10}$ $KMnO_4$/litre	120	136
KCNS, grm./litre	0·08	0·10
Thiosulphates, etc., grm./litre	1·86	1·76

In the lower grade ore sample the cyanide tailings are constant at 0·015 ounce of gold per ton through five cycles of agitation. The higher grade sample, however, shows distinct signs of fouling, or lowered extraction, of the gold, the cyanide residue going from 0·015 ounce of gold per ton in the first cycle to 0·022 ounce in the fifth, and the extraction of the gold falling from 94·6 per cent to 92·0 per cent.

On this account another cycle of tests was run on the higher grade ore. The amount of the lime in the solutions was raised to a titration of 0·6 to 0·8 pound per ton and the filtered solutions were aerated for one hour prior to grinding a fresh batch of ore. Conditions otherwise were similar to the previous cycles of agitation.

Results of Cyanidation:

Higher Grade Ore:

Cycle No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
1	24	0·275	0·015	94·6	1·0	0·70	0·61	6·80
2	24	0·275	0·015	94·6	1·0	0·62	1·23	5·45
3	24	0·275	0·015	94·6	0·96	0·80	1·23	5·25
4	24	0·275	0·015	94·6	1·0	0·68	1·31	5·40
5	24	0·275	0·015	94·6	1·0	0·66	1·31	5·40

Analysis of the final solution resulted as follows:

Reducing power	156 ml. $\frac{N}{10}$ $KMnO_4$/litre
KCNS	0·18 grm./litre
Thiosulphate, etc	1·48 grm./litre

The results of the last cycle test on the higher grade ore sample go to show that high lime, coupled with aeration of the barren solution, succeeds in keeping the cyanide residues at an even 0·015 ounce of gold per ton and the extraction at 94·6 per cent throughout.

The cyanide consumption increase is probably due to the effect of aeration causing oxidation of cyanide. An examination of the final solutions does not show any evidence as to the reason for the more consistent tailing results. Only traces of ferrocyanide were found in these solutions and it can only be presumed that any changed condition of the solution is embraced in the alteration or state of the oxygen-sulphur compounds, such as thiosulphates, etc., that have been formed.

This completes the work on the ore shipments, the remaining tests being performed on the two shipments of mill tailing also received on October 7, 1938, assaying 0·02 and 0·03 ounce of gold per ton, respectively.

SCREEN ANALYSES ON MILL TAILING

Test No. 6

A screen analysis was made on the samples of mill tailing as received. The different sizes were assayed for gold and sulphur.

Results:

Mesh	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S
Lower Grade Tailing Sample:					
− 65 + 100 } −100 + 150 }	8·7	0·01	0·28	3·9	1·8
−150 + 200	10·0	0·01	0·94	4·5	6·7
−200	81·3	0·025	1·56	91·6	91·5
	100·0	0·022	1·39	100·0	100·0
Higher Grade Tailing Sample:					
− 65+100 } −100+150 }	9·2	0·01	0·38	3·0	2·5
−150+200	9·4	0·015	0·91	4·6	6·1
−200	81·4	0·035	1·57	92·4	91·4
	100·0	0·031	1·40	100·0	100·0

INFRASIZING

Test No. 7

The −200-mesh product from the two tailing shipments was passed through the Haultain infrasizer, with the following results:

Lower Grade Sample:

Mesh	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S
+56 } −56+40 }	17·4	0·05	3·32	37·5	36·7
−40+28	11·3	0·03	2·03	14·6	14·6
−28+20	11·9	0·03	1·64	15·5	12·4
−20+14	10·6	0·02	1·36	9·1	9·2
−14+10	10·0	0·015	1·09	6·5	6·9
Dust bag fines	38·8	0·01	0·82	16·8	20·2
Totals	100·0	0·023	1·57	100·0	100·0

Higher Grade Sample:

Mesh	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S
+56 ... } −56+40 ... }	19·5	0·07	3·28	36·9	39·5
−40+28	11·5	0·05	1·94	15·4	13·8
−28+20	11·7	0·05	1·58	15·7	11·4
−20+14	10·5	0·04	1·30	11·4	8·4
−14+10	9·8	0·02	1·07	5·4	6·5
Dust bag fines	37·0	0·015	0·89	15·2	20·4
Totals	100·0	0·037	1·62	100·0	100·0

The results of the infrasizing tests show that the gold follows the sulphides throughout the different sizes and that the distributions are not peculiar to any particular size particle.

FLOTATION AND CYANIDATION

Test No. 8

Following the results obtained in the previous test, it was decided to float off the sulphides in the tailings and to regrind and agitate the sulphide concentrates with a view to improving the overall extraction of the gold.

Accordingly, the tailings were washed, conditioned with 2 pounds of soda ash and 1·0 pound of copper sulphate per ton, and floated with 0·10 pound of amyl xanthate and 0·07 pound of pine oil per ton. The flotation concentrates were washed, reground in cyanide solution of a strength of 3 pounds of potassium cyanide per ton, and the pulps bottle-agitated for 48 hours.

Results:

Product	Weight, per cent	Assay		Distribution per cent		Ratio of concen- tration
		Au, oz./ton	S, per cent	Au	S	
Flotation of Lower Grade Tailing:						
Feed	100·00	0·020	1·38	100·0	100·0	....
Flotation concentrate	6·25	0·245	16·23	76·5	73·5	16 : 1
Tailing	93·75	0·005	0·39	23·5	26·5	....
Flotation of Higher Grade Tailing:						
Feed	100·00	0·030	1·40	100·0	100·0	....
Flotation concentrate	5·87	0·407	16·31	77·1	68·4	17 : 1
Tailing	94·13	0·0075	0·47	22·9	31·6	....

The pH of the pulp was 8·5.

Prior to agitation the flotation concentrates were reground to pass 99·0 −325 mesh.

Results of Cyanidation of Flotation Concentrates:

Product	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
Lower grade concentrate	48	0·245	0·05	79·6	2·9	0·45	4·45	20·2
Higher grade concentrate................	48	0·407	0·06	85·3	2·8	0·40	5·55	20·6

Summary of Results:

—	Per cent	
	Lower grade ore	Higher grade ore
Gold recovered in flotation concentrate..............................	76·5	77·1
Gold extracted from flotation concentrate..............................	60·9	65·8

This additional extraction of the gold gives a final cyanide residue of 0·008 ounce of gold per ton in the lower grade tailing and 0·010 in the higher grade.

CYANIDATION

Test No. 9

The tailings were agitated in cyanide solutions of a strength of 1 pound of potassium cyanide per ton for periods of 24 and 48 hours.

Results of Cyanidation:

Product	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
Lower grade tailing....	24	0·02	0·01	50·0	1·0	0·3	0·3	4·4
Lower grade tailing....	48	0·02	0·01	50·0	1·0	0·25	0·3	4·5
Higher grade tailing...	24	0·03	0·0125	58·3	1·0	0·3	0·3	4·4
Higher grade tailing...	48	0·03	0·01	66·6	0·9	0·25	0·3	4·5

REGRINDING AND AGITATION

Test No. 10

The tailings were reground in ball mills in cyanide solutions of a strength of 1 pound of potassium cyanide per ton to pass 96·4 and 94·2 per cent −200 mesh respectively. The pulps were agitated for different periods of time.

Screen tests showed the grinding as follows:

Mesh	Weight, per cent	
	Lower grade ore	Higher grade ore
−100+150	0·8	0·9
−150+200	2·8	4·9
−200	96·4	94·2
	100·0	100·0

Results of Cyanidation:

Product	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
Lower grade tailing	24	0·02	0·01	50·0	1·0	0·35	0·3	4·3
Lower grade tailing	36	0·02	0·01	50·0	0·9	0·30	0·3	4·4
Lower grade tailing	48	0·02	0·01	50·0	1·0	0·25	0·3	4·5
Higher grade tailing	24	0·03	0·01	66·6	1·0	0·30	0·3	4·4
Higher grade tailing	36	0·03	0·01	66·6	1·0	0·30	0·3	4·4
Higher grade tailing	48	0·03	0·01	66·6	0·9	0·25	0·3	4·5

CONCENTRATION AND MICROSCOPIC EXAMINATION

Test No. 11

Portions of the tailing samples were concentrated on the Haultain superpanner, with the following results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Lower Grade Tailing:				
Feed	100·00	0·02	100·0	
Panner concentrate	1·17	0·86	50·5	85·5 : 1
Tailing	98·83	0·01	49·5	
Higher Grade Tailing:				
Feed	100·00	0·03	100·0	
Panner concentrate	1·33	1·22	52·3	75·2 : 1
Tailing	98·67	0·015	47·7	

Microscopic slides were made of the concentrates and upon examination gave the following information:

The superpanner concentrates from the cyanide tailings of both samples (lower grade and higher grade) are entirely similar. Pyrite predominates, with a considerable quantity of arsenopyrite and rare grains of chalcopyrite and magnetite. Native gold is not visible in the sections.

SUMMARY

On the ore sample received on May 11, 1938, the investigative work showed that a cyanide residue of 0·015 ounce of gold per ton was obtained in 21 hours' agitation at a grind of 81·0 per cent −200 mesh. The cycle cyanidation tests showed little fouling of cyanide solutions after five cycles of grinding and agitation.

The work on the mill products received on June 20, 1938 showed that aeration of the cyanide solutions in a high-lime circuit counteracted the fouling effects of the excess amounts of $FeCO_3$ in these products to some extent. Grinding in water and aeration in a lime pulp prior to cyanidation were also beneficial, the reducing power of the final cyanide solutions being lowered from 260 to 60 in the thickener froth product. Owing to part of the gold in these products being enclosed in arsenopyrite and pyrite, regrinding in cyanide solutions followed by agitation resulted in additional extraction of the gold.

On the ore samples received on October 7, 1938, the cycle tests showed that on the higher grade sample a gradual fouling of the solutions takes place, the extraction of the gold being lowered from a normal 94·5 per cent extraction to 92·0 per cent in the final cycle. A circuit high in lime and aeration of solutions were successful in keeping the extraction constant at 94·5 per cent, the cyanide residue remaining at 0·015 ounce gold per ton throughout. Straight cyanidation of these ores gave extractions of 93·5 and 94·5 per cent respectively at a grind of 81·0 per cent −200 mesh. When a jig was used in concentrating the sulphides after the primary grind and the resulting concentrates were reground and agitated, the overall extraction was raised to 94·7 per cent and 95·2 per cent on the two shipments. Scavenging of the cyanide tailings by means of flotation and regrinding and agitation of the resulting concentrates gave overall extractions of 95·3 and 96·0 per cent respectively and final cyanide residues of 0·011 ounce of gold per ton on both shipments.

On the two tailing shipments received on October 7, 1938, it was shown that by additional agitation the cyanide residue could be reduced from 0·03 and 0·02 ounce of gold per ton to 0·01 ounce in both cases. Concentration by flotation of the sulphides in these tailings, and regrinding and agitation of these concentrates, gave final residues of 0·008 and 0·010 ounce gold per ton. Portions of the tailing samples were concentrated on a Haultain superpanner and the concentrates examined under the microscope. No free gold was visible.

CONCLUSIONS

When an influx of ore high in $FeCO_3$ and graphite takes place, the use of a high-lime circuit and aeration of the barren solutions prior to returning to the mill circuit should result in correction of the fouling conditions and consequent stabilizing of the value of the cyanide residues.

Concentration of the sulphides after the primary grind or a scavenging operation on the cyanide residues should be considered as a means of increasing the extraction.

The results obtained from further agitation of the two samples of mill tailing submitted suggest that the time of agitation could be increased with beneficial results.

Ore Dressing and Metallurgical Investigation No. 763

GOLD ORE FROM THE POWELL ROUYN GOLD MINES, LIMITED, NORANDA, QUEBEC

Shipment. A shipment of 53 sacks of ore, weighing 2,700 pounds, was received on November 1, 1938, from W. E. Leonard, Mine Manager, Powell Rouyn Gold Mines, Limited, Noranda, Quebec.

Previous shipments were received in 1934, covered by Investigation No. 602, Bureau of Mines Report No. 748, and in 1937 covered by an investigation which was not published.

The property of the Powell Rouyn Gold Mines, Limited is situated in Rouyn Township, northwestern Quebec, adjoining the Noranda mine.

Purpose of the Investigation. The company requested an investigation to determine the best method of recovery. The development of the mine has been greatly increased both laterally and vertically since the previous tests. The sample submitted for investigation is stated to be representative of the ore as to grade and character.

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

The *gangue* consists of translucent white quartz and highly siliceous, greenish to brownish grey rock material with a small quantity of fine, disseminated carbonate (calcite), which gives a very slight microchemical test for iron.

The sections are rather sparsely mineralized as regards *metallic minerals.* In their decreasing order of abundance, those present are: pyrite, specular hematite, chalcopyrite, galena, and native gold.

Disseminated pyrite is visible as coarse to fine euhedral to anhedral crystals and grains. Much contains numerous inclusions of gangue, but some is entirely or almost free from such inclusions. A considerable quantity of specular hematite is present in gangue as medium to small, irregular grains and rods, which are so closely compacted locally as to form almost dense masses and stringers. Where included in pyrite it is usually associated with gangue. Small quantities of chalcopyrite and galena occur in gangue and in pyrite as small, irregular grains. The galena is present almost entirely in pyrite and shows a preference for those grains that are more or less free from inclusions of gangue.

Two grains of native gold are visible. Both occur in gangue, one alone, the other against pyrite, and both are 42 microns (−280 mesh) in size.

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

Gold	0·17 oz./ton
Silver	0·17 "
Copper	0·01 per cent
Arsenic	Nil
CaO	1·33 per cent
MgO	0·86 "
SiO_2	80·53 "
Iron	3·15 "
Sulphur	0·92 "

Investigative Procedure. The ore was treated by amalgamation, cyanidation, and concentration, both separately and in combination, to determine the recovery of gold by various methods.

Results. Ninety-seven per cent of the gold can be recovered by flotation, with a ratio of concentration of 30 : 1. Cyanidation of this concentrate gives an extraction of gold of 96 per cent.

Straight cyanidation at a grind of 87 per cent −200 mesh gives the maximum extraction of 94 per cent within 24 hours.

Amalgamation tests made on grinds varying from 33 per cent to 93 per cent −200 mesh show that from 61·8 to 70·6 per cent of the gold is in the free state.

Removal of free gold by jigs or traps before agitation results in the minimum tailing being obtained within 24 hours.

Experimental Tests

The results of the investigation in detail follow:

TRAP CONCENTRATION

Test No. 1

To determine the presence of free-milling gold in the ore, a sample was ground 68 per cent −200 mesh in a ball mill at a dilution of 4 parts of ore to 3 parts of water. The ground pulp was passed over a hydraulic classifier.

The concentrate obtained weighed approximately 0·3 per cent of the feed. It was examined microscopically and was found to contain free gold. The concentrate was amalgamated. After separating the mercury, the amalgamated residue was returned to the trap tailing, mixed, and filtered. A sample from this product was assayed.

Results:

Assay, Au, oz./ton		Recovery of gold, per cent
Feed	Tailing	
0·17	0·11	35·3

AMALGAMATION FOLLOWED BY CYANIDATION

Test No. 2

This was made to note the amount of amalgamable gold recoverable at various degrees of grinding followed by cyanidation of the residues.

Amalgamation

Samples of ore were ground for different lengths of time in ball mills at a dilution of 4 : 3 (water grind).

The pulps were amalgamated by barrel amalgamation with 10 per cent by weight of mercury and 2·0 pounds of lime per ton of ore. After removing the mercury, and sampling, the remaining pulp was held for cyanidation tests.

Results of Amalgamation:

Test No.	Grind, per cent −200 mesh	Assay, Au, oz./ton		Recovery, per cent
		Feed	Tailing	
2A	33	0·17	0·065	61·8
2B	39	0·17	0·065	61·8
2C	57	0·17	0·065	61·8
2D	73	0·17	0·065	61·8
2E	83	0·17	0·06	64·7
2F	93	0·17	0·05	70·6

The test shows that 61 per cent of the gold is freed at a grind of 33 per cent −200 mesh. The gold apparently is rather coarse.

Cyanidation of Amalgamation Residues

Samples from each amalgamation tailing were repulped in cyanide solution immediately after filtering. The dilution was 1 part of solid to 2 parts of cyanide solution (1·0 pound of sodium cyanide per ton strength). Lime was used to give protective alkalinity to the solution.

The extraction of gold was noted at varying periods of agitation.

Results:

Grind, per cent −200 mesh	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton ore		Final titration, lb./ton solution	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
33	16	0·065	0·025	61·5	0·42	3·35	1·3	0·70
33	24	0·065	0·025	61·5	0·46	3·50	1·3	0·70
39	16	0·065	0·025	61·5	0·40	3·75	1·3	0·65
39	24	0·065	0·025	61·5	0·40	3·85	1·3	0·65
57	16	0·065	0·015	76·9	0·40	3·60	1·3	0·70
57	24	0·065	0·015	76·9	0·41	3·65	1·3	0·70
73	16	0·065	0·015	76·9	0·16	2·20	0·9	0·35
73	24	0·065	0·01	84·6	0·20	2·30	0·8	0·3
73	48	0·065	0·01	84·6	0·20	2·30	0·8	0·3
83	16	0·06	0·01	83·3	0·20	2·20	0·8	0·35
83	24	0·06	0·01	83·3	0·20	2·35	0·8	0·30
83	48	0·06	0·01	83·3	0·20	2·40	0·8	0·25
93	16	0·05	0·01	80·0	0·22	2·30	0·8	0·35
93	24	0·05	0·01	80·0	0·22	2·40	0·8	0·30
93	48	0·05	0·01	80·0	0·24	2·60	0·7	0·25

Summary:

Grind, per cent −200 mesh	Recovery, per cent		
	By amalgamation	By cyanidation	Overall
33	61·8	23·5	85·3
39	61·8	23·5	85·3
57	61·8	29·4	91·2
73	61·8	32·4	94·1
83	64·7	29·4	94·1
93	70·6	23·5	94·1

The tests show that a minimum tailing of 0·01 ounce of gold per ton was obtained within 24 hours at a grind of 73 per cent −200 mesh. Finer grinding gave slightly higher recoveries by amalgamation but the overall extraction by cyanidation of the residues remains the same.

STRAIGHT CYANIDATION

Test No. 3

This was made to note the response of the ore to straight cyanidation.

Samples of the ore were ground in cyanide solution, dilution 4 : 3, for various lengths of time. Samples from each grind were diluted to 1 part of solid to 2 parts of cyanide solution (1·0 pound of sodium cyanide per ton) and agitated for 16, 24, and 48 hours.

The solution from the grind was made up to the required volume and strength for use in agitating the pulp.

In one series of tests the effect of adding litharge and lead nitrate to the solutions during agitation was noted.

Results:

Test No.	Grind, per cent −200 mesh	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton ore		Final titration, lb./ton solution	
			Feed	Tailing		NaCN	CaO	NaCN	CaO
3A	55	16	0·17	0·025	85·3	0·24	5·85	1·0	0·50
3B	55	24	0·17	0·015	91·2	0·26	5·95	1·0	0·65
3C	55	48	0·17	0·015	91·2	0·28	6·00	1·0	0·60
3D	72	16	0·17	0·02	88·2	0·24	6·70	1·0	0·75
3E	72	24	0·17	0·015	91·2	0·25	6·90	1·0	0·65
3F	72	48	0·17	0·015	91·2	0·25	7·00	1·0	0·65
3G	86	16	0·17	0·015	91·2	0·3	7·60	1·0	0·65
3H	86	24	0·17	0·015	91·2	0·3	7·65	1·0	0·65
3I	86	48	0·17	0·015	91·2	0·4	7·75	0·95	0·65

In the series Test No. 3 A to 3 I, the lime added for the grind and agitation period was 6·75 pounds per ton of ore.

In the following series, litharge and lead nitrate were used in the solutions. The grind used in these tests was 87 per cent −200 mesh.

Results:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton ore				Final titration, lb./ton solution	
		Feed	Tailing		NaCN	CaO	PbO	$Pb(NO_3)_2$	NaCN	CaO
3J	16	0·17	0·015	91·2	0·44	2·60	....	....	0·8	0·15
3K	16	0·17	0·015	91·2	0·44	2·60	1·0	....	0·8	0·15
3L	16	0·17	0·015	91·2	0·44	2·60	....	1·0	0·8	0·15
3M	24	0·17	0·01	94·1	0·44	2·60	....	....	0·8	0·15
3N	24	0·17	0·01	94·1	0·44	2·60	1·0	....	0·8	0·15
3O	24	0·17	0·01	94·1	0·44	2·60	....	1·0	0·8	0·15
3P	48	0·17	0·01	94·1	0·45	2·75	....	....	0·8	0·10
3Q	48	0·17	0·01	94·1	0·45	2·75	1·0	....	0·8	0·10
3R	48	0·17	0·01	94·1	0·45	2·75	....	1·0	0·8	0·10

The use of lead salts did not increase the extraction.

In the series Test No. 3 J to 3 R, the lime added for the grind and agitation period was 5·0 pounds per ton of ore.

The results of the two series show the effect of high and low lime in the solution and indicate that the minimum tailing is obtained when the lime is low.

INFRASIZER ANALYSES OF A CYANIDE TAILING

Test No. 4

This was made at a grind of 97 per cent −200 mesh and a portion of the cyanide tailing was classified by the infrasizer to note the distribution of gold in the ore particles of various sizes below 200 mesh.

A sample of ore was ground in cyanide solution, 1·0 pound of sodium cyanide per ton, using lime for protective alkalinity. After grinding, the pulp was diluted to 1 part of solid to 1·5 parts of solution (1·0 pound of sodium cyanide per ton) and agitated for 24 hours.

Results of Cyanidation:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton ore		Final titration, lb./ton solution	
Feed	Tailing		NaCN	CaO	NaCN	CaO
0·17	0·01	94·1	0·81	4·85	1·0	0·20

Infrasizer Test

A sample of the tailing was classified by the Haultain infrasizer and the various products were assayed for gold and sulphur.

Results:

Products, size in microns	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S
Feed	100·00	0·011	1·17	100·00	100·00
+56	1·81	0·025	2·41	22·68	21·53
−56+40	8·69				
−40+28	15·65	0·01	1·10	13·52	14·65
−28+20	16·43	0·01	0·87	14·19	12·16
−20+14	13·89	0·01	0·80	12·00	9·45
−14+10	10·60	0·01	0·86	9·16	7·77
−10	32·93	0·01	1·23	28·45	34·45

This test indicates that gold is enclosed in both sulphide and gangue particles down to −10 microns in size. A tailing of 0·01 ounce of gold per ton is the lowest that can be realized by cyanidation.

CYANIDATION CYCLE TEST

Test No. 5

This was made to note whether a uniform extraction could be maintained.

A sample of ore was ground to 77 per cent −200 mesh in cyanide solution (1·0 pound of sodium cyanide per ton). The pulp was diluted to 1 part of solid to 1·5 parts of solution, made up to 1·0 pound of sodium cyanide per ton, and agitated for 24 hours.

After filtering, the solution was made up to the original volume and strength, and the required amount was used to grind the second charge of ore. The remaining solution was used to dilute the pulp for the agitation period.

The same grind and procedure was used in each cycle.

The cyanide tailing was divided into two portions, one of which was sampled and assayed and the other amalgamated in order to remove any particles of undissolved gold. The amalgamation tailing was sampled and assayed. It was noted that there was a lower amalgamation tailing in the first cycle only.

In order to produce the most adverse conditions, the gold in the solution was not precipitated during the test. The solution from the final cycle was analysed to determine the amount of fouling.

Results:

Cycle No.	Tailing, A, cyanidation; B, amalgamation	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton ore		Final titration, lb./ton solution	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
1	A	0·17	0·015	91·2	0·30	3·40	0·88	0·38
	B	0·17	0·01	94·1	....	....	....	....
2	A	0·17	0·015	91·2	0·36	2·10	0·90	0·30
	B	0·17	0·015					
3	A	0·17	0·015	91·2	0·36	2·15	1·06	0·20
	B	0·17	0·015					
4	A	0·17	0·015	91·2	0·32	2·00	1·04	0·26
	B	0·17	0·015					
5	A	0·17	0·015	91·2	0·40	2·04	1·00	0·24
	B	0·17	0·015					

Analysis of Final Solution:

Reducing power	72 ml. $\frac{N}{10}$ $KMnO_4$/litre
NaCNS	0·07 grm./litre
Ferrous iron	0·04 "
Total alkalinity	0·74 lb. /ton (equivalent to CaO)

The test shows no evidence of fouling after five cycles. A uniform extraction was also obtained.

The minimum tailing of 0·01 ounce per ton was not attained in this test, in which the grind was 77 per cent −200 mesh. Previous tests (Test No. 3) showed that 87 per cent −200 mesh was necessary to get maximum extraction.

CYANIDATION FOLLOWED BY REGRINDING OF THE SAND TAILING

Test No. 6

Previous cyanidation tests show that the gold is associated with both gangue and sulphide.

This test was made to note whether the slime portion of the tailing could be rejected and a further extraction obtained by regrinding and re-cyaniding the sand portion of the tailing.

A sample of ore was ground to 75 per cent −200 mesh in cyanide solution and agitated for 24 hours in a solution having 1·0 pound of sodium cyanide per ton.

The tailing was filtered, sampled, and deslimed on a Wilfley table. The table concentrate and sand were reground to 93 per cent −200 mesh and cyanided for 18 hours. The slime was sampled and rejected. A screen test shows these to be 87 per cent −325 mesh.

Cyanidation of the Ore:

Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton ore		Final titration, lb./ton solution	
Feed	Tailing		NaCN	CaO	NaCN	CaO
0·17	0·02	88·24	0·80	4·80	1·00	0·16

Table Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Units	Distribution, per cent
Table feed	100·00	0·02	2·0000	100·0
Table sand	47·85	0·031	1·4785	73·9
Table slime	52·15	0·01	0·5215	26·1

Cyanidation of Reground Sands:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton ore		Final titration, lb./ton solution	
Feed	Tailing		NaCN	CaO	NaCN	CaO
0·031	0·01	67·74	0·31	2·10	0·92	0·14

Summary:

	Per cent
Extraction from ore	88·24
Extraction from reground sand	5·89
Total extraction	94·13
Gold in sand tailing	2·80
Gold in slime tailing	3·07
	100·00

Assay of combined tailing.......Au, 0·01 oz./ton

The test shows that the slime has a value of 0·01 ounce of gold per ton after 24 hours of agitation. The reground sand is reduced to the same value within 18 hours.

These results indicate that an 0·01 ounce tailing is the lowest that can be obtained at an economic degree of grinding.

SETTLING TESTS

Test No. 7

A sample of ore was ground in cyanide solution to 80 per cent −200 mesh and agitated for 18 hours at a dilution of 1 : 2 in a solution having 1·0 pound of sodium cyanide per ton, using lime for protective alkalinity.

The pulp was transferred to a tall glass cylinder, 2 inches inside diameter, with a graduated scale. The pulp level was read at five-minute intervals for one hour.

The dilution was then reduced to 1 : 1·5 and the readings were repeated.

The tailing assayed 0·01 ounce gold per ton. The extraction was 94·1 per cent.

Results of Settling Tests:

Time, minutes	Dilution of 1 : 2		Dilution of 1 : 1·5	
	Pulp level	Drop, feet	Pulp level	Drop, feet
0	3·10	0·00	2·32	0·00
5	2·95	0·15	2·23	0·09
10	2·85	0·10	2·185	0·045
15	2·73	0·12	2·12	0·065
20	2·61	0·12	2·075	0·045
25	2·47	0·14	2·01	0·065
30	2·33	0·14	1·965	0·045
35	2·19	0·14	1·91	0·055
40	2·08	0·11	1·865	0·045
45	1·97	0·11	1·81	0·055
50	1·88	0·09	1·75	0·06
55	1·78	0·10	1·69	0·06
60	1·69	0·09	1·63	0·06
Rate of settling		1·41 ft./hr.		0·69 ft./hr.

—	Dilution of 1 : 2	Dilution of 1 : 1·5
NaCN, lb./ton of solution	0·92	0·92
CaO, lb./ton of solution	0·44	0·44
Overflow	Clear	Clear
Critical point	50 minutes	30 minutes

CYANIDATION OF JIG TAILING

Test No. 8

This was to note the effect of removing free gold by jigging.

The sample of ore was ground in water to 85 per cent −200 mesh and jigged on a Denver Laboratory Mineral Jig. The jig concentrate was amalgamated and the amalgamated residue was mixed with the jig tailing and filtered.

Three portions of this material were repulped in cyanide solution (1·0 pound of sodium cyanide per ton) at a dilution of 1 : 2 and agitated for 16, 24, and 48 hours.

Results:

Jig feed	Au, 0·17 oz./ton
Jig tailing	Au, 0·115 oz./ton
Recovery	32·35 per cent

Cyanidation:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton ore		Final titration, lb./ton solution	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
16	0·115	0·015	87·0	0·30	2·70	0·8	0·25
24	0·115	0·01	91·3	0·30	3·70	0·8	0·30
48	0·115	0·01	91·3	0·35	4·70	0·8	0·40

Recovery by amalgamation..................................	32·35 per cent
Extraction by cyanidation, 67·65×91·3........................	61·76 "
Overall extraction..	94·11 " within 24 hours.

CONCENTRATION

Several flotation tests were made at various grinds, to note the response of the ore to flotation.

FLOTATION

Test No. 9

A sample of ore was ground in water, dilution 4 : 3, with 2 pounds of soda ash per ton to 87 per cent −200 mesh.

After transferring the pulp to a flotation cell it was conditioned for 6 minutes with 0·1 pound of potassium amyl xanthate, then 0·087 pound of pine oil per ton was added and the agitation continued for 3 minutes. A good froth was obtained and the concentrate was removed.

The pulp was conditioned for 5 minutes with 0·5 pound of copper sulphate per ton, 3 minutes with 0·1 pound of amyl xanthate and 0·058 pound of pine oil per ton, and a further concentrate was recovered.

The two concentrates were recleaned together in a smaller cell without reagents, producing a final concentrate and a cleaner tailing (middling).

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed............................	100·00	0·17	17·4686	100·00	....
Flotation concentrate............	1·79	8·72	15·6088	89·35	55·9 : 1
Cleaner tailing..................	1·28	0·885	1·1328	6·48	78·1 : 1
Flotation tailing................	96·93	0·0075	0·7270	4·17	....

Combining the concentrate and cleaner tailing, gives the following values:

Feed............................	100·00	0·17	17·4686	100·00	....
Rougher concentrate............	3·07	5·45	16·7416	95·84	32·6 : 1
Flotation tailing................	96·93	0·0075	0·7270	4·16	....

JIGGING FOLLOWED BY FLOTATION

Test No. 10

Samples of ore were ground in water to 60, 74, and 80 per cent −200 mesh and were then jigged.

The jig tailings were repulped in a flotation cell and conditioned for 20 minutes with 2 pounds of soda ash per ton, for 6 minutes with 0·1 pound of amyl xanthate per ton, and then after adding 0·1 pound of pine oil per ton a concentrate was removed. The addition of 0·2 pound of copper sulphate and 0·1 pound of amyl xanthate per ton resulted in obtaining a small amount of concentrate.

The results of jigging and flotation at 60 and 80 per cent −200 mesh were as follows:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·17	17·4079	100·00	....
Jig and flotation concentrates	1·91	8·46	16·1586	92·82	52·4 : 1
Cleaner tailing	0·59	0·465	0·2743	1·58	170 : 1
Flotation tailing, 60 per cent −200 mesh	48·76	0·01	0·4876	2·80	....
Flotation tailing, 80 per cent −200 mesh	48·74	0·01	0·4874	2·80	....
Bulk concentrate	2·50	6·57	16·4329	94·40	40 : 1

The rougher concentrate was cleaned and the jig concentrate was added to the flotation concentrate. Removals of the coarse gold by jigging did not give a lower tailing than did straight flotation.

The results of jigging and flotation at 74 per cent −200 mesh were as follows:

Recovery by Amalgamation of Jig Concentrate:

Feed	0·17 Au, oz./ton
Amalgamation tailing	0·11 "
Recovery	35·3 per cent

Flotation of Jig Residue:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed (jig tailing)	100·00	0·11	10·943	100·00	....
Flotation concentrate	2·12	4·70	9·964	91·05	47 : 1
Flotation tailing	97·88	0·01	0·979	8·95	....

Recovery of gold by jig (as amalgam)	35·3 per cent
Recovery of gold in flotation concentrate, 91·05×64·7	58·9 "
Loss of gold in flotation tailing	5·8 "
	100·0 "

FLOTATION

Test No. 11

Several tests were made to determine the maximum recovery of gold by flotation at different grinds with different reagents.

Samples of ore were ground for various times in ball mills with 2·0 pounds of soda ash per ton at a dilution of 4 parts solids to 3 parts water.

In Test No. 11 A the ore was ground 78 per cent −200 mesh, and 0·2 pound of amyl xanthate and 0·145 pound of pine oil per ton were added to the cell, and a concentrate was removed.

Flotation Test No. 11 A:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·17	17·0000	100·00	....
Flotation concentrate	2·89	5·55	16·0289	94·3	34·6 : 1
Flotation tailing	97·11	0·01	0·9711	5·7	....

In Test No. 11 B the ore was ground 84 per cent −200 mesh and the same reagents were used.

Flotation Test No. 11 B:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·17	17·0000	100·00	....
Flotation concentrate	3·17	5·06	16·0317	94·3	31·5 : 1
Flotation tailing	96·83	0·01	0·9683	5·7	....

In Test No. 11 C the ore was ground 81 per cent −200 mesh. After adding 0·1 pound of amyl xanthate and 0·145 pound of pine oil per ton and removing as much concentrate as possible, 0·2 pound of copper sulphate and 0·1 pound of amyl xanthate per ton were added to the cell. A small amount of concentrate was obtained.

Flotation Test No. 11 C:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·17	17·0000	100·00	....
Flotation concentrate	3·45	4·51	15·5517	91·48	29 : 1
Flotation tailing	96·55	0·015	1·4483	8·52	....

In Test No. 11 D the ore was ground to 85 per cent −200 mesh and the same reagents were added to the pulp as in Test No. 11 C.

Flotation Test No. 11 D:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·17	17·000	100·00	....
Flotation concentrate	3·50	4·58	16·035	94·32	28·6 : 1
Flotation tailing	96·50	0·01	0·965	5·68	....

Copper sulphate apparently is of no benefit, as a lower grade concentrate and somewhat higher tailing loss were obtained.

A sample of the flotation tailing from Test No. 11 D was panned on a Haultain superpanner. The panner concentrate was examined microscopically and no free gold was observed. The panner tailing was assayed in duplicate and was found to contain 0·01 ounce of gold per ton.

Test No. 12

This was to check previous flotation tests by noting whether a flotation tailing of lower than 0·01 ounce of gold per ton could be obtained.

Test No. 12 A

A sample of ore was ground to 78 per cent −200 mesh, 4 : 3 dilution, with 2·0 pounds of soda ash and 0·132 pound of Barrett No. 4 oil per ton.

The pulp was floated with 0·2 pound of amyl xanthate and 0·116 pound of pine oil per ton. The method used was to add half of the xanthate, condition for 5 minutes, add the pine oil and remove the concentrate obtained. The remaining xanthate was then added, conditioned, and a further amount of concentrate was removed.

Test No. 12 A:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·17	17·0000	100·00	....
Flotation concentrate	3·18	5·04	16·0318	94·30	31·5 : 1
Flotation tailing	96·82	0·01	0·9882	5·70	....

Test No. 12 B

This was similar in detail to the above except that the ore was ground 86 per cent −200 mesh.

Test No. 12 B:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·17	17·000	100·00	....
Flotation concentrate	3·00	5·51	16·515	97·15	33·3 : 1
Flotation tailing	97·00	0·005	0·485	2·85	....

Tests No. 12 C and 12 D

These were similar to Test No. 12 B. The ore was ground 85 per cent −200 mesh and under the same conditions with two samples to check the results obtained.

Test No. 12 C:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·17	17·0000	100·00	....
Flotation concentrate	3·48	4·61	16·0348	94·32	28·7 : 1
Flotation tailing	96·52	0·01	0·9652	5·68	....

Test No. 12 D:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·17	17·0000	100·00	....
Flotation concentrate	3·08	5·20	16·0308	94·30	32·5 : 1
Flotation tailing	96·92	0·01	0·9692	5·70	....

The tailings from Tests No. 12 C and 12 D were assayed in duplicate.

Although the conditions of flotation were maintained as closely as possible, a tailing of 0·005 ounce of gold per ton was not realized.

FLOTATION FOLLOWED BY DESLIMING

Test No. 13

This was to note whether the slime portion of the flotation tailing could be rejected.

A sample of ore was ground to 80 per cent −200 mesh with 2·0 pounds of soda ash per ton, dilution 4 : 3.

The reagents used in flotation were 0·2 pound of amyl xanthate and 0·1 pound of pine oil per ton. The flotation was continued until a froth barren of sulphides was obtained. The flotation tailing was filtered and deslimed on a Wilfley table.

The table sand was reground to 93 per cent −200 mesh, returned to the flotation cell, and floated with the same reagents. No appreciable amount of sulphides appeared in the froth.

Both concentrates were combined.

A screen test showed the slime to be 80 per cent −325 mesh.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·19*	18·87	100·00	....
Flotation concentrate	3·86	4·62	17·83	94·49	26 : 1
Sand tailing	53·40	0·0075	0·40	2·12	....
Slime tailing	42·74	0·015	0·64	3·39	....
Combined tailing	96·14	0·011	1·04	5·51	....

*Feed assay calculated from the products.

It is apparent that no benefit is to be derived by discarding the slime portion of the flotation tailing.

FLOTATION FOLLOWED BY CYANIDATION OF THE CONCENTRATE

Test No. 14

This was to accumulate concentrate for a cyanidation test. Several charges of ore were treated with varying amounts of different reagents to note their effect on producing a minimum flotation tailing. The amounts added to each sample of pulp are shown in the table of results.

The concentrates were combined for cyanidation, filtered, sampled, then reground to 99·5 per cent −325 mesh in a solution of 3·0 pounds of sodium cyanide per ton. The pulp was split into two parts, one was cyanided for 48 hours and the other for 72 hours in a solution of 3·0 pounds of sodium cyanide per ton at a dilution of 1 : 3. Lime was used for protective alkalinity during the grind and agitation.

The cyanide solution from the 72-hour test was analysed for indications of fouling.

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration	Flotation reagents, lb./ton dry weight of ore		
						$CuSO_4$	Amyl xanthate	Pine oil
Feed	100·00	0·21	20·733	100·00	....	....	....	....
Concentrate	3·37	5·76	19·411	93·63	30 : 1	....	....	....
Tailings 1, 2, and 3	47·70	0·02	0·954	4·60	....	0·3	0·2	0·116
Tailing 4	16·30	0·005	0·082	0·40	....	...... 0·2	0·1 0·1	0·116 0·058
Tailing 5	16·26	0·0075	0·122	0·59	....	...... 0·1	0·1 0·1	0·087 ....
Tailing 6	16·37	0·01	0·164	0·78	....	None	0·1 0·1	0·116 ....
Combined tailings	96·63	0·014	1·322	6·37	....	....	....	....

In Samples 1, 2, and 3 the same reagents were used. Copper sulphate was added to the cell and conditioned before adding the xanthate, which was followed by pine oil.

In Sample 4 no copper sulphate was added to the cell until the concentrate obtained with 0·1 pound of amyl xanthate and 0·116 pound of pine oil per ton was removed, then 0·2 pound was added and conditioned, followed by the addition of 0·1 pound of amyl xanthate and 0·058 pound of pine oil per ton.

In Sample 5 the copper sulphate was reduced to 0·1 pound per ton and the second addition of pine oil was omitted.

In Sample 6 no copper sulphate was added.

The effect of adding copper sulphate to the pulp in Samples 1, 2, and 3 appears to be to depress gold into the tailing. The best results were obtained in this test when 0·2 pound per ton was added to the cell after removing the main portion of the concentrate.

Cyanidation of Flotation Concentrate:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Final titration, lb./ton solution	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
14A	48	5·76	0·405	92·97	9·40	15·80	2·6	0·30
14B	72	5·76	0·425	92·62	12·10	18·60	2·2	0·25

Copper in concentrate........ 0·28 per cent

Summary of Overall Results, Test No. 14:

—	48-hour Test	72-hour Test
	Per cent	Per cent
Gold recovered in flotation concentrate	93·63	93·63
Gold extracted by cyanidation—		
93·63×92·97	87·05	....
93·63×92·62	....	86·72
Gold loss in flotation tailing	6·37	6·37
Gold loss in cyanide tailing—		
93·63×7·03	6·58	....
93·63×7·38	....	6·91
Total	100·00	100·00

Combined tailing loss, Au, oz./ton	0·027	0·028
Cyanide consumed, lb./ton ore	0·32	0·41
CaO consumed, lb./ton ore	0·53	0·62

Analysis of Cyanide Solution from the 72-hour Test:

Reducing power	600 ml. $\frac{N}{10}$ $KMnO_4$/litre
NaCNS	0·53 grm./litre
Ferrous iron	0·04 "
Total alkalinity	1·85 lb./ton as CaO

Test No. 14 B indicated that no increase in extraction was obtained after 48 hours' agitation.

Test No. 15

This was to note the effect of shorter periods of agitation on the extraction of gold from the concentrate.

The ore was ground 70, 78, and 86 per cent −200 mesh. Two tests at each grind were made, one with and the other without copper sulphate.

The concentrate from the six samples of ore was combined for cyanidation, reground to 99·5 per cent −325 mesh in cyanide solution and cyanided in two portions, 1 : 3 dilution, in a solution of 3·0 pounds of sodium cyanide per ton for 16 hours and 24 hours respectively.

In detail this test followed Test No. 14 as closely as possible.

Flotation:

Product	Grind, per per cent −200 mesh	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Flotation reagents added to flotation cell for each pulp, lb./ton dry weight ore		
						$CuSO_4$	Amyl xanthate	Pine oil
Feed	....	100·00	0·17	17·1011	100·00	....	....	....
Concentrate		3·11	5·20	16·1720	94·57	....	....	....
Tailing 1	70	16·22	0·025	0·4055	2·37	None	0·15	0·116
Tailing 2	70	16·11	0·01	0·1611	0·94	0·2	0·15	0·116
Tailing 3	78	16·13	0·0075	0·1210	0·71	None	0·15	0·116
Tailing 4	78	16·10	0·005	0·0805	0·47	0·2	0·15	0·116
Tailing 5	86	16·18	0·005	0·0805	0·47	None	0·2	0·116
Tailing 6	86	16·15	0·005	0·0805	0·47	0·2	0·2	0·116
Combined tailings		96·89	0·0096	0·9291	5·43	....	....	....

Ratio of concentration .... 32 : 1
Copper in the flotation concentrate .... 0·14 per cent

Cyanidation of Flotation Concentrate:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton concentrate		Final titration, lb./ton solution	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
15A	16	5·20	0·29	94·42	5·95	11·1	2·8	0·5
15B	24	5·20	0·29	94·42	6·05	11·8	2·7	0·35

Overall extraction, 94·57×94·42	89·29 per cent
Loss in flotation tailing	5·43 "
Loss in cyanide tailing, 94·57×5·58	5·28 "
	100·00 "

Assay of combined tailing .... 0·019 Au, oz./ton

—	Test No. 15A	Test No. 15B
NaCN consumed, lb./ton ore	0·186	0·189
CaO consumed, lb./ton ore	0·35	0·37
Reducing power of the solution, in ml. $\frac{N}{10}$ $KMnO_4$/litre	328	374

The test indicates that grinding to approximately 80 per cent −200 mesh is required to produce a minimum flotation tailing. In the coarser grinds the use of copper sulphate gave the lowest tailing, but in the finest grind (86 per cent −200 mesh) the same result was obtained without it.

