CANADA

DEPARTMENT OF MINES AND RESOURCES

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

MINES AND GEOLOGY BRANCH

JOHN McLEISH, DIRECTOR

BUREAU OF MINES

W. B. TIMM, CHIEF

INVESTIGATIONS IN ORE DRESSING AND METALLURGY

(Testing and Research Laboratories)

January to June, 1936

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

ORE DRESSING AND METALLURGY, JANUARY TO JUNE 1936

I

REVIEW OF INVESTIGATIONS

W. B. Timm
Chief of Division

During the half yearly period ending June 30, 1936, forty-nine investigations were completed and reports thereon were furnished to the parties submitting the ores or other materials for examination and test. In addition numerous tests of minor importance were made.

Of the major investigations, the nineteen reports included in Section II were published as separates. Thirty reports of somewhat less importance are synopsized in Section III.

The results of these investigations were used in the erection of eight new milling plants, and changes and alterations were made in six operating plants which resulted in increased recoveries.

The investigations were carried out under the direction of W. B. Timm, Chief, Division of Ore Dressing and Metallurgy. The work on metallic ores was performed by C. S. Parsons, R. J. Traill, A. K. Anderson, W. R. McClelland, J. D. Johnston, H. L. Beer, and W. S. Jenkins. The work on non-metallic minerals and industrial products was performed by R. K. Carnochan and R. A. Rogers. The microscopic and spectrographic examination of ores and mill products was done by M. H. Haycock. The metallurgical work on iron and steel products including mechanical and physical tests was performed by G. S. Farnham. The chemical work was performed by a staff of chemists under the direction and supervision of H. C. Mabee, Chief Chemist.

Of the total of 49 reports issued, thirty-two were on gold ores or gold-bearing mill products, indicating the great activity in the search for new gold mines. A number of these ores were complex, either containing cyanicides such as copper, arsenic, and pyrrhotite, or gold in a finely divided state that made treatment difficult, and necessitated detailed investigation requiring considerable time to complete. Products from six operating mills were submitted for investigation as to means of increasing recovery.

The treatment of one silver-lead ore, one lead ore, and one zinc ore was investigated. The silver-lead ore was from the Skagit River area near Hope, B.C., and the lead and the zinc ores were both from Ontario.

Magnetic concentration tests were carried out on a sample of titaniferous iron sand from the Thunder Bay district, Ontario, and the treatment of a tungsten ore containing scheelite and gold from the Indian Path Mines, Limited, Lunenburg, Nova Scotia, was also investigated.

The possibility of producing a grade of barite from the mill tailing of the Kamloops Homestake Mines was investigated.

In the mineragraphic laboratory 455 polished sections of ores and mill products were prepared and examined microscopically, and two thin sections were made.

From the microscopic examination of these polished sections, 38 investigations were completed, of which 31 were of gold ores, 4 of base metal ores, and one of a tungsten ore. Two special studies were made on metallurgical problems. Thirty-one spectrographic analyses were made mostly in connexion with the work of the Division.

In co-operation with a representative of the Aluminum Company of Canada, an investigation was made of the practicability of using the quartz spectrograph for determining minor constituents in aluminium alloys quantitatively, particularly as applied in plant operation.

In the radium-measuring laboratory only one sample was tested for radioactivity. This was a sample of spodumene from Wekusko, Manitoba.

In the research chemical laboratories investigations were carried out for Eldorado Gold Mines, Limited, in connexion with the manufacture of sodium uranate, the finished product at the plant in Port Hope being somewhat off colour compared with the material usually marketed. The investigation embraced a study of the methods of precipitation and effect of impurities in the Port Hope material. With the co-operation of the Ceramics Division, which made tests using lead glaze, a solution of the problem was obtained.

An investigation was conducted in the treatment of gold ores from the Bridge River district, B.C., embracing direct cyanidation, concentration by flotation, and washing and cyanidation of the calcine. This work is still in progress.

Upon request a new process for the extraction of gold from ores, known as the Harrison Chemical Process, was tested under the supervision and direction of the patentee. The results were no improvement upon those obtained by the standard cyanide process on the ores tested.

In the non-metallic laboratories the grindability of samples from Consolidated Sand and Gravel, Ltd., Toronto, Ontario, and the crushing of slate from Kingsbury and Ste. Hénédine, Quebec, for making roofing granules, were investigated. Tests were made on gypsum from Island Point, Boularderie island, Cape Breton, Nova Scotia.

In addition, numerous minor tests were conducted on sandstone, silica sand, clay, graphite, asbestos, mica, talc, garnet, spodumene, nepheline, syenite, beryl, calcite, quartz, diatomite, etc. Some testing was done on a norbide nozzle for sandblasting, also on the adjustments of an Andrews classifier. Several lots of silica sand were prepared from sandstone, for experiments being carried out at the Central Experimental Farm.

The appointment of G. S. Farnham to fill the position created by the resignation of H. H. Bleakney in the early part of 1935, allowed the work in ferrous metallurgy to be resumed. Most of the problems dealt with have been incidental to the manufacture and use of ferrous material, results of the work being of interest only to those directly concerned.