The results indicate that no increase in extraction is obtained after 16 hours' agitation.

Test No. 16

This was to show the effect of regrinding the flotation concentrate in water, prior to agitation.

Samples of ore were ground to 80 per cent −200 mesh with 2·0 pounds of soda ash per ton and floated with 0·1 pound of potassium amyl xanthate and 0·116 pound of pine oil per ton. No copper sulphate was used.

The combined concentrate was reground 99·5 per cent −325 mesh in water, filtered, washed, and repulped in a solution of 3·0 pounds of sodium cyanide per ton. The pulp was agitated 48 hours. Lime was used for protective alkalinity and during the grind and agitation.

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·188	18·8412	100·00	....
Flotation concentrate	3·05	5·82	17·7510	94·21	33 : 1
Tailing 1	24·16	0·01	0·2416	1·29	....
Tailing 2	24·11	0·01	0·2411	1·28	....
Tailing 3	24·54	0·01	0·2454	1·30	....
Tailing 4	24·14	0·015	0·3621	1·92	....
Combined tailing	96·95	0·0112	1·0902	5·79	....

Copper in flotation concentrate.................................... 0·32 per cent

Cyanidation of Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Final titration, lb./ton solution	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
48	5·82	0·185	96·82	8·30	3·45	2·7	0·4

Lime used in grind.................................... 14·2 lb./ton concentrate
Overall extraction of gold, 94·21×96·82................ 91·21 per cent
Loss of gold in flotation tailing....................... 5·79 "
Loss of gold in cyanide tailing, 94·21×3·18............ 3·00 "
100·00 "

Combined tailing assay.............................. Au, 0·163 oz./ton

Cyanide Solution Analysed:

Reducing power...................................... 480 ml. $\frac{N}{10}$ $KMnO_4$/litre
NaCNS.. 0·53 grm./litre
Ferrous iron.. Trace
Total alkalinity..................................... 2·24 lb./ton equivalent to CaO
Consumption of cyanide per ton of ore................. 0·25 lb.
Lime consumed: Grind, 0·43 lb.; agitation, 0·11 lb.;—total of 0·54 lb./ton ore.

The results of flotation show that more than 0·1 pound of amyl xanthate per ton was required.

The increased extraction over Test No. 14 A may be due to grinding in water and washing out cyanicides by filtering.

An infrasizer analysis was made on the cyanide residues from Tests Nos. 14, 15, and 16. The residues were combined and classified by the infrasizer. The various products were assayed for gold and sulphur.

Results:

Products (microns)	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S
Feed	100·00	0·3596	27·48	100·00	100·00
+56	0·17	1·10	*	6·47	....
−56+40	1·93				
−40+28	5·47	0·80	49·04	12·18	9·76
−28+20	8·48	0·56	46·92	13·21	14·48
−20+14	10·53	0·40	40·82	11·71	15·64
−14+10	11·47	0·31	34·21	9·90	14·28
−10	61·95	0·27	20·34	46·53	45·84

*All material was used for the gold assay.

The results of the flotation tests as shown in Test No. 15 indicate the possibility of obtaining a flotation tailing of 0·005 ounce of gold per ton. This represents a recovery of 97·1 per cent of the gold by flotation.

Cyanidation extracts 96·8 per cent of the gold in the concentrate, or an overall recovery of 94 per cent.

Test No. 17

This was to check results of grinding the concentrate in water, and to note the extraction within 24 hours.

Samples of ore were floated similarly to that of Test No. 12 B, in which a tailing of 0·005 ounce of gold per ton was obtained.

The concentrate was reground in water with lime to 99·3 per cent −325 mesh, filtered, washed, and repulped in a solution of 3·0 pounds of sodium cyanide per ton solution at a dilution of 1 : 3, and agitated for 24 hours. Lime was used for protective alkalinity.

Flotation:

Reagents to Ball Mill (grind 86 per cent −200 mesh):

Soda ash	2·0 lb./ton
Barrett No. 4 oil	0·132 "

Reagents to the Flotation Cell:

Potassium amyl xanthate	0·1 lb./ton
Pine oil	0·116 "

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Units, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·17	17·0	100·00	....
Concentrate	3·13	5·25	16·4344	96·68	32 : 1
Tailing 1	32·26	0·005	0·1618	9·95	....
Tailing 2	32·28	0·0075	0·2421	1·42	....
Tailing 3	32·33	0·005	0·1617	0·95	....
Combined tailings	96·87	0·0058	0·5656	3·32	....

Copper in flotation concentrate.................................... 0·32 per cent

Cyanidation:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Final titration, lb./ton solution	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
24	5·25	0·35	93·33	7·47	4·33	2·7	0·35

Lime used in grind.................................... 21·3 lb./ton concentrate

Overall extraction of gold, 96·68×93·33............................ 90·23 per cent
Loss of gold in flotation tailing.................................... 3·32 "
Loss of gold in cyanide tailing, 96·68×6·67........................ 6·45 "

100·00 "

Assay of combined tailing.. Au, 0·0166 oz./ton

Analysis of Cyanide Solution:

Reducing power.................................... 492 ml. $\frac{N}{10}$ $KMnO_4$/litre
NaCNS.................................... 0·52 grm./litre
Ferrous iron.................................... Trace
Total alkalinity.................................... 2·21 lb./ton as CaO
Cyanide consumed.................................... 0·233 lb./ton ore

Lime Consumed:

	Lb./ton ore
In agitation	0·14
In grind	0·67
Total lime consumed	0·81

It was noted that a lower cyanide tailing was obtained in Test No. 16 in which the concentrate was cyanided for 48 hours.

With the extraction of 96·8 per cent of the gold in the concentrate as obtained in Test No. 16, this test shows an overall extraction of 93·6 per cent.

Summary of Results of Cyanidation of Flotation Concentrates

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent		Reagents consumed				Final titration, lb./ton solution	
						Lb./ton concentrate		Lb./ton ore			
		Feed	Tailing	From concentrate	Over-all	NaCN	CaO	NaCN	CaO	NaCN	CaO
Grinding in Cyanide Solution:											
15A........	16	5·20	0·29	94·4	89·3	5·95	11·1	0·186	0·35	2·8	0·5
15B........	24	5·20	0·29	94·4	89·3	6·05	11·8	0·189	0·37	2·7	0·35
14A........	48	5·76	0·405	93·0	87·1	9·40	15·8	0·32	0·53	2·6	0·3
14B........	72	5·76	0·425	92·6	86·7	12·10	18·6	0·41	0·62	2·2	0·25
Grinding in Water:											
17..........	24	5·25	0·35	93·3	90·2	7·47	4·33	0·23	0·81	2·7	0·35
16..........	48	5·82	0·185	96·8	91·2	8·30	3·45	0·25	0·54	2·7	0·4

SUMMARY

Amalgamation and Cyanidation

The results of the investigation disclose that the ore contains free gold.

Up to 70 per cent of the gold is free-milling or amalgamable gold, and 60 per cent of this is freed at a grind of 33 per cent −200 mesh.

Part of the gold is locked up in both the sulphides and the gangue particles and could not be freed by grinding them to 99 per cent −325 mesh.

The maximum extraction of 94 per cent of the gold by amalgamation and cyanidation was obtained at a grind of 73 per cent −200 mesh, amalgamating the ore by barrel amalgamation and cyaniding the residue for 24 hours.

Straight cyanidation gave the same extraction at a grind of 87 per cent −200 mesh. It was noted that the use of a minimum amount of lime gave the best extraction at similar grinds.

A cycle test shows no evidence of fouling, and a uniform extraction was obtained after five cycles.

An infrasizer analysis on a −200-mesh portion of a cyanide tailing shows that the gold is enclosed in both sulphide and gangue particles down to −10 microns in size and that 0·01 ounce of gold per ton is the lowest cyanide tailing to be obtained.

A desliming test shows that the cyanide slime is reduced to 0·01 ounce of gold per ton within 24 hours and the reground sand to the same value in 18 hours.

Settling tests indicate that the pulp settles readily with a clear overflow.

Jigging and cyaniding the residue gave an extraction of 94 per cent, and 32 per cent of the gold was recovered by amalgamation of the jig concentrate.

Flotation

The investigation indicates that under proper conditions a minimum tailing of 0·005 ounce of gold per ton can be attained at a grind of 86 per cent −200 mesh.

The concentrate carries over 5 ounces of gold per ton at a ratio of concentration of 30 : 1.

The best results were obtained when the froth was pulled off rapidly from the cell in a short time.

Discrepancies in results were noted when the same reagents and conditions were used. This is probably due to the presence of free gold.

Cyanidation of the concentrate indicates that the gold freed by grinding goes into solution rapidly, 94 per cent within 16 hours. A long period of agitation was not beneficial. When ground in water and filtered prior to cyanidation, 96·8 per cent extraction was obtained within 48 hours.

An infrasizer analysis of the cyanide residue shows that 46 per cent of the gold in the residue is contained in the −10-micron particles.

A recleaned concentrate of over 8 ounces of gold per ton, with a ratio of concentration of 56 : 1, was obtained.

CONCLUSIONS

To obtain maximum recovery by either straight cyanidation or flotation a grind of approximately 85 per cent −200 mesh is indicated.

When the free gold has been removed from the circuit before agitation, the minimum cyanide tailing is obtained within 24 hours from material ground 73 per cent −200 mesh.

A tailing of 0·01 ounce of gold per ton, representing an extraction of 94 per cent, from the grade of ore represented by this sample under investigation is the lowest that can be expected by cyanidation. The infrasizer shows this clearly.

The results of flotation indicate that a flotation tailing ranging from 0·01 ounce to 0·005 ounce of gold per ton can be obtained. When the concentrate is cyanided an overall recovery of approximately 94 per cent is obtained. This equals that obtained by straight cyanidation.

Three methods of treatment are applicable to this ore. The first is cyanidation, with amalgamation of a jig concentrate obtained from the ball mill discharge, when the alkalinity of the solutions should be kept as low as possible. Only sufficient lime should be added to obtain satisfactory thickener operation.

The second method is to float and cyanide the concentrates. When the optimum recovery is obtained by flotation the overall extraction should equal that obtained by cyanidation.

An alternative to either of the above would be to float and smelt the concentrate.

The choice of method, obviously, will depend on the economics of each and the advantage of proximity to a smelter.

The results of this investigation apply to ore of grade and character similar to the sample submitted.

Ore Dressing and Metallurgical Investigation No. 764

GOLD ORE FROM ATHONA MINES (1937), LIMITED, GOLDFIELDS, SASKATCHEWAN

Samples. Nine special samples of gold ore from Athona Mines (1937), Limited, Goldfields, Saskatchewan, were received in November, 1938, from J. J. Byrne, President, Athona Mines (1937), Limited, 80 King Street West, Toronto, Ontario.

The specimens were examined without the aid of a glass and also under the binocular microscope. Polished sections were prepared and examined under the reflecting microscope. The following report is based upon all observations.

Table I gives a list of the samples, with the information supplied by Athona Mines (1937), Limited.

TABLE I

List of Samples

Sample No.	Number of polished sections	Description supplied by the Athona Mines (1937), Limited
1 and 2	6	Specimens similar; "H" vein mineralization. Higher grade ore zones.
3	2	Quartz-chalcopyrite mineralization; low-grade ore zones. Chalcopyrite makes up only a very small percentage of sulphides.
4	2	Quartz-galena mineralization; low-grade ore zones.
5	2	Quartz-sphalerite in red granite; low-grade ore zones.
6	2	Quartz-pyrite mineralization with some galena in gabbro-granite contact zone material; low-grade ore zones.
7	None	Specimen same as No. 6.
8 and 9	2	Quartz-pyrite mineralization in red granite; low-grade ore zones.

General Description. The most prominent constituent of the samples is a rather coarse-textured, light greyish white quartz. Some specimens contain pink granitic material and others a fine-textured grey rock, which is probably a somewhat altered phase of the gabbro referred to by Athona Mines (1937), Limited, in the list of specimens. The quartz has invaded the granite along veinlets the boundaries of which are somewhat hazy, being gradational over a distance of a millimetre or more. Some of the granite shows the effect of high-temperature silicification. The grey gabbroic material also shows evidence of high-temperature alteration by the presence within it of metacrysts of feldspathic material up to a centimetre or more in size.

The sulphides are confined chiefly to the quartz, but rare pyrite occurs in the gabbroic material. Sphalerite and galena are most abundant, occurring as irregular masses, stringers, and grains in the quartz, usually accompanied by a little carbonate. In places the masses send out into the quartz short tongues that taper rapidly and die out. A very small quantity of pyrite is disseminated in the quartz and within the sulphide masses; that occurring in quartz is euhedral and shows cubic forms, but in the sulphide masses it shows the effect of active corrosion and replacement, chiefly by sphalerite. Chalcopyrite is comparatively rare, and is associated with sphalerite and with pyrite. Sphalerite and galena show a distinct tendency to occur together, the sphalerite usually predominating. Sphalerite masses commonly enclose irregular grains and stringers of galena, and in places galena has veined the sphalerite which contains inclusions of pyrrhotite and chalcopyrite. Native gold occurs chiefly in the quartz, along stringers that also carry a little carbonate and are usually continuous with stringers of the sulphides. Lesser quantities of gold occur with sphalerite and galena, and in rare cases gold is associated with chalcopyrite.

Paragenesis. The order of deposition of the minerals is clearly exhibited in the polished sections. The quartz has penetrated and invaded the granitic rock, with a certain amount of high-temperature replacement and silicification of the host. Apparently pyrite was disseminated essentially at this time. Later incipient fracturing, chiefly in the quartz, provided channels for the solutions bearing the later sulphides and the native gold; the solutions at this phase must have carried also a little carbonate. Sphalerite was the first sulphide to start to deposit during this period of mineralization, and a little pyrrhotite and chalcopyrite accompanied it as tiny inclusions, often arranged parallel to the crystallographic directions of the sphalerite, a structure usually attributed to unmixing. Galena was the last sulphide to be deposited. Native gold is definitely associated with the later sulphides, hence its deposition must have taken place during the later sulphide stage of mineralization and not during the quartz-pyrite stage. Two inferences may be drawn from this fact: (1) If the deduction that gold deposition accompanied only the deposition of the later sulphides is correct, the earlier pyrite would not be expected to carry gold and the ore should not prove to be refractory. (2) The later sulphides, sphalerite, galena, etc., may be taken as indicators of ore in the mine.

Table II, below, represents graphically what is thought to be the paragenesis of the minerals.

TABLE II

Paragenesis of the Minerals

Minerals	Decreasing age and probably decreasing temperature		
	Granite (host)	Quartz pyrite stage	Later sulphide stage
Granite	——		
Quartz		——	
Pyrite		? ——	
Sphalerite			——
Pyrrhotite			- - - -
Chalcopyrite			— ——
Galena			— — ——
Mineral X			— ? —
Native gold			- - —— - - -
Carbonate			- —— -

Detailed Descriptions of Samples

Samples 1 and 2. Samples 1 and 2 are similar in character. The gangue is quartz. In this gangue occur masses and irregular stringers of sphalerite and galena and occasional disseminated grains of pyrite. The sphalerite masses contain irregular grains and veinlets of galena, and rare tiny dots of chalcopyrite and pyrrhotite. The sphalerite-galena masses send out short tongues tapering out to points in the quartz. The pyrite occurring in the quartz is euhedral, cubic forms being prominent; the pyrite enclosed in the sulphides, notably sphalerite, has been corroded and replaced to some extent by this mineral. Native gold is present as irregular grains (1) in the quartz, and (2) in the sulphide masses associated with sphalerite and galena.

Sample 3. The gangue is quartz. The sulphide mineralization is commonly irregular stringers and small grains of chalcopyrite, which is associated with some sphalerite and galena. The chalcopyrite contains small irregular grains of Mineral X, the tests on which are given in Table III.

Native gold occurs as irregular grains (1) in quartz, and (2), rarely, in chalcopyrite.

TABLE III

Tests on Mineral X from Sample 3

Colour:	Faint bluish white, very similar to galena.
Hardness:	B; softer than chalcopyrite. No cleavage noticeable.
Crossed nicols:	Rather weakly but distinctly anisotropic; two positions of extinction, light to dark grey.
Etch tests:	HNO_3—after a slight delay mineral tarnishes differentially iridescent to grey without effervescence. A structure strongly suggestive of graphic structure is brought out.
	HCl—differentially practically negative to grey, with development of graphic structure.
	KCN—negative.
	$FeCl_3$—differentially iridescent with development of blues and browns; graphic structure is brought out.
	KOH—negative.
	$HgCl_3$—negative.
Microchemical tests:	Owing to the small quantity of material available and to the fact that any sample free from chalcopyrite could not be obtained, microchemical analysis proved very unsatisfactory. A test for lead was obtained, and the presence of bismuth was inconclusively indicated during a microchemical analysis.
Identification:	The mineral most closely conforming to the above tests is *Matildite* ($(Ag_2,Pb)S.Bi_2S_3$), but identification is tentative only.

Sample 4. The gangue is chiefly quartz with a lesser quantity of pink granitic material. Stringers of sphalerite and galena, usually associated together, occur in the quartz near the quartz-granite boundaries, and tongues of quartz carrying these sulphides extend into the granite. Quartz which contains gold grains also penetrates into the granite. Grains of native gold occur in the quartz, usually alone but occasionally associated with galena. Pyrite is rare.

Sample 5. The gangue is chiefly pink granitic material with veinlets of quartz. The boundaries between quartz and granite are gradational over a millimetre or more, giving the quartz veinlets a somewhat hazy appearance. Coarse, massive sphalerite is present in the quartz; chalcopyrite and galena are comparatively rare. Pyrite occurs as euhedral crystals in the quartz and as corroded grains in sphalerite. Native gold occurs as irregular grains (1) in quartz, (2) along quartz-sphalerite boundaries, and (3) associated with galena.

Samples 6 and 7. Samples 6 and 7 are similar, and sections were prepared from Sample 6 only. The gangue is chiefly a fine-textured grey rock, possibly an altered phase of the gabbro referred to in the correspondence from Athona Mines (1937), Limited, with minor quartz. Metacrysts of feldspathic material, up to a centimetre or more in size, have been developed in the grey rock, and pyrite has also been disseminated in it. Irregular stringers of sphalerite and galena, usually associated, occur in the quartz. The sphalerite contains small inclusions of pyrrhotite and chalcopyrite. No native gold is visible.

Samples 8 and 9. The samples consist chiefly of pink granitic material with a small quantity of quartz. A considerable amount of pyrite occurs in the quartz. The mineral is somewhat fractured and brecciated, and contains occasional inclusions of chalcopyrite. No native gold is visible.

Grain Size of the Native Gold. Tables IV, V, VI, and VII give grain analyses of the visible native gold in the individual specimens, and Table VIII gives a composite grain analysis of the gold in all samples. The tables are based wholly on measurements made on the gold occurring in polished sections.

TABLE IV

Grain Analysis of the Gold in Samples 1 and 2

Mesh	Microns	Gold in quartz, per cent	Gold in sphalerite, per cent		Totals, per cent
			Alone	Associated with galena	
+ 65	+208	35·8	....	....	35·8
− 65+ 100	+147	....	....	....	....
− 100+ 150	+104	....	....	....	....
− 150+ 200	+ 74	6·4	....	....	6·4
− 200+ 280	+ 52	5·1	....	....	5·1
− 280+ 400	+ 37	....	....	....	....
− 400+ 560	+ 26	3·0	2·5	8·6	14·1
− 560+ 800	+ 19	....	14·5	7·7	22·2
− 800+1100	+ 13	1·3	8·2	3·3	12·8
−1100+1600	+ 9	....	2·9	....	2·9
−1600	....	....	0·7	....	0·7
	....	51·6	28·8	19·6	100·0

TABLE V

Grain Analysis of Gold in Sample 3

Mesh	Microns	Gold in quartz, per cent	Gold in chalcopyrite, per cent	Totals, per cent
+ 65	+208	39·2	...	39·2
− 65+ 100	+147	7·8	...	7·8
− 100+ 150	+104	24·5	...	24·5
− 150+ 200	+ 74	8·5	...	8·5
− 200+ 280	+ 52	6·2	...	6·2
− 280+ 400	+ 37	3·6	...	3·6
− 400+ 560	+ 26	2·0	...	2·0
− 560+ 800	+ 19	3·3	1·6	4·9
− 800+1100	+ 13	1·0	0·8	1·8
−1100+1600	+ 9	1·5	...	1·5
−1600	...	...	...	...
	...	97·6	2·4	100·0

TABLE VI

Grain Analysis of Gold in Sample 4

Mesh	Microns	Gold in quartz, per cent	Gold associated with galena, per cent	Totals, per cent
+ 65	+208	...	...	...
− 65+ 100	+147	9·3	...	9·3
− 100+ 150	+104	6·5	...	6·5
− 150+ 200	+ 74	13·9	...	13·9
− 200+ 280	+ 52	10·1	...	10·1
− 280+ 400	+ 37	14·7	...	14·7
− 400+ 560	+ 26	18·2	...	18·2
− 560+ 800	+ 19	15·3	1·3	16·6
− 800+1100	+ 13	6·5	1·1	7·6
−1100+1600	+ 9	2·7	...	2·7
−1600	...	0·4	...	0·4
	...	97·6	2·4	100·0

TABLE VII

Grain Analysis of the Gold in Sample 5

Mesh	Microns	Gold in quartz, per cent	Gold associated with sphalerite and galena, per cent	Totals, per cent
+ 65	+208	...	...	...
− 65+ 100	+147	...	...	...
− 100+ 150	+104	21·7	29·0	50·7
− 150+ 200	+ 74	...	20·3	20·3
− 200+ 280	+ 52	...	14·5	14·5
− 280+ 400	+ 37	...	10·1	10·1
− 400+ 560	+ 26	...	4·4	4·4
− 560+ 800	+ 19	...	...	...
− 800+1100	+ 13	...	...	...
−1100+1600	+ 9	...	...	...
−1600	...	...	...	...
	...	21·7	78·3	100·0

TABLE VIII

Composite Grain Analysis of the Gold in All Sections

Mesh	Microns	Gold in quartz, per cent	Gold associated with sphalerite and/or galena, per cent	Gold in chalco-pyrite, per cent	Totals, per cent
+ 65	+208	21·0	....	....	21·0
− 65+ 100	+147	6·0	....	....	6·0
− 100+ 150	+104	10·2	....	....	10·2
− 150+ 200	+ 74	11·0	2·1	....	13·1
− 200+ 280	+ 52	7·0	1·5	....	8·5
− 280+ 400	+ 37	6·7	1·0	....	7·7
− 400+ 560	+ 26	8·2	3·4	....	11·6
− 560+ 800	+ 19	6·8	5·6	0·5	12·9
− 800+1100	+ 13	3·1	3·2	0·3	6·6
−1100+1600	+ 9	1·5	0·7	....	2·2
−1600	....	0·1	0·1	....	0·2
	....	81·6	17·6	0·8	100·0

As will be obvious from a glance at Table VIII, the figures for the grain sizes of the gold are somewhat erratic in the larger sizes. This is common where coarse gold is present. In the grain sizes below 19 microns (equivalent of 800 mesh) the percentages decrease rapidly and only 0·2 per cent of the gold measured is smaller than 9 microns (equivalent of 1600 mesh).

Ore Dressing and Metallurgical Investigation No. 765

GOLD ORE FROM THURLOW ISLAND PROPERTY OF THE PIEDMONT MINING COMPANY, LIMITED, VANCOUVER, BRITISH COLUMBIA

Shipment. A shipment of ore consisting of four bags and weighing 440 pounds was received on January 11, 1939, from Mr. J. M. Coady, Secretary, Piedmont Mining Company, Limited (N.P.L.), 553 Granville Street, Vancouver, British Columbia.

Location of Property. This shipment of ore was made from the Douglas Pine, Crown Grant Mineral Claim, situated at Shoal Bay, on Thurlow Island, Vancouver Mining Division, British Columbia. This property is being developed by the Piedmont Mining Company, Limited, of Vancouver, British Columbia.

Sampling and Analysis. After reduction, sampling, assaying and analysing by standard accepted methods, the following results were obtained:

Gold	0·32 oz./ton	Zinc	Nil
Silver	0·51 "	Arsenic	"
Copper	0·19 per cent	Antimony	"
Iron	9·89 "	Bismuth	"
Sulphur	4·14 "	Cadmium	"
Silica	69·17 "	Tellurium	"
Lead	Trace	Selenium	"

Purpose of the Investigation. Mr. Coady, in his letter of December 31, 1938, requested "service as to the best methods of milling this ore". In his letter of March 7, 1939, he stated "... they are not interested in cyanide process, on account of the very advantageous shipping facilities."

Summary of Investigations. The ore lends itself peculiarly to concentration owing to the occurrence of almost 70 per cent of the gold in size between 37 and 9 microns, and over 92 per cent between 37 and 6 microns.

A flow-sheet is recommended embodying gravity concentration and flotation, with which it is reasonable to expect an extraction of over 95 per cent.

Microscopic Investigation. Six polished sections were examined under the microscope.

The *gangue* consists essentially of milky-white quartz and rather abundant carbonate, which appears to be calcite and is very finely disseminated for the most part, but occurs also in narrow veinlets and small patches throughout the quartz.

Small areas of a soft, dark grey rock material were also noted that shows a slight residual schistosity or foliation. This assemblage is locally coloured a rusty brown with stains of iron oxides.

Metallic minerals are visible in large quantities in the sections and are very intimately admixed.

Pyrite predominates, largely as small masses and coarse irregular grains in gangue; a small amount is present as small irregular grains and narrow, discontinuous stringers cutting quartz. It has been severely shattered and is extensively veined with pyrrhotite, much less extensively with chalcopyrite and gangue.

Pyrrhotite is very prevalent as coarse to fine irregular grains and small masses in gangue; also as veins and inclusions in pyrite.

A relatively small amount of chalcopyrite occurs as medium to small irregular grains in pyrite, pyrrhotite, and gangue, less frequently as veins in pyrrhotite. Its total quantity appears sufficient to consume cyanide solutions.

A small amount of "limonite" is seen, mostly as stains in the gangue; rarely, as alteration rims along the edges of pyrite and pyrrhotite grains.

Forty-one grains and one stringer of native gold are visible in the sections. The latter is approximately 120 microns long by 2 microns wide, and occurs along the edge of a pyrrhotite veinlet in pyrite. The modes of occurrence and size of the granular gold are shown in the following table:

Grain Size of Gold:

Microns	Mesh	Gold in gangue, per cent		Gold in pyrrhotite, per cent	Totals, per cent	
		Alone	Associated with pyrrhotite		On mesh	Cumulative
+37	+ 400	11·2	....	....	11·2	....
−37+26	− 400+ 560	8·0	....	....	8·0	19·2
−26+19	− 560+ 800	....	5·4	9·8	15·2	34·4
−19+13	− 800+1100	6·3	3·6	10·5	20·4	54·8
−13+ 9	−1100+1600	9·2	4·7	....	13·9	68·7
− 9+ 6	−1600+2300	15·0	3·1	5·8	23·9	92·6
− 6	−2300	3·1	3·4	0·9	7·4	100·0
		52·8	20·2			
Totals	....	73·0		27·0	100·0	....

Laboratory Investigations

AMALGAMATION—Direct

Grind:

Ore	2,000 grammes
Water	1,500 "
Lime	4 "
Time: Test No. 1	20 minutes
Test No. 2	40 "

Contact:

Time	60 minutes
Mercury	200 grammes

Results:

Test No.	Assay, Au, oz./ton		Extraction, per cent
	Feed	Tailing	
1..........	0·32	0·155	51·56
2..........	0·32	0·165	48·44

Screen Analyses—Tailings:

Mesh	Test No. 1		Test No. 2	
	Weight, per cent	Assay, Au oz./ton	Weight, per cent	Assay, Au, oz./ton
+100..........	4·60	0·11	....	....
−100+150..........	17·14	0·18	3·62	0·26
−150+200..........	17·88	0·19	11·06	0·22
−200..........	60·38	0·14	85·32	0·16
	100·00	0·154	100·00	0·170

Combined Feed to Tests Nos. 1 and 2:

Mesh	Weight, per cent	Assay, oz./ton	
		Au	Ag
+150..........	6·96	0·30	0·29
−150+200..........	12·96	0·255	0·37
−200..........	80·08	0·305	0·51
	100·00	0·298	0·477

CYANIDATION—Direct

Grind:

Ore.......... 2,000 grammes
Water.......... 1,500 "
Lime.......... 5 "
NaCN.......... 2 "

Time: Test No. 3.......... 30 minutes
Test No. 4.......... 45 "
Test No. 5.......... 60 "

Agitation:

Time.......... 24 hours
Dilution.......... L : S : : 2 : 1

Reagent strengths: NaCN.......... 0·2 lb./ton of L.
CaO.......... 0·1 "

Results:

Test No.	Grind, per cent −200 mesh	Reagents consumed, lb./ton		Assay, Au, oz./ton		Extraction, per cent
		NaCN	CaO	Feed	Tailing	
3	81·84	1·4	4·8	0·32	0·14	56·3
4	94·70	1·6	4·8	0·32	0·16	50·0
5	98·20	1·9	4·9	0·32	0·16	50·0

Screen Analyses:

Mesh	Test No. 1		Test No. 2		Test No. 3	
	Weight, per cent	Assay, Au, oz./ton	Weight, per cent	Assay, Au, oz./ton	Weight, per cent	Assay, Au, oz./ton
+200	19·16	0·155	5·30	0·425	1·80	1·07
−200	80·84	0·140	94·70	0·140	98·20	0·16
(Calc.)	100·00	0·144	100·00	0·155	100·00	0·176

GRAVITY CONCENTRATION—AMALGAMATION OF CONCENTRATE—CYANIDATION OF AMALGAMATION TAILING—CYANIDATION OF CONCENTRATION TAILING

Grind:

Ore.......... 2,000 grammes (in duplicate)
Water.......... 1,500 "

Lime.......... 0·5 gramme
Time.......... 30 minutes

Concentration:

Time.......... 30 minutes for 2,000 grammes
Jig and strakes.

Amalgamation:

Regrind: Jig concentrate.......... 250 grammes (combined)
Strake.......... 200 " "

Lime.......... 1 gramme
Time.......... 45 minutes

Contact: L : S : : 4 : 1.
Time.......... 60 minutes

Cyanidation: L : S : : 2 : 1.
Time.......... 16 hours
Lime.......... 1 gramme
Sodium cyanide.......... 0·5 gramme

Cyanidation of Concentration Tailing:

Tailing.......... 1,000 grammes
Water.......... 2,000 "
Lime.......... 2·0 "
Sodium cyanide.......... 1·0 "

Results:

Concentration:

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
6	Feed	100·00	0·32	100·00	....
	Jig concentrate	4·74	1·79	26·47	21·1
	Strake concentrate	5·07	1·64	25·63	19·7
	Tailing	90·19	0·17	47·90	....
7	Feed	100·00	0·32	100·00	....
	Jig concentrate	9·27	0·90	26·00	10·8
	Strake concentrate	6·27	1·28	25·16	15·9
	Tailing	84·46	0·185	48·84	....

Amalgamation:

Test No.	Concentrate	Assay, Au, oz./ton		Extraction, per cent	
		Feed	Tailing	Unit	Feed
8	Jig	1·345	0·235	82·53	21·65
9	Strake	1·460	0·235	83·90	21·30

Cyanidation:

Test No.	Tailing of Test	Assay, Au, oz./ton		Extraction, per cent	
		Feed	Tailing	Unit	Feed
Amalgamation Tailing:					
10	8	0·235	0·035	85·11	4·10
11	9	0·235	0·030	87·23	3·57
Concentration Tailing:					
12	6	0·170	0·065	61·76	29·59
13	7	0·185	0·035	81·08	39·60

Screen Analyses of Concentration Tailings Cyanided:

Mesh	Test No. 12		Test No. 13	
	Weight, per cent	Assay, Au, oz./ton	Weight, per cent	Assay, Au, oz./ton
+150	2·8	0·04	4·7	0·03
−150+200	10·6	0·04	15·3	0·035
−200	86·6	0·065	80·0	0·030
	100·0	0·062	100·0	0·031

Note.—Reagent consumptions not determined in cyanidation investigations as advised that cyanidation was not being considered. Tests were under way before such advice was received.

Recapitulation of Results (using arithmetical averages):

Process	Recovery, per cent
Amalgamation	42·95
Cyanidation of amalgamation tailing	7·67
Cyanidation of concentration tailing	34·59
Total	85·21

FLOTATION—DIRECT

Tests Nos. 14 and 15 (Pilot Tests)

These were in the nature of pilot tests and revealed a tendency that appeared in later tests, namely, a slow-floating concentrate and poor frothing conditions when using copper sulphate and xanthates, and excessive froth with the use of any pine oil in addition to an Aerofloat.

Grind:

Ore	2,000 grammes
Water	1,500 "
Soda ash	1·5 "
$CuSO_4$	0·5 gramme
Time—Test No. 14	30 minutes
Test No. 15	60 "

Conditioning:

Time	5 minutes
Aerofloat No. 15	1 drop
Soda ash	0·5 gramme

Flotation:

Time—Test No. 14	12 minutes
Test No. 15	10 "
Pine oil	4 drops
Reagent 301	0·2 gramme

Results:

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
14	Feed	100·00	0·320	100·00	....
	Concentrate	10·33	2·837	91·59	9·68
	Tailing	89·67	0·030	8·41	....
15	Feed	100·00	0·320	100·00	....
	Concentrate	9·58	2·868	85·87	10·44
	Tailing	90·42	0·050	14·13	....

Tests Nos. 16, 17, and 18 (Variable Grinds)

Grind:

Ore	2,000 grammes
Water	1,500 "
Soda ash	1·5 "
$CuSO_4$	0·5 gramme
Aerofloat No. 25	0·035 "
Time—Test No. 16	30 minutes
Test No. 17	45 "
Test No. 18	60 "

Test No. 16:

Conditioning:

Time	5 minutes
Reagent 301	0·2 gramme

Flotation:

Time	17 minutes
Concentrate	7 minutes
Middling	10 "
Pine oil	0·225 gramme
Soda ash	2·0 grammes

Observation:

A heavy frothless scum for five minutes, then copious froth suddenly.
—pH=9·7

Test No. 17:

Conditioning:

Time	5 minutes
Aerofloat No. 25	0·07 gramme

Flotation:

Time	10 minutes
Concentrate	3 minutes
Middling	7 "
Pine oil	0·125 gramme
Soda ash	1·0 "

Observation:

High-grade scum froth for three minutes, then copious "wild" froth suddenly produced.
—pH=9·45

Test No. 18:

Conditioning:

Time	10 minutes
Aerofloat No. 25	0·07 gramme

Flotation:

Time	12 minutes
Concentrate	3 minutes
Middling	9 "
Pine oil	0·05 gramme

Observation:

Froth slow but very much more stable and uniformly small bubbles.

Results of Tests Nos. 16, 17, and 18:

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
16	Feed	100·00	0·320	100·00	....
	Concentrate	8·52	3·140	80·72	11·74
	Middling	7·52	0·570	12·94	13·30
	Tailing	83·96	0·025	6·34	....
17	Feed	100·00	0·320	100·00	....
	Concentrate	6·15	4·028	77·41	16·26
	Middling	9·86	0·520	16·03	10·14
	Tailing	83·99	0·025	6·56	....
18	Feed	100·00	0·320	100·00	....
	Concentrate	6·58	3·880	79·81	15·20
	Middling	7·56	0·570	13·47	13·23
	Tailing	85·86	0·025	6·72	....

Screen Tests:

Test No.	Percentage distribution	
	+200 mesh	−200 mesh
16	25·2	74·8
17	13·7	86·3
18	2·9	97·1

Tests Nos. 19, 20, and 21 (Reagents Varied)

Grind:

Ore	1,000 grammes
Water	750 "
Soda ash	0·75 gramme
$CuSO_4$	0·25 "
Aerofloat No. 15	0·035 "
Time	30 minutes

Test No. 19:

Conditioning:

Time	5 minutes
Reagent 301	0·05 gramme

Flotation:

Time	7 minutes
Pine oil	0·025 gramme
Aerofloat No. 15	0·035 "

Observation:

Uniform, good froth, fast but too copious.

Test No. 20:

Conditioning:

Time	5 minutes
Reagent 343	0·05 gramme
Pine oil	0·025 "
Aerofloat No. 15	0·035 "

Flotation:

Time	5 minutes

Observation:

Froth better appearance but still too copious.

Test No. 21:

Conditioning:

Time	4 minutes
Reagent 208	0·05 gramme
Pine oil	0·025 "
Aerofloat No. 15	0·035 "

Flotation:

Time	7 minutes

Observation:

Quicker froth of good appearance but more brittle.

Results of Tests Nos. 19, 20, and 21:

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
19	Feed	100·00	0·320	100·00	....
	Concentrate	16·14	1·820	91·78	6·20
	Tailing	83·86	0·031	8·22	....
20	Feed	100·00	0·320	100·00	....
	Concentrate	11·99	2·560	95·91	8·34
	Tailing	88·01	0·015	4·09	....
21	Feed	100·00	0·320	100·00	....
	Concentrate	12·57	2·460	96·62	7·96
	Tailing	87·43	0·012	3·38	....

Screen Tests:

Test No.	Percentage distribution	
	+200 mesh	−200 mesh
19	25·8	74·2
20	26·4	73·6
21	25·0	75·0

Tests Nos. 22, 23, and 24 (Reagents and Treatment Times Varied)

Grind:

Ore	1,000 grammes
Water	750 "
Soda ash	0·75 gramme
Time	30 minutes

Test No. 22:

Conditioning:

Time	5 minutes
Pentasol	0·06 gramme
Aerofloat No. 15	0·070 "

Flotation:

Pine oil—0·025 gramme
Pine oil—After 5 minutes of no froth, 0·050 gramme, after 8 minutes—froth very good.

Time	10 minutes

Test No. 23:

Conditioning:

Time	5 minutes
Reagent 208	0·055 gramme
Aerofloat No. 15	0·070 "
Pine oil	0·025 "

Flotation:

Time	4 minutes

Observation:

Very copious but slightly brittle froth, after 2 minutes' conditioning. Most of concentrate off by end of conditioning.

Test No. 24:

Conditioning:

Time	5 minutes
Reagent 301	0·065 gramme
Aerofloat No. 15	0·070 "
Pine oil	0·025 "

Flotation:

Time	8 minutes

Observation:

Froth sufficiently uniform and well mineralized and not too brittle. Best looking froth attained.

Results of Tests Nos. 22, 23, and 24:

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
22	Feed	100·00	0·320	100·00	....
	Concentrate	12·57	2·32	91·12	7·96
	Tailing	87·43	0·032	8·88	....
23	Feed	100·00	0·320	100·00	....
	Concentrate	12·57	2·267	89·05	7·96
	Tailing	87·43	0·040	10·95	....
24	Feed	100·00	0·320	100·00	....
	Concentrate	13·77	2·240	96·37	7·26
	Tailing	86·23	0·013	3·63	....

Screen Tests:

Test No.	Percentage distribution	
	+200 mesh	−200 mesh
22	23·0	77·0
23	25·8	74·2
24	23·2	76·8

GRAVITY CONCENTRATION—FLOTATION

Tests Nos. 25, 26, and 27

Grind:

Ore	2,000 grammes
Water	1,500 "
Soda ash	1·5 "
Time	30 minutes

Concentration:

Laboratory jig and strakes.
Time—30 minutes for 2,000 grammes to pass over jig and strakes.

Test No. 25:

Conditioning:

Time	10 minutes
$CuSO_4$	0·5 gramme
Aerofloat No. 15	0·07 "

Flotation:

Time	10 minutes
Aerofloat No. 15	0·105 gramme
Pine oil	0·025 "

Test No. 26:

Conditioning:

Time	10 minutes
$CuSO_4$	0·5 gramme
Aerofloat No. 15	0·07 gramme at end of 5 minutes
Potassium ethyl xanthate	0·5 " "

Flotation:

Time	15 minutes
Aerofloat No. 15	0·070 gramme

Test No. 27:

Conditioning:

Time	10 minutes
Aerofloat No. 15	0·105 gramme
Reagent 301	0·05 "

Flotation:

Time	12 minutes

Froth Characteristics:

At least 10 minutes' conditioning appears necessary to prevent scum-froth in early flotation followed by sudden over-frothing. If properly conditioned, froth is ample, controllable, well formed, and well mineralized.

Results of Tests Nos. 25, 26, and 27:

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Percentage: Distribution	Percentage: Cumulative recovery	Ratio of concentration: Unit	Ratio of concentration: Overall
25	Feed	100·00	0·320	100·00	....	....	....
	Jig concentrate	19·25	0·640	38·50	38·50	5·19	3·18
	Strake concentrate	7·79	1·120	27·27	65·77	12·84	
	Flotation feed	73·96	0·148	34·23	....	....	
	Flotation concentrate	4·42	2·160	29·83	95·60	22·62	
	Flotation tailing	69·54	0·020	4·40	....	....	
26	Feed	100·00	0·320	100·00	....	....	4·63
	Jig concentrate	7·18	0·960	21·53	21·53	13·93	
	Strake concentrate	6·80	1·160	23·94	45·47	15·15	
	Flotation feed	86·22	0·169	54·53	....	....	
	Flotation concentrate	7·80	1·885	45·97	91·44	12·82	
	Flotation tailing	78·42	0·035	8·56	....	....	
27	Feed	100·00	0·320	100·00	....	....	6·23
	Jig concentrate	3·62	1·800	20·38	20·38	27·62	
	Strake concentrate	5·64	1·380	24·31	44·69	17·73	
	Flotation feed	90·74	0·195	55·31	....	....	
	Flotation concentrate	6·80	1·924	40·88	85·57	14·71	
	Flotation tailing	83·94	0·055	14·43	....	....	

CONCLUSIONS

Cyanidation was tried only in pilot tests. No further investigation was done as the process is not being considered by the company.

The same remarks apply to amalgamation. Amalgamation of gravity concentrate prior to flotation should be considered as a means of recovering between 40 to 45 per cent of the gold in the ore on the property. (See microscopic report and Tests Nos. 1, 2, 6, 7, 8, and 9.)

In flotation a wide range of grinds, reagents, conditioning and flotation periods, and ratios of concentration, was used. Tests Nos. 21, 24, and 25 all indicate an extraction of over 95·0 per cent.

The occurrence of the gold is such as to make the ore peculiarly adaptable to flotation without danger of erratic tailing assays. It is recommended, however, that gravity concentration by means of a mineral jig and strakes be used on the property to recover gold. Owing to the fineness of the free gold the use of strakes appears as efficient as that of a mineral jig, but to prevent loss of occasional "erratic" gold particles a jig should be considered.

The extraction of copper was not considered of sufficient importance to require calculation of recoveries, etc., owing to the low feed value of 0·19 per cent.

Ore Dressing and Metallurgical Investigation No. 766

CYANIDATION TESTS OF GOLD ORES FROM THE CHESTERVILLE LARDER LAKE GOLD MINING COMPANY, LIMITED, CHEMINIS, ONTARIO

Shipments. Samples of ore, marked "A", "B", and "C", from the Chesterville Larder Lake Gold Mining Company, Limited, Cheminis, Ontario, were received on November 2, November 1, and November 4, 1938, respectively.

The weights of the samples were as follows:

Sample A	570 pounds
Sample B	635 "
Sample C	990 "

Characteristics of the Samples. Six polished sections of each sample were prepared and examined microscopically.

Sample A is an altered basalt carrying disseminated sulphide.

The *gangue* is composed of fine-textured, siliceous, dark-grey rock with rather abundant, fine, disseminated carbonate, most of which appears to be calcite.

Of the *metallic minerals* present, pyrite predominates as coarse to fine subhedral crystals and irregular grains disseminated in gangue. It contains numerous inclusions of gangue, occasional small, irregular grains of chalcopyrite, and rare small grains of pyrrhotite. Some grains are veined with gangue and, rarely, with chalcopyrite. A small quantity of chalcopyrite is present as small, irregular grains in gangue and in pyrite as already noted, and a negligible amount of pyrrhotite occurs in pyrite only. Also visible are occasional patches of a grey alteration product, probably leucoxene, which under crossed nicols is wholly translucent with brownish reflections.

Eight small grains of native gold, ranging in size from 10 microns (+ 1600 mesh) to 4 microns (− 2300 mesh) occur in pyrite, in chalcopyrite, and in gangue.

Sample B is more siliceous than Sample A and contains very little sulphide.

The *gangue* is essentially the same as that in Sample A except that the rock is cut by stringers of milky-white quartz and contains patches of a vivid green, non-metallic mineral, which may be mariposite, a chromium-bearing mica.

Most of the *metallic minerals* occur in the rock portion of the gangue. Again, pyrite predominates as disseminated euhedral to anhedral crystals and grains containing the same inclusions but less veining than that in Sample A. Chalcopyrite, pyrrhotite, and leucoxene (?) are present in the same way and in about the same amounts as in the previous sample. A minor quantity of hard, white mineral, which etch reactions indicate to be gersdorffite (NiAsS), occurs as tiny disseminated grains often intimately associated with pyrite. Rare small, irregular grains of sphalerite are visible in gangue, as are rare tiny grains of magnetite.

One minute grain of native gold, 4 microns in size, occurs in dense pyrite.

Sample C is a mixture of Sample A and Sample B in the ratio expected in mill practice.

The mineral occurrences, both metallic and non-metallic, are the same as those of Sample B described above, with one or two minor exceptions. No sphalerite is visible in the sections, and pyrrhotite inclusions in pyrite are almost absent.

One small, irregular grain of native gold, 6 microns (2300 mesh) in size, occurs in gersdorffite.

Sampling and Assaying. After crushing to −14 mesh, and cutting by standard methods, samples were obtained of each sample of ore which assayed as follows:

—		Sample A	Sample B	Sample C
Gold	oz./ton	0·1275	0·98	0·255
Silver	"	0·045	0·07	0·045
Cr_2O_3	per cent	0·03	0·17	0·10
Nickel	"	0·03	0·07	0·05
Arsenic	"	0·04	0·08	0·06
Sulphur	"	2·29	1·02	1·77
Copper	"	Trace	Trace	Trace
Graphite	"	0·05	0·04	0·04
Iron	"	6·40	5·23	5·96
CaO	"	7·25	12·25	9·18
MgO	"	3·48	7·82	5·07
Al_2O_3	"	1·69	0·90	1·68
Insoluble	"	65·64	50·20	59·61

Purpose of the Investigation. The Chesterville Larder Lake gold ore contains chromium-bearing and nickel-bearing minerals. It was desired to know the effect of chromium and nickel on the precipitation of gold in the pregnant solution.

Experimental Tests

The tests consisted of the following:

1. Determination of the Mineral Constituents that will Dissolve in Cyanide and Lime Solutions, particularly Chromium and Nickel:

Only in two instances were traces of chromium detected by spectrographic method. Cyanide solutions from Samples A, B, and C contained nickel, copper, arsenic, and antimony. The nickel content in the cyanide solutions was as follows: Sample A, 0·016, Sample B, 0·04, and Sample C, 0·027 grammes per litre.

2. Standardization of the Gold Precipitation Method:

The method used not being comparable with the Merrill-Crowe method used in mill practice, it was necessary to run comparative tests against ores known to give very low barren solutions in mill practice. The results of precipitation tests on Chesterville ore were compared with those on Preston East Dome ore. The amount of zinc dust required by the laboratory method was from 0·3 to 0·6 pound per ton of solution, appreciably more than that in mill practice.

3. Determination of the Effect of Nickel on Gold Precipitation:

Samples of Preston East Dome ore were ground and cyanided with and without nickel salts. Six-tenths of a pound of zinc dust was added to the pregnant solutions. The barren solutions assayed as follows:

0·004 ounce gold..nickel—Nil.
0·015 " " ....nickel—0·077 grm./litre.

These results show that nickel in cyanide solutions will have an adverse effect on gold precipitation.

4. Determination of the Effect of Chromium on Gold Precipitation:

To a sample of Chesterville "C" ore, potassium chromate was added. Precipitation tests showed that chromium is precipitated by zinc dust; solutions that assayed 0·0001 and 0·00025 gramme of chromium per litre after precipitation assayed 0·00005 and 0·0001 gramme per litre, respectively. The barren solutions assayed 0·001 and 0·0005 ounce of gold.

5. Precipitation of Gold with Zinc Dust and with Charcoal:

Gold precipitation with activated charcoal showed no encouraging results:

Solution from Chesterville Sample	Ni, grm./litre	Zn, lb./ton	Charcoal, lb./ton	Au, oz./ton	
				Pregnant solution	Barren solution
B	0·049	0·6	....	0·842	0·066
B	0·049	....	0·6	0·842	0·424
C	0·026	0·3	....	0·21	0·01
C	0·026	....	0·6	0·21	0·042

Barium chloride and ammonium sulphate improved precipitation, as shown by the following figures:

Solution from	Ni, grm./litre	$BaCl_2$, lb./ton	$(NH_4)_2SO_4$, lb./ton	Zn, lb./ton	Au, oz./ton	
					Pregnant solution	Barren solution
Preston East Dome	0·077	....	....	0·3	0·88	0·061
Preston East Dome	0·077	0·6	....	0·3	0·88	0·040
Chesterville B	0·023	....	....	0·6	0·738	0·0665
Chesterville B	0·023	0·6	....	0·6	0·738	0·02
Chesterville B	0·023	....	0·6	0·6	0·738	0·014
Chesterville A	0·023	0·6	....	0·6	0·114	0·004
Chesterville A	0·023	0·6	0·6	0·6	0·114	0·002

6. Precipitation of Gold in Cycle Test Solutions:

Barren solutions from Chesterville Sample C ore, from which the gold had been precipitated every cycle with 0·6 pound of zinc dust or aluminium dust, assayed 0·002 ounce of gold at the end of the twelfth cycle. This compares favourably with the results from the Preston East Dome solution, which gave a barren of 0·002 ounce of gold with 0·3 pound of zinc dust. Precipitating at the end of every fourth cycle, the barren solutions at the end of the twelfth cycle assayed 0·12 and 0·014 ounce of gold, using zinc and aluminium respectively.

Barren solutions from Sample B ore, from which the gold had been precipitated every cycle with 0·6 pound of zinc or aluminium, assayed 0·434 and 0·005 ounce of gold respectively. The high gold value, namely 0·434 ounce, is due to nickel and copper, which are high in Sample B ore solutions, coating the zinc particles, thus making the zinc passive to gold precipitation.

The solution, before precipitation, assayed 0·155 gramme of nickel and 0·069 gramme of copper per litre. After precipitation the analyses were 0·140 gramme of nickel and 0·055 gramme of copper, indicating that these metals are partly precipitated.

7. Grinding in Lime Circuit:

When grinding in lime solution, discarding the grinding solution, and cyaniding with fresh solution, the dissolution of base metals was slightly lower than when cyanide was added to the grinding circuit.

8. Recovery of Gold by Charcoal Flotation:

Activated charcoal was added to the grind. After 24 hours' cyanidation, the charcoal was recovered by flotation. Gold recoveries of 96·1, 95·8, and 93·5 per cent were made from 10, 5, and 2 pounds of charcoal per ton of ore.

Experimental Tests

Tests Nos. 1, 2, and 3

These were to determine the mineral constituents that will dissolve in cyanide and lime solutions, particularly chromium and nickel.

Samples of ore were ground in lime and cyanide solutions. The pulp was agitated in bottles for 24 hours. During agitation, sodium cyanide in the pulp solution was kept at about 1·0 pound and the lime at about a quarter of a pound per ton of solution.

Some chromium-bearing minerals dissolve in alkaline solutions. The Chesterville Larder Lake ore was ground in lime solution to determine whether the contained chromium-bearing minerals would dissolve. The pulp was agitated for 24 hours. The alkalinity in the solution was kept at about a quarter of a pound of lime during agitation. No cyanide was used.

Tests Nos. 1 and 2:

—	Sample B ore		Sample A ore	
	Test No. 1C	Test No. 1B	Test No. 2A	Test No. 2C
Agitation solution	Sodium cyanide and lime	Lime only	Sodium cyanide and lime	Lime only
Dilution, liquid to solid	1·5 : 1	1·5 : 1	1·5 : 1	1·5 : 1
Cyanide consumed, lb./ton of solid	1·16	....	0·81	....
Lime consumed, lb./ton of solid	4·1	6·5	4·1	7·3
Cyanide residue, Au, oz./ton of solid	0·01	....	0·015	....
Fineness of grind, per cent −200 mesh	85·3	85·3	79·2	79·2

Spectrographic analyses of the solutions of Tests Nos. 1 and 2 were as follows:

LEGEND:

1. Strong spectrum, is an essential constituent.
2. Weak spectrum, is a minor essential constituent.
3. Strong traces.
4. Traces.
5. Very faint traces.

Element	Sample B ore		Sample A ore	
	Test No. 1C	Test No. 1B	Test No. 2A	Test No. 2C
	Sodium cyanide and lime solution	Lime solution	Sodium cyanide and lime solution	Lime solution
Sodium	1	2	1	3
Nickel	1	Nil	1	Nil
Calcium	2	1	2	1
Silicon	3	3	2	4
Iron	3	4	3	Nil
Aluminium	4	Nil	4	5
Copper	3	3	3	4
Zinc	5	5	4	Nil
Molybdenum	4	Nil	3	Nil
Potassium	4	Nil	3	Nil
Strontium	3	4	3	Nil
Arsenic	4	4	5	Nil
Lead	Nil	5	4	4
Vanadium	4	Nil	4	Nil
Gold	3	....	3	....
Silver	3	....	3	....
Antimony	5	Nil	5	Nil
Magnesium	4	2	5	2
Chromium	Nil	Nil	Nil	Nil
Cobalt	Nil	Nil	....	....

Cyanidation Test No. 3:

	Sample A ore	Sample B ore	Sample C ore
	Test No. 3A	Test No. 3B	Test No. 3C
Dilution, liquid to solid	1 : 1	1 : 1	1 : 1
Cyanide consumed, lb./ton of solid	1·00	1·16	1·04
Lime consumed, lb./ton of solid	3·3	3·3	3·5
Cyanide residue, Au. oz./ton of solid	0·013	0·005	0·010
Fineness of grind per cent −200 mesh,	92·3	93·5	92·0

The analyses of the pregnant solutions of Test No. 3 were as follows:

Solution Analyses:

Grm./litre	Sample A ore	Sample B ore	Sample C ore
	Test No. 3A	Test No. 3B	Test No. 3C
Chromium (determined spectrographically)	Nil	Nil	Nil
Nickel	0·016	0·04	0·027
Copper	0·02	0·007	0·009
Antimony	0·003	0·005	0·008
Bismuth	Nil	Nil	Nil
Lead	Trace	Trace	Trace
Magnesium	0·008	0·006	0·005
Arsenic	0·002	0·014	0·012
Iron	0·04	0·05	0·04
Molybdenum	Nil	Nil	Nil
Sulphur	0·096	0·062	Not determined

Summary of the Analyses. Analyses of the solutions show no indication of chromium. The cyanide solutions contain an appreciable amount of nickel, which may be an interfering factor in gold precipitation. Sample B contains more nickel than Sample A or Sample C, hence more nickel will be found in its cyanide solutions.

GOLD PRECIPITATION TESTS

Standard Cyanidation Tests: Samples of ore were ground in lime and cyanide solution to about 90 per cent −200 mesh. The pulp was agitated in bottles for 24 hours. During agitation, sodium cyanide in the pulp solution was kept at about one pound and the lime at about a quarter of a pound per ton of solution.

After 24 hours' agitation the cyanidation pulp was filtered and the gold in the cyanide solution was precipitated by the method described below.

Method of Gold Precipitation: The apparatus for gold precipitation consisted of three Erlenmeyer flasks of 3,000-millilitre capacity: one, where evacuation of air from pregnant solution takes place; the second is for precipitation; and the third, which is fitted with a Buckner filter, is for the barren solution. These flasks are airtight and connected in series by tubing. The "barren solution" flask is connected to the suction.

Three hundred and thirty millilitres of clarified pregnant solution is placed in the "evacuation" flask, which is shaken in such a manner as to give the solution a centrifugal motion. This type of motion spreads the solution along the walls of the flask, thus exposing a larger area of surface and also a thin layer of solution along the wall of the flask. When the air has been evacuated from the solution (this point is readily observed by the disappearance of froth caused by the agitation) the solution is passed into the "precipitation" flask, in which has been placed zinc dust, and agitated for 15 minutes. The solution is then passed through a Buckner filter (filter paper covered with diatomaceous silica) into the "barren solution" flask.

This method not being comparable with the Merrill-Crowe method used in mill practice, it is necessary to run comparative tests against ores known to give very low barren solutions in mill practice. In the case of tests of Chesterville ore, the results of precipitation tests were compared with those of Preston East Dome ore.

Tests Nos. 4, 6, 8, 9, 10, 11, 15, and 31

These were to determine the gold precipitation with different amounts of zinc dust.

Test No.	Ore Sample	Grm./litre			Lb./ton	Au, oz./ton	
		Ni	Cu	Cr	Zn	Pregnant solution	Barren solution
4A	Preston East Dome	Nil	....	....	3·3	0·76	0·0005
6B	Preston East Dome	Nil	....	....	0·6	0·89	0·004
6C	Preston East Dome	Nil	....	....	0·3	0·89	0·006
15C	Preston East Dome	Nil	....	....	0·3	0·204	0·002
8B	Chesterville A	0·014	....	....	3·3	0·093	0·0015
8G	Chesterville A	0·014	....	....	2·0	0·093	0·0015
8F	Chesterville A	0·014	....	....	1·0	0·093	0·001
8E	Chesterville A	0·014	....	....	0·4	0·093	0·001
9B	Chesterville B	0·04	0·006	Nil	3·3	0·6	0·001
10C	Chesterville B	0·027	....	....	0·6	0·6	0·011
10B	Chesterville B	0·027	....	....	0·12	0·6	0·043
11B	Chesterville C	....	....	....	4·0	0·17	Trace
31B	Chesterville C	0·026	....	....	0·3	0·21	0·01

The results indicate that from 0·3 to 0·6 pound of zinc dust is necessary for gold precipitation by the method used, depending on the gold content in the pregnant solution.

Preston East Dome and Chesterville Sample C pregnant solutions (gold content of about 0·2 ounce) gave barren solutions assaying 0·002 and 0·01 ounce of gold per ton, respectively, with 0·3 pound of zinc dust. Six-tenths of a pound of zinc dust gave barren solutions of 0·011 ounce on Chesterville Sample B and 0·004 ounce on Preston East Dome, the pregnant solutions assaying 0·60 and 0·89 ounce respectively. This indicates some interfering element in the Chesterville solutions.

Tests Nos. 4, 5, and 6

These were to determine the effect of nickel on gold precipitation.

To a sample of Preston East Dome ore, nickel sulphate was added. The sample was treated by straight cyanidation.

The results of precipitation tests were as follows:

Effect of Nickel on Gold Precipitation:

Test No.	Ore Sample	Grm./litre	Lb./ton	Au, oz./ton	
		Ni	Zn	Pregnant solution	Barren solution
4A	Preston East Dome....................	Nil	3·3	0·76	0·0005
5A	Preston East Dome....................	0·077	3·3	0·88	0·007
6B	Preston East Dome....................	Nil	0·6	0·89	0·004
5D	Preston East Dome....................	0·077	0·6	0·88	0·015
6C	Preston East Dome....................	Nil	0·3	0·89	0·006
5F	Preston East Dome....................	0·077	0·3	0·88	0·061

The results indicate that nickel is an interfering element.

Test No. 7

This was to determine the effect of chromium on gold precipitation.

To a sample of Chesterville C ore, 0·4 pound of potassium chromate per ton of ore was added. It was then ground in cyanide and lime solution. The pulp was agitated for 24 hours at 1 : 1 dilution.

Chromium in the pregnant solution was precipitated by barium chloride and by lead nitrate. These reagents were added to the solution before clarification. Nickel in the pregnant solution was 0·023 gramme per litre.

Results of Gold Precipitation:

Test No.	$BaCl_2$, lb./ton	$Pb(NO_3)_2$, lb./ton	Zn, lb./ton	Cr, grm./litre		Au, oz./ton	
				Before Zn precipitation	After Zn precipitation	Pregnant solution	Barren solution
7B	....	....	0·6	0·004	0·004	0·186	0·176
7C	2·4	....	0·6	0·0001	0·00005	0·186	0·0005
7D	....	3·0	0·6	0·00025	0·0001	0·186	0·001

The results indicate that chromium in cyanide solution is an interfering element in gold precipitation.

Tests Nos. 7C and D indicate that some chromium is precipitated by zinc dust. Barium chloride and lead nitrate added to the pregnant solution before clarification are effective precipitants of chromium, permitting low barrens to be obtained.

Tests Nos. 5, 6, 8, 10, 12, 13, 14, 30, and 31

A series of tests was carried out with various gold precipitation reagents.

When barium chloride or lead acetate was used, it was added to the pregnant solution before clarification. Ammonium sulphate was added after clarification.

In the case of precipitation with aluminium dust, 1·2 pounds per ton of solution of sodium hydroxide was added to the pregnant solution before de-aeration.

The results of the tests are tabulated in Table I.

TABLE I

Gold Precipitation

Test No.	Grm./litre			Reagent added		$(NH_4)_2SO_4$, lb./ton	Lb./ton			Au, oz./ton	
	Ni	Cu	Cr	Reagent	Lb./ton		Zn	Activated charcoal	Al	Pregnant solution	Barren solution
	Preston East Dome										
5D	0·077						0·6			0·88	0·015
5E	0·077			$BaCl_2$	0·6		0·6			0·88	0·01
5F	0·077						0·3			0·88	0·061
5G	0·077			$BaCl_2$	0·6		0·3			0·88	0·04
6C	Nil						0·3			0·89	0·006
15G	Nil						0·3			0·204	0·002
	Chesterville Sample A										
8F	0·014						1·0			0·093	0·001
8J	0·014			$BaCl_2$	1·0		1·0			0·093	0·001
8H	0·014			Pb acetate	1·0		1·0			0·093	0·0005
12C	0·023	0·019		$BaCl_2$	0·6		0·6			0·114	0·004
12D	0·023	0·019		$BaCl_2$	0·6	0·6	0·6			0·114	0·002
12F	0·023	0·019		$BaCl_2$	0·6		0·3			0·114	0·003
12G	0·023	0·019		$BaCl_2$	0·6	0·6	0·3			0·114	0·0025
	Chesterville Sample B										
10C	0·027						0·6			0·6	0·011
10D	0·027			$BaCl_2$	0·6		0·6			0·6	0·002
13D	0·023		0·00001				0·6			0·738	0·0665
13C	0·023		0·00001	$BaCl_2$	0·6		0·6			0·738	0·02
13B	0·023		0·00001			0·6	0·6			0·738	0·014
13E	0·023		0·00001	$BaCl_2$	0·6				0·6	0·738	0·0095
30B	0·049						0·6			0·842	0·066
30C	0·049							0·6		0·842	0·458
30D*	0·049							0·6		0·842	0·424
	Chesterville Sample C										
14C	0·032			$BaCl_2$	0·6		0·6			0·19	0·007
31B	0·026						0·3			0·21	0·01
31C	0·026							0·6		0·21	0·066
31D*	0·026							0·6		0·21	0·042
31F	0·026							0·3		0·21	0·139

*Tests Nos. 30D and 31D. The activated charcoal was added to pregnant solutions that were not clarified nor de-aerated. The solutions were agitated with charcoal for one hour.

In Precipitation Test No. 31E, 0·6 pound per ton of activated charcoal was added to Chesterville Sample C pregnant solution (solution not clarified nor de-aerated) and agitated for one hour. The solution was then filtered to remove the charcoal, and the barren solution (gold content of 0·042 ounce per ton) was de-aerated, and the gold precipitated with 0·06 pound of zinc dust. The final barren solution assayed 0·015 ounce gold. It is probable that some of the zinc dust was used up in precipitation of nickel and copper, hence, more than 0·06 pound of zinc dust would be necessary for satisfactory gold precipitation by the method used.

Summary of Precipitation Tests. Charcoal precipitation did not give encouraging results. Using the same amounts of reagents (0·6 pound per ton) for precipitation of gold in Chesterville Sample B solution, barren solutions of 0·458 and 0·066 ounce gold were obtained with charcoal and zinc dust, respectively.

Barium chloride and ammonium sulphate improved precipitation of gold. Precipitation with aluminium dust gave encouraging results.

CYCLE TESTS

Tests Nos. 10, 18, 19, 20, and 21

In order to build up the gold content in the cyanide solutions to about half an ounce per ton, cyanidation cycle tests were conducted, that is, the cyanide solution from the previous test is used in the grind and the agitation in the succeeding test. The gold in the cyanide solution was precipitated after the last cycle.

In Test No. 19, half a pound of ammonium sulphate per ton of ore was added to the grind in each cycle.

A sample of Preston East Dome ore, which is high-grade gold, was mixed with silica sand to make the mixture about the same gold content as that of Chesterville Sample C ore.

—	Test No. 10, Chesterville Sample B	Test No. 18, Chesterville Sample C	Test No. 19, Chesterville Sample C	Test No. 20, Chesterville Sample A	Test No. 21, Preston East Dome and sand
Ammonium sulphate, lb./ton	....	....	0·25	....	....
Number of cycles	1	4	4	6	4
Average NaCN consumption, lb./ton	1·12	1·02	1·05	0·88	0·19
Average lime consumption, lb./ton	4·0	4·1	3·1	3·7	1·4
Dilution, liquid to solid	1·5 : 1	1·5 : 1	1·5 : 1	1·5 : 1	1·5 : 1
Agitation period (each cycle)	24 hours	24 hours	24 hours	24 hours	24 hours
Average cyanide residue, Au, oz./ton	0·01	0·01	0·01	0·01	Not determined

The results are tabulated in Table II.

TABLE II

Gold Precipitation

Test No.	Reducing power ml. $\frac{N}{10}$ $KMnO_4$/l.	Grm./litre KCNS	Grm./litre Ferrous iron	Grm./litre Ni	Grm./litre Cu	Grm./litre Cr	$BaCl_2$, lb./ton	$(NH_4)_2SO_4$, lb./ton	Zn, lb./ton	Au, oz./ton Pregnant solution	Au, oz./ton Barren solution
Chesterville Sample B (one cycle)											
10C	60	0·06	0·02	0·027	....	Nil	....	....	0·6	0·6	0·011
10D	60	0·06	0·02	0·027	....	Nil	0·6	....	0·6	0·6	0·002
10B	60	0·06	0·02	0·027	....	Nil	....	....	0·12	0·6	0·043
Chesterville Sample C (four cycles)											
18F	....	....	....	0·063	....	Nil	....	....	0·6	0·448	0·034
18C	....	....	....	0·063	....	Nil	0·6	....	0·6	0·448	0·044
18G	....	....	....	0·063	....	Nil	....	....	0·3	0·448	0·049
18E	....	....	....	0·063	....	Nil	0·6	....	0·3	0·448	0·017
Chesterville Sample C (four cycles)—$(NH_4)_2SO_4$ added to grind											
19B	172	0·16	0·19	0·07	0·013	....	....	....	0·6	0·462	0·003
19C	172	0·16	0·19	0·07	0·013	....	....	....	0·3	0·462	0·008
Chesterville Sample A (six cycles)											
20B	300	0·33	0·12	0·08	0·07	....	....	....	0·6	0·552	0·148
20D	300	0·33	0·12	0·08	0·07	....	0·6	....	0·6	0·552	0·124
20E	300	0·33	0·12	0·08	0·07	....	0·6	....	0·3	0·552	0·132
20J	300	0·33	0·12	0·08	0·07	....	0·6	....	1·2	0·552	0·12
20C	300	0·33	0·12	0·08	0·07	....	....	1·2	0·6	0·552	0·044
20H	300	0·33	0·12	0·08	0·07	....	0·6	1·2	0·6	0·552	0·014
Preston East Dome and Sand (four cycles)											
21B	22	0·008	0·09	Nil	0·003	....	....	....	0·6	0·426	0·004
21C	22	0·008	0·09	Nil	0·003	....	0·6	....	0·6	0·426	0·005
21D	22	0·008	0·09	Nil	0·003	....	....	....	0·3	0·426	0·01
21E	22	0·008	0·09	Nil	0·003	....	0·6	....	0·3	0·426	0·004

Test No.	Sample	Reducing power, ml. $\frac{N}{10}$ $KMnO_4$/l	Grm./litre					Spectrographic determination, Cr	$BaCl_2$, lb./ton	Zn, lb./ton	Al, lb./ton	Au, oz./ton		Remarks
			KCNS	Ferrous iron	Ni	Cu	As					Pregnant solution	Barren solution	
6B	Preston East Dome........	....	....	....	Nil	....	....	....	....	0·6	....	0·89	0·004	
24	Chesterville B........	316	0·25	....	0·18	0·03	0·01	....	0·6	0·6	....	0·988	0·434	Gold precipitated every cycle.
25	Chesterville B........	280	0·21	....	0·15	0·028	0·016	....	0·6	....	0·6	0·85	0·005	
26	Chesterville C........	400	0·39	....	0·16	0·075	0·012	....	0·6	0·6	....	0·66	0·12	Gold precipitated end of fourth, eighth, and twelfth cycles.
27	Chesterville C........	400	0·39	0·24	0·155	0·069	0·031	Very faint trace	0·6	....	0·6	0·542	0·014	
15C	Preston East Dome........	....	....	....	Nil	....	....	....	....	0·3	....	0·204	0·002	
28B	Chesterville C........	280	0·19	0·34	0·087	0·046	....	....	0·6	0·6	....	0·234	0·002	Gold precipitated every cycle.
28C*	Chesterville C........	280	0·19	0·34	0·087	0·046	....	....	0·6	0·3+ 0·3	....	0·234	0·005	
29B	Chesterville C........	308	0·22	0·34	0·12	0·032	....	....	0·6	....	0·6	0·204	0·002	
29C*	Chesterville C........	308	0·22	0·34	0·12	0·032	....	....	0·6	....	0·3+ 0·3	0·204	0·001	

*NOTE.—In these tests (28C and 29C), "double precipitation" was carried out on the twelfth cycle pregnant solutions, that is, after adding 0·6 pound of $BaCl_2$, the solution was clarified, de-aerated, and the gold was precipitated with 0·3 pound of precipitation reagent. After filtering, the solution was again de-aerated and the gold was precipitated with 0·3 pound of reagent.

In the cycle tests, the nickel as well as other fouling compounds were "building up". This will account for the high gold content in the barren solutions of Chesterville Samples C and A.

Ammonium sulphate added to the grind showed marked improvement on gold precipitation.

Tests Nos. 24, 25, 26, 27, 28, and 29 (12-cycle Cyanidation)

In Tests Nos. 25, 27, and 29, in which the gold was precipitated with aluminium dust, 1·2 pounds of sodium hydroxide per ton was used in place of lime for protective alkalinity.

In Tests Nos. 24, 25, 28, and 29 the gold was precipitated every cycle. In Tests Nos. 26 and 27 the gold was precipitated at the end of the fourth, eighth, and twelfth cycles.

—	Chesterville B ore		Chesterville C ore			
	Test No. 24	Test No. 25	Test No. 26	Test No. 27	Test No. 28	Test No. 29
Number of cycles	12	12	12	12	12	12
Agitation period (each cycle), hours	24	24	24	24	24	24
Dilution, liquid to solid	1 : 1	1 : 1	1 : 1	1 : 1	1 : 1	1 : 1
Average NaCN consumed, lb./ton	0·98	1·14	0·97	1·09	0·91	1·09
Average lime consumed, lb./ton	3·6	....	3·6	....	3·7	....
Average NaCN consumed, lb./ton	....	1·2	....	1·2	....	1·2
Cyanide residue, average for eleven cycles, Au, oz./ton	0·01	0·01	0·01	0·01	0·01	0·012
Cyanide residue, twelfth cycle, Au, oz./ton	0·025	0·01	0·02	0·02	0·015	0·01

Nickel and copper are precipitated during the gold precipitation with zinc and with aluminium, as shown by the following:

—	Precipitation with zinc		Precipitation with aluminium	
	Au precipitation every 4th cycle	Au precipitation every cycle	Au precipitation every 4th cycle	Au precipitation every cycle
	Test 26	Test 28	Test 27	Test 29
Nickel, grm./litre	0·16	0·087	0·155	0·12
Copper, grm./litre	0·075	0·046	0·069	0·032

Analyses of the cyanidation solution of Test No. 27, before and after precipitation with aluminium, confirm the nickel and copper precipitation. The analyses were as follows:

—	Pregnant solution	Barren solution
Nickel, grm./litre	0·155	0·140
Copper, grm./litre	0·069	0·055

Aluminium dust precipitation gave lower barren solution than zinc dust. This is due to the fact that for the same weight of reagent, by volume, there is two and a half times as much aluminium as zinc.

"Double precipitation" showed no improvement.

Test No. 32

This was to determine whether grinding in a lime circuit, without cyanide, would result in lower dissolution of base metals during cyanidation than when cyanide is added to the grind.

Test No. 32A. A sample of Chesterville B ore was ground with 1·2 pounds of sodium cyanide and 1·6 pounds of lime per ton of ore. The pulp was agitated for 23 hours and 40 minutes at 1 : 1 dilution.

Test No. 32B. A sample of B ore was ground with 1·6 pounds of lime per ton of ore. The pulp was filtered (13 per cent of grinding solution remained in the filter cake) and was made up with fresh water for cyanidation. The pulp was agitated 24 hours.

The pulp solution was kept at 2·0 pounds of sodium cyanide and a quarter of a pound of lime during agitation.

—	Test No. 32A	Test No. 32B
NaCN consumed, lb./ton of solid	1·44	0·6
Total lime, lb./ton of solid	2·2	2·8
Cyanide residue, Au, oz./ton	0·01	0·008

Cyanide Solution Analyses:

—	Test No. 32A	Test No. 32B	Test No. 32B
	Cyanide solution	Cyanide solution	Grinding solution
Nickel, grm./litre	0·04	0·037	Nil
Copper, grm./litre	0·017	0·015	Trace
Antimony, grm./litre	0·005	0·003	Nil
Arsenic, grm./litre	0·022	0·01	Trace

Grinding in lime circuit slightly decreased the dissolution of base metals during cyanidation. The cyanide consumption was appreciably lower.

Test No. 33

An attempt was made to recover the gold by cyaniding the ore to which activated charcoal had been added, and then recovering the charcoal by flotation.

Samples of Chesterville C ore were ground and cyanided by the method of standard cyanidation, with the exception that activated charcoal was added to the grind. After 24 hours' agitation the charcoal was recovered by flotation.

Charcoal Added:

Test No. 33A..	10 lb./ton ore.
Test No. 33B..	5 "
Test No. 33C..	2 "

Reagents Added to Flotation:

Fuel oil No. 34—0·18 lb./ton—10-minute conditioning.
Pine oil—0·09 lb./ton—15-minute float.

Results:

Test No.	Product	Weight, grammes	Assay, Au, oz./ton	Total Au, mg.	Distribution, Au, per cent	Ratio of concentration
33A	Feed (solid)..........................	1,001·0	0·255	8·754	100·0	
	Charcoal concentrate................	62·3	3·93	8·36	95·5	16·1 : 1
	Flotation tailing.....................	938·7	0·01	0·321	3·7	....
	Flotation tailing solution............	4,280·0	0·0005	0·073	0·8	....
33B	Feed (solid)..........................	1,002·3	0·252	8·645	100·0	....
	Charcoal concentrate................	50·3	4·75	8·18	94·6	19·9 : 1
	Flotation tailing.....................	952·0	0·01	0·326	3·8	....
	Flotation tailing solution............	4,065·0	0·001	0·139	1·6	....
33C	Feed (solid)..........................	1,001·6	0·255	8·739	100·0	....
	Charcoal concentrate................	54·9	3·93	7·38	84·4	18·3 : 1
	Flotation tailing.....................	946·7	0·01	0·324	3·7	....
	Flotation tailing solution............	4,035·0	0·0075	1·035	11·9	....

The ratio of concentration is low, owing to gangue slime. This could be improved in mill practice by recleaning the concentrate. The middling product from the recleaning circuit would be returned either to the cyanidation circuit or to the rougher flotation.

Gold Recovery. Assuming that the flotation tailing pulp is thickened to 50 per cent solid and discharged to waste; and the overflow solution is used again in the mill circuits:

Test No. 33A:

	Gold loss, per cent
Flotation tailing (in solid)	3·7
Flotation tailing solution to waste after thickening	0·2
Total loss	3·9
Gold recovery	96·1 per cent

Test No. 33B:

	Gold loss, per cent
Flotation tailing (in solid)	3·8
Flotation tailing solution to waste after thickening	0·4
Total loss	4·2
Gold recovery	95·8 per cent

Test No. 33C:

	Gold loss, per cent
Flotation tailing (in solid)	3·7
Flotation tailing solution to waste after thickening	2·8
Total loss	6·5
Gold recovery	93·5 per cent

SUMMARY AND CONCLUSIONS

The solubility in cyanide solution of the chromium-bearing minerals in the Chesterville ore is very low. Spectrographic determinations showed no chromium except in two solutions that assayed 0·00001 gramme per litre and a very faint trace, respectively. This amount of chromium is not likely to cause high gold values in the barren, as is shown by Test No. 7 to which potassium chromate was added. In this test, solutions assaying 0·0001 and 0·00025 gramme of chromium per litre gave barrens assaying 0·0005 and 0·001 ounce of gold, respectively. Analyses of solutions before and after precipitation showed that chromium is precipitated by zinc dust. An increased amount of zinc dust would, therefore, be necessary to attain satisfactory precipitation should chromium be present in cyanide solutions.

Nickel-bearing and copper-bearing minerals in the ore dissolve readily in cyanide solution. Solutions from tests of Samples A, B, and C assayed 0·016, 0·04, and 0·027 gramme of nickel and 0·02, 0·007, and 0·009 gramme of copper per litre, respectively.

Tests of Preston East Dome solutions, with and without nickel, showed this metal to be an interfering factor in gold precipitation. (Tests Nos. 4, 5, and 6). Barren solutions assayed 0·004 ounce of gold (no nickel present) and 0·015 ounce of gold (nickel content, 0·077 gramme per litre).

Barium chloride and ammonium sulphate improved gold precipitation. Six-tenths of a pound of barium chloride added to the pregnant solution before clarification decreased the gold content in the barrens from 0·0665 to 0·02 ounce per ton. Ammonium sulphate added to the grind in a four-cycle test (gold precipitated at the end of the fourth cycle) decreased the barrens from 0·034 to 0·002 ounce gold.

Precipitation with activated charcoal did not give encouraging results. Six-tenths of a pound of charcoal gave a barren of 0·424 ounce of gold; the same amount of zinc dust gave 0·066 ounce of gold in Sample B ore solution. Three-tenths of a pound of zinc dust and 0·6 pound of charcoal added to Sample C ore solution gave barrens which assayed 0·01 and 0·042 ounce of gold respectively. By precipitation with 0·6 pound of charcoal followed by that of zinc dust (0·06 pound), a barren solution was obtained that assayed 0·015 ounce of gold. It is probable that more than 0·06 pound of zinc is required for satisfactory precipitation by the method used.

Barren solutions of 0·002 ounce gold content at the end of the twelfth cycle were obtained with 0·6 pound of zinc and aluminium dust from Sample C ore solution in which the gold was precipitated every cycle. This value compares favourably with the test of Preston East Dome solution, which gave a barren of 0·002 ounce of gold with 0·3 pound of zinc dust. Precipitating every fourth cycle with 0·6 pound of zinc and aluminium, the barrens at the end of the twelfth cycle assayed 0·12 and 0·014 ounce of gold respectively. Precipitating every cycle, some nickel and copper are precipitated during every cycle; thus the amounts of these two metals in solution were low. In the case of precipitation every fourth cycle, the nickel and copper were "building up", as is shown by the analyses of solutions at the end of the twelfth cycles: (1) precipitation every cycle, nickel 0·087, copper 0·046 gramme per litre; (2) precipitation every fourth cycle, nickel 0·16, copper 0·075 gramme per litre.

Sample B ore solution, in which the gold was precipitated every cycle with 0·6 pound of zinc and aluminium dust, gave barrens, at the end of the twelfth cycle, assaying 0·434 and 0·005 ounce of gold; the pregnant solutions assayed 0·18 and 0·15 gramme of nickel per litre respectively. Lower barren with precipitation by aluminium dust is due to the fact that, for the same weight of reagent, by volume there is two and a half times as much aluminium as zinc. It is probable that with a larger amount of zinc a low barren could be obtained.

It is evident that owing to the nickel and copper in the ore, the zinc consumption in mill practice will be high. In mill practice low barrens will probably be obtained at the cost of high zinc consumption. It may be necessary to send part of the solution after precipitation to waste, in order to maintain low copper and nickel in the solution.

Sodium sulphide and ammonium sulphide were added to cyanide solutions to precipitate copper and nickel. Cyanide compounds of these metals are too stable and will not react to form insoluble sulphides.

By grinding in lime solution, discarding the grinding solution, and cyaniding with fresh water, the base metal content in the cyanidation solution was decreased slightly.

Extraction of gold by flotation of activated charcoal gave recoveries of 96·1, 95·8, and 93·5 per cent, with 10, 5, and 2 pounds of charcoal per ton of ore, respectively. The ratio of concentration was low owing to gangue slime, but could be improved in mill practice by recleaning the concentrate. The middling product from the recleaning circuit should be returned either to the cyanidation circuit or to the rougher flotation.

Ore Dressing and Metallurgical Investigation No. 767

GOLD ORE FROM THE WOOD CADILLAC MINES, LIMITED, KEWAGAMA, CADILLAC TOWNSHIP, QUEBEC

Shipment. Five bags of gold ore, weighing 314 pounds, were received on March 9, 1939, from the Wood Cadillac Mines, Limited, Kewagama, Quebec. The shipment was submitted by Mill Builders, Limited, Haileybury, Ontario.

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

The *gangue* is an assemblage of milky-white translucent quartz and dark grey to black, siliceous rock with a small quantity of finely disseminated carbonate.

In the polished sections *metallic minerals* are more abundant than gangue. In their approximate order of decreasing abundance they are: pyrite, arsenopyrite, magnetite, chalcopyrite, pyrrhotite, and native gold. Pyrite predominates as coarse-textured masses and large euhedral to subhedral crystals and grains intimately associated with arsenopyrite. It contains numerous fractures filled with gangue and many inclusions of the same material. Arsenopyrite is very abundant as coarse crystals and irregular grains admixed with pyrite. Like the latter mineral it contains numerous inclusions of gangue and fractures healed with gangue. A considerable quantity of magnetite is present in one section as small, irregular grains in gangue and in pyrite. Small amounts of both chalcopyrite and pyrrhotite are visible as rare, small grains in pyrite and arsenopyrite.

Sixty-nine grains of native gold were observed and measured. All occur in arsenopyrite. Their modes of occurrence and grain sizes are given in the table below:

Grain Size and Mode of Occurrence of Native Gold:

Mesh	Gold in arsenopyrite		Totals, per cent
	Associated with gangue, per cent	Alone, per cent	
+ 400	....	10·7	10·7
− 400+ 560	18·8	....	18·8
− 560+ 800	11·0	2·4	13·4
− 800+1100	13·1	11·2	24·3
−1100+1600	6·7	13·3	20·0
−1600+2300	2·6	0·7	3·3
−2300	4·0	5·5	9·5
Totals	56·2	43·8	100·0

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

Gold	1·63 oz./ton (average of six assays)
Silver	0·37 " " "
Copper	0·04 per cent
Iron	13·45 "
Arsenic	1·41 "
Sulphur	5·12 "

Results of Investigation. Experimental work embraced concentration of free gold by jigs and blankets, amalgamation, and cyanidation.

Around 60 per cent of the gold is recovered by amalgamation from jig and blanket concentrates.

Direct cyanidation of the ore showed a tailing loss in gold of 0·035 ounce per ton.

The ore settles readily and the solution shows a very slight tendency to fouling.

The sample of ore submitted is probably of a much higher grade than that constituting the normal mill feed. In the conclusions the results of the investigation in this regard are discussed.

Experimental Tests

BARREL AMALGAMATION

Tests Nos. 1 and 2

Two samples of the ore, −14 mesh, were ground in a ball mill to have 50·5 and 77 per cent −200 mesh respectively. The ground pulps were barrel-amalgamated with mercury in an Abbé grinding jar for one hour. The results obtained indicate the free amalgamable gold at the respective fineness of grinding shown and do not necessarily indicate the recovery by amalgamation in mill operation.

Results:

Test No.	Feed assay, Au, oz./ton	Tailing assay, Au, oz./ton	Recovery, per cent
1	1·63	0·56	65·6
2	1·63	0·42	74·2

CONCENTRATION, AMALGAMATION, AND CYANIDATION

Test No. 3

A charge of ore, −14 mesh, was ground in a ball mill to a fineness of 65·5 per cent −200 mesh and the pulp was run over a Denver mineral jig. The jig tailing was run over a blanket strake. The combined jig and blanket concentrates were amalgamated. The blanket tailing was reground in cyanide to a fineness of 90·6 per cent −200 mesh and agitated for 24 hours.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Jig Concentration:				
Feed	100·0	1·63	100·0	....
Concentrate	6·33	14·8	57·47	15·8 : 1
Tailing	93·67	0·74	42·53	....
Blanket Concentration:				
Feed	100·0	0·74	100·0	....
Concentrate	3·03	5·22	21·38	33 : 1
Tailing	96·97	0·6	78·62	....

Amalgamation of Combined Concentrates:

Feed assay, Au, oz./ton	Tailing assay, Au, oz./ton	Recovery, per cent
12·38	2·48	79·9

Cyanidation (Pulp Dilution 2 : 1):

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
24	0·6	0·015	97·5	1·0	0·25	0·8	4·5

Summary:

	Au, per cent
Recovered in jig concentrate	57·47
Recovered in blanket concentrate	9·09
Recovered by amalgamation	53·18
Extraction by cyanidation	32·6*
Overall extraction	85·78

*Cyanidation of blanket tailing only.

Test No. 4

This was a duplicate of Test No. 3 with the period of cyanide agitation increased to 48 hours.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Jig Concentration:				
Feed	100·0	1·63	100·0	....
Concentrate	9·18	11·13	62·67	10·89 : 1
Tailing	90·82	0·67	37·33	....
Blanket Concentration:				
Feed	100·00	0·67	100·0	....
Concentrate	3·05	5·76	26·2	32·79 : 1
Tailing	96·95	0·51	73·8	....

Amalgamation of Combined Concentrates:

Feed assay, Au, oz./ton	Tailing assay, Au, oz./ton	Recovery, per cent
10·14	1·73	82·9

Cyanidation (Pulp Dilution 2 : 1):

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
48	0·51	0·015	97·06	0·92	0·2	0·56	4·6

Summary:

	Au, per cent
Recovered in jig concentrate	62·67
Recovered in blanket concentrate	9·78
Recovered by amalgamation	60·06
Extraction by cyanidation	26·74*
Overall extraction	86·80

*Cyanidation of blanket tailing only. The amalgamation tailing was not cyanided.

These tests indicate primarily the recovery of free-milling gold. Subsequent tests show the action of cyanide on the gold occurring in dense sulphides.