In the chemical laboratories 2,729 samples of ores, minerals, and metal products were analysed and reports issued thereon. This work involved a total of 8,160 chemical and assay determinations and of 37 different mineral constituents.

This record shows an increase of 28 per cent over that of the same period in 1935.

Of the total number of samples completed, 2,541 consisted of metallic ores, including gold and other precious metal ores; 163 were non-metallic minerals; and 25 were metallic products.

Included in the above total are 70 field samples, such as diamond drill cores and channel samples from mining properties and samples submitted by officers of the Geological Survey, Ottawa.

In addition, numerous other samples were submitted for identification and valuation.

II

INVESTIGATIONS THE RESULTS OF WHICH ARE RECORDED IN DETAIL

Ore Dressing and Metallurgical Investigation No. 665

GOLD ORE FROM BANKFIELD GOLD MINES, LIMITED, GERALDTON, ONTARIO

Shipment. A sample shipment of ore, weight 150 pounds, was received on December 5, 1935, from the Bankfield Gold Mines, Limited, Geraldton, Ontario. The shipment was submitted by John W. MacKenzie, mine superintendent.

Characteristics of the Ore. The *gangue* consists largely of a fine-textured, predominantly green and somewhat schistose, chloritic rock with associated white vein quartz and a small amount of impure carbonate.

The *metallic minerals* present are: pyrite, arsenopyrite, chalcopyrite, magnetite, pyrrhotite, and native gold.

Pyrite is coarsely- to finely-disseminated and contains inclusions of gangue, chalcopyrite, magnetite, and pyrrhotite; it also contains rare tiny grains of native gold.

Arsenopyrite is common; like the pyrite it is disseminated and often is associated with this mineral. It contains tiny grains of chalcopyrite and pyrrhotite.

A small amount of magnetite is disseminated in the gangue; it is also included in the pyrite.

Chalcopyrite and pyrrhotite, both in very small amounts, are disseminated in the gangue and included in pyrite and arsenopyrite.

Five grains of native gold ranging from 1600 mesh to 560 mesh in size were visible in the sections studied. Four of these are within pyrite, the remaining one is attached to the border of a pyrite grain.

The information obtained from the microscopic examination indicates that a certain proportion of the gold occurs in the pyrite and is finely divided.

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

Gold	0·365	oz./ton
Silver	0·05	"
Arsenic	0·95	per cent
Copper	0·03	"
Iron	11·89	"
Sulphur	3·69	"

Calculations based on the analyses indicated the distribution of metallic minerals to be approximately as follows:

Pyrite	6·15	per cent
Arsenopyrite	2·07	"
Oxide iron (magnetite)	8·31	"

Experimental Tests

Test work carried out on the ore consisted of cyanidation, amalgamation, blanket concentration, flotation, and combination of cyanidation, and table concentration.

CYANIDATION

A series of standard cyanide tests was run on samples at different finenesses of grinding. The cyanide solution had a strength equivalent to 1 pound of potassium cyanide per ton. Lime was added to maintain a protective alkalinity and the pulp dilution was 3 : 1.

Time of Agitation: 24 hours:

Test No.	Grinding size, mesh	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
1	− 48	0·365	0·08	78·08	0·30	5·10
2	−100	0·365	0·04	89·04	0·30	7·90
3	−150	0·365	0·035	90·41	0·30	8·90
4	−200	0·365	0·035	90·41	0·93	9·20

Time of Agitation: 48 hours:

Test No.	Grinding size, mesh	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
5	− 48	0·365	0·065	82·19	0·30	5·25
6	−100	0·365	0·04	89·04	0·30	8·90
7	−150	0·365	0·03	91·78	0·75	13·25
8	−200	0·365	0·03	91·78	1·08	13·40

Screen Tests:

Mesh	Weight, per cent		
	−48 mesh	−100 mesh	−150 mesh
+ 65	7·2	....	....
− 65+100	17·1	....	....
−100+150	11·2	8·1	....
−150+200	12·3	12·3	6·5
−200	52·2	79·6	93·5
Total	100·0	100·0	100·0

The results indicate that fine grinding is necessary for high gold extraction. The consumption of cyanide increases with the fineness of the ore and time of contact but is not excessively high. The lime consumption is rather high, due probably to the presence of soluble salts in the ore. An analysis of the solution from a water pulp indicated the presence of certain water-soluble mineral salts.

TEST FOR WATER-SOLUBLE SALTS IN THE ORE

A sample, 200 grammes in weight, of –200-mesh ore, was agitated for a short time in warm distilled water. After filtering, the clear solution was analysed, with the following results:

Total solids (other than SiO_2)	0·24 grm./litre
Calcium as $CaCO_3$	0·054 "
Magnesium as $MgCO_3$	0·0049 "
SO_3	0·072 "
SO_3 as $CaSO_4$	0·122 "
Iron	Faintest trace
Alkaline to phenolphthalein.	