CYANIDATION

Tests Nos. 5, 6, 7, and 8

Charges of ore were ground directly in cyanide to have 77·5 and 88·2 per cent −200 mesh respectively and were agitated for 24- and 48-hour periods. The solution strength was 1 pound of sodium cyanide per ton at a pulp dilution of 2 : 1, and lime (4·9 pounds of lime per ton of ore) was added to maintain protective alkalinity.

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

Mesh	Weight, per cent	
	Tests 5 and 6	Tests 7 and 8
+ 65	0·2	....
− 65+100	2·7	0·6
−100+150	9·5	4·4
−150+200	10·1	6·8
−200	77·5	88·2
	100·0	100·0

Results:

Feed: gold, 1·53 oz./ton

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
5	24	0·04	97·5	0·96	0·2	0·68	4·5
6	48	0·045	97·2	0·92	0·2	0·76	4·9
7	24	0·035	97·8	0·92	0·2	0·76	4·5
8	48	0·035	97·8	0·96	0·2	1·08	4·9

The tests show a rather high tailing. The table on "Grain Size and Mode of Occurrence of Native Gold" shows that in the specimens examined 33 per cent of the gold occurs in arsenopyrite and in grain sizes of −560 mesh, and, therefore, the gold loss in the tailing is probably due to very fine gold in grains of arsenopyrite. This is typical of the ores in the Cadillac area.

Test No. 9

In order to give the sulphides a preferential treatment, two charges of ore were ground in cyanide to have 77 per cent −200 mesh and the pulp was agitated for 24 hours. The cyanide tailings were then tabled and a bulk sulphide concentrate made, which was reground to have 98·6 per cent −200 mesh and again cyanided for 24 hours. The results are as follows:

Cyanidation of Ore:

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore		Pulp dilution
	Feed	Tailing		NaCN	CaO	NaCN	CaO	
9A	1·63	0·04	97·55	0·96	0·2	0·68	4·6	2 : 1
9B	1·63	0·04	97·55	0·96	0·2	0·68	4·6	2 : 1

Table Concentration of Combined Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·04	100·0	....
Concentrate	38·43	0·11	87·29	2·6 : 1
Tailing	61·57	0·01	12·71	....

Cyanidation of Reground Concentrates:

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore		Pulp dilution
	Feed	Tailing		NaCN	CaO	NaCN	CaO	
9A	0·11	0·04	63·6	1·9	0·3	3·1	9·1	3 : 1
9B	0·11	0·04	63·6	1·9	0·3	3·1	9·1	3 : 1

Summary:

	Per cent
Gold extraction, initial cyanidation	97·55
Gold extraction by re-treatment of concentrated sulphides	1·36
Overall extraction	98·91

Calculated final tailing: Au, 0·021 oz./ton.

Test No. 10

The coarse free gold was removed by jigging and on blankets, and the sulphides were concentrated by flotation. The combined concentrates were amalgamated and the amalgamation tailing was cyanided. By this procedure it was possible to obtain information on the amount of gold in the sulphides which is refractory to cyanide treatment.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Jig and Blanket Concentration:				
Feed	100·0	1·63	100·0	....
Combined concentrate	5·4	17·396	57·6	18·5 : 1
Tailing	94·6	0·73	42·4	....
Flotation:				
Feed	100·0	0·73	100·0	....
Flotation concentrate	11·17	5·78	88·4	8·95 : 1
Tailing	88·83	0·095	11·6	....

Amalgamation of Combined Concentrates:

Feed	Au, 10·0 oz./ton
Tailing	Au, 2·49 oz./ton
Recovery, per cent	75·1

Cyanidation of Amalgamation Tailing:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore		Pulp dilution
	Feed	Tailing		NaCN	CaO	NaCN	CaO	
24	2·49	0·62	75·1	1·8	0·2	4·7	8·48	2·61 : 1

Grind, 96·9 per cent −200 mesh.

Discussion of Results:

The combined jig, blanket and flotation concentrates contain 95·08 per cent of the gold in the feed. The rest, 4·92 per cent, is in the flotation tailing. From the previous tests it has been shown that over 97 per cent of this gold can be extracted by cyanidation.

By amalgamation, 71·4 per cent of the gold is recovered. Of the remaining 23·68 per cent, 17·78 per cent is extracted by cyanidation.

Summary:

	Per cent
Gold recovered by amalgamation	71·4
Gold in flotation tailing	4·92
Gold extracted by cyanidation of amalgamation tailing	17·78
Unrecovered gold in sulphides	5·9
Total, per cent	100·0

A certain proportion of the gold in the sulphides (largely arsenopyrite) is, therefore, shown to be refractory to cyanidation, and this extremely fine gold is the determining factor in the method of treatment for ores of the Cadillac area.

SETTLING TESTS

Test No. 11

A series of settling tests at different ratios of liquid to solid and with varying amounts of lime was made. After grinding in cyanide, with the requisite amount of lime added, the pulp was transferred to a glass tube of 2 inches diameter and the level of solid in decimals of a foot was read for a 1-hour period. The solution was then titrated for cyanide and alkalinity.

The grinding is shown by the following screen test:

Mesh	Weight, per cent
+100	0·5
−100+150	4·6
−150+200	7·1
−200	87·8
	100·0

Results:

—	Test A	Test B	Test C	Test D
Ratio of liquid to solid	1·5 : 1	2 : 1	1·5 : 1	2 : 1
Lime added per ton of solid, pounds	4·0	4·0	8·0	8·0
Alkalinity of solution at end of test, CaO, lb./ton	0·35	0·25	0·8	0·65
Titration for cyanide, NaCN, lb./ton	0·5	0·4	0·6	0·5
Overflow	Clear	Clear	Clear	Clear
Rate of settling, ft./hour	0·67	1·23	0·64	1·24

The settling rate is satisfactory.

CYCLE TEST

Test No. 12

A series of cyanidation tests was run in five cycles. Fresh ore was used in each cycle and the solution from the preceding cycle after treatment with zinc dust was used for both grinding and agitation. The ore charge was 1,000 grammes and the pulp dilution 2 : 1 at a cyanide strength of 1·0 pound of sodium cyanide per ton. The agitation period was 24 hours.

The grind was about 89 per cent −200 mesh.

Results:

Cycle	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
1	1·63	0·03	98·16	0·96	0·2	0·88	4·6
2	1·63	0·03	98·16	1·0	0·2	0·8	5·1
3	1·63	0·04	97·54	1·0	0·2	0·6	5·1
4	1·63	0·048	97·05	1·0	0·2	0·6	5·1
5	1·63	0·033	97·97	1·0	0·2	0·6	4·96

Analysis of Solution, Cycle 5:

Reducing power	190 ml. $\frac{N}{10}$ $KMnO_4$/litre
NaCNS	0·19 grm./litre
Iron	0·003 "
Copper	0·02 "

The results do not indicate any serious degree of fouling.

CONCLUSIONS

The results show that around 60 per cent of the gold in the sample is recoverable by amalgamation and from 37 to 38 per cent is extractable by cyanidation.

In view of the high grade of the sample submitted, it may be assumed that in practice the mill feed will be much lower in grade. The ores in the Cadillac area contain coarse free gold and very fine gold associated with arsenopyrite that is refractory to cyanide treatment. The high-grade character of ore, such as the sample examined, is largely due to the presence of abundance of free-milling gold and when the proportion of this falls off the larger proportion of fine refractory gold will lower appreciably the overall extraction.

It is this fact that should be borne in mind in considering the results of this investigation.

Ore Dressing and Metallurgical Investigation No. 768

GOLD-SILVER ORE FROM THE MOUNT ZEBALLOS GOLD MINES, LIMITED, ZEBALLOS RIVER DISTRICT, BRITISH COLUMBIA

Shipment. Five bags of ore (channel sample rejects), weighing 468 pounds, were received on February 18, 1939, from G. F. MacDonnell, consulting engineer for the Conwest Exploration Company, Limited, 85 Richmond Street West, Toronto, Ontario, and the Royal Bank Building, Vancouver, British Columbia.

Location of the Property. The property of the Mount Zeballos Gold Mines, Limited is situated on Spud Creek, Zeballos River area, Clayoquot Mining Division, Vancouver Island, British Columbia.

Sampling and Analysis. The ore sample was mixed and sampled by standard methods, and a feed sample cut out assayed as follows:

Gold	3·17 oz./ton (average of five assays)
Silver	1·50 " " "
Copper	0·09 per cent
Lead	0·37 "
Zinc	0·35 "
Iron	6·01 "
Arsenic	1·35 "
Sulphur	3·82 "

Characteristics of the Ore. As the ore was received in a finely crushed condition, it was not possible to carry out a standard microscopic examination. From the analysis of the ore and its response to metallurgical treatment it is found to resemble closely the ore from the Privateer mine, Zeballos River area. The report on ore from the Privateer mine is covered in Ore Dressing and Metallurgical Investigation No. 746, Bureau of Mines Report No. 792 (January to June, 1938).

The ore was said to contain some oxidized material as numerous channel samples were taken near the portals of the tunnels, where the ore in place is badly weathered.

Purpose and Results of the Investigation. The object of the investigation was to determine a method of metallurgical treatment of the ore from which a satisfactory flow-sheet could be worked out.

Treatment by concentration, amalgamation, flotation, and cyanidation was tried. Recovery of the gold by jigs and blankets followed by amalgamation of the resultant concentrates gave an extraction of over 80 per cent of the gold. Of the remaining gold, 18 per cent is recovered by either flotation or cyanidation, giving an overall recovery of over 98 per cent. The flotation and cyanidation tailings were the same, at 0·055 ounce gold per ton.

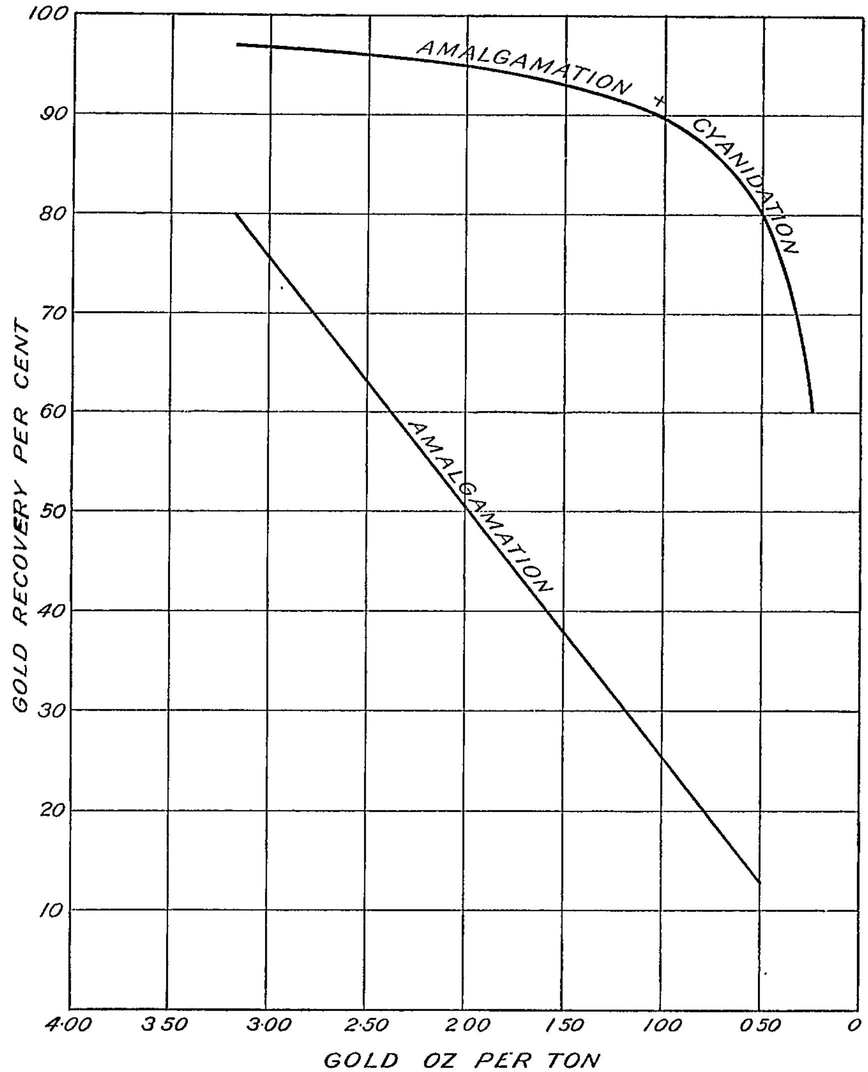


Figure 2.

EXPERIMENTAL TESTS

BARREL AMALGAMATION

Tests Nos. 1 and 2

Ore was ground in a ball mill to have 49·5 per cent pass a 200-mesh screen in Test No. 1 and 77·4 per cent pass a 200-mesh screen in Test No. 2. The pulps were amalgamated with mercury in a ball mill for one hour. The amalgamation tailings were assayed for gold.

Results:

Feed: gold, 3·17 oz./ton

Test	Tailing assay, Au, oz./ton	Recovery, per cent
1	0·54	82·97
2	0·46	85·49

These results only show the amount of gold freed at the degree of comminution indicated and may be somewhat higher than results obtained by amalgamation of jig and blanket concentrates in mill operation.

CONCENTRATION, AMALGAMATION, AND FLOTATION

Test No. 3

The ore was ground in a ball mill to have 63·3 per cent −200 mesh. The pulp was passed over a small Denver mineral jig and a jig concentrate obtained. The jig tailing was passed over a corduroy blanket strake and a blanket concentrate removed. The combined jig and blanket concentrates were amalgamated with mercury. The blanket tailing was reground with 1·6 pounds of soda ash and 0·054 pound of Aerofloat No. 31 per ton to a fineness of 68 per cent −200 mesh, and was floated using 0·16 pound of potassium amyl xanthate and 0·10 pound of pine oil per ton. A bulk concentrate was obtained.

Samples of the jig and blanket tailings were taken for assay and the gold and silver contents of the concentrates were calculated from these results. The flotation products were all sampled and assayed.

Screen tests on the jig feed and flotation feed showed the grinding as follows:

Mesh	Weight, per cent	
	Jig feed	Flotation feed
+ 65	3·0	0·4
− 65+100	8·0	4·9
−100+150	14·3	14·0
−150+200	11·4	12·6
−200	63·3	68·1
	100·0	100·0

Results:

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Jig Concentration:						
Feed	100·0	3·17	1·50	100·00	100·00	....
Concentrate	2·3	118·50	46·10	85·98	70·69	43·5 : 1
Tailing	97·7	0·455	0·45	14·02	29·31	....
Blanket Concentration:						
Feed	100·0	0·455	0·45	100·00	100·00	....
Concentrate	1·2	14·86	10·33	39·20	27·55	83·33 : 1
Tailing	98·8	0·28	0·33	60·80	72·45	....

Flotation:

Product	Weight, per cent	Assay						Distribution per cent		Ratio of concentration
		Oz./ton		Per cent						
		Au	Ag	Cu	Pb	As	S	Au	Ag	
Feed	100·0	0·30	0·39	......	......	......	......	100·0	100·0	....
Concentrate	9·9	2·54	3·02	0·35	1·10	7·37	......	83·54	75·10	10·1 : 1
Tailing	90·1	0·055	0·11	......	......	......	0·36	16·46	24·90	....

Amalgamation of Combined Jig and Blanket Concentrates:

Feed assay, oz./ton		Tailing assay, oz./ton		Recovery, per cent	
Au	Ag	Au	Ag	Au	Ag
88·10	35·61	6·50	6·54	92·62	81·63

Summary:

—	Au, per cent	Ag, per cent
Recovered in jig concentrate	85·98	70·69
Recovered in blanket concentrate	5·50	8·07
Recovered in flotation concentrate	7·12	15·85
Overall recovery	98·60	94·61
Recovered by amalgamation	84·73	64·29

Test No. 4

The ore was ground to have 83·3 per cent −200 mesh and was treated similarly to Test No. 3. The blanket tailing was not reground but was conditioned prior to flotation with 2·8 pounds of soda ash, 0·05 pound of Aerofloat No. 31, and 0·14 pound of potassium amyl xanthate per ton. Pine oil was used as frother.

Samples were taken for assay as in the previous test.

A screen test on the jig tailing showed the grinding as follows:

Mesh	Weight, per cent
+100	1·1
−100+150	6·4
−150+200	9·2
−200	83·3
	100·0

Results:

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Jig Concentration:						
Feed	100·0	3·17	1·50	100·00	100·00	....
Concentrate	0·8	316·89	113·10	79·97	60·32	125 : 1
Tailing	99·2	0·64	0·60	20·03	39·68	....
Blanket Concentration:						
Feed	100·0	0·64	0·60	100·00	100·00	....
Concentrate	0·9	24·86	16·01	34·97	24·02	111·1 : 1
Tailing	99·1	0·42	0·46	65·03	75·98	....

Flotation:

Product	Weight, per cent	Assay						Distribution, per cent		Ratio of concentration
		Oz./ton		Per cent						
		Au	Ag	Cu	Pb	As	S	Au	Ag	
Feed	100·0	0·43	0·49	......	......	......	......	100·00	100·00	....
Cleaner concentrate	9·8	3·40	3·40	0·34	1·32	8·20	......	77·21	67·69	10·2 : 1
Middling	14·5	0·34	0·55	......	......	......	......	11·42	16·20	....
Tailing	75·7	0·065	0·105	......	......	......	0·35	11·37	16·11	....

Amalgamation of Combined Jig and Blanket Concentrates:

Feed assay, oz./ton		Tailing assay, oz./ton		Recovery, per cent	
Au	Ag	Au	Ag	Au	Ag
172·31	65·03	9·80	8·51	94·31	86·91

Summary:

—	Au, per cent	Ag, per cent
Recovered in jig concentrate	79·97	60·32
Recovered in blanket concentrate	7·00	9·53
Recovered in flotation concentrate	11·55	25·29
Overall recovery	98·52	95·14
Recovered by amalgamation	82·02	60·71

Test No. 4A:

This was for obtaining information on the nature of the gold in the flotation tailing. The initial procedure was similar to that in Test No. 4.

A sample of ore was ground to have 86·1 per cent −200 mesh and the pulp was run over a Denver mineral jig and a blanket strake in series. The blanket tailing was thickened and conditioned in a flotation cell with 1·6 pounds of soda ash per ton and 0·035 pound of Aerofloat No. 31 per ton. Reagents used in flotation were: potassium amyl xanthate 0·1 pound per ton, copper sulphate 0·5 pound per ton, and pine oil 0·093 pound per ton.

The combined jig and blanket concentrates were amalgamated with mercury and the flotation tailing was studied by both hydro-classification and superpanning.

Results:

Jig and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	3·17	100·0	....
Concentrate	2·3	117·01	84·9	43·48 : 1
Tailing	97·7	0·49	15·1	....

Amalgamation of Concentrates:

Feed assay, Au, oz./ton	Tailing assay, Au, oz./ton	Recovery, per cent
117·01	7·36	93·71

Flotation of Blanket Tailing:

Product	Weight, per cent	Assay					Distribution of gold, per cent	Ratio of concentration
		Au, oz./ton	Per cent					
			Cu	Pb	As	S		
Feed	100·00	0·43	......	......	......	......	100·0	....
Concentrate	9·52	4·02	0·43	1·52	9·55	......	88·5	10·5 : 1
Tailing	90·48	0·055	......	......	0·36	0·44	11·5	....

Summary:

	Per cent
Gold recovered in jig and blanket concentrates	84·9
Gold recovered in flotation concentrate	13·4
Overall recovery	98·3
Gold recovered by amalgamation	79·56
Gold recovered as shipping concentrate (flotation concentrate + amalgamation residue)	18·74
Overall recovery	98·30

EXAMINATION OF FLOTATION TAILING

Hydro-Classification of Tailing

A sample of flotation tailing was pulped in water and fed through a hydraulic classifier. The two products, underflow (sand) and overflow (fine) were assayed.

The results are as follows:

Product	Weight, per cent	Assay: Au, oz./ton	Assay, Per cent: As	Assay, Per cent: S	Distribution, per cent: Au	Distribution, per cent: As	Distribution, per cent: S
Feed	100·00	0·06*	0·30*	0·54*	100·0	100·0	100·0
Underflow	26·44	0·18	0·28	0·60	34·3	23·9	29·3
Overflow	73·56	0·055	0·32	0·52	65·7	76·1	70·7

*Calculated from assays of products.

The fine carries the major portion of the gold and sulphides, and the gold is associated with the sulphides.

This is brought out more clearly by superpanning.

A sample of flotation tailing was treated on the Haultain superpanner and the products were assayed and examined under the microscope.

Results of Superpanning:

Product	Weight, per cent	Assay: Au, oz./ton	Assay, Per cent: As	Assay, Per cent: S	Distribution, per cent: Au	Distribution, per cent: As	Distribution, per cent: S
Feed	100·00	0·058	0·36	0·44	100·0	100·0	100·0
Sulphides	0·39	3·03	18·88*	12·04*	20·2	20·5	10·6
Sand	20·75	0·035	0·24	0·26	12·4	13·8	12·3
Fine	78·86	0·05	0·30	0·43	67·4	65·7	77·1

*Calculated.

Four grains of gold weighing 0·71 milligram were removed at the peak of the pan. These were not included in the assays.

Microscopic Examination:

1. Sand..........A few grains of quartz seen with tiny specks of sulphide attached.
2. Fine...........A few grains of free sulphide mineral noted. No attached mineral seen.

From the analyses it is apparent that the sulphides in the flotation tailing are very fine, and the gold, apart from the free gold, is closely associated with the arsenopyrite.

The high grade of this ore is primarily due to free gold. Should the proportion of free gold decrease, the principal gold will be that associated with the sulphides, and the extremely fine gold in these sulphides would probably remain the same. The result would be a lower recovery as the content of free gold dropped. This should be given careful consideration in developing any method of treatment for an ore like this.

CONCENTRATION, AMALGAMATION, AND CYANIDATION

Test No. 5

A sample of −14-mesh ore was run over the Denver jig and a blanket strake in series. The blanket tailing was filtered and the residue quartered. One quarter was retained for assay and the other three were ground separately to have 47·1, 73·1, and 87·5 per cent −200 mesh. Each lot was passed over a blanket strake and samples of the blanket tailing from each grind were cyanided for 24 and 48 hours respectively. The jig concentrate and the primary and secondary blanket concentrates were combined and barrel-amalgamated. Cyanidation was carried out in a pulp dilution of 1·5 : 1 at a cyanide strength of 1 pound of sodium cyanide per ton.

Results:

Jig and Blanket Concentration (−14-mesh Feed):

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Feed...............	100·0	3·17	1·50	100·00	100·00	....
Concentrate.........	5·2	49·66	19·18	81·46	66·50	19·23 : 1
Tailing............	94·8	0·62	0·53	18·54	33·50	....

Assays of Secondary Blanket Tailings for Each Respective Grind:

Grind, per cent −200 mesh	Au, oz./ton	Ag, oz./ton	Cu, per cent
47·1..	0·17	0·23	0·02
73·1..	0·23	0·31	0·04
87·5..	0·33	0·40	0·03

Summary of Secondary Blanket Concentration:

Product	Weight, per cent	Assay			Distribution, per cent		Ratio of concentration
		Oz./ton		Cu, per cent			
		Au	Ag		Au	Ag	
Feed...............	100·0	0·62	0·53	....	100·00	100·00	....
Concentrate.........	1·2	31·08	17·82	....	60·16	40·35	83·33 : 1
Tailing............	98·8	0·25*	0·32*	0·03*	39·84	59·65	....

*Calculated from assays of each grind.

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Cyanidation of Blanket Tailings:

Grind, per cent −200 mesh	Agitation, hours	Feed assay, oz./ton		Tailing assay, oz./ton		Extraction, per cent		Final solution, lb./ton solution		Reagents consumed, lb./ton ore	
		Au	Ag	Au	Ag	Au	Ag	NaCN	CaO	NaCN	CaO
47·1	24	0·17	0·23	0·055	0·065	61·8	71·7	1·0	0·15	1·10	8·77
	48	0·17	0·23	0·05	0·07	64·6	69·5	0·9	0·20	1·87	9·66
73·1	24	0·23	0·31	0·06	0·09	73·9	70·9	1·0	0·10	1·40	8·85
	48	0·23	0·31	0·07	0·055	69·5	82·2	1·0	0·20	1·40	10·70
87·5	24	0·33	0·40	0·075	0·095	77·3	76·2	0·9	0·10	1·96	8·84
	48	0·33	0·40	0·07	0·055	78·8	86·2	1·0	0·20	1·90	11·70

NOTE.—The high-lime consumption may be due to the oxidized material contained in the sample submitted. (See "Characteristics of the Ore".)

The analysis of the final solution from the 48-hour cyanidation of the finest ground ore resulted as follows:

Reducing power	196 ml.$\frac{N}{10}$ $KMnO_4$/litre
KCNS	0·24 grm./litre
Copper	0·06 "

Summary:

—	Au, per cent	Ag, per cent
Recovered in jig and primary blanket concentrate	81·46	66·50
Recovered in secondary blanket concentrate	11·15	13·52
Extraction by cyanidation*	5·82	17·22
Overall recovery	98·43	97·24
Recovered by amalgamation	75·66	54·25

*87 per cent −200-mesh grind. 48 hours' agitation.

Test No. 6

The procedure consisted in giving the ore a preliminary grind to have 63 per cent −200 mesh and then passing the pulp over a Denver jig and a blanket strake in series. The combined jig and blanket concentrates were barrel-amalgamated and the tailing was added to the blanket tailing. The combined tailings were reground to a fineness of 93 per cent −200 mesh in cyanide solution and filtered. The cake was divided into two equal portions and each part repulped, using the solution in which the ore was ground, and made up to a pulp dilution of 2 : 1. The strength of the solution was 1 pound of sodium cyanide per ton. Agitation was carried out for 24 and 48 hours respectively.

Results of Cyanidation:

Agitation, hours	Tailing assay, oz./ton		Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Au	Ag	NaCN	CaO	NaCN	CaO
24	0·105	0·155	1·04	0·10	2·12	9·80
48	0·105	0·145	1·00	0·10	2·20	10·80

Summary:

—	Au, oz./ton	Ag, oz./ton
Feed assays	3·17	1·50
Tailing assays	0·105	0·145
Overall recovery	96·68 per cent	90·3 per cent

The fineness of the cyanidation feed is shown by the following screen test:

Mesh	Weight, per cent
+150	2·0
−150+200	5·0
−200	93·0
	100·0

SETTLING TESTS

Test No. 7

A series of settling tests at different ratios of liquid to solid and with varying amounts of lime was made. After grinding in cyanide, with the requisite amount of lime, the pulp was transferred to a glass tube of 2 inches diameter and the level of solids in decimals of a foot was read for a 1-hour period. At the end of the test the solution was titrated for cyanide and alkalinity.

The grinding is shown by the following screen test:

Mesh	Weight, per cent
+100	0·4
−100+150	5·2
−150+200	9·2
−200	85·2
	100·0

Results:

—	Test No.			
	7A	7B	7C	7D
Ratio of liquid to solid	1·5 : 1	2 : 1	1·5 : 1	2 : 1
Lime added per ton of solid, lb	7·0	7·0	12·0	12·0
Alkalinity of solution at end of test, lime, lb./ton	Trace	0·05	0·25	0·20
Titration for cyanide, sodium cyanide, lb./ton	0·10	0·10	0·50	0·30
Overflow	Slightly cloudy	Slightly cloudy	Clear	Clear
Rate of settling, foot/hour	0·16	0·22	0·26	0·34

The rate of settling is slower than normal.

Filtration was found to be slow.

The rate of settling and filtering is probably aggravated by the oxidized material to which reference has already been made.

SUMMARY AND CONCLUSIONS

The results indicate an overall tailing from amalgamation and cyanidation having a gold content of 0·105 ounce per ton. Tests have shown that this gold is largely held in grains of arsenopyrite, and is extremely fine. This is a very important consideration in an ore of this character in which around 80 per cent of the gold is free-milling.

The accompanying empirical curve (Figure 2) shows how the overall recovery falls as the grade of the ore is lowered. It is assumed that 80 per cent of the gold is recoverable by amalgamation. The points of maximal recovery on the curves are taken from actual tests on the present sample, gold, 3·17 ounces per ton. The other points are calculated on assumed feed values of 2·50 ounces, 1·00 ounce, 0·50 ounce, and 0·25 ounce of gold per ton, it being assumed in each case that the decrease is due to falling-off in free-milling gold. As the free-milling gold diminishes, more gold proportionately will be carried in the sulphides.

The lower curve represents recovery of free-milling gold by amalgamation, the upper, overall recovery (amalgamation plus cyanidation) assuming a constant tailing loss of 0·10 ounce of gold per ton. By these curves it is shown how rapidly the overall extraction may fall as the grade of the ore drops.

The results obtained by concentration, amalgamation, and flotation indicate a final tailing carrying 0·055 ounce of gold per ton. Around 80 per cent of the gold is recovered by amalgamation and 18 per cent as a shipping concentrate, the grade of which is between 4 and 5 ounces of gold per ton. The ratio of concentration is 10·5 : 1.

Settling of solids in solution was a little slow, but as the sample contained some oxidized ore, this condition may not exist in regular run-of-mine ore.

Ore Dressing and Metallurgical Investigation No. 769

GOLD ORE FROM THE "HARRY A. INGRAHAM TRUST",
YELLOWKNIFE, NORTHWEST TERRITORIES

Shipment. A bulk sample comprising 822 pounds of gold ore was received on February 14, 1939, from Harry A. Ingraham, Yellowknife Hotel, Yellowknife, Northwest Territories.

Location of the Property. The property of the "Harry A. Ingraham Trust" from which the present shipment was received is known as Mineral Claim VIC-No. 5, 40 miles northeast of the settlement of Yellowknife, Northwest Territories.

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

Gold	4·67 oz./ton
Silver	1·22 "
Copper	Trace
Iron	4·06 per cent
Arsenic	0·04 "
Lead	Nil
Sulphur	0·15 per cent
Graphitic carbon	0·61 "

Characteristics of the Ore. Twelve polished sections were examined microscopically.

The *gangue* consists of grey to white quartz and soft, green to black rock material. In some hand specimens much of the latter megascopically appears to be graphitic shear material with crystals of mica (biotite?) oriented across its schistosity. Under the microscope the soft rock material cuts quartz in narrow veins which, in places, widen into lakes. Rusty brown stains of iron oxides locally colour the quartz along narrow fractures.

Metallic minerals are sparse and for the most part occur in rock material. In the sections, arsenopyrite is the most abundant, as coarse to fine disseminated crystals and small granular masses. It is somewhat fractured and veined with gangue and native gold and contains inclusions of these two minerals along with those of marcasite and pyrrhotite. Marcasite is present as small granular masses and discontinuous stringers in gangue, usually closely associated with aresnopyrite, less often with pyrite. It contains numerous inclusions of gangue and occasional grains of gold.

Pyrite occurs largely as small masses in gangue but is not so abundant as arsenopyrite or marcasite. Occasional medium to small irregular grains of chalcopyrite are associated with the other sulphides, but the total quantity does not appear sufficient to affect cyanide solutions adversely. Rare, tiny grains of pyrrhotite are visible in pyrite and arsenopyrite, but its total amount is negligible.

Native gold is, comparatively, very prevalent in the ore. It occurs both in gangue and in sulphides. Some 5,500 grains were measured and the results are tabulated below. It should be noted, however, that there is slightly more fine gold than is shown in the table as it was impossible to measure some very fine films and veinlets occurring in arsenopyrite, or all the very tiny particles in the gangue.

Grain Size of Native Gold:

Mesh	Gold in gangue, per cent		Gold in sulphides, per cent		Totals, per cent
	In quartz	In rock material	In arsenopyrite	In marcasite	
+ 65	1·90	6·08	....	....	7·98
− 65+ 100	0·47	4·61	....	0·66	5·74
− 100+ 150	0·48	2·44	0·46	0·34	3·72
− 150+ 200	0·88	3·49	0·34	0·34	5·05
− 200+ 280	1·33	3·90	1·01	0·64	6·88
− 280+ 400	1·68	4·70	0·50	0·48	7·36
− 400+ 560	2·76	5·96	0·93	0·33	9·98
− 560+ 800	5·17	5·66	0·61	0·15	11·59
− 800+1100	2·60	3·05	0·35	0·08	6·08
−1100+1600	6·81	8·28	0·51	0·13	15·73
−1600+2300	5·74	7·12	0·22	0·02	13·10
−2300	2·23	4·51	0·05	....	6·79
	32·05	59·80	4·98	3·17	100·00
	91·85		8·15		

Experimental Tests

Concentration, amalgamation, and cyanidation tests were conducted on samples of the ore to see how much of the gold could be recovered.

The microscopic examination revealed that the ore, which is high grade, contained considerable quantities of both coarse and very fine gold, the latter requiring fine grinding to liberate it.

Experimental tests showed that about 70 per cent of the gold could be concentrated by gravity methods and recovered as bullion by amalgamation of the concentrate. Most of the residual gold can be recovered by cyanidation of the combined amalgamation tailing and gravity concentration tailing if these products are ground 80 to 90 per cent through 200 mesh.

A fair recovery of the gold in the blanket tailing and amalgamation tailing can also be obtained by flotation.

The tests are described in detail as follows:

STRAIGHT CYANIDATION

Tests Nos. 1 to 3

Portions of the ore at −14 mesh were ground in a ball mill in cyanide solution, containing 1 pound of sodium cyanide per ton, to different degrees of fineness. The pulps were agitated for 24- and 48-hour periods. The cyanide residues of Tests Nos. 1 and 2 were infrasized and the products assayed. The residue from Test No. 3 was treated on the Haultain superpanner and examined for the presence of free gold. None was found.

Screen tests showed the grinding as follows:

Mesh	Weight, per cent	
	Tests Nos. 1 and 2	Test No. 3
− 65+100	1·0	....
−100+150	6·8	1·4
−150+200	11·4	8·6
−200	80·8	90·0
	100·0	100·0

Summary of Cyanidation Results:

Test	Agitation, hours	Grind, per cent −200 mesh	Tailing assay, Au, oz./ton	Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
					NaCN	CaO	NaCN	CaO
1	24	80·8	0·14	97·0	1·00	0·25	0·80	8·2
2	48	80·8	0·10	97·9	0·90	0·22	0·90	8·2
3	48	90·0	0·07	98·5	0·96	0·22	1·08	8·6

The pregnant solutions from the three tests were analysed for reducing power and KCNS, with the following results:

Test No.	Reducing power, ml. $\frac{N}{10}$ $KMnO_4$/litre	KCNS, grm./litre
1	18	0·02
2	20	0·02
3	24	0·02

The above results indicate no appreciable fouling of the cyanide solutions during the grinding and agitation periods.

Infrasizer Analysis, Cyanide Tailing of Test No. 1:

Size, in microns	Weight, per cent	Assay, Au, oz./ton	Assay, units	Distribution of gold content, per cent
+40	39·31	0·29	11·3999	73·02
−40+28	15·95	0·09	1·4355	9·19
−28+20	12·04	0·06	0·7244	4·63
−20+14	8·26	0·06	0·4956	3·18
−14+10	5·84	0·06	0·3504	2·24
−10	18·60	0·065	1·2090	7·74
Totals	100·00	0·156	15·6128	100·00

Infrasizer Analysis, Cyanide Tailing of Test No. 2:

Size, in microns	Weight, per cent	Assay, Au, oz./ton	Assay, units	Distribution of gold content, per cent
+40	40·72	0·207	8·4290	69·39
−40+28	15·35	0·085	1·3048	10·74
−28+20	11·42	0·055	0·6281	5·17
−20+14	8·08	0·05	0·4040	3·33
−14+10	5·67	0·045	0·2552	2·10
−10	18·76	0·06	1·1256	9·27
Totals	100·00	0·09	12·1466	100·00

These analyses show that the ore requires fine grinding for successful treatment by cyanidation.

Summary of Results Obtained on Superpanner with Cyanide Tailing from Test No. 3:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	0·06*	100·0
Panner concentrate	0·10	0·675	1·1
Panner sand	49·70	0·08	71·7
Panner slime	50·20	0·03	27·2

*Calculated.

Under the microscope, the panner concentrate was seen to consist of graphitic carbon, pyrite, arsenopyrite, and magnetite. No gold was visible.

GRAVITY CONCENTRATION FOLLOWED BY BULK FLOTATION

Test No. 4

The ore at −14 mesh was ground in a ball mill to pass 53·0 per cent −200 mesh. The pulp was passed through a Denver jig and the coarse gold caught in the jig concentrate. The jig tailing was passed over a corduroy blanket. The combined jig and blanket concentrates were amalgamated and the amalgamation residue added to the blanket tailing. This product was ground in a ball mill to pass 76·3 per cent −200 mesh with 2 pounds of soda ash and 0·08 pound of Barrett No. 4 oil per ton and was transferred to a flotation machine. The pulp was conditioned with 1·0 pound of copper sulphate per ton and a concentrate was removed by the addition of 0·10 pound of amyl xanthate and 0·07 pound of pine oil per ton.

A screen test showed the grinding to be as follows:

Mesh	Weight, per cent	
	Initial grind	Flotation grind
− 48+ 65	1·5	....
− 65+100	10·0	2·1
−100+150	20·5	9·8
−150+200	15·0	11·8
−200	53·0	76·3
	100·0	100·0

Gravity Concentration Results, Test No. 4:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	4·67	100·00	....
Jig and blanket concentrates	5·78	56·34	69·70	17·3 : 1
Blanket tailing	94·22	1·50	30·30	....

The jig and blanket concentrates were amalgamated and the amalgamation residue was added to the blanket tailing. This product assayed 1·55 ounces of gold per ton and showed a recovery of 66·8 per cent of the total gold by amalgamation.

Flotation of the Blanket Tailing and Amalgamation Residue:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	1·55	100·0	....
Flotation concentrate	6·19	16·66	66·6	16·1 : 1
Flotation middling	8·54	3·27	18·0	11·7 : 1
Flotation tailing	85·27	0·28	15·4	....

Summary of Results:

	Per cent
Gold recovered by amalgamation	66·8
Gold recovered in rougher flotation concentrate	28·1
Overall recovery	94·9

GRAVITY CONCENTRATION FOLLOWED BY GRAPHITE FLOTATION AND CYANIDATION

Test No. 5

In order to determine whether the removal of the graphitic carbon by flotation, prior to cyanidation, would result in a lower cyanide residue, the ore was ground to pass 65·9 per cent −200 mesh and passed through a Denver jig and blanket as in the previous test. The concentrates were amalgamated and the amalgamation residue was added to the blanket tailing. This product was transferred to a Denver flotation machine and the bulk of the graphite was removed by flotation concentration, 0·05 pound of pine oil per ton being added during this treatment. The flotation tailing was split into two parts (A and B) and was reground in cyanide solution to pass 87·0 and 89·4 per cent −200 mesh respectively. In part B, 0·5 pound of lead nitrate was added during the grind. The pulps were agitated for 24 hours.

The initial grind prior to jig concentration was as follows:

Mesh	Weight, per cent
− 48+ 65	0·6
− 65+100	5·1
−100+150	14·2
−150+200	14·2
−200	65·9
	100·0

Results:

Jig and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	4·67	100·0	....
Jig and blanket concentrates	7·38	46·46	73·4	13·5 : 1
Tailing	92·62	1·34	26·6	....

After amalgamation the amalgamation residue was added to the jig tailing, this product assaying 1·43 ounces of gold per ton and giving a recovery of 69·4 per cent by amalgamation.

Flotation of Graphitic Concentrate:

Product	Weight, per cent	Assay		Distribution per cent		Ratio of concentration
		Au, oz./ton	C, per cent	Au	C	
Feed	100·00	1·43	0·51*	100·0	100·0	....
Graphite concentrate	10·29	8·60	3·95	56·7	77·2	9·7 : 1
Tailing	89·71	0·69	0·13	43·3	22·8	....

*Calculated.

Cyanidation of Flotation Tailings:

Test No.	Grind, per cent −200 mesh	Assay Au. oz./ton		Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
5A	87·0	0·69	0·06	91·3	0·92	0·20	0·76	7·6
5B	89·4	0·69	0·06	91·3	0·92	0·20	0·76	7·5

Screen Analysis:

Mesh	Weight, per cent	Assay, Au, oz./ton	Assay units	Distribution of gold, per cent
Cyanide Tailing of Test No. 5A:				
− 65+100	0·2	0·28	0·185	13·4
−100+150	2·8			
−150+200	10·0	0·115	1·150	18·1
−200	87·0	0·05	4·350	68·5
Totals	100·0	0·064	6·355	100·0
Cyanide Tailing of Test No. 5B:				
− 65+100	0·3	0·08	0·248	4·2
−100+150	2·8			
−150+200	7·5	0·155	1·162	19·8
−200	89·4	0·05	4·471	76·0
Totals	10·00	0·06	5·88	100·0

Another portion of the cyanide residue from Test No. 5B was treated on the superpanner with the following results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	0·10	100·0
Panner concentrate	0·20	0·85	1·6
Panner sand	46·90	0·195	87·9
Panner slime	52·90	0·02	10·5

The panner slime screened 99·2 per cent−325 mesh. No free gold was visible in the panner concentrate when placed under the microscope, sulphides, carbon, and iron oxide predominating.

Summary:

	Per cent
Gold recovered by amalgamation	69·4
Gold recovered in flotation concentrate	17·4
Gold extracted by cyanidation	12·0
Overall recovery	98·8

AMALGAMATION OF JIG AND BLANKET CONCENTRATES WITH CYANIDATION OF COMBINED TAILINGS

Test No. 6

A sample of the ore at −14 mesh was treated in a jig to remove coarse free gold. The primary jig tailing was riffled into four equal parts, which were ground 33, 39, 61, and 67 per cent through 200 mesh respectively. The samples of reground primary jig tailing were again treated in a jig, which overflowed on a corduroy blanket set at a slope of 2·5 inches per foot. Separate assay samples were taken from each of the blanket tailings produced from the different sizes of feed to the secondary jig. Samples from each of the blanket tailings were treated by cyanidation for periods of 24 and 48 hours.

All blanket concentrates and all hutch products were combined, reground 77 per cent through 200 mesh, and amalgamated with mercury in a jar mill for one hour. Both amalgam and amalgamation tailing were assayed for gold and a sample of the amalgamation tailing was also treated by cyanidation for 48 hours. In all the foregoing cyanidation tests solutions were maintained at approximately 1·0 pound of sodium cyanide per ton and from 0·20 to 0·40 pound of lime per ton.

The amalgam obtained from the combined concentrates was found to contain 1,130·52 milligrams of gold, corresponding to an assay value of 44·47 ounces per ton of combined concentrates. The amalgamation tailing from the combined concentrates assayed 5·06 ounces per ton in gold. These two assay values added together give an assay value of 49·53 ounces per ton in gold for the combined concentrates as produced. The average value of the blanket tailing, calculated from the weights and assays of the individual samples, was found to be 1·15 ounces per ton in gold.

Summary of Concentration Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent total gold	Extraction, per cent contained gold	Extraction, per cent total gold
Combined concentrates	8·3	49·53	79·58	....	....
Blanket tailing	91·7	1·15	20·42	....	....
Feed (cal.)	100·0	5·17	100·00	....	....
Concentrates amalgamated	....	5·06	....	89·78	71·45
Amalgamation tailing cyanided	....	0·185	....	9·85	7·84

Combined ratio of concentration: 12·05 : 1.

From the above summary it can be seen that 89·78+9·85=99·63 per cent of the gold contained in the combined concentrates is recoverable by the double treatment. This figure and the combined ratio of concentration were used to calculate final recoveries, as follows:

Example (Test No. 6A):

Product	Weight, per cent	Assay, Au, oz./ton	Units	Distribution, per cent total gold
Feed	100·0	5·17	517·000	100·00
Blanket tailing	91·7	1·18	108·207	20·93
Combined concentrates (by difference)	8·3	....	408·794	79·07

	Per cent
Extraction from concentrates—79·07 × 0·9963	78·78
Extraction from blanket tailing in 24 or 48 hours—$\frac{1·18-0·28}{1·18}$ × 20·93	15·96
Total extraction	94·74

Screen Analysis, 48-hour Cyanide Tailing from Test No. 6B:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold	
			Per cent content	Per cent total
+ 65	4·08	0·35	6·69	0·24
− 65+100	14·73	0·43	29·67	1·05
−100+150	28·03	0·30	39·38	1·40
−150+200	14·59	0·17	11·62	0·41
−200	38·57	0·07	12·64	0·45
Cyanide tailing	100·00	0·21	100·00	3·55

Screen Analysis of Cyanide Tailing from Amalgamated Concentrates:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold	
			Per cent content	Per cent total
+150	6·30	0·32	11·03	0·03
−150+200	16·56	0·33	29·89	0·09
−200	77·14	0·14	59·08	0·17
Cyanide tailing	100·00	0·18	100·00	0·29

Complete Summary of Results, Test No. 6:

Test No.	Grind, per cent −200 mesh	Blanket slope, inches per foot	Distribution of gold, per cent		Assay, Au, oz./ton		Period of agitation, hours	Total extraction, of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton ore	
			Combined concentrates	Blanket tailing	Blanket tailing	Blanket tailing cyanided			NaCN	CaO	NaCN	CaO
6A	33	2·5	79·07	20·93	1·18	0·28	24	94·74	1·0	0·25	0·32	6·35
						0·28	48	94·74	1·0	0·45	0·32	8·64
6B	39	2·5	78·36	21·64	1·22	0·24	24	95·45	1·0	0·20	0·34	6·22
						0·20	48	96·16	0·95	0·40	0·43	8·52
6C	61	2·5	79·78	20·22	1·14	0·16	24	96·86	1·0	0·20	0·29	6·73
						0·16	48	96·86	1·0	0·35	0·40	7·94
6D	67	2·5	80·84	19·16	1·08	0·13	24	97·39	1·0	0·15	0·22	6·04
						0·13	48	97·39	1·0	0·40	0·38	8·21

SUMMARY AND CONCLUSIONS

A concentration of 70 to 75 per cent of the gold can be made by a simple flow-sheet consisting of jigs at the ball mill discharge followed by blanket concentration of the classifier overflow. The combined concentrates representing a concentration of 70 to 80 per cent of the gold can be barrel-amalgamated to give an overall recovery of some 70 per cent by amalgamation. Flotation of the blanket tailing gave an additional recovery of some 17 per cent of the gold in a flotation concentrate. Jig and blanket concentration followed by cyanidation of the amalgamation residue and blanket tailing gave an overall extraction of 97 to 98 per cent of the gold. Straight cyanidation of the ore resulted in an extraction of 98·5 per cent and a cyanide residue of 0·07 ounce of gold per ton at a grind of 90 per cent −200 mesh.

The microscopic examination, screen analyses, panner and infrasizing tests all show that some 40 per cent of the gold is in a fine state of subdivision and that fine grinding is essential to free it from the gangue or sulphides and permits its extraction by cyanidation. The infrasizing tests on the cyanide residue show that some 70 per cent of the gold is in the larger-size material. In the superpanner tests the slime assayed 0·02 to 0·03 ounce of gold per ton and was 50 per cent by weight, yet carried only from 10 to 27 per cent of the gold in the feed.

It appears that the most suitable flow-sheet would be to grind in cyanide solution followed by jigs and blankets to remove the coarse gold, the resulting concentrates to be amalgamated. The blanket tailing and amalgamation residue should be reground in cyanide solution sufficiently fine to permit economic extraction of the gold remaining.

The ore on which this work has been done contains a considerable amount of coarse free gold, which is readily recoverable, and some very fine gold, which is more difficult to liberate and extract. Should the character of the ore change so that it contain less coarse gold, lower recoveries may be expected.

Ore Dressing and Metallurgical Investigation No 770

ORE AND FLOTATION CONCENTRATE FROM THE MONETA PORCUPINE MINE AT TIMMINS, ONTARIO

Shipment. A shipment of 9 sacks of ore, net weight approximately 1,000 pounds, and 3 cans of flotation concentrate, net weight approximately 100 pounds, was received on November 15, 1938.

The samples were submitted by W. E. Segsworth, President, Moneta Porcupine Mines, Limited, 67 Yonge Street, Toronto, Ontario, on the recommendation of C. W. Dowsett, Consulting Engineer.

Location of Property. This property is in Tisdale Township, Porcupine mining division, Ontario, and near the town of Timmins.

Characteristics of the Ore. A complete description of ore from this property will be found in Investigation No. 714, Bureau of Mines Report No. 785.

The flotation concentrate was chiefly pyrite and carried some gangue material consisting of silicates and carbonates. It was ground 72 per cent through 200 mesh and carried about 14 per cent of moisture.

Sampling and Assaying. The ore was submitted to find out whether it contained nickel and chromium, and after sampling it was assayed for these elements only and was reported as follows:

Nickel	Trace
Chromium	0·16 per cent

A sample cut from the concentrate was assayed, and was reported as follows:

Gold	3·10 oz./ton
Silver	0·60 "
Total iron	29·40 per cent
Ferrous iron	3·90 "
Sulphur	29·30 "
Carbon dioxide	5·60 "
Graphitic carbon	0·76 "
Sulphate (SO_3)	0·18 "
Insoluble (acid)	23·98 "
CaO (acid soluble)	4·26 "
MgO (acid soluble)	1·56 "

Experimental Tests

A series of small-scale tests of the sample of concentrate was made to determine the best conditions for its treatment by cyanidation, and if possible to find a reason for the erratic tailing assays being obtained at the mill.

The results of these tests show that—

(1) The concentrate must be ground extremely fine in order to expose the fine particles of gold to attack by the cyanide solution.

(2) High lime, i.e. more lime than is necessary for mere protective alkalinity or settling, is beneficial, and this is believed to be due to the presence of carbonates in the concentrate.

(3) Water washing of the concentrate before it comes in contact with the cyanide solution was also found beneficial. This may be due to the presence in the concentrate of flotation reagents, such as xanthate, or to the presence of oxidation products, the concentrate having been some time in transit. As the series of tests progressed it was found that lime and cyanide consumption went higher and higher if the concentrate were not washed before coming in contact with the cyanide solution.

The following tests, typical of a number conducted, are described in detail:

CYANIDATION OF CONCENTRATE WITHOUT WASHING

Tests Nos. 1, 2, and 3

Samples of the concentrate as received were ground in cyanide solution in ball mills for 2 hours. Infrasizing tests on the products showed them to be about 60 per cent finer than 10 microns.

The pulps were agitated in cyanide solution for periods of 6, 16, and 24 hours at 2·5 : 1 dilution. The lime in the cyanide solution was kept down to about 0·10 pound of lime per ton. The cyanide tailings were assayed for gold.

Summary of Tests Nos. 1, 2, and 3:

Feed: gold, 3·10 oz./ton.

Test No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
				NaCN	CaO	NaCN	CaO
1	6	0·40	87·10	0·76	0·12	5·36	6·82
2	16	0·233	92·48	0·96	0·14	7·21	9·80
3	24	0·215	93·06	0·96	0·14	7·73	10·88

An infrasizer analysis of the tailing from Test No. 2 shows a gradual drop in the gold and sulphur assays from the coarsest to the finest fraction. At the same time the ratio of sulphur to gold increases steadily from the coarsest to the finest fraction. This shows the need for fine grinding of the sulphides to expose the gold to attack by the cyanide solution.

Infrasizer Analysis, Tailing of Test No. 2:

Size of particle, in microns	Weight, per cent	Assay		Per cent content	
		Au, oz./ton	S, per cent	Au	S
+40	1·48	1·28	44·00	8·11	2·29
−40+28	5·24	0·58	43·70	13·02	8·06
−28+20	8·73	0·39	41·44	14·58	12·73
−20+14	11·24	0·28	37·34	13·48	14·77
−14+10	11·52	0·225	32·56	11·10	13·20
−10	61·79	0·15	22·50	39·71	48·95
Average tailing	100·00	0·233	28·41	100·00	100·00

CYANIDATION OF WASHED CONCENTRATE

Tests Nos. 4, 5, and 6

Samples of the concentrate were stirred in water, then filtered and washed. The washed concentrate was ground in cyanide solution for two hours, made up to 2·5 : 1 dilution, and agitated for periods of 6, 16, and 24 hours. Alkalinity of the solutions was kept low as in Tests Nos. 1, 2 and 3.

Summary of Tests Nos. 4, 5, and 6:

Feed: gold, 3·10 oz./ton.

Test No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
				NaCN	CaO	NaCN	CaO
4	6	0·29	90·65	0·98	0·06	5·86	7·63
5	16	0·21	93·23	0·68	0·12	7·35	10·94
6	24	0·19	93·87	0·84	0·08	8·13	12·00

Extractions are somewhat higher than in Tests Nos. 1, 2, and 3 and at the same time reagent consumption is up a little.

COMPARING HIGH AND LOW LIME IN CYANIDATION

Tests Nos. 7 and 8

Samples of the concentrate were water-washed and ground in cyanide solution for two hours. The pulps were agitated for 24 hours at 2·5 : 1 dilution with a very low lime content in one of them and a comparatively high one in the other.

Owing to a difference in the condition of the grinding jars used, one batch of pulp was ground noticeably finer than the other but still the coarser was agitated with high lime and gave a slightly lower tailing assay than the finer.

Summary of Tests Nos. 7 and 8:

Feed: gold, 3·10 oz./ton.

Test No.	Tailing assay, Au, oz./ton	Extraction, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
			NaCN	CaO	NaCN	CaO
7	0·206	93·35	0·72	0·06	8·46	12·02
8	0·199	93·58	0·76	0·54	6·75	33·70

Screen Analysis, Test No. 7:

Size of particles, in microns	Weight, per cent	Assay		Per cent content	
		Au, oz./ton	S, per cent	Au	S
+40	1·25	0·235	32·37	1·43	1·46
−40+28	3·75	0·33	40·63	6·01	5·51
−28+20	7·27	0·375	42·31	13·24	11·13
−20+14	10·74	0·275	38·88	14·35	15·11
−14+10	12·22	0·22	33·21	13·06	14·69
−10	64·77	0·165	22·22	51·91	52·09
Average tailing	100·00	0·206	27·63	100·00	100·00

Screen Analysis, Test No. 8:

Size of particles, in microns	Weight, per cent	Assay		Per cent content	
		Au, oz./ton	S, per cent	Au	S
+40	2·74	0·34	32·30	4·67	3·23
−40+28	5·80	0·385	38·82	11·20	8·23
−28+20	9·27	0·36	40·55	16·73	13·74
−20+14	12·24	0·25	37·24	15·34	16·66
−14+10	13·14	0·185	31·60	12·19	15·17
−10	56·81	0·14	20·69	39·87	42·97
Average tailing	100·00	0·199	27·35	100·00	100·00

GRINDING IN CYANIDE SOLUTION, FILTERING, AND REPULPING IN FRESH SOLUTION

Test No. 9

A sample of the concentrate was water-washed and ground in cyanide solution for two hours. The pulp was filtered and repulped in fresh cyanide solution and agitated for 24 hours.

A large quantity of lime was added in order to keep the lime in solution comparatively high but, owing possibly to oxidation of the sample, the lime was consumed very rapidly and it could not be maintained higher than 0·25 pound per ton for any length of time.

The cyanide tailing was infrasized and the fractions assayed for gold and sulphur.

Infrasizer Analysis, Test No. 9:
Feed: gold, 3·10 oz./ton

Size of particle, in microns	Weight, per cent	Assay		Per cent content	
		Au, oz./ton	S, per cent	Au	S
+40	1·73	0·73	30·54	6·33	1·90
−40+28	3·86	0·38	38·08	7·35	5·28
−28+20	7·10	0·38	42·48	13·52	10·83
−20+14	10·84	0·28	38·91	15·21	15·14
−14+10	13·05	0·20	34·00	13·08	15·93
−10	63·42	0·14	22·36	44·51	50·92
Average tailing	100·00	0·1995	27·85	100·00	100·00

Extraction.. 93·56 per cent

Reagents:

	NaCN	CaO
Final titration, lb./ton solution	0·52	0·23
Consumed, lb./ton concentrate	5·39	31·04

In Tests Nos. 8 and 9 the fractions finer than 10 microns have been reduced to the lowest gold content obtained in any of the tests and the comparatively higher lime content of the solutions is believed to be responsible for this.

From the analysis of the concentrate it will be found that there is not enough lime and magnesia to account for all the carbon dioxide present. It is, therefore, probable that some of the carbonates are in the form of $FeCO_3$ or some similar compound, which in the absence of sufficient lime would form ferrocyanides, which in turn would absorb oxygen from the solution and cause a drop in extraction.

Repulping in fresh cyanide solution in Test No. 9 did not appear to help the extraction as compared with Test No. 8. In order to determine whether any further extraction was possible from this material all the fractions finer than 10 microns were collected from the tests conducted and agitated for 24 hours in cyanide solution containing 3·0 pounds of sodium cyanide per ton and almost 1·0 pound of lime per ton.

The assay of this composite fraction-sample was calculated to be 0·149 ounce per ton in gold and after the extra period of agitation it was reduced to 0·135 ounce per ton in gold. This checks rather closely the minimum tailing obtained in Tests Nos. 8 and 9.

Five individual assays were made on this final tailing and each bead weighed exactly 0·135 milligrams and the five beads together weighed 0·675 milligrams. This would indicate that the gold in the tailing is very uniformly distributed and must, therefore, be very fine, as the whole sample was finer than 10 microns.

Therefore, this would appear to be the minimum tailing possible from this concentrate at any grind within economic limits.

No gold is visible under the microscope in polished sections of this material. This is confirmatory evidence of the extreme fineness of the gold in the ore.

The sample of ore submitted was assayed for nickel and chromium, both of which were found in small amounts.

A cyanidation test was conducted on the ore to see if either the nickel or chromium were soluble.

No chromium was found in the pregnant solution but a small amount of nickel, 0·002 gramme per litre, was present. This would indicate that the chromium was present as an insoluble compound, perhaps a silicate, whereas the nickel is in some soluble form. The chromium determination was made spectrographically on the evaporated residue from a sample of solution.

CONCLUSIONS

The results of tests conducted on the sample of flotation concentrate show that—

(1) The concentrate should be ground as fine as it is economically possible to grind it;

(2) A liberal quantity of lime should be used to neutralize the carbonates present, particularly ferrous carbonate; and

(3) Water-washing of the concentrate before it comes in contact with the cyanide solution is beneficial. This may be due to the presence of xanthates in the concentrate which slow up dissolution of the gold.

The washing of the concentrate could best be done in a washing thickener placed ahead of the filter.

The concentrate should then be ground in cyanide solution as at present and the lime should be kept reasonably high in the grinding circuit to prevent the formation of ferrocyanides by the ferrous carbonate.

The grinding can be done more efficiently in practice in a well designed mill-classifier circuit than it can be done in batch grinding in small-scale tests with the possibility of slightly better results.

Although the small-scale tests did not show it, in view of the high-grade of the product being treated it would be wise to use a large volume of solution in the primary thickener to wash out the high gold solution coming from the grinding circuit, and so to keep it out of contact with the pulp in the agitators.

The greater part of the gold is very readily soluble and will be extracted in the first few hours of treatment. The solution containing this gold should be removed as soon and as completely as possible and precipitated.

Keeping these points in mind, the best possible extraction of the gold should be obtained, the limiting factor being that of very fine gold not exposed to attack by the cyanide solution.

Ore Dressing and Metallurgical Investigation No. 771

GOLD ORE FROM THE ATHONA MINES (1937), LIMITED, GOLDFIELDS, SASKATCHEWAN

Shipment. Ten samples of gold ore, having a combined weight of 1,178 pounds, were received on January 31, 1939, from the Athona Mines (1937), Limited, Goldfields, Saskatchewan. The shipment was submitted by Norman W. Byrne, Resident Engineer.

In November, 1938, nine special samples were received for microscopic examination from J. J. Byrne, President, Athona Mines (1937), Limited, 80 King Street West, Toronto, Ontario.

Characteristics of the Ore. The nine special specimens were examined without the aid of a glass and also under the binocular microscope. Polished sections were then prepared and examined under the reflecting microscope. The following is based upon all observations.

Table I gives a list of the samples, with the information supplied by Athona Mines (1937), Limited:

TABLE I

List of Samples

Sample	Number of polished sections	Description supplied by the Athona Mines (1937), Limited.
1 and 2	6	Specimens similar; "H" vein mineralization. Higher grade ore zones.
3	2	Quartz-chalcopyrite mineralization; low-grade ore zones. Chalcopyrite makes up only a very very small percentage of sulphides.
4	2	Quartz-galena mineralization; low-grade ore zones.
5	2	Quartz-sphalerite in red granite; low-grade ore zones.
6	2	Quartz-pyrite mineralization with some galena in gabbro-granite contact zone material; low-grade ore zones.
7	None	Specimen same as No. 6.
8 and 9	2	Quartz-pyrite mineralization in red granite; low-grade ore zones.

General Description. The most noticeable constituent of the samples is a rather coarse-textured, light greyish-white quartz. Some specimens contain pink granitic material and others, a fine-textured, grey rock, which is probably a somewhat altered phase of the gabbro referred to by Athona Mines (1937), Limited in the list of specimens. The quartz has invaded the granite along veinlets the boundaries of which are somewhat hazy, being gradational over a distance of a millimetre or more. Some of the granite shows the effect of high-temperature silicification. The grey gabbroic material also shows evidence of high-temperature alteration by the presence within it of metacrysts of feldspathic material up to a centimetre or more in size.

The sulphides are confined chiefly to the quartz, but rare pyrite occurs in the gabbroic material. Sphalerite and galena are most abundant, occurring as irregular masses, stringers and grains in the quartz, usually accompanied by a little carbonate. In places the masses send out into the quartz short tongues that taper rapidly and die out. A very small quantity of pyrite is disseminated in the quartz and within the sulphide masses; where it occurs in quartz it is euhedral and shows cubic forms, but where it occurs in the sulphide masses it shows the effect of active corrosion and replacement, chiefly by sphalerite. Chalcopyrite is comparatively rare, and is associated with sphalerite and with pyrite. Sphalerite and galena show a distinct tendency to occur together, with the sphalerite usually predominating. Sphalerite masses commonly enclose irregular grains and stringers of galena, and in places galena has veined the sphalerite. The latter mineral contains inclusions of pyrrhotite and chalcopyrite. Native gold occurs chiefly in the quartz along stringers that also carry a little carbonate and are usually continuous with stringers of the sulphides. Lesser quantities of gold occur with sphalerite and galena, and in rare cases gold is associated with chalcopyrite.

Paragenesis. The order of deposition of the minerals is clearly exhibited in the polished sections. Obviously, the quartz has penetrated and invaded the granitic rock, with a certain amount of high-temperature replacement and silicification of the host. Apparently, pyrite was disseminated essentially at this time. Later incipient fracturing, chiefly in the quartz, provided channels for the solutions bearing the later sulphides and the native gold; the solutions at this phase must have carried also a little carbonate. Sphalerite was the first sulphide to start to deposit during this period of mineralization, and a little pyrrhotite and chalcopyrite accompanied it as tiny inclusions, often arranged parallel to the crystallographic directions of the sphalerite, a structure usually attributed to unmixing. Galena was the last sulphide to be deposited. Native gold is definitely associated with the later sulphides, hence its deposition must have taken place during the later sulphide stage of mineralization and not during the quartz-pyrite stage. Two inferences may be drawn from this fact: (1) If the induction that gold deposition accompanied only the deposition of the later sulphides is correct, the earlier pyrite would not be expected to carry gold and the ore should not prove to be refractory; (2) the later sulphides, sphalerite, galena, etc., may be taken as indicators of ore in the mine.

Table II, below, represents graphically what is thought to be the paragenesis of the minerals:

TABLE II

Paragenesis of the Minerals

Minerals	Decreasing age and probably decreasing temperature		
	Granite (host)	Quartz pyrite stage	Later sulphide stage
Granite	——		
Quartz		————	
Pyrite		? ———	—
Sphalerite			——
Pyrrhotite			– – – –
Chalcopyrite			– —
Galena			– – ——
Mineral X			– ? –
Native gold			— – ——– –
Carbonate			– ———– –

Detailed Description of Samples

Samples 1 and 2. Samples 1 and 2 are similar in character. The gangue is quartz. In this gangue occur masses and irregular stringers of sphalerite and galena and occasional disseminated grains of pyrite. The sphalerite masses contain irregular grains and veinlets of galena, and rare tiny dots of chalcopyrite and pyrrhotite. The sphalerite-galena masses send out short tongues which taper out to points in the quartz. The pyrite that occurs in the quartz is euhedral, cubic forms being noticeable; the pyrite enclosed in the sulphides, notably sphalerite, has been corroded and replaced to some extent by this mineral. Native gold is present as irregular grains (1) in the quartz, and (2) in the sulphide masses associated with sphalerite and galena.

Sample 3. The gangue is quartz. The sulphide mineralization is commonly irregular stringers and small grains of chalcopyrite which is associated with some sphalerite and galena. The chalcopyrite contains small, irregular grains of Mineral X, the tests on which are given in Table III.

Native gold occurs as irregular grains (1) in quartz, and (2), rarely, in chalcopyrite.

TABLE III

Tests on Mineral X from Sample 3

Colour	Faint bluish white, very similar to galena.
Hardness	B; softer than chalcopyrite. No cleavage noticeable.
Crossed Nicols	Rather weakly but distinctly anisotropic; two positions of extinction, light to dark grey.
Etch Tests	HNO_3—after a slight delay mineral tarnishes differentially iridescent to grey without effervescence. A structure strongly suggestive of graphic structure is brought out.
	HCl—differentially practically negative to grey, with development of graphic structure.
	KCN—negative.
	$FeCl_3$—differentially iridescent with development of blues and browns; graphic structure is brought out.
	KOH—negative.
	$HgCl_2$—negative.
Microchemical Tests	Owing to the small quantity of material available and to the fact that any sample free from chalcopyrite could not be obtained, microchemical analysis proved unsatisfactory. A test for lead was obtained, and the presence of bismuth was inconclusively indicated during a microchemical analysis.
Identification	The mineral most closely conforming to the above tests is *Matildite* ($(Ag_2,Pb)S.Bi_2S_3$), but identification is tentative only.

Sample 4. The gangue is chiefly quartz with a lesser quantity of pink granitic material. Stringers of sphalerite and galena, usually associated, occur in the quartz near the quartz-granite boundaries, and tongues of quartz carrying these sulphides extend into the granite. Quartz which contains gold grains also penetrates into the granite. Grains of native gold occur in the quartz, usually alone but occasionally associated with galena. Pyrite is rare.

Sample 5. The gangue is chiefly pink granitic material with veinlets of quartz. The boundaries between quartz and granite are gradational over a millimetre or more, giving the quartz veinlets a somewhat hazy appearance. Coarse, massive sphalerite is present in the quartz; chalcopyrite and galena are comparatively rare. Pyrite occurs as euhedral crystals in the quartz and as corroded grains in sphalerite. Native gold occurs as irregular grains (1) in quartz, (2) along quartz-sphalerite boundaries, and (3) associated with galena.

Samples 6 and 7. These are similar, and sections were prepared from Sample 6 only. The gangue is chiefly a fine-textured grey rock, possibly an altered phase of the gabbro referred to in the correspondence from Athona Mines (1937), Limited, with minor quartz. Metacrysts of feldspathic material, up to a centimetre or more in size, have been developed in the grey rock, and pyrite has also been disseminated in it. Irregular stringers of sphalerite and galena, usually associated, occur in the quartz. The sphalerite contains small inclusions of pyrrhotite and chalcopyrite. No native gold is visible.

Samples 8 and 9. These consist chiefly of pink granitic material with a small quantity of quartz. A considerable amount of pyrite occurs in the quartz. The mineral is somewhat fractured and brecciated, and contains occasional inclusions of chalcopyrite. No native gold is visible.

Grain Size of the Native Gold. Tables IV, V, VI, and VII give grain analyses of the visible native gold in the individual specimens, and Table VIII gives a composite grain analysis of the gold in all samples. The tables are based wholly on measurements made on the gold occurring in polished sections.

TABLE IV

Grain Analysis of the Gold in Samples 1 and 2

Mesh	Microns	Gold in quartz, per cent	Gold in sphalerite, per cent		Totals, per cent
			Alone	Associated with galena	
+ 65	+208	35·8	....	....	35·8
− 65+ 100	+147	....	....	....	....
− 100+ 150	+104	....	....	....	....
− 150+ 200	+ 74	6·4	....	....	6·4
− 200+ 280	+ 52	5·1	....	....	5·1
− 280+ 400	+ 37	....	....	....	....
− 400+ 560	+ 26	3·0	2·5	8·6	14·1
− 560+ 800	+ 19	....	14·5	7·7	22·2
− 800+1100	+ 13	1·3	8·2	3·3	12·8
−1100+1600	+ 9	....	2·9	....	2·9
−1600	....	....	0·7	....	0·7
		51·6	28·8	19·6	100·0

TABLE V

Grain Analysis of Gold in Sample 3

Mesh	Microns	Gold in quartz, per cent	Gold in chalco-pyrite, per cent	Totals, per cent
+ 65	+208	39·2	....	39·2
− 65+ 100	+147	7·8	....	7·8
− 100+ 150	+104	24·5	....	24·5
− 150+ 200	+ 74	8·5	....	8·5
− 200+ 280	+ 52	6·2	....	6·2
− 280+ 400	+ 37	3·6	....	3·6
− 400+ 560	+ 26	2·0	....	2·0
− 560+ 800	+ 19	3·3	1·6	4·9
− 800+1100	+ 13	1·0	0·8	1·8
−1100+1600	+ 9	1·5	....	1·5
−1600	....	....	....	....
		97·6	2·4	100·0

TABLE VI

Grain Analysis of Gold in Sample 4

Mesh	Microns	Gold in quartz, per cent	Gold associated with galena, per cent	Totals, per cent
+ 65	+208	....	....	....
− 65+ 100	+147	9·3	....	9·3
− 100+ 150	+104	6·5	....	6·5
− 150+ 200	+ 74	13·9	....	13·9
− 200+ 280	+ 52	10·1	....	10·1
− 280+ 400	+ 37	14·7	....	14·7
− 400+ 560	+ 26	18·2	....	18·2
− 560+ 800	+ 19	15·3	1·3	16·6
− 800+1100	+ 13	6·5	1·1	7·6
−1100+1600	+ 9	2·7	....	2·7
−1600	....	0·4	....	0·4
		97·6	2·4	100·0

TABLE VII

Grain Size of the Gold in Sample 5

Mesh	Microns	Gold in quartz, per cent	Gold associated with sphalerite and galena, per cent	Totals, per cent
+ 65	+208	....	....	....
− 65+ 100	+147	....	....	....
− 100+ 150	+104	21·7	29·0	50·7
− 150+ 200	+ 74	....	20·3	20·3
− 200+ 280	+ 52	....	14·5	14·5
− 280+ 400	+ 37	....	10·1	10·1
− 400+ 560	+ 26	....	4·4	4·4
− 560+ 800	+ 19	....	....	....
− 800+1100	+ 13	....	....	....
−1100+1600	+ 9	....	....	....
−1600	....	....	....	....
		21·7	78·3	100·0

TABLE VIII

Composite Grain Analysis of the Gold in All Sections

Mesh	Microns	Gold in quartz, per cent	Gold associated with sphalerite and/or galena, per cent	Gold in chalcopyrite, per cent	Totals, per cent
+ 65	+208	21·0	....	....	21·0
− 65+ 100	+147	6·0	....	....	6·0
− 100+ 150	+104	10·2	....	....	10·2
− 150+ 200	+ 74	11·0	2·1	....	13·1
− 200+ 280	+ 52	7·0	1·5	....	8·5
− 280+ 400	+ 37	6·7	1·0	....	7·7
− 400+ 560	+ 26	8·2	3·4	....	11·6
− 560+ 800	+ 19	6·8	5·6	0·5	12·9
− 800+1100	+ 13	3·1	3·2	0·3	6·6
−1100+1600	+ 9	1·5	0·7	....	2·2
−1600	....	0·1	0·1	....	0·2
Totals		81·6	17·6	0·8	100·0

As will be obvious from a glance at Table VIII, the figures for the grain sizes of the gold are somewhat erratic in the larger sizes. This is common where coarse gold is present. In the grain sizes below 19 microns (equivalent of 800 mesh) the percentages decrease rapidly and only 0·2 per cent of the gold measured is smaller than 9 microns (equivalent of 1600 mesh).

Sampling and Assaying. The ten samples received on January 31, 1939, were sampled separately and the assays are as follows:

Sample	Weight, pounds	Assay, oz./ton	
		Gold	Silver
1	113	0·065	0·08
2	118	0·055	0·045
3	108	0·15	0·06
4	130	0·393	0·06
5	123	0·875	0·54
6	102	0·13	0·16
7	139	0·08	0·07
8	114	0·32	0·18
9	129	0·17	0·06
10	102	0·145	0·05
Total	1178		

For purposes of testing, the above samples were combined in three groups, as follows:

Group I: Samples 1, 2, and 9.

Group II: Samples 3, 6, 7, and 10.

Group III: Fifty pounds of Sample 5 and the whole of Sample 8.

Sample 4 was tested separately.

Analyses of Groups:

Group	Assay, oz./ton		Assay, per cent			
	Au	Ag	Fe	Cu	As	S
I..................	0·09	0·10	1·60	0	0	0·32
II..................	0·11	0·09	1·65	0	0	0·30
III..................	0·48	0·35	1·34	Trace	0	0·29

Purpose of Investigation. The investigation had the object of determining an economic method of treating the ore.

On account of the location of the property and the low-grade nature of the ore it would be advisable to treat a comparatively large tonnage with a minimum of equipment and at a low operating cost.

Results of Investigation. By straight cyanidation of the ore of the three groups the tailing was, Au, 0·005 ounce per ton.

Barrel amalgamation of Sample 4 (Au, 0·393 ounce per ton) gave a gold recovery of 87 per cent.

The tailings from flotation tests were as follows:

Group II (Au, 0·11 oz./ton).. 0·005 oz./ton
Group III (Au, 0·48 oz./ton).. 0·025 " and
Groups II and III, mixed, 1 : 1 (Au, 0·295 oz./ton).................. 0·015 "

By flotation, classification of the flotation tailing for removal of fine, and cyanidation of sand and concentrate the recovery was 97 per cent.

By primary cyanidation, tabling, and secondary cyanidation of reground table sand the recovery was around 97 per cent.

The results by jigs and blankets indicated a recovery of 71·5 per cent by amalgamation on a mixture of Groups II and III ores.

Settling tests showed that the ore has a satisfactory rate of settling.

The results of the tests follow in detail:

Experimental Results

CYANIDATION

Tests Nos. 1 to 6

A series of standard cyanidation tests was carried out on the ore to determine its amenability to cyanide treatment.

Samples were given a moderate grind in cyanide and agitated for 24- and 48-hour periods in solution of strength of 1 pound of sodium cyanide per ton and a pulp dilution of 2 : 1.

Results of Tests Nos. 1 to 6:

Ore sample	Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
			Feed	Tailing		NaCN	CaO	NaCN	CaO
Group I.......	1	24	0·09	0·005	94·4	1·0	0·35	0·10	3·30
	2	48	0·09	0·005	94·4	1·0	0·35	0·10	3·30
Group II......	3	24	0·11	0·005	95·4	1·0	0·35	0·20	3·30
	4	48	0·11	0·005	95·4	1·0	0·35	0·20	3·30
Group III.....	5	24	0·48	0·01	97·9	1·0	0·30	0·40	3·40
	6	48	0·48	0·005	98·9	0·96	0·30	0·48	3·40

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

Mesh	Weight, per cent		
	Group I	Group II	Group III
+ 65	1·3	0·2	2·1
− 65+100	9·6	13·7	10·4
−100+150	17·9	19·2	18·8
−150+200	15·3	15·1	14·8
−200	55·9	51·8	53·9
	100·0	100·0	100·0

The results show that the ore is amenable to cyanidation at moderate grinding.

AMALGAMATION

Test No. 7 (Sample 4)

A barrel-amalgamation test was carried out on Sample 4 ore. The grinding was to a fineness of 61·2 per cent −200 mesh.

Assay feed	0·393 Au, oz./ton
Assay tailing	0·05 Au, oz./ton
Recovery	87·2 per cent

The gold in this sample is largely free-milling, and no further tests were carried out on it. A comparison of the microscopic examination of the various samples and the results obtained in the above test indicates a large proportion of free-milling gold in the higher grade ore samples.

FLOTATION, CLASSIFICATION OF TAILING, AND CYANIDATION

Group II Ore

The object of these tests was to make a rough concentrate by flotation, and then classify the flotation tailing into sand and fine products. Classification was carried out on laboratory Wilfley tables. The fine was sufficiently low in gold to be discarded. The combined flotation concentrate and sand were treated by cyanidation.

Test No. 8

Flotation:

Ore charge	4,000 grm.
Soda ash	2·0 lb./ton
Aerofloat No. 25	0·07 "
Potassium amyl xanthate	0·10 "
Pine oil	0·031 "

Results of Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·11	100·0	....
Flotation concentrate	1·44	6·27	82·1	69·4 : 1
Flotation tailing	98·56	0·02	17·9	....

Classification of Flotation Tailing on Table:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	0·02	100·0	....
Sand	32·7	0·03	49·3	3·06 : 1
Fine	67·3	0·015	50·7	....

The sand was combined with the flotation concentrate and cyanided for 24 hours.

Results:

Feed: gold, 0·387 oz./ton.

Product	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore		Pulp dilution
			NaCN	CaO	NaCN	CaO	
Cyanide tailing	0·015	96·1	1·04	0·46	0·32	3·08	2 : 1

Overall recovery .... 87·4 per cent

The grinding is indicated by the following screen tests:

Mesh	Weight, per cent		
	Flotation tailing	Sand	Cyanide tailing
+ 65	0·3	1·6	1·2
− 65+100	4·1	12·7	11·3
−100+150	13·3	33·1	25·9
−150+200	14·2	24·7	26·6
−200	68·1	27·9	35·0
Totals	100·0	100·0	100·0

A screen analysis of the fine indicates the distribution of the gold at the different sizes of grinding to be as follows:

Mesh	Weight, per cent	Assay, Au, oz./ton	Units	Distribution of gold, per cent
+100	1·0	0·02	0·1620	10·6
−100+150	8·1			
−150+200	11·8	0·02	0·1770	11·6
−200	79·1	0·015	1·1865	77·8
Totals	100·0	0·015	1·5255	100·0

Test No. 9

This was a flotation test only and less soda ash was used, and a small amount of copper sulphate was added to activate the sphalerite, which carries gold.

The results indicated a decided improvement in flotation.

Reagents:

Soda ash	0·5 lb./ton
Aerofloat No. 25	0·07 "
Potassium amyl xanthate	0·1 "
Copper sulphate	0·5 "
Pine oil	0·09 "
Conditioning time	5 minutes
Flotation time	5 "
Grind	65·6 per cent −200 mesh

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·09*	100·00	....
Concentrate	2·06	4·24	94·69	48·5 : 1
Tailing	97·94	0·005	5·31	....

*Calculated from products.

Test No. 10

Duplicate of Test No. 9.

Results:

Product	Weight, per cent	Assay				Distribution of gold, per cent	Ratio of concentration
		Au, oz./ton	Per cent				
			Cu	Zn	S		
Feed	100·0	0·10*	....	....	....	100·0	....
Concentrate	3·1	3·02	0·14	0·18	....	92·8	33·1 : 1
Tailing	96·9	0·0075	....	....	0·04	7·2	....

*Calculated from products.

The use of low soda ash and a small amount of copper sulphate to float the sphalerite indicates a marked improvement in flotation results.

Group III Ore

The following tests were carried out similarly to those on Group II ore.

Test No. 11

Flotation:

Ore charge .... 4,000 grm.

Soda ash .... 1·0 lb./ton
Aerofloat No. 25 .... 0·07 "
Potassium amyl xanthate .... 0·1 "
Pine oil .... 0·062 "

Grind .... 68 per cent −200 mesh

Results:

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·48	100·00	....
Flotation concentrate	1·33	33·86	93·83	75·2 : 1
Flotation tailing	98·67	0·03	6·17	....

Classification of Flotation Tailing on Table:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·04	100·00	....
Sand	33·62	0·05	41·98	2·97 : 1
Slime	66·38	0·035	58·02	....

The sand was combined with the flotation concentrate and cyanided for 24 hours.

Feed: gold, 1·45 oz./ton.

Product	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore		Pulp dilution
			NaCN	CaO	NaCN	CaO	
Cyanide tailing..............	0·06	95·86	0·96	0·34	0·48	3·32	2 : 1

Overall recovery.. 92·43 per cent

Screen Test on Sand:

Mesh	Weight, per cent
+ 65	1·6
− 65+100	13·3
−100+150	26·6
−150+200	29·7
−200	28·8
Total	100·0

Screen Analysis of Fine:

Mesh	Weight, per cent	Assay, Au, oz./ton	Units	Distribution, per cent
+100	1·1	0·54	0·594	16·39
−100+150	5·0	0·115	0·575	15·87
−150+200	14·4	0·06	0·864	23·85
−200	79·5	0·02	1·590	43·89
Totals	100·0	0·036	3·623	100·00

These tests were carried out with one stage of grinding. From the results of the cyanidation, it is evident that regrinding of the mixed sand and concentrate is necessary.

Test No. 12

Flotation test only.

Reagent additions the same as in tests on Group II ore.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	0·42*	100·00	....
Concentrate	2·0	19·56	94·11	50 : 1
Tailing	98·0	0·025	5·89	....

*Calculated.

Test No. 13

Duplicate of Test No. 12.

Results:

Product	Weight, per cent	Assay				Distribution of gold, per cent	Ratio of concentration
		Au, oz./ton	Per cent				
			Cu	Zn	S		
Feed	100·0	0·40*	....	....	....	100·0	....
Concentrate	2·2	16·94	0·21	5·64	....	93·8	45·5 : 1
Tailing	97·8	0·025	....	....	0·03	6·2	....

*Calculated.

A portion of the tailing from this test was panned on the Haultain superpanner. Three small grains of gold (weight, 0·59 milligram) were noted. A very small amount of sulphide was seen, all of which was free from gangue.

The assays of the products and the distribution of gold and sulphur are as follows:

Test No. 13:

Product	Weight, per cent	Assay, Au, oz./ton	Assay, S, per cent	Distribution, per cent	
				Gold	Sulphur
Feed	100·0	0·025	0·04	100·0	100·0
Sand	58·6	0·03	0·03	70·3	44·5
Fine	41·4	0·017	0·05	29·7	55·5

These results indicate that the gold in the flotation tailing is associated with the gangue as well as with the sulphide minerals.

Mixture of Ores of Groups II and III

Test No. 14

The procedure was the same as in Tests Nos. 8 and 9. The charge was composed of a mixture of the ores of Groups II and III in equal amounts, which gave a calculated feed of Au, 0·295 ounce per ton.

The primary grind was 66·1 per cent −200 mesh. The reagents used were as follows:

Soda ash	0·5 lb./ton
Aerofloat No. 25	0·07 "
Potassium amyl xanthate	0·1 "
Copper sulphate	0·5 "
Pine oil	0·09 "

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Flotation:				
Feed	100·0	0·295	100·0	....
Concentrate	2·1	13·35	95·0	47·6 : 1
Tailing	97·9	0·015	5·0	....
Classification of Flotation Tailing on Table:				
Feed	100·00	0·015	100·0	....
Sand	31·97	0·025	54·6	3·13 : 1
Fine	68·03	0·01	45·4	....

The sand product was combined with the flotation concentrate and reground. Two lots of this reground product were cyanided for 24 and 48 hours, respectively, in a solution of 1·0 pound of sodium cyanide per ton strength, at a pulp dilution of 2 : 1.

Cyanidation Results:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
24	1·03	0·04	96·1	1·0	0·25	0·80	1·50
48	1·03	0·01	99·0	0·9	0·25	1·00	1·50

The distribution of the gold in the fine is indicated by the following screen analysis:

	Weight, per cent	Assay, oz./ton	Units	Distribution of gold, per cent
+100	3·30	0·027	0·0890	8·43
−100+150	15·11	0·01	0·1511	14·31
−150+200	14·17	0·01	0·1417	13·42
−200	67·42	0·01	0·6742	63·84
Totals	100·00	0·01	1·0560	100·00

The primary grind and regrind of sand and flotation concentrate are indicated by the following screen tests:

Mesh	Weight, per cent	
	Primary grind	Regrind
+ 65	0·4	....
− 65+100	4·6	0·4
−100+150	13·8	5·2
−150+200	15·1	10·7
−200	66·1	83·7
Totals	100·0	100·0

Summary of Test No. 14:

	Per cent
Gold recovered in flotation concentrate	95·00
Gold recovered in sand	2·73
Gold extraction by cyanidation of sand and concentrate	99·0
Overall recovery	96·75

PRIMARY GRIND IN CYANIDE WITH SHORT PERIOD OF AGITATION FOLLOWED BY CLASSIFICATION AND REGRINDING OF SAND AND SECONDARY AGITATION

Group II Ore

Test No. 15

A sample of ore was ground in cyanide to have 40 per cent −200 mesh and was agitated for 4 hours at a pulp dilution of 2 : 1. The tailing was tabled on a Laboratory Wilfley to give two products, sand and fine. The sand product was thickened and reground and was given a further period of agitation for 15 and 24 hours.

The results are as follows:

Primary Cyanidation—4 Hours' Agitation:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton		Reagents consumed, lb./ton ore	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
Cyanide tailing	0·11	0·06	45·4	0·92	0·40	0·06	2·20

Tabling of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	0·06	100·0	....
Sand	54·4	0·106	96·2	1·84 : 1
Fine	45·6	0·005	3·8	....

Secondary Cyanidation after Regrinding of Sand:

Agitation, hours	Regrind, per cent −200 mesh	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution	
		Feed	Tailing		NaCN	CaO
15	47·4	0·106	0·03	71·70	0·9	0·45
24	47·4	0·106	0·01	90·56	0·9	0·40
24	84·3	0·106	0·01	90·56	0·9	0·20

Summary of Test No. 15:

	Per cent
Gold extraction, primary agitation	45·40
Gold extraction, secondary agitation	48·44
Overall extraction	93·84

NOTE.—The same cyanide solution was used throughout the test.

The following screen tests indicate the grinding at the different stages:

Mesh	Weight, per cent			
	Primary grind	Table fine	1st regrind	2nd regrind
+ 48	1·0			
− 48+ 65	10·0	0·5	0·3	
− 65+100	21·6	1·8	7·6	0·2
−100+150	18·2	5·0	22·5	4·8
−150+200	9·2	8·0	22·2	10·7
−200	40·0	84·7	47·4	84·3
Totals	100·0	100·0	100·0	100·0

Group III Ore

Test No. 16

This was similar to Test No. 15.

Primary Cyanidation—4 Hours' Agitation:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore		Pulp dilution
	Feed	Tailing		NaCN	CaO	NaCN	CaO	
Cyanide tailing	0·48	0·12	75·0	0·88	0·20	0·14	2·60	2 : 1

Tabling of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	0·12	100·0	....
Sand	63·7	0·182	97·0	1·57 : 1
Fine	36·3	0·01	3·0	....