Reducing power=10·7 c.c. $\frac{N}{10}$ $KMnO_4$ per litre.

AMALGAMATION

Test No. 9

A sample of ore was ground in a water pulp to a fineness of 58·6 per cent –200 mesh and then barrel-amalgamated with 100 grammes of mercury for one hour.

Gold in feed, oz./ton	Gold in tailing, oz./ton	Recovery, per cent
0·365	0·180	50·68

The recovery indicates the free-milling gold at the grinding indicated above.

BLANKET CONCENTRATION

Test No. 10

A sample of ore was ground in a water pulp (4 : 3) to a grinding fineness of approximately 68 per cent –200 mesh and then passed over a corduroy blanket. The blanket concentrate was panned to remove gangue and excess sulphides. Free gold was visible in the panned concentrate.

The results are as follows:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·37	100·00	
Concentrate	1·64	8·82	39·00	60·98 : 1
Tailing	98·36	0·23	61·00	

FLOTATION

Test No. 11

A sample of ore was ground in a soda ash pulp to a fineness of 78·6 per cent –200 mesh. Soda ash, 4 pounds per ton, and Aerofloat No. 31, 0·07 pound per ton, were used in the grinding. The pulp was floated with 0·20 pound per ton of sodium ethyl xanthate and 0·05 pound per ton of pine oil.

The results are as follows:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·364	100·00	10·07 : 1
Concentrate	9·93	3·26	88·87	
Tailing	90·07	0·045	11·13	

BLANKET CONCENTRATION—CYANIDATION

Test No. 12

A sample of the ore was ground dry to pass a 65-mesh screen and then fed in a pulp to a corduroy blanket strake.

The blanket concentrate was amalgamated with mercury in an iron mortar.

The amalgamation tailing was combined with the blanket tailing and reground in an Abbé grinding jar. Samples of the reground tailing were then treated by cyanide, in solutions of strength equivalent to 1 pound of potassium cyanide per ton.

The results of the test are as follows:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·365	100·0	56·5 : 1
Blanket concentrate	1·77	7·025	34·06	
Blanket tailing	98·23	0·245	65·94	

Cyanidation of Blanket and Amalgamation Tailings:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton tailing		Pulp dilution
	Feed	Tailing		KCN	CaO	
24	0·245	0·045	81·63	0·98	6·02	3·27 : 1
48	0·245	0·050	79·59	1·29	6·36	3·22 : 1

Screen Test of Reground Tailings:

Mesh	Weight, per cent
− 65+100	0·2
−100+150	2·0
−150+200	8·0
−200	89·8
	100·0

The results obtained by blanket concentration of the ore followed by cyanidation indicate a poor extraction of gold.

Test No. 13

A sample of ore was ground dry to pass a 100-mesh screen and treated by agitation for 24 hours. The cyanide tailing was filtered and then, after repulping, fed to a laboratory Wilfley table. The table concentrate (sulphides) was then reground and re-cyanided. The results of the test are as follows:

Cyanide Treatment of −100-mesh Ore:

Cyanide strength: 1 lb. KCN/ton

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing.....	0·365	0·05	86·30	0·64	6·50	2·5 : 1

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed..............................	100·00	0·05	100·00	
Concentrate........................	9·43	0·30	55·54	10·6 : 1
Tailing............................	90·57	0·025	44·46	

Cyanide Treatment of Reground Table Concentrate:

Cyanide strength: 2 lb. KCN/ton

Time of agitation: 24 hours

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing.....	0·30	0·16	46·67	0·30	6·25	3 : 1

A screen test indicated that the −100-mesh ore is ground to a fineness of 79 per cent −200 mesh and the reground concentrate to a fineness of 98 per cent −200 mesh.

Recovery of Gold:

Gold recovery by cyanidation of raw ore..........................	86·30	per cent
Gold recovered in table concentrate of cyanide tailing— 55·54 × 13·70 = 7·61 per cent.		
Gold recovery by cyanidation of table concentrate—46·67 × 7·61	3·55	"
Overall recovery..	89·85	"

Test No. 14

In this test the ore was ground in a cyanide and lime pulp. The ore was ground to an approximate fineness of 78 per cent −200 mesh at a pulp dilution of 0·75 : 1. The ground ore was then transferred to bottles, made up to a pulp dilution of 2·5 : 1 and cyanide added to give a strength equivalent to 1 pound of potassium cyanide per ton. Agitation was carried out for 24 hours, after which the tailing was filtered, washed, and fed to a Wilfley table. The table concentrate was reground in a pebble jar to a fineness of approximately 100 per cent −200 mesh, and re-cyanided for 24 hours as in the previous test.

Consumption of Reagents during Grinding:

KCN.. 0·25 lb./ton
CaO.. 9·75 "

Cyanide Treatment of Raw Ore:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing.....	0·365	0·04	89·04	0·13	3·0	2·5 : 1

Total Consumption of Reagents:

KCN.. 0·38 lb./ton
CaO.. 12·75 "

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz/ton	Distribution, per cent	Ratio of concentration
Feed..............................	100·00	0·04	100·00	
Concentrate..............................	8·66	0·44	62·53	11·55 : 1
Tailing..............................	91·34	0·025	37·47	

Cyanide Treatment of Reground Table Concentrate:

Cyanide strength: 2 lb. KCN/ton
Time of agitation: 24 hours

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing.....	0·44	0·13	70·45	1·2	9·25	3 : 1

Recovery of Gold:

Gold recovery during grinding and by cyanidation of ore......... 89·04 per cent
Gold recovered in table concentrates of cyanide tailing—
62·53 × 10·96=6·85 per cent.
Gold recovery by cyanidation of table concentrate—
70·45 × 6·85.. 4·83 "
Overall recovery.................................... 93·87 "

From the results obtained in this test, it is apparent that soluble salts present in the ore have a decided action on the lime consumption. In view of this the subsequent tests were carried out employing a water pulp.

Test No. 15

The ore was ground in a water pulp to a fineness of approximately 78 per cent −200 mesh. After grinding, the ore was dewatered and then repulped in a cyanide solution of strength equivalent to 1 pound of potassium cyanide per ton at a pulp dilution of 2·5 : 1. Agitation was carried out for 24 hours. The tailing was tabled and the concentrate reground and re-cyanided as in the previous tests.

Cyanidation of Ore (Grinding in Water):

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing.....	0·365	0·055	84·93	0·25	5·00	2·5 : 1

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Pulp dilution
Feed..	100·00	0·055	100·00	
Concentrate.................................	7·30	0·499	66·29	13·7 : 1
Tailing......................................	92·70	0·02	33·71	

Cyanidation of Reground Table Concentrate:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing.....	0·499	0·155	68·94	1·44	6·55	3 : 1

Recovery of Gold:

Gold recovery by cyanidation of ore............................ 84·93 per cent
Gold recovered in table concentrate of cyanide tailing—
66·29 × 15·07=9·99 per cent.
Gold recovery by cyanidation of table concentrate—
68·94 × 9·99.. 6·89 "
Overall recovery.................................... 91·82 "

Test No. 16

This was a duplicate of Test No. 15.

Cyanidation of Ore (Grinding in Water):

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing.....	0·365	0·045	87·67	0·25	5·00	2·5 : 1

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·045	100·0	
Concentrate	14·12	0·24	72·46	7·08 : 1
Tailing	85·88	0·015	27·54	

Cyanidation of Reground Table Concentrate:

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing	0·24	0·11	54·16	0·93	5·40	3 : 1

Screen Test on Reground Table Concentrate:

Mesh	Weight, per cent
+325	9·2
−325	90·8
	100·0

Recovery of Gold:

Gold recovery by cyanidation of ore .... 87·67 per cent
Gold recovered in table concentrate of cyanide tailing—
72·46 × 12·33 = 8·93.
Gold recovery by cyanidation of table concentrate—54·16 × 8·93. 4·84 "
Overall recovery .... 92·51 "

Summary of Tests Nos. 13 to 16:

Test No.	Grinding	Reagents consumed, lb./ton		Extraction of gold, per cent
		KCN	CaO	
13	Dry	0·67	7·09	89·95
14	Cyanide pulp	0·48	13·55	93·87
15	Water pulp	0·36	5·47	91·82
16	"	0·38	5·76	92·51

Test No. 17

A sample of ore was ground to a fineness of approximately 78 per cent −200 mesh in a water pulp and then passed over a corduroy blanket strake. The blanket concentrate was panned to remove gangue and excess sulphides in order to produce a high-grade concentrate. This concentrate was then amalgamated with mercury. The blanket tailing was dewatered, combined with the amalgamation tailing, and treated by cyanide for 24 hours in a solution of strength equivalent to 1 pound of potassium cyanide per ton.

The cyanide tailing was filtered and washed and conditioned preparatory to floating. The flotation concentrate was then roasted for two hours in a muffle furnace. After roasting the concentrate was treated by cyanide for 24 hours.

Details of the test are as follows:

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·365	100·00	153·8 : 1
Concentrate	0·65	19·47	34·67	
Tailing	99·35	0·24	65·33	

Cyanidation of Blanket Tailing and Amalgamation Tailing of Blanket Concentrate:

Product	Assay, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton of tailing		Pulp dilution
	Tailing		KCN	CaO	
Cyanide tailing	0·035	90·41	0·25	5·00	2·5 : 1

Flotation of Cyanide Tailing:

Conditioned for 15 minutes with soda ash, 2 pounds per ton, and for 3 minutes with copper sulphate, 0·5 pound per ton, and potassium amyl xanthate, 0·2 pound per ton. Pine oil—0·05 pound per ton.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·035	100·00	14·31 : 1
Concentrate	6·99	0·30	59·91	
Tailing	93·01	0·015	40·09	

Roast of Flotation Concentrate:

Weight of charge, grammes	Weight of calcine, grammes	Loss of weight, grammes	Loss of weight, per cent	Gold in charge, oz./ton	Gold in calcine, oz./ton
220	180	40	18·2	0·30	0·366

Temperature Log:

Time	Temperature, degrees C.	Remarks	Time	Temperature, degrees C.	Remarks
2·05	400	Charge in.	3·00	620	
2·15	580		3·30	680	Heat off
2·30	620		4·00	....	Charge out

Cyanidation of Roasted Flotation Concentrate:

Cyanide strength: 2 lb. KCN/ton

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton of calcine		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing.....	0·366	0·095	74·04	2·1	38·14	3 : 1

The roasted concentrate was not washed prior to cyanidation. This probably accounts for the high consumption of lime.