Secondary Agitation after Regrinding of Sand:

Agitation, hours	Regrind, per cent −200 mesh	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton of solution	
		Feed	Tailing		NaCN	CaO
15	44·0	0·182	0·075	58·8	1·0	0·45
24	44·0	0·182	0·02	89·0	1·0	0·40
24	82·1	0·182	0·01	94·5	1·0	0·20

Summary of Test No. 16:

	Per cent
Gold extraction, primary agitation	75·0
Gold extraction, secondary agitation	22·9
Overall extraction	97·9

The distribution of the gold in the fine is indicated by the following screen analysis:

Mesh	Weight, per cent	Assay, Au, oz./ton	Units	Distribution of gold, per cent
+150	4·6	0·015	0·069	6·74
−150+200	6·2	0·01	0·062	6·07
−200	89·2	0·01	0·892	87·19
Totals	100·0	0·01	1·023	100·00

Test No. 17

This was a duplicate of Test No. 16.

Primary Cyanidation—4 Hours' Agitation:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore		Pulp dilution
	Feed	Tailing		NaCN	CaO	NaCN	CaO	
Cyanide tailing	0·48	0·13	72·9	0·92	0·25	0·06	2·50	2 : 1

Tabling of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	0·13	100·00	....
Sand	63·7	0·183	90·07	1·57 : 1
Fine	36·3	0·035	9·93	....

Secondary Agitation after Regrinding of Sand:

Agitation, hours	Regrind, per cent −200 mesh	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution	
		Feed	Tailing		NaCN	CaO
48	65·9	0·183	0·02	89·0	1·0	0·25
24	90·2	0·183	0·01	94·5	0·9	0·20

Summary:

	Per cent
Gold extraction, primary agitation	72·9
Gold extraction, secondary agitation	22·1
Overall extraction	95·0

The grinding is indicated by the following screen tests:

Mesh	Weight, per cent		
	Table fine	1st regrind	2nd regrind
+ 65	1·0	....	....
− 65+100	1·9	2·1	0·1
−100+150	5·0	13·2	2·7
−150+200	6·6	18·8	7·0
−200	85·5	65·9	90·2
Totals	100·0	100·0	100·0

JIGS, BLANKETS, AND CYANIDATION

Test No. 18

The object was to recover the free gold and gold-bearing sulphides in a jig and blanket. The concentrates were to be amalgamated and the amalgamation residue to be treated by cyanidation.

This is a very simple method of treatment, but the results were not encouraging. The blanket tailing was examined on the Haultain superpanner and although no free gold was seen there was a definite amount of fine free sulphide that had passed over the blanket. From the information known about the association of the gold, there were sufficient sulphides present to account for the high gold loss in the tailing.

A composite sample, consisting of equal parts of Groups II and III ore, was ground to have 75·4 per cent −200 mesh and passed over a Denver mineral jig and blanket strake in series. The jig and blanket concentrates were reground and barrel-amalgamated with mercury for 1 hour. The amalgamation tailing was cyanided for 24 hours.

The results are as follows:

Jig and Blanket Concentration:

Calculated feed: gold, 0·295 oz./ton.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·295	100·00	....
Concentrate	3·88	5·87	77·19	25·77 : 1
Tailing	96·12	0·07	22·81	....

Amalgamation of Concentrate:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
5·87	0·435	92·59

Cyanidation of Amalgamation Tailing:

Feed: gold, 0·435 oz./ton.

Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton concentrate		Pulp dilution
		NaCN	CaO	NaCN	CaO	
0·04	90·8	1·8	0·25	0·54	3·07	3·7 : 1

Screen Test of Initial Grind:

Mesh	Weight, per cent
+ 65	0·1
− 65+100	2·0
−100+150	9·5
−150+200	13·0
−200	75·4
	100·0

Summary of Test No. 18:

	Per cent
Gold recovered by amalgamation	71·47
Gold recovered by cyanidation	5·19
Overall recovery	76·66

SETTLING TESTS

Test No. 19

A series of settling tests at different ratios of liquid to solid and with varying amounts of lime was made. After grinding in cyanide, with the requisite amount of lime added, the pulp was transferred to a glass tube of 2 inches diameter and the level of solids in decimals of a foot read for a 1-hour period. At the end of the test the solution was titrated for cyanide and alkalinity in terms of lime.

The grinding is shown by the following screen tests:

Mesh	Weight, per cent	
	Series A and B	Series C and D
+ 48	0·2	......
− 48+ 65	8·4	......
− 65+100	17·9	1·2
−100+150	19·5	9·5
−150+200	11·8	14·9
−200	42·2	74·4
Totals	100·0	100·0

Results:

—	Series A	Series B	Series C	Series D
Ratio of liquid to solid	1·5 : 1	2 : 1	1·5 : 1	2 : 1
Lime added, lb./ per ton solids	2·0	2·0	3·0	3·0
Alkalinity at end of test, CaO, lb./ton	0·20	0·15	0·20	0·15
Titration for cyanide, NaCN, lb./ton	0·30	0·30	0·30	0·30
Overflow	Clear	Clear	Clear	Clear
Ratio of settling, ft./hr	1·43	2·07	0·67	1·26
Remarks	Slows up after 20 minutes	Slows up after 30 minutes	Increases slightly after 30 minutes	Increases slightly after 30 minutes

CYANIDATION AND FLOTATION CONCENTRATE

Test No. 20

The remainder of the samples were mixed and fed to a small mill unit to make a flotation concentrate for cyanidation tests. The feed assayed 0·19 ounce of gold per ton.

The grade of concentrate obtained was as follows:

Gold	2·46 oz./ton
Silver	1·92 "
Copper	0·21 per cent
Iron	8·66 "
Zinc	1·10 "

A jig was used in the circuit to remove free gold. Unfortunately, insufficient ore was available to make a satisfactory mill run, but the results of cyanidation of the flotation concentrate are given. These preliminary results are encouraging and viewed in the light of the small-scale flotation tests indicate a high recovery. Some fouling of the solutions occurs, but a shortage of concentrate prevented the making of a study of this condition.

The results of these preliminary tests are as follows:

The concentrate, without regrinding, was agitated for 24 and 48 hours respectively in a solution of sodium cyanide—2 pounds per ton, at a pulp dilution of 3 : 1. Additions of cyanide were made during the tests and lime was added to maintain a protective alkalinity.

Results:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
24	2·46	0·08	96·70	2·32	0·20	6·24	8·40
48	2·46	0·06	97·56	2·20	0·26	8·00	10·22

Analysis of the 48-hour Solution:

Reducing power	261·5 ml. $\frac{N}{10}$ $KMnO_4$/litre
KCNS	0·19 grm./litre
Zinc	0·14 "
Copper	0·26 "

Considerable fouling of the solution is indicated.

CONCLUSIONS

It is apparent that the gold in the ore is readily attacked by cyanide. A tailing of 0·005 ounce gold per ton was obtained from all three groups, the percentage recovery being governed by the amount of gold in the feed.

Sample 4, with a gold content of 0·393 ounce per ton, yields 87·2 per cent of its gold to amalgamation.

Flotation on the lower grade samples (*see* Tests Nos. 9 and 10) produces a tailing equal to that obtained by straight cyanidation. On the higher grade sample, it is apparent that to obtain a minimum tailing, a gold jig, trap or unit cell will be necessary to remove coarse free gold.

It is worthy of note that sphalerite carries gold. The microscopic examination discloses this fact that is substantiated by the tests made with and without copper sulphate additions. When the sphalerite is activated and recovered in the concentrate the gold in the flotation tailing is appreciably lower.

Flotation of the higher grade ore does not produce so low a tailing as does straight cyanidation. However, by classifying the flotation tailing, and cyaniding the sand portion together with the flotation concentrate, the final tailing is but slightly higher than that obtained by straight cyanidation.

The same result can be obtained by cyaniding the ore at a coarse grind for a short period of time, as in Test No. 15, followed by discarding the slime and cyaniding the sand portion.

Concentration by jigs and blankets with amalgamation of free gold and cyanidation of amalgamation residue did not give encouraging results. The gold associated with the sulphides is extremely fine and it is difficult to make a good recovery on the blankets.

The results obtained by cyanidation of flotation concentrate indicated a high recovery, but some fouling of the solution by zinc and copper was indicated. A shortage of ore prevented a detailed study of this condition and further tests would be necessary to determine the degree of fouling over a period of continuous operation.

4174—10½

Ore Dressing and Metallurgical Investigation No. 772

GOLD ORE FROM THE AUGITE PORCUPINE MINES, LIMITED, TIMMINS, ONTARIO

Shipment. Two shipments of gold ore were received from the Augite Porcupine Mines, Limited, now the Aunor Gold Mines, Limited, Timmins, Ontario, on March 3 and May 15, 1939, respectively. The first consisted of 26 bags and weighed 1,500 pounds; the second of 25 bags, weighing 1,395 pounds. The shipments were submitted by Stanley S. Saxton, Manager.

Characteristics of the Ore:

Six polished sections from each of the two shipments were prepared and examined microscopically.

First Shipment. The *gangue* consists of mottled, greenish grey rock and milky vein quartz with a considerable quantity of disseminated carbonate, which gave a very slight microchemical test for iron. The rock material shows distinct schistose structure in the sections and probably represents a chlorite schist; the quartz is cut by narrow sinuous fractures.

The *sulphides* are disseminated throughout gangue but are more prevalent in the schist. Pyrite predominates as fine to coarse grains and small masses containing numerous inclusions of quartz and occasional, small, irregular grains of chalcopyrite. It is slightly shattered and veined with gangue, rarely with chalcopyrite. Chalcopyrite is present largely as medium to small irregular grains in quartz and in pyrite; also as rare narrow veinlets in pyrite as already noted. Rare, small grains of pyrrhotite are included in pyrite but its total amount is negligible.

Eleven grains of native gold were observed and measured. All occur in pyrite, eight grains along fractures and three in dense mineral. In size they range from 40 microns (minus 280 Tyler mesh) down to 12 microns (minus 1100 Tyler mesh). Five grains are associated with chalcopyrite, four grains with gangue, and two are alone. The first association suggests contemporaneous deposition and both later than the host mineral.

Second Shipment. The *gangue* is composed of an aggregate of greenish grey rock minerals with rather abundant fine, disseminated carbonate, and narrow veins of milky-white quartz. In some sections the rock exhibits a distinct schistose structure and may represent a silicified greenstone schist.

The *metallic minerals* are rather thinly scattered throughout gangue. Pyrite predominates as coarse to fine irregular grains and subhedral crystals containing numerous small pits and inclusions of gangue. Some grains are slightly fractured and veined with gangue. Chalcopyrite occurs as occasional small inclusions in pyrite, and as coarse to fine disseminated grains and small masses in quartz. It is much less abundant than pyrite. A negligible amount of pyrrhotite is present as rare tiny grains in pyrite. One or two sections contain occasional, small, irregular grains of a hard, grey, anisotropic mineral which was not definitely identified in reflected light but which may be ilmenite or specular hematite.

No native gold or gold minerals are visible in the sections.

Location of Property. The property is located between that of Delnite Mines, Limited, on the west and that of Buffalo Ankerite Gold Mines, Limited, on the east, in the Porcupine area of Ontario.

Sampling and Assaying. The ore of each shipment was crushed and sampled by standard methods and assayed as follows:

	1st Shipment	2nd Shipment
Gold	0·85 oz./ton	0·30 oz./ton
Silver	0·11 "	0·10 "
Copper	0·02 per cent	Trace
Iron	5·22 "	5·79 per cent
Arsenic	0·04 "	0·03 "
Sulphur	1·60 "	1·43 "

Experimental Tests

Shipment No. 1

The investigation comprised straight cyanidation tests on the ore; blanket concentration with amalgamation and cyanidation; and flotation followed by cyanidation of the concentrate. The minimum tailing obtained by direct cyanidation showed a loss in gold of 0·015 ounce per ton. Flotation results indicated a combined tailing of 0·014 ounce of gold per ton.

CYANIDATION

Test No. 1

Samples of ore were ground in cyanide solution at a dilution of 0·75:1 and a strength of 1 pound of sodium cyanide per ton. The pulps were diluted to 1·5:1, made up to 1 pound of sodium cyanide per ton, and agitated for various periods of time. Lime was added for protective alkalinity.

Screen tests on the cyanide tailings indicate the degree of fineness of grinding in each test.

Results:

Test	Grind, per cent −200 mesh	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton ore	
			Feed	Tailing		NaCN	CaO	NaCN	CaO
1 A	56	24	0·85	0·045	94·71	0·84	0·20	0·54	2·70
1 B	78	24	0·85	0·035	95·88	0·84	0·18	0·54	2·75
1 C	90	24	0·85	0·025	97·06	0·84	0·14	0·54	2·80
1 D	95	24	0·85	0·015	98·24	0·80	0·12	0·60	2·85
1 E	95	48	0·85	0·015	98·24	1·0	0·10	0·90	2·85
1 F	95	72	0·85	0·015	98·24	1·0	0·10	0·90	2·85

The test shows that a minimum tailing is obtained at a fineness of grinding of 95 per cent −200 mesh.

CYANIDATION FOLLOWED BY DESLIMING

Test No. 2

A sample of ore was ground in cyanide solution to have 70 per cent −200 mesh and was agitated for 24 hours at a pulp dilution of 1·5 : 1, cyanide strength of 1·0 pound of sodium cyanide per ton. The cyanide tailing was filtered and tabled to give three products: concentrate, sand tailing, and slime tailing.

The concentrate was reground to have 99 per cent −325 mesh and cyanided at a dilution of 3 :1 in a solution having a strength of 3·0 pounds of sodium cyanide per ton for 18 hours. The table sand was reground to have 95 per cent −200 mesh and was cyanided at a dilution of 1·5 :1 in a solution of 1·0 pound of sodium cyanide per ton for 18 hours.

The slime tailing was assayed and rejected.

Cyanidation of the Ore:

Assay, Au, oz./ton		Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton ore	
Feed	Tailing		NaCN	CaO	NaCN	CaO
0·85	0·03	96·5	1·0	0·2	0·60	2·70

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·03	100·00	....
Concentrate	5·50	0·45	73·40	18·2 :1
Sand tailing	42·50	0·015	18·89	2·3 :1
Slime tailing	52·00	0·005	7·71	....

Cyanidation of Table Products:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton product	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
Concentrate	0·45	0·05	88·9	3·1	0·5	2·58	7·4
Sand tailing	0·015	0·01	33·3	1·0	0·2	0·40	2·1

Summary:

	Per cent
Gold extraction by primary cyanidation	96·50
" " cyanidation of table concentrate	2·28
" " cyanidation of table sand	0·22
Overall extraction	99·00

Distribution of Gold Losses in Tailings:

Slime tailing	0·27
Cyanide residue sand tailing	0·44
" " concentrate	0·29
	1·00

Combined tailing assay: Au—0·0095 oz./ton.
Overall cyanide consumption: 0·90 lb./ton ore.

Test No. 3

The ore was given a primary agitation for 24 hours as in the previous test. The fineness of grind was the same, 70 per cent −200 mesh.

Desliming was carried out in a hydraulic cone classifier, yielding two products: sand and slime. The slime was assayed and rejected and the sand was reground in cyanide solution to have 95 per cent −200 mesh and agitated for 18 hours at a dilution of 1·5:1 in a solution of 1·0 pound of sodium cyanide per ton.

Cyanidation of the Ore:

Assay, Au, oz./ton		Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton sand	
Feed	Tailing		NaCN	CaO	NaCN	CaO
0·85	0·03	96·5	1·0	0·22	0·60	2·70

Desliming of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	0·03	100·00	....
Sand	45·2	0·06	90·87	2·2:1
Slime	54·8	0·005	9·13	....

Cyanidation of Reground Sand:

Assay, Au, oz./ton		Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton ore	
Feed	Tailing		NaCN	CaO	NaCN	CaO
0·06	0·015	75·0	1·0	0·36	1·0	5·7

Summary:

	Per cent
Gold extraction by primary cyanidation..............................	96·50
" " cyanidation of sand..................................	2·39
Overall extraction..	98·89

Distribution of Gold Losses in Tailings:

Slime tailing..	0·32
Cyanide residue of sand...	0·79
	1·11

Combined tailing assay: Au—0·0096 oz. /ton.
Overall cyanide consumption: 1·05 lb./ton ore.

The above tests show the advantages of concentrating and regrinding the sulphides and sand.

BLANKET CONCENTRATION, AMALGAMATION, AND CYANIDATION

Test No. 4

A sample of ore was ground in a water pulp to have 68 per cent −200 mesh and passed over a blanket strake with a slope of 2·5 inches per foot.

The blanket concentrate was amalgamated and the residue from amalgamation was mixed with the blanket tailing, filtered, and repulped in cyanide solution of a strength of 1·0 pound of sodium cyanide per ton. The agitation period was 24 hours.

Results:

Amalgamation:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
0·85	0·37	56·47

Cyanidation of Residue:

Assay, Au, oz./ton		Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton ore	
Feed	Tailing		NaCN	CaO	NaCN	CaO
0·37	0·03	91·9	0·80	0·10	0·43	1·02

Gold recovered by amalgamation	56·47 per cent
Gold extracted by cyanidation	40·00 "
Overall recovery	96·47 "

The test shows that 56 per cent of the gold is amalgamable or free at the grind used.

Test No. 5

The ore was given a grind of 81·5 per cent −200 mesh and the pulp was passed over a blanket strake. The concentrate was amalgamated and the residue was added to the blanket tailing. The combined products were agitated in cyanide, 1 pound of sodium cyanide per ton, at a pulp dilution of 1·5:1 for 24 hours.

The results are as follows:

Initial feed	Au, 0·85 oz./ton
Final tailing	Au, 0·025 "
Recovery	97·05 per cent

Reagent consumption:

NaCN	0·60 lb./ton
CaO	2·67 "

Test No. 6

This was a duplicate of Test No. 5.

The results are as follows:

Initial feed	Au, 0·85 oz./ton
Final tailing	Au, 0·025 "
Recovery	97·05 per cent

Reagent consumption:

NaCN	0·60 lb./ton
CaO	2·67 "

INFRASIZER TEST ON CYANIDE TAILING

Test No. 7

A sample of ore was ground to have 95 per cent −200 mesh in cyanide solution and cyanided for 24 hours in a solution having 1·0 pound of sodium cyanide per ton at a dilution of 1·5:1.

A portion of the dry tailing was classified by the infrasizer and the various products were assayed for gold and sulphur.

Results:

Products, in microns	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S
Feed	100·00	0·017	1·38	100·00	100·0
+56	5·22	0·045	3·51	13·29	13·2
−56+40	10·53	0·045	2·40	28·81	18·3
−40+28	14·25	0·02	1·79	16·12	18·4
−28+20	14·75	0·015	1·38	12·52	14·7
−20+14	13·40	0·01	1·14	7·58	11·0
−14+10	10·30	0·01	0·97	5·83	7·2
−10	31·55	0·01	0·75	17·85	17·2

The results indicate that fine grinding is necessary to expose the gold to the action of cyanide solution.

FLOTATION

Tests Nos. 8 and 9

Two samples of ore were ground in water with 1·5 pounds of soda ash and 0·07 pound of Barrett No. 4 oil per ton to have 83 per cent and 93 per cent −200 mesh respectively.

The pulps were transferred to a Denver Sub-A flotation machine and were conditioned with 0·2 pound of potassium amyl xanthate per ton; 0·075 pound of pine oil per ton was used as frother. A rougher concentrate was recovered.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Test No. 8: Grind, 83 per cent −200 mesh				
Feed	100·00	0·89*	100·00	....
Concentrate	5·57	15·78	98·94	18:1
Tailing	94·43	0·01	1·06	....
Test No. 9: Grind, 93 per cent −200 mesh				
Feed	100·00	0·95*	100·00	....
Concentrate	6·91	13·64	99·02	14·5:1
Tailing	93·09	0·01	0·98	....

*Calculated from products.

Analysis of Concentrate, Test No. 8:

Silver	1·56 oz./ton
Copper	0·17 per cent
Arsenic	0·67 "

CYANIDATION OF FLOTATION CONCENTRATE

Tests Nos. 10 and 11

Flotation concentrates were obtained as in the previous tests and were reground in water to have 99 per cent −325 mesh. The products were agitated in cyanide solution, 3 pounds of sodium cyanide per ton, at a pulp dilution of 3 : 1 for periods of 18 and 48 hours.

Results:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
10	18	12·14	0·175	98·56	2·7	0·1	4·80	6·85
11	48	15·00	0·125	99·17	2·6	0·2	6·90	5·00

Total extraction of gold in terms of feed	96·4 per cent
Combined tailing	Au, 0·014 oz./ton

Analysis of Solution, Test No. 11:

Reducing power	378 ml. $\frac{N}{10}$ $KMnO_4$/litre
NaCNS	0·47 grm./litre
Total iron	0·36 "

Shipment No. 2

This investigation comprised cyanidation tests on the ore and concentration by flotation followed by cyanidation of the concentrates. A tailing of 0·01 ounce of gold per ton was obtained by straight cyanidation and by the latter method an overall tailing loss in gold of 0·018 ounce per ton was obtained.

CYANIDATION

Tests Nos. 12 to 15

Four samples of −14-mesh ore were ground in cyanide to different degrees of fineness and then agitated at a pulp dilution of 1·5 :1 for a period of 24 hours. The cyanide strength was 1 pound of sodium cyanide per ton, and a protective alkalinity of around 0·2 pound of lime per ton of solution was maintained.

The results are as follows:

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
12	0·30	0·02	93·33	0·94	0·26	0·19	2·81
13	0·30	0·01	96·66	0·88	0·20	0·28	2·90
14	0·30	0·01	96·66	0·84	0·16	0·34	2·96
15	0·30	0·01	96·66	0·94	0·12	0·43	3·02

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

Mesh	Test No. 12	Test No. 13	Test No. 14	Test No. 15
+ 65	7·0	0·2	....	....
− 65+100	12·6	2·6	0·6	0·1
−100+150	14·7	8·0	4·0	1·6
−150+200	10·3	9·8	6·4	3·7
−200	55·4	79·4	89·0	94·6
Totals	100·0	100·0	100·0	100·0

Test No. 16

The ore was given a moderate grind (55 per cent −200 mesh) and a short period of agitation (4 hours). The primary cyanide tailing was tabled to give two products, sand and fine. The sand was reground and agitated for a further 24 hours; the fine was discarded.

Cyanide solutions had a strength of 1 pound of sodium cyanide per ton at a pulp dilution of 1·5:1.

The results are tabulated as follows:

Primary Agitation, 4 Hours:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
Cyanide tailing	0·30	0·025	91·66	0·88	0·26	0·20	2·70

Tabling of Primary Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·025	100·0	....
Sand	56·48	0·04	91·3	1·77:1
Fine	43·52	0·005	8·7	....

The sand was reground to have 66·2 per cent −200 mesh and was agitated for 24 hours:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton sand	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
Cyanide product......	0·04	0·015	62·5	0·92	0·30	0·15	2·55

Summary:

	Per cent
Gold extraction by primary cyanidation..............................	91·66
" " re-treatment of sand..............................	4·76
Overall extraction..	96·42

Final overall tailing=Au, 0·0106 oz./ton.

A screen analysis of the fine indicates the distribution of gold in the different screen sizes to be as follows:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
+100..	1·7	Trace	....
−100+150..	1·9	0·105	29·3
−150+200..	2·9	0·005	2·1
−200..	93·5	0·005	68·6
Totals....................................	100·0	0·006	100·0

Test No. 17

Fine grinding of the sand was carried out. The primary grind was the same as in Test No. 16.

Primary Agitation, 4 Hours:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
Cyanide tailing.......	0·30	0·02	93·33	1·00	0·24	0·20	2·64

Tabling of Primary 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	....
Sand......................................	46·19	0·035	86·55	2·16 : 1
Fine......................................	53·81	0·005	13·45	....

The sand was reground to have 89·9 per cent −200 mesh and was agitated for 24 and 48 hours:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton sand	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
24	0·035	0·02	42·86	1·00	0·30	0·25	2·55
48	0·035	0·015	57·14	0·90	0·25	0·40	2·69

Summary:

		Per cent
Gold extraction by	primary cyanidation	93·33
" "	re-treatment of sand (48 hours)	3·30
Overall extraction		96·63

Final overall tailing=Au, 0·0096 oz./ton.

Test No. 18

The ore was given a primary grind to have 79 per cent −200 mesh.

Primary Agitation, 4 Hours:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
Cyanide tailing	0·30	0·02	93·33	0·92	0·22	0·16	2·67

Tabling of Primary Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·02	100·0	....
Sand	44·43	0·038	86·1	2·25:1
Fine	55·57	0·005	13·9	....

The sand was reground to have 86·4 per cent −200 mesh and was agitated for 24 hours:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton sand	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
Cyanide tailing	0·038	0·02	47·3	1·00	0·24	0·21	2·64

Summary:

	Per cent
Gold extraction by primary cyanidation	93·33
" " re-treatment of sand	2·73
Overall extraction	96·06

Final overall tailing=Au, 0·0116 oz./ton.

A screen analysis of the fine indicates the distribution of the gold in the different screen sizes to be as follows:

Mesh	Weight, per cent	Assay, Au, oz./ton	Units	Distribution of gold, per cent
+150	1·6	Trace	....	....
−150+200	2·6	0·02	0·052	9·8
−200	95·8	0·005	0·479	90·2
Totals	100·0	0·005	0·531	100·0

SETTLING TESTS

These tests were made of the cyanide tailings.

Samples of ore were ground in a cyanide-lime pulp to have a fineness of from 56 per cent −200 mesh to 95 per cent −200 mesh.

After agitating for 24 hours at a dilution ratio of 1·5:1, each pulp was transferred to a tall glass cylinder, 2 inches inside diameter. Readings of the pulp level in decimals of a foot were taken at 5-minute intervals over a period of 1 hour.

The results are given in the following table:

—	Grinding, per cent −200 mesh			
	56	78	90	95
Rate of settling, ft./hr	1·175	0·94	0·455	0·34
Pulp dilution	1·5 : 1	1·5 : 1	1·5 : 1	1·5 : 1
Agitation, hours	24	24	24	24
Final titration, NaCN, lb./ton	0·84	0·84	0·84	0·80
CaO, lb./ton solution	0·20	0·18	0·14	0·12
Overflow	Clear	Clear	Clear	Clear

The rate is a little below normal on the finely ground products.

FLOTATION

A number of flotation tests were carried out in order to recover the gold in a concentrate that could subsequently be treated by cyanidation. The flotation tailing resulting from these tests carried 0·01 ounce of gold per ton, which was the same as by direct cyanidation of the ore. The overall tailing from flotation and cyanidation of concentrates was somewhat higher than by the straight cyanide method.

The initial flotation tests gave a dirty, slimy concentrate, which was probably due to the carbonates and schistose material in the gangue. This was overcome by the use of either starch solution or sodium silicate, with a small addition of copper sulphate in each case. These reagents improved the ratio of concentration and produced a clean sulphide concentrate and a satisfactory froth. The results of the flotation tests are as follows:

Tests Nos. 19, 20, 21, 22, and 23

Denver Sub-A Laboratory Machine:

Test No.	Grind, per cent −200 mesh	Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
19	76·3	Feed	100·00	0·25*	100·0	
		Concentrate	16·33	1·47	96·6	6·12 : 1
		Tailing	83·67	0·01	3·4	
20	85·4	Feed	100·00	0·25*	100·0	
		Concentrate	8·49	2·78	96·3	11·77 : 1
		Tailing	91·51	0·01	3·7	

*Calculated from products.

Reagents Used, lb./ton Feed:

Test No.	Soda ash	Aerofloat No. 25	Potassium amyl xanthate	Pine oil
19	1·5	0·07	0·2	0·031
20	1·5	0·07	0·2	0·031

The soda ash and Aerofloat were added to grinding.

These tests showed a very dirty concentrate.

The following tests were made using either starch or sodium silicate and the concentrate was cleaned:

Test No.	Grind, per cent −200 mesh	Product	Weight, per cent	Assay			Distribution of gold, per cent	Ratio of concentration
				Au, oz./ton	Per cent			
					Cu	S		
21	75·8	Feed	100·00	0·24*			100·00	
		Cleaner concentrate	3·22	6·82			93·01	31 : 1
		Middling	8·54	0·09			3·26	
		Tailing	88·24	0·01		0·26	3·73	
22	75·8	Feed	100·00	0·23*			100·00	
		Cleaner concentrate	3·76	5·52			92·02	26·6 : 1
		Middling	2·39	0·36			3·82	
		Tailing	93·85	0·01		0·06	4·16	
23	75·8	Feed	100·00	0·21*			100·00	
		Cleaner concentrate	3·33	5·88	0·14		91·08	30 : 1
		Middling	1·45	0·665			4·49	
		Tailing	95·22	0·01		0·02	4·43	

*Calculated from products.

Reagents Used, Lb/ton Feed:

Test No.	Soda ash	Aerofloat No. 25	Starch, 5 per cent solution	Sodium silicate	Copper sulphate	Potassium amyl xanthate	Pine oil
21	1·5	0·07	0·5	....	0·5	0·2	0·031
22	1·5	0·035	0·5	....	0·25	0·2	0·062
23	1·5	0·07	....	1·0	0·5	0·2	0·062

In Test No. 23 the sodium silicate and copper sulphate were added before the xanthate.

Tests Nos. 24, 25, 26, 27, and 28

These five tests were identical. A rougher concentrate was made in a 2,000-gramme laboratory Fagergren cell. The rougher concentrate was cleaned in a 500-gramme Denver Sub-A machine.

Reagents Used:

	Lb./ton
Soda ash	1·5
Aerofloat No. 25	0·035
Sodium silicate	1·0
Copper sulphate	0·25
Potassium amyl xanthate	0·2
Pine oil	0·124

No reagents were added in the cleaner cell.

Summarized Results of Tests Nos. 24, 25, 26, 27, and 28:

Product	Weight, per cent	Assay				Distribution of gold, per cent	Ratio of concentration
		Oz./ton		Per cent			
		Au	Ag	Cu	S		
Feed	100·00	0·23*	....	....	....	100·0	....
Cleaner concentrate	4·44	4·74	0·74	0·09	....	93·3	22·5 : 1
Middling	1·69	0·285	....	....	....	2·1	....
Tailing	93·87	0·011	....	....	0·04	4·6	....

*Calculated from products.

CYANIDATION OF FLOTATION CONCENTRATE

Test No. 29

The combined cleaner concentrates from Tests Nos. 24 to 28 inclusive were reground in a water pulp to a fineness of 98 per cent −325 mesh. The pulp was filtered and washed to remove flotation agents and two samples were repulped in cyanide solution of strength, 3 pounds of sodium cyanide per ton, and at a dilution ration of 3:1. Agitation was carried out for 24- and 48-hour periods.

Results:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
29 A	24	4·74	0·13	97·26	2·8	0·40	4·40	8·80
29 B	48	4·74	0·14	97·04	2·7	0·25	4·90	9·25

The overall recovery by flotation and cyanidation of the concentrate, as shown by the results of the above tests, is as follows:

	Per cent
Gold recovered in flotation concentrate..............................	95·40
Gold extraction by cyanidation of concentrate (24 hours' agitation), 97·26 per cent of 95·40 per cent....................................	92·79
This gives an indicated overall tailing of Au..	0·018 oz./ton

CONCLUSIONS

The investigation indicates that the higher grade of the ore in the first shipment is due primarily to its free-gold content. It is shown (Test No. 4) that at a grind of 68 per cent −200 mesh, 56 per cent of the gold is amalgamable or free-milling.

Comparing results obtained by primary cyanidation, desliming and regrinding of the sand, it is found that the final combined tailing for the ore of each shipment is as follows: (Reference, Tests Nos. 2 and 17)

First shipment..	Au, 0·0095 oz./ton
Second shipment..	Au, 0·0096 oz./ton

Flotation results give a loss of gold in the tailing of 0·01 ounce per ton for both grades of ore. The use of starch or sodium silicate as dispersants with a small amount of copper sulphate minimizes contamination of the concentrate by carbonates or other talcy material in the gangue. A high ratio of concentration is attained. The concentrate is amenable to cyanidation after regrinding and shows a combined tailing loss of from 0·014 to 0·018 ounce of gold per ton.

Jigs, traps, or a unit cell in the primary grinding-classifying circuit would be essential in any method of treatment on ore similar to that of the first shipment.

Ore Dressing and Metallurgical Investigation No. 773

CONCENTRATE FROM PORCHER ISLAND MINES, LIMITED, PORCHER ISLAND, BRITISH COLUMBIA

Shipment. A box of flotation concentrate, weight 84 pounds, was received on January 31, 1939, from the Porcher Island Mines, Limited, Surf Point, Porcher Island, British Columbia. The sample was submitted at the request of A. M. Richmond, 108 Merchants Exchange Building, Vancouver, B.C.

Purpose of Investigation. The sample was submitted to determine if the concentrate could be treated locally by cyanidation, either directly or after roasting, in order to effect any savings over the present method of shipping the concentrate to a smelter at Tacoma, Wash.

Character of the Concentrate. The concentrate consists of almost pure pyrite containing gold in an extremely fine condition.

Analysis of the concentrate was as follows:

Gold	7·66 oz./ton
Silver	3·31 "
Iron	40·00 per cent
Sulphur	44·71 "
Arsenic	Nil

Results of Investigation. By cyanidation of the raw concentrate only 32·6 per cent of the contained gold was extracted.

After subjecting the concentrate to a dead roast, 91·2 per cent of the gold in the calcine was extracted by cyanide.

Extraction after a chloridizing roast was not satisfactory.

Experimental Tests

DIRECT CYANIDATION OF CONCENTRATE

Test No. 1

A sample of concentrate, 1,000 grammes, was ground in a cyanide solution with 2 pounds of lime per ton to a fineness of 89 per cent −325 mesh. The ground pulp was transferred to a bottle and made up to a pulp dilution of 3 : 1 and a cyanide strength of 2 pounds of sodium cyanide per ton, and agitated for 24 hours. Cyanide was added during the period of the run to maintain the strength at 2 pounds of sodium cyanide per ton.

Results:

Total cyanide added (including grinding)........................ 9·6 lb./ton
Total lime added (including grinding)........................... 4·0 "

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Final solution, lb./ton		Reagents consumed, lb./ton concentrate	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
Cyanide tailing.......	7·66	5·16	32·6	1·82	0·12	4·14	3·64

In order to determine the distribution of the gold in the tailing, a sample was infrasized in the Haultain infrasizer. The results show the distribution of the gold in the different sizes of the pyrite grains and indicate conclusively its extreme fineness.

Infrasizer Analysis:

Size in microns	Weight, per cent	Assay, Au, oz./ton	Units	Distribution, per cent
Over 56....................................	9·13	4·82	44·0066	8·59
56—40....................................	18·91	4·54	85·8514	16·76
40—28....................................	18·79	4·16	78·1664	15·26
28—20....................................	15·06	3·90	58·7340	11·46
20—14....................................	11·21	4·30	48·2030	9·41
14—10....................................	7·76	4·96	38·4896	7·51
Minus 10....................................	19·14	8·30	158·8620	31·01
Total.........................	100·00	5·12	512·3130	100·00

The results of this test show that economic cyanidation of the raw concentrate is not possible.

ROASTING TESTS

Roasting of the concentrate followed by cyanidation of the calcine was carried out in several tests. A decided improvement in extraction was noted, but the final tailing still carried almost an ounce of gold per ton.

Test No 2

A sample of concentrate, 2,000 grammes in weight, was roasted in an open tray in an electric muffle furnace for 6 hours. The temperature range was as follows:

1st hour.. 300° C.—450° C.
2nd hour.. 450° C.—550° C.
3rd to 5th hours.. 550° C.—740° C.
6th hour.. 740° C.—740° C.

The charge was rabbled frequently during the period of roasting.

The loss in weight during roasting was 30·9 per cent. The calcine was ground in water to a fineness of 89 per cent—325 mesh and an analysis of the grinding water indicated that 12·33 grammes of soluble salts (sulphates and iron) had been dissolved out.

The washed calcine assayed as follows:

Gold	11·14 oz./ton
Sulphur	0·85 per cent

Cyanidation of Calcine. Four samples of the calcine were agitated in cyanide solution, as follows:

Sample	Pulp dilution	NaCN, lb./ton	Agitation, hours
A	3:1	2	24
B	3:1	3	24
C	3:1	2	48
D	2:1	2	24

The results are shown in the following table:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Final solution, lb./ton		Reagents consumed, lb./ton	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
2A	24	11·14	1·12	89·9	1·86	0·18	4·42	3·46
2B	24	11·14	1·20	89·2	2·72	0·20	4·84	3·40
2C	48	11·14	1·10	90·1	2·00	0·18	5·00	3·46
2D	24	11·14	1·12	89·9	1·46	0·17	4·08	3·66

Test No. 3

A charge of concentrate was roasted under conditions similar to those in Test No. 2. The calcine was ground in water and after washing assayed as follows:

Gold	10·75 oz./ton
Sulphur	0·80 per cent

Two samples of the calcine were agitated in cyanide solution, at a strength of 2 pounds of sodium cyanide per ton, with 2 pounds of lime per ton at a pulp dilution of 3:1 for 24 hours. The solution was filtered and the calcine repulped in fresh cyanide solution and agitated for an additional 24 hours. There was a slight improvement in extraction by the two-stage agitation.

Results:

Test No	Product	Assay, Au, oz./ton		Extraction of gold, per cent	Final solution, lb./ton		Reagents consumed, lb./ton	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
3A	Cyanide tailing...	10·75	0·945	91·2	2·1	0·20	6·6	2·95
3B	Cyanide tailing...	10·75	0·95	91·1	1·9	0·15	3·9	2·95

Test No. 4

A chloridizing roast was tried in order to determine if such treatment would promote the solubility of the gold in the calcine.

A charge of concentrate, 2,000 grammes, was mixed with sodium chloride in an amount equal to 5 per cent of the weight of the charge, and was roasted in an open tray under similar conditions to the two previous tests. Maximum temperature, 735° C.

The calcine weighed 1,553 grammes, indicating a loss in weight of 26 per cent.

The calcine was ground in water to wash out soluble salts and a screen test of the ground calcine indicated 98 per cent −325 mesh.

The analysis of washed calcine was as follows:

Gold..	9·60 oz./ton
Sulphur..	0·92 per cent

Soluble salts removed from the calcine during grinding were as follows:

Sulphates..	94·3 grammes
Iron..	14·5 "
Gold...	Nil

Two samples of the ground calcine were cyanided for 24 and 48 hours respectively. The solution strength was 2 pounds of sodium cyanide per ton and lime was added to give a protective alkalinity of 0·10 pound of lime per ton. Pulp dilution was 3 : 1.

The cyanide and lime consumptions were considerably higher than on a straight calcine.

The results are as follows:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Final solution, lb./ton		Reagents consumed, lb./ton	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
4A	24	9·60	1·56	83·7	2·0	0·10	11·00	5·70
4B	48	9·60	1·40	85·4	2·1	0·15	12·70	8·55

The chloridizing roast gives a lower extraction and a higher consumption of cyanide and lime than a straight dead roast and also accounts for a loss of gold by volatilization.

CONCLUSIONS

The results obtained in roasting and cyaniding the calcine of Porcher Island Mines concentrate do not indicate any saving by roasting over shipping the concentrate to a smelter.

The condition of the gold in the pyrite is apparently similar to that found in the ore of the Surf Point mine, which was investigated in the Ore Dressing and Metallurgical Laboratories in 1932 and reported under Investigation No. 456, Mines Branch Report No. 736. In this ore the gold occurring in the pyrite was sub-microscopic.

The present investigation has shown that in the cyanide tailing of the raw concentrate 31 per cent of the gold is in pyrite grains of minus-10-micron size.

Roasting apparently exposes much of this fine gold to the action of cyanide, but the remaining 9 per cent, which represents a tailing containing about 1 ounce of gold per ton, is so locked in the fine particles of calcine that the cyanide has no opportunity of dissolving it.

Roasting a concentrate of the grade indicated will result in a large dust loss unless suitable measures be adopted to overcome this.

The present costs per ton of concentrate for freight, treatment, and sacking are given as $10.75.

The tailing loss in roasting and cyanidation is about $22.46 per ton of concentrate, assuming a calcine cyanide tailing of gold, 0·95 ounce per ton, and gold at $33.78 per ounce.

When the cost of erecting a roasting and cyanide plant and operating charges are added to the above tailing loss, it is evident that the present method is the more economic.

Ore Dressing and Metallurgical Investigation No. 774

GOLD ORE FROM THE NORTH ORE ZONE OF THE HARD ROCK GOLD MINES, LIMITED, GERALDTON, ONTARIO

Shipment. One carload of gold ore, weight 70,300 pounds, was received on July 19, 1938, from J. C. Dumbrille, Mine Manager, Hard Rock Gold Mines, Limited, Geraldton, Ontario. Previously, shipments of ore had been received in September 1935, and February 1937, and are covered by Bureau of Mines Reports Nos. 771 and 785, respectively.

Location of the Property. The property of the Hard Rock Gold Mines, Limited, from which the present shipment was received is in the Little Long Lac area, Thunder Bay district, northern Ontario.

Sampling and Analysis. A representative sample of the ore was cut from the carload shipment by standard methods, and it assayed as follows:

Gold	0·33 oz./ton
Silver	0·03 "
Arsenic	2·45 per cent
Iron	22·80 "
Tellurium	Nil
Lead	0·02 per cent
SiO_2	40·45 "
Al_2O_3	8·38 "
Sulphur	11·95 "
Copper	0·04 "

Upon the assumption that all sulphur came from either pyrite or arsenopyrite, the above chemical analysis is equivalent to 20·4 per cent pyrite and 5·3 per cent arsenopyrite.

Characteristics of the Ore. From a sample of ore removed from the carload shipment six polished sections were prepared.

Microscopic examination of the polished sections showed that the ore is highly siliceous and contains only a minor quantity of carbonate; pyrite and lesser arsenopyrite are abundantly disseminated throughout this gangue. Both minerals occur as moderately coarse to fine grains. A small quantity of magnetite as irregular, disseminated grains and rare tiny grains of pyrrhotite occur in the pyrite. An occasional small grain of chalcopyrite is present in the gangue. The sections were traversed with low-power and high-power oil-immersion lenses and no native gold was visible.

Conclusion from Microscopic Examination. An appreciable proportion of the gold in the ore is present in the sulphides and is largely, if not wholly, sub-microscopic.

Investigative Work. The metallurgical work described in Section A of this report was taken from the report of A. H. Ross, consulting metallurgist for the Hard Rock Gold Mines, Limited. The operation of the test mill, together with all sampling and chemical and gold analyses, was performed by the Bureau of Mines. Section B covers the cyanidation tests, and Section C the roasting and cyanidation tests.

Section A

TEST MILL RUNS

General Flow-Sheet Used in Test Mill Runs. Ore was fed at a rate of 550 pounds per hour to a Deco ball mill (2 feet by 3 feet), in closed circuit with a Dorr Simplex classifier. The classifier overflow discharged by gravity into a Denver conditioner (2 feet by 3 feet), giving a conditioning period of 13·8 minutes. From the conditioner, the pulp flowed by gravity to the five pyrite flotation cells. The rough pyrite concentrate was pumped into two cleaner flotation cells arranged in series. The cleaned pyrite concentrate went to storage and the middling from the cleaner cells was pumped to a standard Wilfley table. The arsenopyrite which was tabled out together with the table tailing went to separate storage bins.

The flotation tailing from the rough pyrite cells was pumped to a Denver conditioner (2 feet by 3 feet), giving a conditioning period of 14 minutes. The feed to the six arsenopyrite flotation cells was the discharge from this conditioner. The rough arsenopyrite concentrate from these cells was pumped to two cleaner cells arranged in parallel, and the flotation tailing from the rougher cells was discarded as mill tailing. The cleaned arsenopyrite concentrate went to storage and the middling from the cleaner cells was pumped to a standard Deister Plat -O table. The arsenopyrite table concentrate and the table tailing went to separate storage bins.

All flotation cells were Denver Sub-A No. 7 cells, each of which had a volume of 1 cubic foot and a period of flotation of 2·2 minutes.

The above flow-sheet was modified slightly in Mill Run No. 6, as can be noted by reference to Figure 4.

Mill Run No. 1

The flow-sheet was the same as shown in Figure 3, with the exceptions that seven cells were used for the rough pyrite float and eight cells were employed for the rough arsenopyrite float.

Reagents Added, Lb./ton Original Ore:

A. For Pyrite:

To Ball Mill:

Soda ash	1·45
Potassium amyl xanthate	0·022

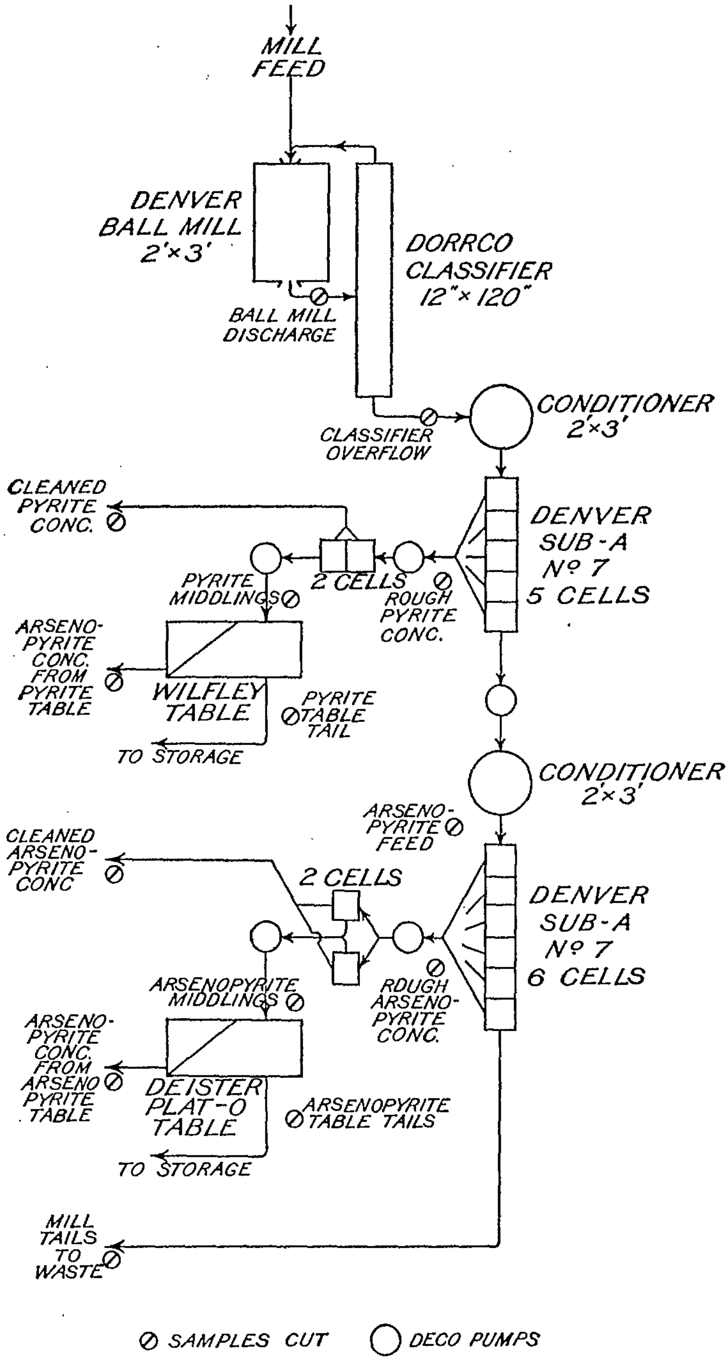

Figure 3. Flow-sheet of Mill Runs Nos. 1 to 4 on Sample 4.

To Conditioner:

Pine oil Yarmor F	0·05
Conditioned 13·8 minutes.	

During Flotation:

Potassium amyl xanthate	0·073 to 2nd cell
" "	0·12 to 4th cell
Time of float	15·4 minutes
The pH of solution feeding to 1st cell was 8·2.	

B. For Arsenopyrite:

To Pump:

Soda ash	3·1

To Conditioner:

Copper sulphate	1·45 } Conditioned 14 minutes
Reagent 301	0·06 }

During Flotation:

Reagent 301	0·04 to 2nd cell
Tarol pine oil	0·02 to 3rd cell
Reagent 301	0·04 to 5th cell
Copper sulphate	0·96 } to 6th cell
Aerofloat No. 25	0·032 }
Tarol pine oil	0·02 to 7th cell
Time of float	17·6 minutes
The pH of solution feeding to 1st cell was 9·75.	

Average specific gravity of ball mill discharge was 1,750 (61 per cent solids) and of classifier overflow, 1,221 (26 per cent solids).

Results:

Ball mill discharge assayed:

Gold	0·57 oz./ton
Arsenic	4·01 per cent

Classifier overflow assayed:

Gold	0·33 oz./ton
Arsenic	2·71 per cent
Iron	23·16 "
Insoluble	44·76 "
Sulphur	12·79 "

The above analysis of the classifier overflow is equivalent to 5·9 per cent of arsenopyrite and 21·8 per cent of pyrite.

Results—Continued

Product	Weight, per cent		Assay		Distribution of gold, per cent		Ratio of concentration	
	Section	Head	Au, oz./ton	As, per cent	Section	Head	Section	Head
Pyrite Rougher Section—								
Feed	100·0	100·0	0·33	2·71	100·0	100·0	....	....
Concentrate	15·5	15·5	1·15	1·17	54·0	54·0	6·47	6·47
Tailing	84·5	84·5	0·18	3·06	46·0	46·0	....	....
Pyrite Cleaner Section—								
Feed	100·0	15·5	1·15	1·17	100·0	54·0	....	....
Concentrate	66·5	10·3	1·53	0·78	88·8	48·0	1·5	9·7
Tailing	33·5	5·2	0·385	1·89	11·2	6·0	....	....
Pyrite Table Section—								
Feed	100·0	5·2	0·385	1·89	100·0	6·0	....	....
Concentrate	13·5	0·7	1·01	4·56	35·3	2·1	7·85	143
Tailing	86·5	4·5	0·29	1·50	64·7	3·9	....	....
Arsenopyrite Rougher Section—								
Feed	100·0	84·5	0·18	3·06	100·0	46·0	....	....
Concentrate	13·3	11·3	1·19	19·26	88·2	40·6	7·5	8·9
Tailing	86·7	73·2	0·025	0·36	11·8	5·4	....	....
Arsenopyrite Cleaner Section—								
Feed	100·0	11·3	1·19	19·26	100·0	40·6	....	....
Concentrate	92·1	10·4	1·25	20·87	96·7	39·2	1·1	9·6
Tailing	7·9	0·9	0·52	8·83	3·3	1·4	....	....
Arsenopyrite Table Section—								
Feed	100·0	0·9	0·52	8·83	100·0	1·4	....	....
Concentrate	11·8	0·1	0·93	14·76	21·2	0·3	8·5	1000
Tailing	88·2	0·8	0·465	8·08	78·8	1·1	....	....

In actual mill operation, and the tables in closed circuit with the rougher cells, the calculated results would be:

Product	Weight, per cent	Ratio of concentration	Assay		Distribution of gold, per cent
			Au, oz./ton	As, per cent	
Classifier overflow	100·0	....	0·33	2·71	100·0
Cleaned pyrite concentrate	14·8	6·75	1·15	1·01	51·9
Pyrite table concentrate	0·7	143·0	1·01	4·56	2·1
Cleaned arsenopyrite concentrate	11·2	8·9	1·19	19·30	40·3
Arsenopyrite table concentrate	0·1	1000·0	0·93	14·76	0·3
Flotation tailing	73·2	....	0·025	0·36	5·4

Screen Analyses:

Mesh	Ball Mill Discharge			Classifier overflow
	Weight, per cent	Assay		Weight, per cent
		Au, oz./ton	As, per cent	
+ 48	3·4	0·126	0·68	....
+ 65	6·4	0·15	0·85	....
+100	15·4	0·25	1·30	2·1
+150	22·6	0·30	3·19	10·3
+200	18·8	0·85	6·00	13·5
−200	33·4	0·775	5·64	74·1
	100·0	....	....	100·0

In Mill Runs Nos. 2, 3, and 4 the flow-sheet was similar to that of Mill Run No. 1 as shown in Figure 3. The results obtained with different proportions of the flotation reagents did not differ greatly from those of Mill Run No. 1.

In Mill Run No. 5 the flow-sheet gave results indicating the recovery that could be obtained by the introduction of a Deister Plat-O table in closed circuit with the rougher cells. These results approximated those of the previous mill runs.

Mill Run No. 6

The flow-sheet was similar to that shown in Figure 4.

Reagents Added, Lb./ton Original Ore:

A. For Pyrite:

To Ball Mill:

Soda ash	1·45
Potassium amyl xanthate	0·022

To Conditioner:

Pine oil Yarmor F	0·05
Conditioned 13·8 minutes.	

During Flotation:

Potassium amyl xanthate	0·073 to 1st cell
" "	0·12 to 3rd cell
" "	0·073 to 4th cell
Time of flotation	11·0 minutes
The pH of solution feeding to 1st cell was 8·1.	
To 1st pyrite cleaner cell—Pine oil Yarmor F	0·022

B. For Arsenopyrite:

To Pump:

Soda ash	3·1

To Conditioner:

Copper sulphate	1·45 } Conditioned 14 minutes
Reagent 301	0·06 }

During Flotation:

Reagent 301	0·12 to 2nd cell
Tarol pine oil	0·02 to 3rd cell
Reagent 301	0·12 to 4th cell
Aerofloat No. 25	0·032 } to 5th cell
Copper sulphate	0·96 }
Tarol pine oil	0·02 to 6th cell
Time of flotation	13·2 minutes
The pH of solution feeding to rougher cells was 9·6.	

Average specific gravity of ball mill discharge was 1,730 (60 per cent solids), and of classifier overflow (feed to pyrite rougher cells) was 1,230 (26·5 per cent solids).

Results:

Ball mill discharge assayed:

Gold	0·53 oz./ton
Arsenic	3·60 per cent

Classifier overflow assayed:

Gold	0·29 oz./ton
Arsenic	1·80 per cent
Sulphur	11·44 "
Iron	21·94 "
Insoluble	44·02 "

The above analysis of the classifier overflow is equivalent to 3·9 per cent of arsenopyrite and 20·0 per cent of pyrite.

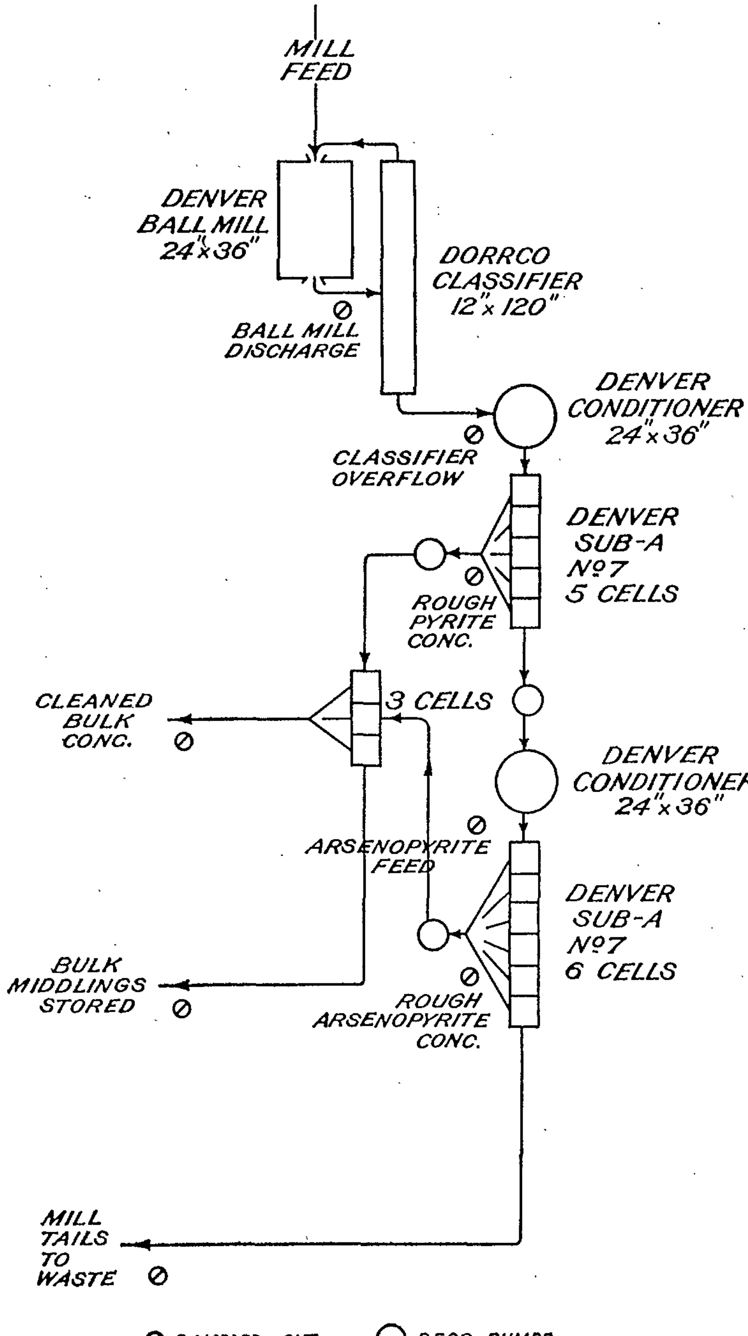

Figure 4. Flow-sheet of Mill Run No. 6 on Sample 4.

Results—Continued

Product	Weight, per cent		Assay		Distribution of gold, per cent		Ratio of concentration	
	Section	Head	Au, oz./ton	As, per cent	Section	Head	Section	Head
Pyrite Rougher Section—								
Feed	100·0	100·0	0·29	1·80	100·0	100·0	....	....
Concentrate	12·8	12·8	1·28	0·90	56·5	56·5	7·8	7·8
Tailing	87·2	87·2	0·145	2·44	43·5	43·5	....	....
Arsenopyrite Rougher Section—								
Feed	100·0	87·2	0·145	2·44	100·0	43·5	....	....
Concentrate	10·3	9·0	1·23	21·90	87·8	38·2	9·7	11·1
Tailing	89·7	78·2	0·02	0·23	12·2	5·3	....	....
Bulk Concentrate Cleaner Section—								
Feed	100·0	21·8	1·26	9·55	100·0	94·7	....	....
Concentrate	....	....	1·21	8·40	No calculation possible owing to inaccurate sampling			
Tailing	....	....	0·19	3·11				

Summary:

Product	Weight, per cent	Ratio of concentration	Assay		Distribution of gold, per cent
			Au, oz./ton	As, per cent	
Classifier overflow	100·0	....	0·29	1·80	100·0
Rougher bulk concentrate	21·8	4·6	1·26	9·55	94·7
Flotation tailing	78·2	....	0·02	0·23	5·3

Screen Analyses:

Mesh	Ball Mill Discharge			Classifier overflow		
	Weight, per cent	Assay		Weight, per cent	Assay	
		Au, oz./ton	As, per cent		Au, oz./ton	As, per cent
+ 65	14·3	0·105	0·86	....	....	....
+100	18·2	0·23	1·42	4·2	0·032	0·34
+150	22·0	0·405	3·75	14·5	0·085	0·64
+200	17·7	0·815	7·80	15·7	0·21	1·35
−200	27·8	0·885	6·07	65·6	0·375	3·15
	100·0	....	....	100·0	....	....

The cleaned bulk concentrate analysed as follows:

Gold	1·21 oz./ton
Arsenic	8·40 per cent
Sulphur	43·86 "
Iron	43·76 "
Insoluble	3·84 "

Mr. F. D. Reid, consulting engineer for the Hard Rock Gold Mines, Limited, asked whether an arsenical concentrate assaying two ounces or more in gold could be made. The next mill run was made with this point in view.

The pH of the solution feeding to the pyrite cells was raised to 8·7 in this run, to see whether gold recovery in a cleaned pyrite concentrate could be raised without increasing the content of arsenic in the pyrite. No attempt was made to maintain overall recovery.

In continuous mill operation these results could be bettered considerably.

Mill Run No. 7

The flow-sheet was the same as shown in Figure 5.

Reagents Added, Lb./ton Original Ore:

A. For Pyrite:

To Ball Mill:

Soda ash	1·92
Potassium amyl xanthate	0·022

To Conditioner:

Pine oil Yarmor F	0·05
Conditioned 13·8 minutes.	

During Flotation:

Potassium amyl xanthate	0·073 to 1st cell
" "	0·12 to 3rd cell
" "	0·10 to 4th cell
Time of flotation	11·0 minutes
The pH of solution feeding to 1st cell was 8·75.	
To 1st pyrite cleaner cell—Pine oil Yarmor F	0·022

B. For Arsenopyrite:

To Pump:

Soda ash	3·8

To Conditioner:

Copper sulphate	1·92	Conditioned 14 minutes
Reagent 301	0·06	
Reagent 208	0·06	
Aerofloat No. 25	0·032	
Time of flotation	2·2 minutes	
The pH of solution feeding to arsenopyrite rougher cells was 9·7.		

Average specific gravity of ball mill discharge was 1,780 (62 per cent solids) and of classifier overflow (feed to pyrite flotation cells) was 1,220 (25·5 per cent solids).

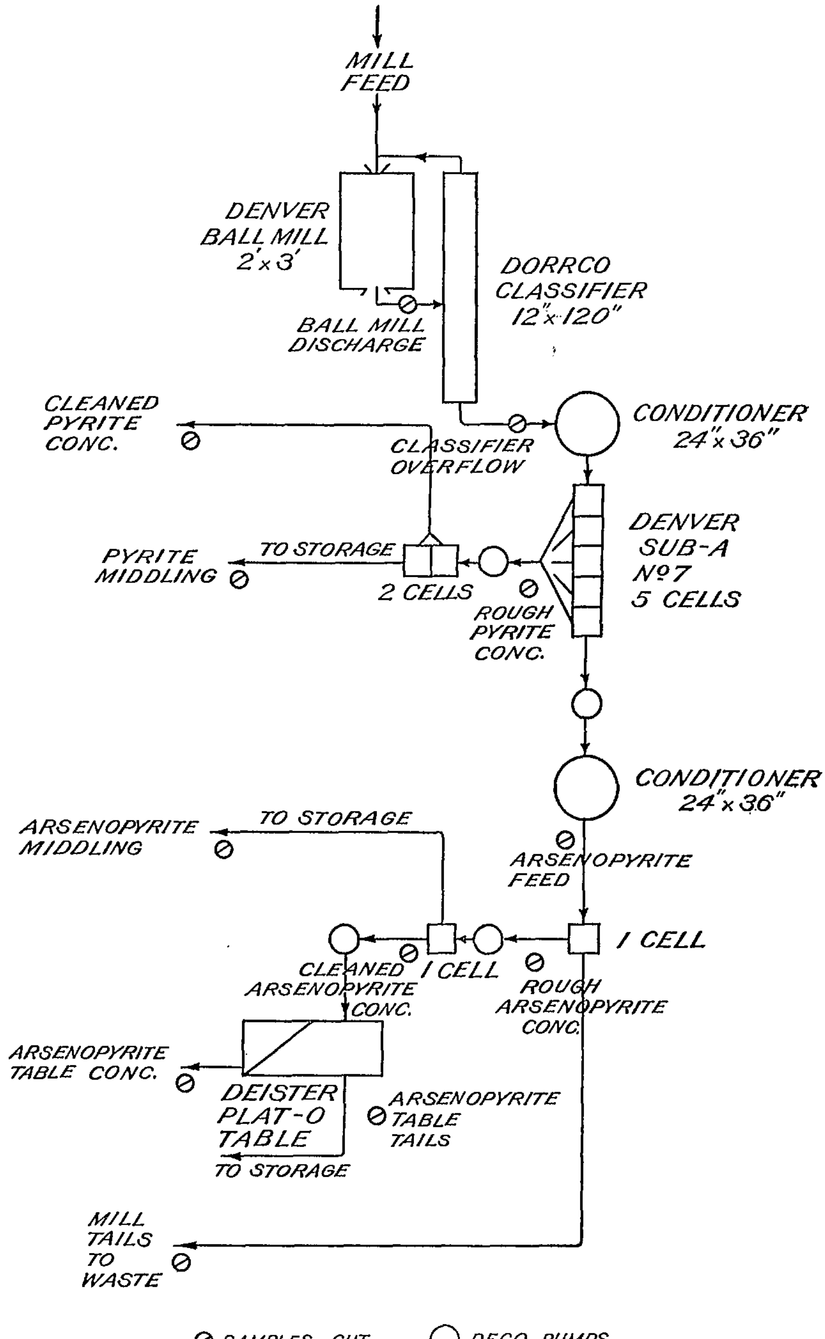

Figure 5. Flow-sheet of Mill Runs Nos. 7 and 8 on Sample 4.

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Results:

Ball mill discharge assayed:

Gold	0·485 oz./ton
Arsenic	3·37 per cent

Classifier overflow assayed:

Gold	0·27 oz./ton
Arsenic	1·95 per cent
Sulphur	10·00 "

The above analysis of the classifier overflow is equivalent to 4·2 per cent of arsenopyrite and 17·2 per cent of pyrite.

Results—Concluded

Product	Weight, per cent		Assay		Distribution of Au, per cent		Ratio of concentration	
	Section	Head	Au, oz./ton	As, per cent	Section	Feed	Section	Feed
Pyrite Rougher Section—								
Feed	100·0	100·0	0·27	1·95	100·0	100·0	....	....
Concentrate	18·0	18·0	0·93	2·24	62·0	62·0	5·55	5·55
Tailing	82·0	82·0	0·125	2·32	38·0	38·0	....	....
Pyrite Cleaner Section—								
Feed	100·0	18·0	0·93	2·24	100·0	62·0	....	....
Concentrate	81·9	14·75	1·00	1·40	88·2	54·7	1·22	6·8
Tailing	18·1	3·25	0·61	8·02	11·8	7·3	....	....
Arsenopyrite Rougher Section—								
Feed	100·0	82·0	0·125	2·32	100·0	38·0	....	....
Concentrate	4·7	3·86	1·54	29·00	58·0	22·0	21·2	25·9
Tailing	95·3	78·14	0·055	0·92	42·0	16·0	....	....
Arsenopyrite Cleaner Section—								
Feed	100·0	3·86	1·54	29·00	100·0	22·0	....	....
Concentrate	91·7	3·54	1·62	32·25	96·6	21·25	1·09	28·3
Tailing	8·3	0·32	0·64	11·80	3·4	0·75	....	....
Arsenopyrite Table Section—								
Feed	100·0	3·54	1·62	32·25	100·0	21·25	....	....
Concentrate	39·0	1·38	2·00	39·75	48·1	10·2	2·56	72·5
Tailing	61·0	2·16	1·38	27·44	51·9	11·05	....	....

This indicates that 1·38 per cent of the original ore can be obtained in the form of arsenopyrite concentrate, assaying 2·00 ounces of gold per ton and 39·75 per cent of arsenic (86·5 per cent of arsenopyrite).

Screen Analyses:

Mesh	Weight, per cent		
	Ball mill discharge	Classifier overflow	Mill tailing
+ 35	3·2	....	....
+ 48	4·8	0·1	....
+ 65	9·5	0·1	....
+100	20·2	1·9	1·5
+150	23·1	12·4	13·8
+200	17·3	18·6	15·9
−200	21·9	66·9	68·8
	100·0	100·0	100·0

The concentrate from each of the five pyrite rougher cells was sampled and assayed, with the following results:

Cell	Assay	
	Au, oz./ton	As, per cent
1	1·14	1·64
2	0·87	1·72
3	0·66	4·27
4	0·60	6·00
5	0·58	5·85

Miscellaneous Tests

A sample of 292 grammes of mill tailing from a mill run was panned on a Haultain superpanner. In this sample, which assayed 0·02 ounce of gold per ton, four very small pieces of free gold were visible on the panner.

A sample of the daily water used in the present Hard Rock mill was received on September 1, 1938. The pH of this water was 8·0.

Section B

CYANIDATION TESTS ON PRODUCTS FROM MILL RUNS

Cyanidation of Concentrates. In Tests Nos. 1 to 5 inclusive, the following products were cyanided:

Analysis of concentrate	Cleaned arsenopyrite concentrate, Mill Run No. 1	Cleaned pyrite concentrate, Mill Run No. 1	Cleaned bulk concentrate, Mill Run No. 6
Au, oz./ton	0·82	2·00	1·18
As, per cent	11·36	0·90	8·16
Fe, "	35·90	44·58	41·55
S, "	32·94	49·86	42·87
Insoluble, per cent	....	....	5·60

Test No. 1

Samples of raw concentrate were ground in a ball mill with lime to 99 per cent −325 mesh. The pulps were transferred to bottles and agitated for 24 hours. The pulp solutions were kept at about 2 pounds of sodium cyanide and less than 0·5 pound of lime per ton of solution. The ratio of dilution was 2 : 1.

Results of Cyanidation Tests:

Test No.	Product	Lime added to grind, lb./ton solids	Reagents consumed, lb./ton solids		Assay, Au, oz./ton		Extraction, per cent	
			NaCN	CaO	Feed	Cyanide tailing	—	Overall on mill feed
1 A	Arsenopyrite concentrate, Mill Run No. 1..........	30	5·4	43·5	0·82	0·595	27·44	11·14
1 B	Pyrite concentrate, Mill Run No. 1.............	30	2·6	53·5	2·00	0·165	91·75	49·54
1 C	Bulk concentrate, Mill Run No. 6.................	40	3·9	50·5	1·18	0·47	60·17	56·98

The analyses of the pregnant solutions were as follows:

—	A	B
Reducing power, ml. $\frac{N}{10}$ $KMnO_4$/litre..................................	804	492
KCNS, grm./litre..	0·78	0·58
Ferrous iron, grm./litre..	Trace	Nil
Arsenic, grm./litre...	Trace	0·002
NaCN, lb./ton solution..	1·94	2·28
Protective alkalinity as CaO, lb./ton solution............................	0·12	0·44
Total " " "	1·16	1·53
pH..	11·00	11·30

To determine the distribution of the gold in the cyanide tailing, infra-sizing analyses were made. The results are as follows:

Microns	Arsenopyrite concentrate, Mill Run No. 1			Pyrite concentrate, Mill Run No. 1			Bulk concentrate, Mill Run No. 6		
	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent
−10.....	45·0	0·405	29·63	32·8	0·14	26·15	36·3	0·34	26·28
+10−14.....	12·0	0·710	28·96	10·0	0·175	25·11	11·0	0·525	28·07
+14−20.....	13·1			15·2			14·1		
+20−28.....	13·0	0·86	32·85	17·2	0·195	35·36	15·9	0·56	35·30
+28−40.....	10·5			14·6			13·7		
+40−56.....	5·0	0·823	8·56	8·6	0·23	13·38	8·0	0·54	10·35
+56.........	1·4			1·6			1·0		
Feed........	100·0	0·615	100·00	100·0	0·176	100·00	100·0	0·47	100·00

Test No. 2

This was to determine whether a longer period of agitation would increase the gold extraction.

Samples of raw concentrate were ground in a ball mill with lime, to 99 per cent −325 mesh. The pulps were transferred to agitation bottles. Sodium cyanide was kept at about 2 pounds and the lime at less than 0·5 pound per ton of solution.

Arsenopyrite concentrate was repulped with fresh water every 24 hours.

Results:

Product	Agitation, hours	Lime added to grind, lb./ton of solids	Reagents consumed, lb./ton of solids		Assay, Au, oz./ton		Extraction, per cent	
			NaCN	CaO	Feed	Cyanide tailing	—	Overall on mill feed
Arsenopyrite concentrate, Mill Run No. 1	24	40	5·1	81·5	0·82	0·595	27·44	11·14
	48	40	8·3	124·5	....	0·58	29·27	11·88
	72	40	10·6	158·0	....	0·575	29·88	12·13
	72*	40	10·6	160·0	....	0·56	31·71	12·87
Pyrite concentrate, Mill Run No. 1	24	30	2·6	53·5	2·00	0·165	91·75	49·54
	43	30	2·8	63·5	....	0·17	91·50	49·41
	67	30	3·65	66·5	....	0·17	91·50	49·41
Bulk concentrate, Mill Run No. 6	24	40	2·8	57·5	1·18	0·475	59·75	56·58
	42	40	4·25	73·0	....	0·465	60·59	57·38
	66	40	5·6	78·0	....	0·46	61·02	57·79

*NOTE.—Aniline hydrochloride, 0·7 pound per ton of solids, was added to the last 24-hour cyanidation.

The results indicate that a maximum of gold extraction is obtained in less than 24 hours' cyanidation of the raw pyrite concentrate. Longer periods of cyanidation of the raw bulk concentrate and the arsenopyrite concentrate slightly increase the gold extractions.

Test No. 3

Amalgamation tests were carried out on the raw arsenopyrite concentrate and the pyrite concentrate.

Samples of raw concentrate, 1,000 grammes, were agitated at 1:1 dilution for 1½ hours in a ball mill jar with 4 balls, 7·5 millilitres of mercury. Thirty pounds of lime was added to the arsenopyrite concentrate pulp, and 24 pounds of lime to the pyrite concentrate.

Results:

Product	Assay, Au, oz./ton		Extraction of gold, per cent
	Feed	Amalgamation tailing	
Arsenopyrite concentrate, Mill Run No. 1	0·82	0·765	6·7
Pyrite concentrate, Mill Run No. 1	2·00	1·54	23·0

The amalgamation tailing was ground to 99 per cent −325 mesh and cyanided for 24 hours with fresh water.

Results of Cyanidation:

Product	Assay, Au, oz./ton		Extraction of gold, per cent		Extraction, Au, per cent	
	Amalgamation tailing	Cyanide tailing	By cyanidation	By amalgamation	—	Overall on mill feed
Arsenopyrite concentrate, Mill Run No. 1	0·765	0·60	21·57	6·7	26·83	10·89
Pyrite concentrate, Mill Run No. 1	1·54	0·165	89·29	23·0	91·75	49·55

Test No. 4

This was to determine the effect of litharge on gold extraction.

Samples of raw concentrate were ground with lime and cyanide to 99 per cent −325 mesh. The pulps were agitated for 24 hours at 2 : 1 dilution. Sodium cyanide and lime in the cyanidation solutions were kept at about 2 pounds and 0·5 pound per ton of solution, respectively.

Results:

Product	Reagents added to grind, lb./ton			Reagents consumed, lb./ton solids		Assay, Au, oz./ton		Extraction, Au, per cent	
	NaCN	CaO	PbO	NaCN	CaO	Feed	Cyanide tailing	—	Overall on mill feed
Pyrite concentrate, Mill Run No. 1	2·0	30	Nil	4·4	36·2	2·00	0·17	91·50	49·41
	2·0	30	0·10	4·2	35·6	2·00	0·175	91·25	49·28
	2·0	30	0·40	4·1	36·4	2·00	0·175	91·25	49·28
Bulk concentrate, Mill Run No. 6	2·0	34	nil	3·6	41·7	1·18	0·46	61·02	57·79
	2·0	34	0·10	3·8	42·7	1·18	0·475	59·75	56·58
	2·0	34	0·40	3·5	41·7	1·18	0·465	60·59	57·38

Litharge did not improve the extraction of gold.

Test No. 5—Cycle Tests

Samples of raw concentrate were ground with lime and cyanide to 99 per cent −325 mesh.

Half of the pregnant solution was made barren. It was aerated and used in the succeeding grind. The remaining half was aerated and used in the cyanidation test.

Litharge, 0·10 pound per ton of concentrate, was added to the grind. The pulp was agitated for 24 hours at 2 : 1 dilution.

Results of Cycle Tests:

Product	Cycle	Reagents consumed, lb./ton of solids		Assay, Au, oz./ton		Extraction, Au, per cent	
		NaCN	CaO	Feed	Cyanide tailing	—	Overall on mill feed
Pyrite concentrate, Mill Run No. 1	1st	3·35	49·5	2·00	0·17	91·50	49·41
	2nd	3·5	39·0	2·00	0·175	91·25	49·27
	3rd	3·8	38·5	2·00	0·17	91·50	49·41
	4th	3·9	39·5	2·00	0·18	91·00	49·14
	5th	3·45	38·0	2·00	0·175	91·25	49·27
Bulk concentrate, Mill Run No. 6	1st	3·9	50·5	1·18	0·47	60·17	56·98
	2nd	3·2	40	1·18	0·485	58·90	55·78
	3rd	3·6	39	1·18	0·465	60·59	57·38
	4th	3·3	40·5	1·18	0·47	60·17	56·98
	5th	4·0	42·0	1·18	0·48	59·32	56·18
Arsenopyrite concentrate, Mill Run No. 1	1st	4·9	60·0	0·82	0·605	26·22	10·64
	2nd	4·3	62·0	0·82	0·615	25·00	10·15
	3rd	5·2	54·0	0·82	0·605	26·22	10·64
	4th	5·4	55·0	0·82	0·615	25·00	10·15
	5th	4·6	55·0	0·82	0·615	25·00	10·15

The analyses of the pregnant solutions of the 5th cycle were as follows:

—	Pyrite concentrate	Bulk concentrate	Arsenopyrite concentrate
Reducing power, ml. $\frac{N}{10}$ $KMnO_4$/litre	1416	1992	2380
KCNS, grm./litre	1·55	1·94	2·53
Ferrous iron, grm./litre	0·07	0·05	0·05
Arsenic, "	Trace	Trace	0·002
NaCN, lb./ton solution	1·64	1·47	2·28
Protective alkalinity as CaO, lb./ton solution	0·18	0·10	0·20
Total " " "	1·12	1·54	1·60
pH	9·90	10·20	11·00

The KCNS is rather high. This would indicate that some of the old cyanide solution would have to be discharged to waste and fresh water be added to the circuit.

Test No. 6

This was to determine whether cyanidation of the pyrite concentrate and the arsenopyrite concentrate would give tailings assaying 0·17 and 0·60 ounce of gold, respectively, regardless of the gold in the concentrates.

Samples of raw concentrate were ground (99 per cent −325 mesh) with lime and sodium cyanide. The pulp was agitated 24 hours.

Pyrite concentrate from Mill Run No. 7 assayed 1·15 per cent of arsenic. The arsenopyrite concentrate from Mill Run No. 7 assayed 28·56 per cent of arsenic.

Results:

Product	Reagents consumed, lb./ton		Assay, Au, oz./ton		Assay, As per cent
	NaCN	CaO	Feed	Cyanide tailing	
Pyrite concentrate, Mill Run No. 1	3·35	49·5	2·00	0·17	0·90
" " Mill Run No. 7	2·0	29·5	0·985	0·175	1·15
Arsenopyrite concentrate, Mill Run No. 1	4·9	60·0	0·82	0·605	11·36
" " Mill Run No. 7	5·3	39·0	1·56	1·37	28·56

The results of the cyanidation tests on the pyrite concentrate ground to 99 per cent −325 mesh indicate that the cyanide tailing would be 0·17 ounce of gold per ton, regardless of the gold in the concentrate, provided the arsenic content in the concentrate is about 1 per cent. Increase of arsenopyrite in the concentrate will increase the gold in the cyanide tailing.

The arsenopyrite concentrate used in this test assayed 11·36 and 28·56 per cent of arsenic. This accounts for the difference in the gold in the cyanide tailings.

Cycle tests did not indicate any decrease in extraction after five cycles of 24 hours each during the cyanidation of the pyrite, bulk, or arsenopyrite concentrates.

GENERAL CONCLUSIONS, SECTIONS A AND B

The largest proportion of the gold in this ore is associated with the sulphides and is largely, if not wholly, sub-microscopic. The gold associated with the arsenopyrite appears to be in a finer state than that associated with pyrite. Infrasizing tests reveal that 29·6 per cent of the gold remaining in cyanide tailing from the arsenopyrite concentrate is −10 microns in size and that 30 per cent lies between 10 and 14 microns; 26 per cent of the gold remaining in cyanide tailing from the pyrite concentrate is −10 microns in size and 25 per cent lies between 10 and 14 microns. As extremely fine grinding fails to expose the surfaces of the gold particles, sodium cyanide or any other cyanide compound will not be able to dissolve the gold; the ore is, therefore, refractory to all forms of cyanidation. Some other process, such as roasting or smelting, is necessary to obtain the gold in a condition suitable for its recovery.

In the cyanidation of any concentrate containing an appreciable amount of arsenopyrite, milling difficulties may be expected owing to fouling of solutions. More efficient milling operation may, therefore, be expected from cyanidation of a pyrite concentrate than from cyanidation of a bulk concentrate. If a bulk concentrate be cyanided, 91 tons of cyanide tailing, assaying 0·47 ounce per ton in gold and requiring re-treatment at a later date, will accumulate daily from a 300-ton mill feed. The possibility of making a separation between pyrite and arsenopyrite in this tailing would be very slight, but the arsenopyrite would be placed in good condition for future metallurigcal treatment by making a selective float and stockpiling it separately.

Overgrinding of the sulphides occurs in the grinding-classifying circuit, and is unlikely to be remedied by placing a unit flotation cell in this circuit. The ball mill discharges at about 29 per cent −200 mesh and a separation between pyrite and arsenopyrite could not be accomplished by flotation at such a coarse grind. The only possibility of decreasing this overgrinding appears to be as follows:

If the mineralization structure be such that the pyrite is liberated at a grind of 50 per cent −200 mesh, for example, a very rough arsenopyrite concentrate might be removed after flotation of the pyrite, the arsenopyrite middling from the arsenopyrite cleaner cells being reground and returned to the feed of the rough arsenopyrite flotation circuit. It is doubtful, however, whether a satisfactory separation can be made between pyrite and arsenopyrite at a 50 per cent −200-mesh grind.

The separation by flotation of pyrite and arsenopyrite depends largely upon a difference in pH in the solutions during the two floats. Test work has indicated that the most efficient separation can be obtained when the pH of the pulp feeding to the pyrite cells is 8·3. No difficulty in obtaining this pH is expected with the water at the mine, which has a pH of 8·0.

Section C

ROASTING AND CYANIDATION TESTS

Purpose of Investigation. A large part of the gold in the ore was known to be refractory to ordinary methods of cyanide leaching, irrespective of whether the ore is treated direct or the gold is concentrated in pyrite or arsenopyrite products and then treated by cyanidation.

In this investigation a process of roasting and cyanidation of the calcine was sought by which the refractoriness could be overcome and a higher extraction of the gold obtained. This was requested by Mr. J. C. Dumbrille, Manager, Hard Rock Gold Mines, Limited, Geraldton, Ontario.

Products Used in Investigation. The concentrate used was produced in the laboratory investigation carried out by a representative of the company in co-operation with engineers of this Department and previously reported. The ore used was similar.

The arsenopyrite concentrate was representative of flotation tests, Mill Runs Nos. 5 and 9, and the bulk concentrate used represented concentrate made by an engineer of the Department and flotation concentrate Mill Run No. 10.

Summary of Results of Straight Cyanidation. These products were subjected to straight cyanidation and the results illustrate the refractoriness of the gold.

Arsenopyrite Concentrate Mill Run No. 5—Au, 0·83 oz./ton.

Concentrate ground 99 per cent −325 mesh.

Screen Analysis:

Mesh, microns	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent
−56+40	6·4	0·823	8·56
−40+20	23·5	0·86	32·85
−20+10	25·1	0·71	28·96
−10	45·0	0·405	29·63

Cyanidation:

Agitation, hours	Reagents consumed, lb./ton		Tailing, Au, oz./ton	Extraction, per cent
	NaCN	CaO		
24	5·74	41·2	0·62	24·39
24	5·42	43·3	0·595	27·44

Arsenopyrite Concentrate Mill Run No. 9—Au, 1·33 oz./ton.

Similarly ground

Cyanidation:

Agitation, hours	Reagents consumed, lb./ton		Tailing, Au, oz./ton	Extraction, per cent
	NaCN	CaO		
24	5·64	41·7	1·08	19·4
	5·46	53·8	1·12	16·4

Bulk Concentrate Mill Run No. 10—Au, 1·18 oz./ton.

Screen Analysis:

Mesh, microns	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent
−56+40	9·0	0·54	10·35
−40+20	29·6	0·56	35·30
−20+10	25·1	0·525	28·07
−10	36·3	0·34	26·28

Cyanidation:

Agitation, hours	Reagents consumed, lb./ton		Tailing, Au, oz./ton	Extraction, per cent
	NaCN	CaO		
24	2·8	57·6	0·475	59·75
42	4·25	73·0	0·465	60·59
66	5·60	78·0	0·46	61·02

Description of Investigative Procedure. The concentrate or ore was roasted in fireclay trays in a globar electric, muffle-type furnace controlled in temperature. An opening in the rear of the muffle connected with a pipe and exhaust fan permitted the fumes to be drawn from the furnace. The exhaust was operated in such a manner that a minimum of fumes escaped from the front of the furnace.

Experience, checked by preliminary test work, indicated that slow roasting was more favourable than fast for attaining the maximum of extraction by cyanide leaching of the calcine. Generally speaking, charges were placed in the cold furnace and roasting started at 300-350° C., this temperature being maintained for ½ to 1 hour, then the temperature was raised in 50-degree steps each ½ hour or 1 hour and finished at 600-850° C. Roasting at finishing temperatures below 600° C. resulted in definitely lower extractions.

Rabbling of the charge was performed about every 10 to 15 minutes.

Conditions were varied as to rate of heating, temperature, time, etc., to determine the best procedure in producing a calcine to give the highest extraction by cyanidation. In cyanidation of calcine, some variations in time of agitation, alkalinity, fineness of grind, etc., were tried.

Analyses of Products Used:

Sample Arsenopyrite Concentrate Mill Run No. 5:

Gold, oz./ton	Arsenic, per cent	Sulphur, per cent
0·83	13·44	28·92

Sample Arsenopyrite Concentrate Mill Run No. 9:

Gold, oz./ton	Arsenic, per cent	Sulphur, per cent	Calcium oxide, per cent	Magnesia, per cent
1·33	22·34	27·97	0·37	0·31

Sample Bulk Concentrate (small test):

Analysis not made.

Sample Bulk Concentrate Mill Run No. 10:

Gold, oz./ton	Arsenic, per cent	Sulphur, per cent
1·18	7·68	Not determined.

Sample Ore:

Gold, oz./ton	Arsenic, per cent	Sulphur, per cent	Iron, per cent	Alumina, per cent	Silica, per cent	Copper, per cent	Lead, per cent
0·33	2·45	11·95	22·80	8·38	40·45	0·04	0·02

Experimental Tests

INVESTIGATIVE WORK ON ARSENOPYRITE CONCENTRATE

The following is a summary of the results obtained on roasting and cyaniding the calcine of the arsenopyrite concentrate.

In all tests except No. 20, fireclay trays 11¾ x 6⅝ x 1⅝ inches (I.D.) were used. In Test No. 20 a tray measuring 16 x 8½ x 2 inches was used. Tests Nos. 3 to 12 were conducted on arsenopyrite concentrate Mill Run No. 5, gold, 0·83 ounce per ton. Tests Nos. 17 to 46 were conducted on Arsenopyrite Concentrate Mill Run No. 9, gold, 1·33 ounces per ton.