Recovery of Gold:

Gold recovery by amalgamation of blanket concentrate and cyanidation of tailing: $\frac{0{\cdot}365-0{\cdot}035}{0{\cdot}365} \times 100$.................. 90·41 per cent

Gold recovered in flotation concentrate of cyanide tailing—59·91 × 8·59=5·15 per cent.

Gold recovery by cyanidation of roasted concentrate: 74·04 × 5·15   3·81 "

Overall gold recovery..............................   94·22 "

Test No. 18

In this test a sample of ore was ground in a water pulp to a fineness of approximately 78 per cent −200 mesh and then passed over a corduroy blanket strake. The blanket concentrate was panned to produce a high-grade product and then amalgamated with mercury.

The blanket tailing was treated by cyanide for 24 hours and then fed to a laboratory Wilfley table.

The table concentrate was added to the amalgamation tailing and reground preparatory to agitation in cyanide for 24 hours.

The details of the test are as follows:

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed..................................	100·00	0·365	100·00	
Concentrate...........................	1·67	9·49	43·43	59·88 : 1
Tailing...............................	98·33	0·21	56·57	

Recovery of Gold:

Gold recovery by amalgamation: $\frac{9{\cdot}49-2{\cdot}306^*}{9{\cdot}49} \times 100$............ 75·70 per cent

*By calculation.

Cyanidation of Blanket Tailing:

Cyanide strength: 1 lb. KCN/ton

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton of tailing		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing.....	0·21	0·025	88·10	0·475	5·25	2·5 : 1

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·025	100·00	9·12 : 1
Concentrate	10·97	0·066	28·78	
Tailing	89·03	0·02	71·22	

Cyanidation of Reground Table Concentrate and Amalgamation Tailing:

Cyanide strength : 2 lb. KCN/ton. Time of agitation : 24 hours.

Product	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Pulp dilution
	Feed	Tailing		KCN	CaO	
Cyanide tailing	0·405	0·33	18·52	2·88	4·10	3 : 1

Recovery of Gold:

Gold recovery by amalgamation of blanket concentrate: 75·70 × 43·43	32·88 per cent
Gold recovery by cyanidation of blanket tailing: 88·10 × 56·57	49·84 "
Percentage of total gold in table concentrate: 28·78 per cent of (56·57−49·84) = 1·94 per cent.	
Percentage of total gold in amalgamation tailing: 43·43−32·88 = 10·55 per cent.	
Recovery of gold by cyanidation of amalgamation tailing and flotation concentrate: 18·52 per cent of (1·94+10·55)	2·31 "
Overall gold recovery	85·03 "

Test No. 19

BLANKET CONCENTRATION

This test was carried out for the purpose of determining the gold recovery by amalgamation of the blanket concentrate.

The sample was ground to approximately the same fineness as in the previous tests and the pulp then passed over a corduroy blanket. The concentrate was panned to increase the grade and was then amalgamated in a mortar with mercury for half an hour.

The results are as follows:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·365	100·00	151·5 : 1
Concentrate	0·66	18·426	33·32	
Tailing	99·34	0·245	66·68	

Amalgamation of Blanket Concentrate:

Gold in concentrate	18·426 oz./ton
Gold in amalgamation tailing	6·02 "
Gold recovery	67·33 per cent

SETTLING TESTS

Test No. 20

Ratio of liquid to solids : 2·5 : 1.
Water pulp.
Grinding : 78 per cent −200 mesh.
Rate of settling : 0·515 foot per hour.
Overflow solution : slightly cloudy.

Time	Settlement of solids, in feet	Cumulative settlement	Time	Settlement of solids, in feet	Cumulative settlement
10·05	...	...	11·05	0·045	0·480
10·10	0·050	0·050	11·10	0·045	0·525
10·15	0·035	0·085	11·15	0·035	0·560
10·20	0·035	0·120	11·20	0·045	0·605
10·25	0·035	0·155	11·25	0·050	0·655
10·30	0·040	0·195	11·30	0·040	0·695
10·35	0·035	0·230	11·35	0·055	0·750
10·40	0·040	0·270	11·40	0·040	0·790
10·45	0·040	0·310	11·45	0·045	0·835
10·50	0·040	0·350	11·50	0·055	0·890
10·55	0·040	0·390	11·55	0·050	0·940
11·00	0·045	0·435	12·00	0·045	0·985
...	...	...	12·05	0·045	1·030

Test No. 21

Ratio of liquid to solids : 2·5 : 1.
Lime pulp : 10 pounds per ton of ore.
Grinding : 78 per cent −200 mesh.
Rate of settling : 0·565 foot per hour.
Overflow solution : Clear.
Titration for alkalinity : 0·5 CaO, lb./ton.

Time	Settlement of solids, in feet	Cumulative settlement	Time	Settlement of solids, in feet	Cumulative settlement
1·55	...	...	3·00	0·050	0·570
2·00	0·050	0·050	3·05	0·045	0·615
2·05	0·045	0·095	3·10	0·045	0·660
2·10	0·035	0·130	3·15	0·055	0·715
2·15	0·040	0·170	3·20	0·050	0·765
2·20	0·040	0·210	3·25	0·050	0·815
2·25	0·050	0·260	3·30	0·055	0·870
2·30	0·035	0·295	3·10	0·050	0·920
2·35	0·045	0·340	3·40	0·055	0·975
2·40	0·045	0·385	3·45	0·055	1·030
2·45	0·045	0·430	3·50	0·050	1·080
2·50	0·045	0·475	3·55	0·050	1·130
2·55	0·045	0·520			

SUMMARY

The results of the tests conducted on the sample of ore submitted indicate certain features that influence the method of treatment to be adopted.

At a fineness of grinding of 58 per cent −200 mesh, about 50 per cent of the gold was found to be free milling (*see* Test No. 9). The gold occurring in the pyrite is very closely associated, necessitating very fine grinding of the sulphides. At a fineness of 90 per cent −325 mesh the tests indicate that gold still remains tied up in the sulphides.

The presence of certain water-soluble salts in the ore has a tendency to increase the consumption of lime. Grinding in a water pulp and the use of blankets for removal of the free gold will overcome this difficulty. Concentration of the cyanide tailing on tables or by flotation with roasting of the sulphide concentrate and subsequent cyanidation indicated a recovery of 94·22 per cent (*see* Test No. 17). Regrinding of the concentrate without roasting, followed by re-cyanidation, indicated a recovery of 92·51 per cent. Primary grinding in cyanide, followed by agitation in cyanide, filtering and washing of cyanide tailing, tabling of tailing and regrinding and cyanidation of table concentrate, indicated a recovery of 93·87 per cent (*see* Test No. 14). Grinding in cyanide, however, consumes a considerable excess of lime over grinding in a water pulp.

Settling tests carried out on both a water pulp and a lime pulp indicate a rate of settling that is well within the normal limits for satisfactory settling.

Ore Dressing and Metallurgical Investigation No. 666

GOLD ORE FROM STURGEON RIVER GOLD MINES, LIMITED, STURGEON RIVER AREA, THUNDER BAY DISTRICT, ONTARIO

Shipments. Two boxes of ore, total weight 820 pounds, were received December 18, 1935, from C. M. Bowyer, Mine Manager, Coniagas Mines, Limited, Nezah, Ontario.

It was desired that preliminary tests be conducted for the recovery of the gold.

After crushing, cutting and grinding by standard methods, a sample of the ore was obtained that assayed as follows:—

Gold	0·84 oz./ton	Sulphur	0·50 per cent
Silver	0·23 "	Copper	0·03 "
Iron	2·64 per cent	Arsenic	Trace

Characteristics of the Ore. The *gangue* is chiefly green silicified chloritic schist with smaller quantities of vein quartz.

The *metallic minerals* present in the polished sections are pyrite, chalcopyrite, pyrrhotite, and native gold.

Pyrite is disseminated sparingly in the chloritic gangue as medium to fine cubic crystals and irregular grains. It contains small inclusions of chalcopyrite, and rare tiny grains of pyrrhotite and native gold.

Chalcopyrite in small quantity occurs chiefly as irregular small grains in the chloritic gangue. A very small quantity is also present in pyrite, as noted above.

Pyrrhotite is extremely rare, occurring as tiny grains in pyrite.

Only five tiny grains of native gold were observed in the polished sections. All were between 400 mesh and 100 mesh in size. Three occur along the borders of quartz stringers in highly siliceous gangue and two are enclosed in pyrite.

Conclusion. The ore consists of silicified chlorite schist and white vein quartz. Pyrite is sparingly disseminated in the chlorite gangue. Small quantities of chalcopyrite and pyrrhotite are present, but the amount is probably too small to affect treatment.

The meagre information concerning the mode of occurrence of the gold indicates that it occurs both in siliceous gangue and in pyrite, and is very finely divided.

Experimental Tests

The test work included cyanidation, blanket concentration, hydraulic classification, amalgamation, and flotation. Amalgamation followed by cyanidation of the amalgamation tailing gave an overall recovery of 98·7 per cent of the gold.

BARREL AMALGAMATION

Tests Nos. 1 and 2

In these tests, the ore at -14 mesh was ground in a ball mill to pass 76·5 per cent through 200 mesh in Test No. 1 and 83·8 per cent in Test No. 2. The pulp was then amalgamated with mercury for one hour in a jar mill. The amalgamation tailings were assayed for gold.