TABLE I

Test No.	Roasting time		Assay, Au, oz./ton		Extraction, per cent	Agitation time, hours	Reagents, consumed, lb./ton ore		Cyanide Test No.
	Total time, hours	Final temp., °C.	Calcine	Tailing			NaCN	CaO	
3	9	1 hr. 850	1·20	0·315	73·8	24	0·38	9·4	3–1
	....	....	....	0·295	75·4	48	0·70	11·1	3–2
	....	....	....	0·30	75·0	72	0·60	14·6	3–3
4	8	1 hr. 850	1·17	0·225	80·8	48	0·60	11·9	4–1
5	8	1 hr. 850	1·18	0·25	78·8	24	0·37	7·0	5–1
6	7	2 hr. 600	1·18	0·375	68·2	48	4·04	25·2	6–1
7	7	2 hr. 600	1·17	0·34	70·9	24	2·53	15·7	7–1
12	8	1¼ hr. 750	1·17	0·295	74·8	48	0·35	5·2	12–2
17	8	1 hr. 850	2·04	0·475	76·7	48	3·60	3·2	17–1
	....	....	....	0·445	78·2	48	2·80	11·0	17–3
18A	10½	3 hr. 800	2·00	0·32	84·0	24	0·30	8·4	18A–1
	....	....	....	0·32	84·0	24	3·0	8·0	18A–2
18B	....	....	1·96	0·34	82·5	48	1·69	8·4	18B–2
	....	....	1·96	0·325	83·8	48	2·31	10·0	18–7
20	10½	2 hr. 700	2·01	0·38	81·1	48	1·10	7·1	20–2
22	10½	2 hr. 750	2·01	0·39	80·7	48	0·88	8·4	22–1
	....	....	....	0·33	83·6	48	0·18	4·2	22–5
24A	10	3 hr. 750	2·06	0·30	85·4	48	0·88	10·0	24–A
24B	10	3 hr. 750	2·04	0·31	84·8	48	0·60	10·0	24–B
26	10	2 hr. 700	2·06	0·46	77·7	24	0·24	15·0	26–1
27	10½	3 hr. 700	2·00	0·32	84·0	48	1·64	11·0	27–1
29	9	1 hr. 800	2·02	0·40	80·2	48	1·08	8·4	29–1
31	9	3 hr. 800	2·01	0·39	80·6	48	0·52	4·9	31–1
	....	....	....	0·37	81·6	48	0·64	19·7	31–2
37	7	1 hr. 800	1·96	0·44	77·6	48	0·48	6·7	37–1
39	7	1 hr. 800	2·05	0·40	81·4	48	0·60	11·9	39–1
	....	....	....	0·38	81·4	48	1·24	11·9	39–2
40	13	7hr. 450	2·02	0·61	69·8	48	2·88	16·0	40–4
41	Similar to 40—Discarded.								
43	14	6 hr. 550	1·94	0·63	67·5	48	2·24	25·2	43–1
44	15	6 hr. 650	1·97	0·46	76·7	48	1·50	15·0	44–1
45	8	2 hr. 450	1·95	0·56	71·3	48	2·50	22·0	45–1
46	8	1 hr. 550	1·93	0·72	62·7	48	1·24	8·0	46–1

The above table shows the results of twenty-three roasting tests on which a total of fifty-six cyanidation tests were conducted.

The conditions and variations in the tests were, briefly:

Unless otherwise stated the calcines were treated as follows: Ground to 95 to 97 per cent −325 mesh, made up to 6:1, bottle-agitated 3 hours, filtered, and washed. Portions were then agitated in lime cyanide solution containing 2 pounds of sodium cyanide and about 0·2 pound of lime per ton of solution. During the first 5 or 6 hours of the cyaniding the lime was consumed rather rapidly and the ratio was kept higher than 0·2 pound per ton. In Tests Nos. 3, 4, 5, 6, 8, and 12 when the agitation in cyanide solution was continued for longer than 24 hours the solution was changed at the end of half the total interval. In the other tests fresh solution was not added.

Test No. 3. Two charges of 1,000 grammes each. Temperature held 2 hours at 350° C., 1 hour at 450° C.; heat turned off ½ hour, then raised to 850° C. and held 1 hour. Door of furnace opened 2 inches to allow restricted air draught. Calcine ground to 95 per cent −325 mesh, bottle-agitated 3 hours 6:1, filtered, and washed. Sample portions were agitated in cyanide 24, 48, and 72 hours.

Test No. 4. One charge of 1,000 grammes. Roasting conditions similar to Test No. 3 except that furnace door was opened 7 inches to allow a greater supply of air.

Test No. 5. One charge of 1,000 grammes. Same as Test No. 4 except in length of time of agitation in cyanide solution.

Test No. 6. One charge of 1,000 grammes. Two tests were made in which 2 per cent of sodium chloride was added to the concentrate to determine its influence on the calcine. Temperature held 2 hours at 350° C., then brought to 600° C.

Test No. 7. One charge of 1,000 grammes. Same as Test No. 6 except that calcine was cyanided 24 hours.

Test No. 12. Two charges of 1,000 grammes each. Similar to Test No. 4 except that final temperature was 750° C.

Test No. 17. Two charges of 1,500 grammes each. Similar to Test No. 4. In Cyanide Test No. 17-1, solution of 5 pounds of sodium cyanide per ton was used; in Cyanide Test No. 17-3 the calcine was cyanided directly after grinding without the usual preliminary wash treatment. In this and the following tests Arsenopyrite Concentrate Mill Run No. 9 (Au, 1·33 oz./ton) was used.

Test No. 18. Two charges of 1,200 grammes each. Temperature held 2 hours at 300° C., 1½ hours at 350° C., ½ hour at 400° C., ½ hour at 450° C., ½ hour at 500° C., ½ hour at 550° C., ½ hour at 600° C., then raised to 800° C. Calcine in dish nearer furnace door (Test No. 18A) showed greater loss than calcine at rear of furnace (Test No. 18B). Cyanide Test No. 18-7 was made by cyaniding after grinding without the preliminary wash treatment. In Cyanide Test No. 18A-2, solution of 5 pounds of sodium cyanide per ton was used.

Test No.	Ignition loss, per cent	Total S	As	Extraction, per cent
18A	35·6	0·26	3·84	84·0
18B	33·3	0·22	5·80	82·5
18A-2	Cyanided in solution containing 5 pounds of sodium cyanide per ton.			

Test No. 20. One charge of 700 grammes. This test was made to determine the effect of roasting concentrate in shallow layers; the charge was about one-eighth inch deep. The temperature was brought up gradually to 750° C. The charge was not rabbled. The results of an infrasizing test on the cyanide tailings are also given.

Test No. 22. Two charges of 1,200 grammes each. Temperature held for 1½ hours at 350° C., 1 hour at 400° C., 1 hour at 450° C., finally for 2 hours at 750° C. Routine cyanidation gave an extraction of 80·7 per cent. By agitating the calcine 45 minutes with hot solution of 15 per cent caustic soda, filtering and cyaniding, an extraction of 83·6 per cent was obtained (Cyanide Test No. 22-5).

Test No. 24. Two charges of 500 grammes each. Temperature held 1 hour at 300° C., 1 hour at 350° C., increased in 50-degree jumps, held ½ hour at each range. The ignition loss, 36·2 per cent, was higher than usual.

Test No. 26. One charge of 600 grammes. Previous to roasting, the concentrate was treated with 5 per cent acetic acid to dissolve the calcium carbonate and prevent the possible formation of interfering calcium compounds during calcination.

Test No. 27. Two lots of 500 grammes each. In this test 5 per cent coke was added to the concentrate with an idea of obtaining more complete removal of the arsenic. No marked removal resulted—As, 3·0 per cent.

Test No. 29. One lot of 1,000 grammes. Temperature brought up gradually to 800° C.

Test No. 31. One lot of 1,200 grammes. Temperature held 1 hour at 300° C., 1 hour at 350° C., 1 hour at 400° C., 1 hour at 450° C., ½ houir at 550° C., 3 hours at 800° C. Cyanide Test No. 31-2 was cyanided in a l me-cyanide solution saturated with lime and containing 2 pounds of sodium cyanide per ton. Cyanide Test No. 31-1 was cyanided with low lime.

Test No. 37. One lot of 1,000 grammes. Similar to Test No. 27 except that the final temperature was 800° C.

Test No. 39. One lot of 1,000 grammes. Temperature brought up gradually to 800° C. In Cyanide Test No. 39-1 the calcine was ground 30 minutes to give 95 per cent −325 mesh, whereas in Cyanide Test No. 39-2 the calcine was ground 2 hours.

Test No. 40. Two lots of 750 grammes each. Tests were made to determine the effect of roasting at temperatures less than 700° C. The calcines were aerated for at least 9 hours, filtered, and cyanided. In Test No. 40 the temperature was held at 300°, 350°, and 400° C. each for 2 hours, then at 450° C. for 7 hours.

Test No. 41. Two lots of 1,000 grammes each. Similar to Test No. 40. Discarded.

Test No. 43. Two lots of 750 grammes each. Similar to Test No. 40 except that final temperature was 550° C.

Test No. 44. Two lots of 750 grammes each. Similar to Test No. 40 except that final temperature was 650° C.

Test No. 45. Two lots of 750 grammes each. Test to determine the effect of roasting for a short time at 450° C. in contrast to Test No. 40.

Test No. 46. Similar to Test No. 45 except that final temperature was 550° C.

Typical Calcine Analyses:

Test No.	Arsenic, per cent	Sulphur, per cent	Ferrous iron, per cent	Water soluble sulphur, per cent	Alkali soluble sulphur, per cent	Alkali soluble arsenic, per cent
3	1·92	0·27	Nil	....	....	....
4	1·84	0·25	Nil	....	....	....
18A	3·84	0·26	0·15	0·09	0·14	0·92
18B	5·80	0·22	0·22	0·07	0·07	1·00
24A	3·07	0·20	Nil	0·17	0·19	0·92
24B	3·46	0·17	Nil	0·15	0·17	0·69

The alkali-soluble sulphur and arsenic were determined by treating the calcine with a 10 per cent solution of sodium carbonate and determining the arsenic and sulphur in the soluble portion.

The lower arsenic content in Tests Nos. 3 and 4 is probably due to the fact that this sample analysed only 13·44 per cent arsenic and 28·92 per cent sulphur, whereas in Tests Nos. 18 and 24 the concentrates analysed 22·34 per cent arsenic and 27·97 per cent sulphur. The excess sulphur in the concentrates from Tests Nos. 3 and 4 probably assisted in removing the arsenic during roasting, whereas in Tests Nos. 18 and 24 the tendency would be to the formation of more ferric arsenate.

Examination of Calcine Cyanide Tailing by Infrasizing

The cyanide tailing from Test No. 20 (Au, 0·38 oz./ton) was infrasized in the Haultain infrasizer, with the following results:

Mesh, microns	Weight, per cent	Assay, Au, oz./ton	Units	Distribution, per cent
+40	2·07	0·44	0·91	2·41
−40+20	13·23	0·60	7·94	21·02
−20+10	21·91	0·46	10·08	26·69
−10	62·79	0·30	18·83	49·88
Totals	100·00	0·38	37·76	100·00

Interesting in this test is that about 63 per cent of the calcine tailing is −10 microns in size and carries 50 per cent of the total gold content of the sample. This illustrates the extremely fine character of the gold, and the difficulty in extracting it by cyanidation.

Superpanning, on the Haultain superpanner, of calcines and calcine cyanide tailings failed to reveal any particles of free gold.

The extreme fineness of the gold in the calcined concentrate is also shown by the results of the following chemical test:

Test to Determine the Form of Gold in Arsenopyrite Concentrate

A sample of twenty-five grammes of the calcine from arsenopyrite concentrate (24-A retains) prepared from ore from Hard Rock Gold Mines, Limited, was leached with hydrochloric acid. A calculated quantity of chloride of tin was used to keep the iron reduced and prevent solution of the gold. The residue from this treatment would be expected to contain the gold and the insoluble silicates, the other metallic constituents having been removed. After the residue was filtered and thoroughly washed, it was almost white but showed a faint purplish tint. This residue, designated as "bulk residue", was analysed spectrographically.

Part of the bulk residue was dispersed in distilled water and the fine was decanted, this process being repeated several times. The result was a fraction containing the sand, designated as "coarse fraction", and a liquid containing the fine material, some of which remained in suspension causing the liquid to appear turbid with a faint purplish cast. The fine was dispersed and allowed to settle for five minutes, after which the liquid was decanted and filtered. The fraction that settled out here was designated as "fine fraction". When dried the coarse fraction was white and the fine fraction possessed a faint purplish tint.

The filtrate was evaporated. During this procedure a distinct ring was observed. This ring was typically similar to the gold ring produced when treating gold solutions in this manner. The dry residue from the filtrate was dissolved in a small quantity of aqua regia, evaporated carefully, and taken up with a small quantity of distilled water and analysed spectrographically.

The results of the spectrographic analyses of the four fractions are shown in the following table:

Element	Bulk residue	Coarse residue	Fine residue	Filtrate
Au	Present	Nil	Strong	Present
Cu	Present	Present	Present	Nil
As	Present	Faint trace	Strong	Present
Sn (added) (as $SNCl_2$, and probably carried through because some of the tin forms insoluble tin oxide).	Strong	Strong	Strong	Very faint
Fe	Strong	Strong	Faint trace	Nil

These indicate that a considerable proportion of the gold in the bulk residue was so finely divided that it passed into the filtrate, and that very little or no gold was retained in the coarse silicate residue. This, coupled with the purplish colour noted at various points in the test, indicates very definitely that part of the gold is colloidal in form. Assuming that the form of the gold was not altered by roasting and leaching of the roasted product, then it seems probable that an important proportion of the gold in the sulphides at the mine of the Hard Rock Gold Mines, Limited, is colloidal in character. There is no evidence as to whether or not any gold occurs also in solid solution in the sulphides.

SUMMARY OF INVESTIGATIVE WORK ON ARSENOPYRITE CONCENTRATE

It would appear that the maximum extraction of gold content obtainable is in the neighbourhood of 84 to 85 per cent. (Tests Nos. 18 and 24).

The favourable roasting conditions are apparently as follows:

Free admission of air while arsenic and sulphur are being driven off. Holding initial roasting temperature at 300 to 350° C. for a period of 2 to 2½ hours with a gradual increase in temperature over the next three or four hours and finishing at 750 to 800° C. for three hours.

In cyaniding the calcine it would appear preferable to grind in water to −325 mesh, agitate in water for several hours to remove soluble salts, filter or thicken, and repulp in solution (at 2 : 1 or 3 : 1 ratio) of 2 pounds of cyanide with 2 or more pounds of lime to start with for neutralization, then maintaining lime alkalinity about ½ pound per ton in agitation for 48 hours.

Cyanidation, without pre-wash treatment, gives approximately a similar extraction but results in higher chemical consumption. Higher lime, cyanide, or longer agitation apparently do not contribute to a lower tailing.

The extreme fineness of the gold particles in the calcine, as shown by tests embracing infrasizing, superpanning, and chemical separation, leads to the conclusion that no further extraction can be hoped for by ordinary hydrometallurgical means.

INVESTIGATIVE WORK ON BULK CONCENTRATE

The procedure followed was essentially the same as observed in the arsenopyrite investigative work.

The results obtained by roasting and cyaniding the calcines are shown in the following table:

TABLE II

Test No.	Roasting time		Assay, Au, oz./ton		Extraction, per cent	Agitation time, hours	Reagents consumed, lb./ton ore		Cyanide Test No.
	Total time, hours	Final temp., °C.	Calcine	Tailing			NaCN	CaO	
1	7	1 hr. 600	1·58	0·345	78·1	20	4·48	6·9	1–1
2	7	1 hr. 600	1·48	0·29	80·4	20	4·40	2·0	2–1
21	8	1½ hr. 650	1·74	0·46	73·6	48	1·32	14·0	21–1
	....	....	....	0·34	80·5	90	0·84	14·0	21–2
23	9	2½ hr. 650	1·76	0·26	85·2	72	1·02	10·0	23–2
25A	9	2 hr. 650	1·72	0·23	86·6	96	1·80	15·0	25A
25B	9	2 hr. 650	1·74	0·26	85·1	96	2·28	15·0	25B
28	10	2 hr. 650	1·745	0·22	87·4	72	0·60	11·2	28–1
30	9	1 hr. 800	1·79	0·23	87·2	72	0·96	8·4	30–1
32	9	3 hr. 800	1·80	0·295	83·6	72	1·36	16·8	32–1
	....	....	....	0·29	83·9	72	1·12	32·3	32–2

Remarks:

Test No. 1. Charge of 540 grammes of bulk concentrate from a small-scale concentration test. Temperature raised gradually to 370° C., held

1 hour, then 470° C., held 1 hour, then 600° C. and held 1 hour. Calcine was ground to 97 per cent −325 mesh, agitated in lime solution until the solution remained alkaline, 20 minutes, filtered, and agitated 20 hours in cyanide solution containing 2 pounds of sodium cyanide and 1 pound of lime per ton of solution.

Test No. 2. Similar to Test No. 1.

Test No. 21. In Test No. 21 and the following tests Bulk Concentrate Mill Run No. 10 (Au, 1·18 oz./ton) was used. Two charges of 1,000 grammes each. Temperature held 1 hour at 350° C., 1 hour at 450° C., 1 hour at 550° C., 1 hour at 600° C., and 1½ hours at 650° C. After water agitation portions were agitated in cyanide for 48 and 90 hours.

Test No. 23. Two charges of 500 grammes each. Similar temperature range as in Test No. 21 except final temperature of 650° C. held for 2½ hours. Ignition loss 34·4 per cent.

Test No. 25. Two lots of 500 grammes each. Temperature held ½ hour at 300° C., 2 hours at 350° C., 1 hour at 400° C., and finally 2 hours at 650° C. Test No. 25A calcined near furnace front gave a better extraction than Test No. 25B roasted in rear of furnace.

Test No. 28. Two lots of 500 grammes each; 5 per cent coke added to the concentrate. Held 2 hours at 300° C., 1 hour at 350° C., 1 hour at 400° C., ½ hour at 500° C., and finally 2 hours at 650° C.

Test No. 30. One lot of 1,000 grammes. Previous to roasting, the ore was treated with 5 per cent acetic acid to dissolve the calcium carbonate. Temperature held 2 hours at 300° C., 1½ hours at 350° C., then raised gradually to 800° C.

Test No. 32. One lot of 1,200 grammes. Temperature held 1 hour at 300° C., 1 hour at 350° C., 1 hour at 400° C., 1 hour at 450° C., ½ hour at 500° C., and finally 3 hours at 800° C.

The calcines were ground 95 to 97 per cent −325 mesh, pulped 6:1, bottle-agitated 3 hours, then filtered and washed. Portions of washed calcine were then agitated in lime-cyanide solution containing 2 pounds of sodium cyanide and 0·2 pound of lime per ton.

Nineteen cyanide tests were made on the eight roast tests. The most favourable extractions only, on each calcine, are shown in the table of results.

Variations in lime and cyanide strength of leaching solutions apparently had no effect.

Long period agitation tends toward obtaining increased extraction (Test No. 21).

Typical Analysis of Bulk Concentrate Calcines:

Test No.	Arsenic, per cent	Sulphur, total per cent	Ferrous iron, per cent	Water soluble sulphur, per cent	Alkali soluble	
					Arsenic, per cent	Sulphur, per cent
23	1·19	0·43	Nil	0·40	0·40	0·43
25A	2·04	0·42	Nil	0·42	0·69	0·42
25B	1·27	0·43	Nil	0·42	0·46	0·43

SUMMARY OF INVESTIGATIVE WORK ON BULK CONCENTRATE

The maximum of gold extraction obtainable from roasting and cyaniding the calcine of the bulk concentrate would appear to be in the neighbourhood of 85 to 87 per cent.

The favourable roasting conditions are apparently as follows:

Free access of air in early stages of roast, to assist removal of arsenic and sulphur; holding initial roasting temperatures to around 300 to 350° C. for the first hour or two, then gradually increasing temperature over a period of several hours to a maximum of 650° C. and holding that temperature for about 2 hours. Carrying the final temperature to 800° C. or above indicates reduced extraction.

In cyanidation, grinding and washing the calcine to remove soluble salts prior to cyanidation is beneficial to extraction and chemical consumption.

The conditions generally are similar to those indicated for treatment of the arsenopyrite concentrate.

Higher extractions appear unattainable owing to the extreme fineness of the gold in the sulphides.

ROASTING AND CYANIDING CALCINE OF RAW ORE

Samples of the ore were ground to different degrees of fineness and subjected to roasting under conditions similar to those employed in roasting the arsenopyrite and bulk concentrates.

Eight roasting tests were conducted, with a total of twenty-four cyanide tests being run on the calcines.

A summary of the results obtained by this procedure is shown in the following table:

TABLE III

Test No.	Roasting time		Assay, Au, oz./ton		Extraction, per cent	Agitation time, hours	Reagents consumed, lb./ton ore		Cyanide Test No.
	Total time, hours	Final temp., °C.	Calcine	Tailing			NaCN	CaO	
8	9	1 hr. 750	0·375	0·055	85·3	24	0·69	10·9	8–1
				0·045	88·0	48	1·08	15·8	8–2
9	9	1 hr. 750	0·435	0·04	90·8	24	0·55	6·9	9–1
10	9	1 hr. 750	0·40	0·05	87·9	24	0·31	4·0	10–1
	....	....	....	0·05	87·9	48	0·91	15·0	10–2
11	9	1 hr. 750	0·415	0·065	84·3	24	0·67	6·0	11–1
	....	....	....	0·055	86·7	48	0·98	10·0	11–2
13	9	1 hr. 750	0·455	0·06	86·8	48	0·98	6·0	13–1
	....	....	....	0·06	86·8	62	1·00	6·0	13–3
15	9	1 hr. 750	0·375	0·05	86·7	48	6·5	13·6	15–1
	....	....	....	0·055	85·3	48	5·0	Nil	15–3
	....	....	....	0·045	88·0	48	2·0	Nil	15–5
16	8	1½ hr. 650	0·37	0·045	87·8	48	3·2	Nil	16–2
19	7	1 hr. 700	0·37	0·05	86·5	48	2·5	14·0	19–1

Remarks:

Test No. 8. Size of material:

$+\frac{1}{4}$ inch	1·6 per cent
$-\frac{1}{4}$ in. + 10 mesh	72·5 "
−10 mesh +100 mesh	20·8 "
−100 mesh	5·1 "
	100·0 "

One charge of 1,500 grammes. Temperature held 2 hours at 350° C., 1 hour at 400° C., 1 hour at 450° C., and 1 hour at 750° C. Calcine ground to 99 per cent −325 mesh, made up 6 : 1, bottle-agitated 2 hours, filtered, and washed. Portions were agitated in cyanide solution containing 2 pounds of sodium cyanide and 0·5 pound of lime per ton of solution for 24 and 48 hours, 3 : 1 dilution.

In Tests Nos. 8, 9, 10, 11, and 13, when the agitation in cyanide solution was continued longer than 24 hours the solution was changed at the end of half the total interval.

Test No. 9. Size of material:

+ 28 mesh	37·7 per cent
− 28+ 35 "	17·1 "
− 35+100 "	22·9 "
−100 "	22·3 "
	100·0 "

One charge of 1,000 grammes. Temperature held 1 hour at 350° C., 1 hour at 450° C., and 1 hour at 750° C. Cyanidation similar to Test No. 8.

Test No. 10. Size of material:

+ 65 mesh	24·7 per cent
− 65+100 "	13·9 "
−100 "	61·4 "

Two charges of 950 grammes each. Temperature held 2 hours at 350° C., 1 hour at 450° C., 1 hour at 750° C. Cyanidation similar to Test No. 8.

Test No. 11. Size of material:

+150 mesh	24·3 per cent
−150 "	75·7 "
	100·0 "

Two charges of 950 grammes each. Roast conditions same as Test No. 10 The calcines were agitated in cyanide solution containing 1 pound of sodium cyanide per ton of solution, 2:1 dilution.

Test No. 13. Size of material similar to Test No. 9. Two charges of 1,000 grammes each. Roast conditions same as Test No. 10. Calcines agitated in cyanide solution containing 1 pound of sodium cyanide per ton of solution.

Test No. 15. Size of material similar to Test No. 9. Two charges of 1,500 grammes each. Roasting same as Test No. 10.

In Cyanide Test No. 15-1, sodium cyanide and lime added to ball mill.

In Cyanide Test No. 15-3, sodium cyanide only added to ball mill. No lime used in test.

In Cyanide Test No. 15-5, sample ground; made up 2 pounds of sodium cyanide per ton solution added; no lime used.

Test No. 16. Size of material similar to Test No. 9. Two charges of 1,500 grammes each. Temperature held 1 hour at 350° C., 1 hour at 450° C., 1½ hours at 650° C. The calcines were ground in cyanide solution and cyanided without adding lime.

Test No. 19. Size of material similar to Test No. 9. Two charges of 1,500 grammes each. Temperature held 1 hour at 300° C., 1 hour at 400° C., 1 hour at 500° C., and 1 hour at 700° C. Calcine cyanided directly after grinding without the preliminary wash treatment.

Typical Analysis of Ore Calcine:

Test No.	Sulphur, per cent	Arsenic, per cent
8	1·11	0·42
10	1·12	0·63

SUMMARY OF INVESTIGATIVE WORK ON ROASTING AND CYANIDING CALCINE OF RAW ORE

The maximum gold extraction obtainable by roasting the ore and cyaniding the calcine is in the neighbourhood of 88 to 90 per cent.

Roasting conditions for this extraction are comparable mainly with those observed in treatment of the concentrates, namely, a long period roast of about 8 hours with low initial temperature to remove arsenic and sulphur, with gradual increasing temperature to a maximum of 700 to 750° C.

In roasting, comparatively coarse grinding appears to give results quite as satisfactory as fine grinding. Cyaniding for 48 hours appears sufficient for maximum of extraction.

Water leaching of calcine prior to cyanide leaching tends to give slightly better extraction and shows less chemical consumption.

III

INVESTIGATIONS THE DETAILS OF WHICH ARE NOT PUBLISHED

Ore or Product	Source of Shipment	Address
Gold-silver-lead	Carleton District	Yarmouth County, N.S.
Wood tar distillates	Forest Products Laboratories	Ottawa, Ont.
Chrome-nickel-magnetite	Canadian Johns-Manville Company, Limited.	Asbestos, Que.
Copper-gold	"Quatsino King" Mineral Claim	Quatsino Sound, B.C.
Concentrate and gold ore	Hard Rock Gold Mines, Limited	Geraldton, Ont.
Gold	Malartic Goldfields, Limited	Norrie, Que.
Gold	Jellicoe Consolidated Gold Mines, Limited.	Geraldton, Ont.
Silver-lead	Jewel Consolidated Mines, Limited	Ainsworth, B.C.
Gold	Claverny Gold Mines, Limited	Amos, Que.
Gold	Hasaga Gold Mines, Limited	Red Lake, Ont.
Pyrite	Northern Pyrites, Limited	Skeena Mining Division, B.C.
Gold-antimony	West Gore Mine	Clarksville, N.S.
Gold	Abbott Mines, Limited	Lake Wanipigow, Man.
Arsenical gold	A. T. Westbrook	Madoc Township, Hastings County, Ont.
Gold	Morris Kirkland Gold Mines, Limited	Larder Lake Area, Timiskaming County, Ont.
Copper-lead-gold	Brunne Copper Lake Telluride Mines, Limited.	Copper Lake, Man.
Cyanide tailing	J-M Consolidated Gold Mines, Limited.	Jackson Manion, Ont.

Testing of a wire rope from the Princess Colliery, Sydney Mines, N.S.

Examination of broken portions of three austenitic manganese steel castings. (Sorel Steel Foundries).

The chemical and metallurgical examination of a shovel tractor shoe casting. (Joliette Steel, Limited.)

Examination of a broken crankshaft from Lycoming Aero Engine No. 144. (Dept. of Transport.)

Examination of a failed Chevrolet axle. (W. Bailey, Purchasing Department, Dept. of Mines and Resources).

Examination of two pieces of welded armour plate. (Department of National Defence.)

Examination of track pins from light tanks and Carden Lloyd carriers. (Department of National Defence.)

Examination of a failed austenitic manganese steel casting. (Joliette Steel, Ltd.)

Identification of worn numbers on copper bird-band. (Lands, Parks and Forests Branch.)

Examination of welded non-skid endless type chain. (Dept. of National Defence.)

Examination of a failed austenitic manganese steel ball mill liner. (Sorel Steel Foundries, Limited.)

Examination of two austenitic manganese steel castings. (Sorel Steel Foundries, Limited.)

Examination of an austenitic manganese steel pin. (Sorel Steel Foundries, Limited.)

Microscopic examination of sample from Conley Gold Mines, Wallace Lake, Man.

Microscopic analysis of chalcopyrite in tailing from the Aldermac Copper Corporation, Limited, Arntfield, Que.

Microscopic examination of hand specimen of gold ore from the Halliwell Gold Mines, Limited, Montreal, Que.

Microscopic examination of amalgamation tailing from Straw Lake Beach Gold Mines, Limited, Emo, Ont.

Microscopic examination of two samples of gold ore from the DeSantis Porcupine Mines, Limited, Timmins, Ont.

Microscopic examination of two hand specimens of mineralized quartz from the Sullivan Consolidated Mines, Ltd., Sullivan Post Office, Abitibi, Que.

Microscopic examination of samples of drill cores from the Calumet Mines, Ltd., Calumet Island, Que.

Microscopic examination of two samples of mill tailing from the Central Patricia Gold Mines, Limited, Central Patricia, Ont.

Chemical analyses and microscopic examination of a sample of copper concentrate from the Granby Consolidated Mining, Smelting and Power Company, Limited, Granby, B.C.

IV

ACTIVITIES OF CHEMICAL, MINERALOGICAL, PHYSICAL TESTING, AND HEAT-TREATING LABORATORIES; ANALYSES AND TESTS, INCLUDING MISCELLANEOUS ITEMS, TESTS, AND SAMPLING JOBS

Chemical Laboratories:

During the half year, January 1 to June 30, 1939, 3,449 samples of ores, minerals, and metal products were analysed by the staff of the Chemical Laboratories and complete reports were issued thereon.

This work included a total of 8,516 chemical and assay determinations, in which 49 different mineral constituents were involved.

The samples were made up from the following:—

	Number of samples
Metallic mill products	2,536
Bureau of Geology and Topography	6
Pyrometallurgical Laboratory	92
Industrial Minerals Division	604
Fuel Testing Laboratory	14
Outside assays	197
	3,449
Total determinations	8,516
Total gold assays	2,800
Total silver assays	484

Mineragraphic Laboratory:

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

A. Investigations Completed:

Gold ores	16
Mill products	3
Silver-lead-zinc ores	2
Special studies	13
Total	34

B. Polished sections were prepared for:

The Mineragraphic Laboratory	274
The Industrial Minerals Division	4
The Metallurgical Laboratory	14
The Bureau of Economic Geology	26
Queen's University	29
Princeton University	13
The University of Manitoba	8
The Hudson Bay Mining and Smelting Company	6
Total	374

C. Thin sections were prepared for:

The Mineragraphic Laboratory	3
The Industrial Minerals Division	5
The Calumet Mines, Limited	25
Total	33

Physical Testing and Heat-treating Laboratories:

Fatigue strength of a welded steel section was determined. (Dominion Bridge Company, Limited.)

Impact strength of a welded steel section was determined. (Dominion Engineering Co., Limited.)

A small piece of poor machining steel examined. (Joliette Steel, Limited.)

Thirteen welded plates and six welded tubes tested in tension. (Ottawa Flying Club.)

Hardness of eleven grinding balls determined and three balls heat-treated. (Hull Iron and Steel Foundries, Limited.)

Experimental plumbing fixtures at Banff, Alberta, examined. (Lands, Parks and Forests Branch.)

Counter weight carburized and heat-treated. (Royal Canadian Air Force.)

Milling cutter carburized and heat-treated. (Royal Canadian Air Force.)

Light alloy piston examined. (Department of Transport.)

Hardness tests made on bronze gunsight parts. (National Research Council.)

Miscellaneous:

Three tons of manganese ore given special reducing roast. (Shawinigan Chemicals, Limited.)

One-half ton tungsten ore given special roast. (Pan American Steel and Alloys Company, Merrickville.)

Lead and zinc ingots representing Canadian production. (New York World's Fair.)

A method worked out for reclaiming the metal content from high-speed steel scale. (Atlas Steels, Limited.)

Testing sample of butyl xanthate for comparative tests. (Societe Commerciale Belgo Canadienne Company, Montreal.)

Two samples of silica infrasized for W. S. Tyler Company of Canada, St. Catharines, Ont.

Sampling Jobs:

Gold ore, 5,800 pounds, from property of John Malloch in Trecesson Township, Abitibi, Que. (Gilman Exploration, Limited, Montreal.)

V

A RÉSUMÉ OF SPECIAL INVESTIGATIONS AND RESEARCH COMPLETED, IN PROGRESS, OR UNDER CONSIDERATION

The Research Section of these laboratories was constituted for special investigations to:

(1) Assist in solving particular difficulties that occasionally arise in the course of determining methods of treating ores submitted by the mining industry for investigation.

(2) Assist mine operators in solving problems in current milling practice.

(3) Develop new procedures or processes to further the use of the mineral resources of the country.

(4) Improve present practice in milling operations with the purpose of increasing recovery or grade of product.

(5) Assist the metal industries by standard and special tests involving physical, chemical, microscopic, and spectrographic examinations.

(6) Investigate the possible production of new alloys.

Something in each of these categories was accomplished in past years, but most attention was directed towards (1), (2), and (5), in connection with which the demand has been heavy and sufficient to keep the staff fully engaged.

During the period under review, January 1 to June 30, special work was needed in determining the method of treatment to be employed on ores from the following:

Hard Rock Gold Mines, Limited, Geraldton, Ont.

Porcher Island Mines, Limited, Porcher Island, B.C.

Delnite Mines, Limited, Timmins, Ont.

Chesterville Larder Lake Gold Mining Co., Ltd., Cheminis, Ont.

Athona Mines (1937), Limited, Goldfields, Saskatchewan.

Northern Pyrites, Limited, B.C.

Moneta Porcupine Mine, Timmins, Ont.

Canadian Johns-Manville Co., Limited, Asbestos Tailing, Asbestos, Que.

Normetal Mining Corporation, Limited, Dupuy, Que.

The investigation of the Hard Rock and Porcher Island ores involved experiments on roasting ore or concentrate, and cyanidation of the calcine. During the Delnite investigation experimental treatment of the ore and the plant tailing was made, in order to determine the cause of fouling of the cyanide solution, and what corrective procedure should be followed.

The investigation of the Chesterville Larder Lake ore had to do principally with difficulty in the precipitation of the gold from the cyanide solution.

The Athona Mines problem concerned the determination of how to treat the largest tonnage of low-grade ore for the best return.

Northern Pyrites was a flotation problem involving the recovery of copper and the production of a clean pyrite tailing.

The further study of the ore from the Moneta Porcupine Mines comprised an attempt to lower tailing losses or to increase the present recovery at the mine.

Table and magnetic concentration was tried in an attempt to produce a marketable iron oxide concentrate containing chromium and nickel from the asbestos tailing from the operations of the Canadian Johns Manville Company. The Shawinigan Water and Power Company, which was interested in and co-operated in this investigation, was responsible for the smelting experiments.

The Normetal ore problem was the treatment of a copper-zinc sulphide ore in which the copper-zinc minerals were closely interlocked and had been subjected to the action of water. The sphalerite had been activated by dissolved copper salts, and caused difficulty in separating the zinc and copper minerals. Numerous tests were conducted embracing pre-treatment, de-activation, etc., but the results were not entirely satisfactory.

Interest is being taken in Canada in the Shelton process for production of electrolytic manganese, the result of experiments carried out in the laboratories of the United States Bureau of Mines at Boulder City, Nev. (R.1.3406). Manganese carbonate (rhodocrosite) or manganous oxide is leached with sulphuric acid to produce a manganous sulphate, which is electrolysed in a diaphragm cell producing metallic manganese and regenerating acid. Ores such as pyrolusite, wad or bog manganese require reduction to a bivalent manganese form to permit leaching by sulphuric acid. This reduction is brought about by heating the ore in an atmosphere of reducing gas at a temperature ranging from 650° to 700° C. At the request of Shawinigan Chemicals, Limited, some 3 tons of pyrolusite ore was treated in this manner producing 2 tons of product, in which 95 per cent of the manganese was soluble in the acid. The oxide was shipped to Shawinigan for experimental electro-deposition.

An iron oxide from the Aldermac Copper Corporation was submitted for investigating the possibility of removing the contained silica. The oxide was of the order of 2300 mesh in fineness and contained 5 to 6 per cent of silica, 800 to 2300 mesh in fineness. Various schemes embracing flotation, magnetic concentration, roasting with chemicals, and straight chemical leaching were tried. By flotation, and by roasting with chemicals and leaching, the silica could be reduced to 2 per cent, and by chemical means a product containing 0·5 per cent of silica was obtained. The specified product had to contain not more than 0·1 per cent of silica. The chemical treatment appeared feasible but the cost would prove definitely uneconomic.

Further data were obtained in the investigation on the behaviour of accessory minerals such as sulphides in milling and cyanidation. This investigation has had frequently to be interrupted through pressure of other work, but in brief it embraces grinding the sulphides, in varying proportions, with clean silica sand, *(a)* in water, *(b)* in low-lime solution, *(c)* in high-lime solution, *(d)* in low-lime cyanide solution, and *(e)* in high-lime cyanide solution. The solutions so obtained are subjected to careful chemical analysis. Samples of sulphide minerals free from other sulphides are, however, difficult to obtain. To date, galena, stibnite, and pyrite have been under investigation separately, and it is proposed to study pyrrhotite (native and artificial), sphalerite, chalcopyrite, arsenopyrite, and carbonates of iron, manganese and such other minerals as are available. Subsequently, the behaviour of combinations of these minerals will be studied. It is hoped that a clearer understanding of the chemical reactions involved in the milling and cyanidation of metallic ores will result.

The effect of the presence of chromium in gold ores on the precipitation of the gold from cyanide solution was studied in connection with the treatment of the sample shipment from Chesterville Larder Lake Mines, in which poor and erratic precipitation of the gold from the cyanide solution had been obtained. Nickel and chromium were found in the cyanide solution. Other work prevented determination of the reason for this abnormal behaviour but a corrective procedure was developed, the details of which are given in Investigation No. 766, pages 61 to 77. This study necessitated an investigation of the methods of chemically determining small amounts of chromium. By a method evolved chromium as low as 0·002 per cent can be determined in this kind of solution; by spectrographic methods quantities of the order of 0·000002 per cent can probably be measured.

Owing to the increasing usefulness and adaptability of the spectroscope for the qualitative and quantitative determination of elements, a spectrographic laboratory has been added to the Ore Dressing and Metallurgical Laboratories. Two rooms are occupied, the smaller being a combined photographic dark room and chemical laboratory, which may be completely darkened by closing the door and lowering a light-proof shade, built into the window. It has the usual dark room facilities, such as sink, safe-lights for various kinds of plate or film, shelf space for photographic chemicals and materials, an enlarger, etc. It also contains an ordinary chemical work bench and sink, and such apparatus and shelf space as are necessary for the chemical work incident to spectrographic analysis. The larger room is the spectrographic laboratory proper. In it are the spectrograph, the control panel for the motor generator set (up to 300 volts D.C.) with transformer (up to 32,000 volts A.C.; 2 KVA), an analytical balance, and an assay balance.

A photo-electric densitometer, essential to quantitative spectrographic analysis will shortly be added.

The spectrograph was made by Baird Associates of Cambridge, Mass. It is of the concave grating type and conforms to the following specifications:

Grating—4 inches wide, 15,000 lines per inch, 3 metres radius of curvature.

Mounting—Modified angle type light distribution: 80 per cent in the 1st order; ghost line intensity 1/500 that of normal spectrum.

Dispersion—5 angstroms per millimetre, thus giving a range of about 1400 angstroms on a 10-inch plate (in the 1st order). By using large exposures, satisfactory pictures can be taken in the 2nd, 3rd, and 4th orders, in cases where very large dispersion is required.

Focussing adjustment and plate racking—Entirely electrically controlled and automatic.

This spectrograph is a very flexible instrument, displaying all the qualities of a large quartz instrument in the ultraviolet region and yet applicable to work in the visible and infra-red regions with no loss of dispersion. This is of importance in the analysis of alkali metals and in absorption spectrophotometry. Thus far, the work in the new spectrographic laboratory has been confined to:

1. The fitting-up of the laboratory itself.
2. The design and construction of such auxiliary equipment as could be made in the mechanical shops of the Bureau, viz.: optical bench, arc stand, variable angle and logarithmic step sectors, etc.
3. Accumulation of pure samples of most of the metals or their salts and the synthesis from these of a large number of standard mixtures (containing various percentages of the particular elements).
4. A semi-quantitative study of the lower limits of detection of the metals. This work is still in progress.
5. Performance of occasional purely qualitative analyses for such elements as beryllium, lithium, nickel, etc., for the information of the chemical laboratory.

When the photo-electric densitometer is ready, accurate quantitative determinations will begin.

In the Metallurgical Laboratory, public demands for physical tests and examinations of metals, alloys, etc., took up most of the time, but a few tests were conducted in the investigation of uranium as an alloying element with steel. Additional ingots of different composition were made and specimens were prepared therefrom for physical testing and microscopic examination. The results were inconclusive and further investigation will be deferred until the vacuum furnace, now being built, is in operation.

The Föppl-Pertz machine for determining damping capacity originally obtained from Baldwin-Southward, has been substantially modified as follows:

(a) The bearing forming the upper pivot for the specimen has been replaced by a simple spring pivot, whereby a very serious source of frictional loss has been removed.

(b) The recording device, consisting of a stylus and moving wax paper, has been replaced by a photographic arrangement. Two improvements have thereby been effected: the frictional loss resulting from the action of the stylus on the paper is completely avoided; the photographic record is considerably larger than that of the stylus and is much more easily interpreted and measured.

These alterations have increased considerably the accuracy of the machine.

Although, in general, damping capacities are used only relatively, it is considered advisable to calibrate the machine on an absolute basis, so that, if desired in any special case, the results may be corrected to absolute units.

The application of methods of X-ray analysis to metallurgical and mineralogical examinations has made the X-ray spectrograph an essential tool of the routine and research laboratory. In April, a Phillips' X-ray diffraction unit was acquired. This machine is furnished with the well known Phillips' X-ray tubes and the whole unit is mounted so that manipulation is simple. It comprises one high-tension generator with cooling device, camera support, two cameras, and two tubes: one with a copper target and the other with a cobalt target. The tubes are interchangeable and radiation and high-tension shocks are avoided by suitable protection. The copper target tube will be used for general crystallographic examinations and in crystallographic examinations of non-ferrous metals and alloys. The cobalt target tube will be used largely in obtaining diffraction patterns from ferrous alloys. A third molybdenum target tube, which will be used in X-ray spectrography involving the identification of new materials, is to be purchased.

The apparatus has been tested and spectrograms have been obtained from a variety of materials. Exposure times have been determined and camera radii have been calculated. Attention is being directed to the development of a technique suitable for the examination of metallographic specimens by microscope and X-ray, using the same specimen.

Another addition to the Metallurgical Laboratory is a vacuum induction furnace, designed by the Ajax Electrothermic Corporation and suitable for melting and pouring in vacuo. Electrical connections are so arranged that this unit or one of the two melting units previously erected can be operated from the same motor generator set. Water-cooling arrangements for coil cable and shell have been fitted. The electrical arrangements have been tested and found to be satisfactory.

A suitable vacuum pump is being added to complete the unit for operation. The unit will be used in determining the effect of gases on metals, in the production of cleaner steels, in making low-carbon materials, and in the manufacture of steels containing easily oxidizable elements such as uranium.

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