Screen tests showed the grinding to be as follows:

Mesh	Weight, per cent		Test No.	Assay, Au, oz./ton		Recovery, per cent
	Test No. 1	Test No. 2		Feed	Tailing	
− 65 + 100	0·6	0·1				
−100 + 150	4·3	2·0	1	0·84	0·22	73·8
−150 + 200	18·5	14·0	2	0·84	0·10	88·1
−200	76·5	83·8				

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

AMALGAMATION, BLANKET CONCENTRATION, CYANIDATION

Test No. 3

The ore at -14 mesh was ground in a ball mill to pass 76·5 per cent through 200 mesh. The pulp was then passed over an amalgamation plate and a sample taken of the amalgamation tailing. This tailing was then passed over a corduroy blanket set at a slope of 2·5 inches per foot. The blanket concentrate was reground to pass 100 per cent through 200 mesh and then barrel-amalgamated with mercury for one hour. The amalgam residue was then combined with the blanket tailing and agitated in a 3 : 1 cyanide solution of a strength of 1·5 pounds of potassium cyanide per ton. Five pounds of lime per ton of tailing was added to maintain protective alkalinity. The different products were assayed for gold. A screen test of the plate amalgamation tailing showed the grinding to be as follows:—

Mesh	Weight, per cent
− 65 + 100	1·9
−100 + 150	4·7
−150 + 200	16·7
−200	76·5

Plate Amalgamation:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
0·84	0·33	60·7

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·33	100·0	
Blanket concentrate	4·72	3·66	52·4	21·2 : 1
Blanket tailing	95·28	0·165	47·6	

Barrel Amalgamation of Blanket Concentrate:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
3·66	0·90	75·4

Cyanidation:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·19	0·02	89·5	0·3	4·1
48	0·19	0·01	94·7	0·4	4·3

Summary:

Bullion recovered by amalgamation—Plates	60·7 per cent
Blankets	15·3 "
Bullion recovered by cyanidation (48 hours)	22·7 "
Overall recovery	98·7 "

HYDRAULIC CLASSIFICATION, BLANKET CONCENTRATION, AMALGAMATION, CYANIDATION

Test No. 4

The ore at −14 mesh was ground in a ball mill to pass 83·8 per cent through 200 mesh. The pulp was passed through a hydraulic classifier and the coarse gold and heavier particles removed. The hydraulic tailing was passed over a corduroy blanket. The hydraulic concentrate and blanket concentrate were combined and barrel-amalgamated for one hour. The amalgamation tailing was then combined with the blanket tailing and agitated in cyanide solution of a strength of 1·5 pounds of potassium cyanide for 24 and 48 hours, 5 pounds of lime per ton of tailing being added to maintain protective alkalinity. The various products were assayed for gold. A screen test showed the grinding to be as follows:—

Mesh	Weight, per cent
− 65 + 100	0·3
−100 + 150	2·2
−150 + 200	13·7
−200	83·8

Hydraulic Classification and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·84	100·0	
Hydraulic and blanket concentrates	3·0	21·53	76·9	33 : 1
Blanket tailing	97·0	0·20	23·1	

The ratio of concentration of the hydraulic classification was 137 : 1.
The ratio of concentration of the blanket concentration was 44 : 1.

Barrel Amalgamation of Concentrates:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
21·53	6·14	71·5

Cyanidation:

Agitation, hours,	Assay, Au, oz./ton		Recovery, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·38	0·01	97·4	0·3	3·6
48	0·38	0·005	98·7	0·3	3·7

Summary:

Bullion recovered by amalgamation	55·0 per cent
" " cyanidation (48 hours)	44·4 "
Overall recovery	99·4 "

HYDRAULIC CLASSIFICATION, BLANKET CONCENTRATION, FLOTATION

Test No. 5

In this test, the ore at −14 mesh was ground in a ball mill to pass 82·5 per cent through 200 mesh. The pulp was passed through a hydraulic classifier and the coarse gold and heavier particles removed. The hydraulic tailing was passed over a corduroy blanket. The blanket tailing was conditioned with 2 pounds of soda ash and 0·05 pound of Aerofloat No. 31 per ton and floated with 0·04 pound of pine oil and 0·20 pound of amyl xanthate per ton. The combined hydraulic, blanket, and flotation concentrates were reground to pass 90 per cent through 200 mesh and barrel-amalgamated with mercury for one hour. The various products were assayed for gold. A screen test showed the grinding to be as follows:—

Mesh	Weight, per cent
− 65+100	0·1
−100+150	1·7
−150+200	15·7
−200	82·5

Hydraulic Classification and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·84	100·0	
Hydraulic and blanket concentrates	0·95	55·05	62·3	105 : 1
Blanket tailing	99·05	0·32	37·7	

The ratio of concentration of the hydraulic classification was............ 222 : 1
The ratio of concentration of the blanket concentration was............. 204 : 1

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·32	100·0	
Flotation concentrate	5·2	5·50	89·6	19·2 : 1
Flotation tailing	94·8	0·035	10·4	

Amalgamation of Concentrates:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
13·00	5·20	60·0

Summary:

Gold recovered in hydraulic and blanket concentrates............ 62·3 per cent
Gold recovered in flotation concentrate.......................... 33·8 "
Bullion recovered by amalgamation of combined concentrates..... 57·7 "

HYDRAULIC CLASSIFICATION, CYANIDATION

Tests Nos. 6, 7, 8, 9

In these tests, the ore was ground to different sizes and the pulp passed through a hydraulic classifier. The hydraulic concentrates were amalgamated with mercury for one hour and the amalgam residue added to the hydraulic tailing. This pulp was agitated in cyanide solution of a strength of 1·5 pounds of potassium cyanide with 5 pounds of lime per ton of tailing, for 24 hours. The different products were assayed for gold. A screen test of the hydraulic tailing showed the grinding as follows:—

Mesh	Weight, per cent			
	Test No. 6	Test No. 7	Test No. 8	Test No. 9
− 65+100	0·3	0·2	....	....
−100+150	3·8	2·8	0·7	0·7
−150+200	14·9	12·1	6·3	6·1
−200	81·0	84·8	92·8	93·1

Hydraulic Classification:

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
6..........	Feed....................	100·00	0·84	100·0	
	Hydraulic concentrate....	0·51	65·22	39·6	196 : 1
	Hydraulic tailing.........	99·49	0·51	60·4	
7..........	Feed....................	100·00	0·84	100·0	
	Hydraulic concentrate....	0·33	83·90	33·0	303 : 1
	Hydraulic tailing.........	99·67	0·565	67·0	
8..........	Feed....................	100·00	0·84	100·0	
	Hydraulic concentrate....	0·35	90·51	37·7	286 : 1
	Hydraulic tailing.........	99·65	0·525	62·3	
9..........	Feed....................	100·00	0·84	100·0	
	Hydraulic concentrate....	0·26	117·84	36·5	385 : 1
	Hydraulic tailing.........	99·74	0·535	63·5	

Amalgamation of Hydraulic Concentrates:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
85·00	38·42	54·8

Cyanidation:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
6..................	24	0·66	0·025	96·2	0·3	3·65
7..................	24	0·69	0·015	97·8	0·3	3·65
8..................	24	0·67	0·02	97·0	0·4	3·65
9..................	24	0·67	0·01	98·5	0·4	3·75

SUMMARY

Bullion recovered by amalgamation of hydraulic concentrate:

Test No. 6.. 21·7 per cent
" 7.. 18·1 "
" 8.. 20·6 "
" 9.. 20·0 "

Bullion recovered by cyanidation of hydraulic tailing plus amalgam residue:

Test No. 6.. 76·2 per cent
" 7.. 80·1 "
" 8.. 77·0 "
" 9.. 78·8 "

Overall recovery:

Test No. 6.. 97·9 per cent
" 7.. 98·2 "
" 8.. 97·6 "
" 9.. 98·8 "

SETTLING TEST

Test No. 10

This test was carried out in a tall glass tube having an inside diameter of $2\frac{1}{8}$ inches. The pulp, having been previously ground with 5 pounds of lime per ton of ore, was transferred to the glass tube and the levels of solids in decimals of feet were read every five minutes. Readings were made for a two-hour period. At the end of the test, the solution was titrated for alkalinity. The result is recorded in the following table:—

Ratio of liquid to solid.............................2·5 : 1
Lime added per ton solid.............................5 pounds
Alkalinity of solution at end of test....................0·30 CaO, lb./ton solution
Overflow solution.....................................Clear
Rate of settling.......................................0·83 foot/hour

A screen test showed the grinding as follows:—

Mesh	Weight, per cent
− 65+100	0·3
−100+150	2·6
−150+200	11·1
−200	86·0

Time	Settling of solids, in feet	Cumulative settling
0 hr. 5 min.	0·065	0·065
0 " 10 "	0·060	0·125
0 " 15 "	0·070	0·195
0 " 20 "	0·070	0·265
0 " 25 "	0·070	0·335
0 " 30 "	0·070	0·405
0 " 35 "	0·075	0·480
0 " 40 "	0·080	0·560
0 " 45 "	0·070	0·630
0 " 50 "	0·080	0·710
0 " 55 "	0·075	0·785
1 " 0 "	0·080	0·865
1 " 5 "	0·085	0·950
1 " 10 "	0·075	1·025
1 " 15 "	0·085	1·110
1 " 20 "	0·080	1·190
1 " 25 "	0·080	1·270
1 " 30 "	0·075	1·355
1 " 35 "	0·080	1·435
1 " 40 "	0·085	1·525
1 " 45 "	0·060	1·580
1 " 50 "	0·065	1·645
1 " 55 "	0·060	1·705
2 " 0 "	0·060	1·765

The rate of settling is approximately normal.

CONCLUSIONS

The ore responds readily to cyanidation and this is the process to be recommended.

Owing to the comparatively large amount of coarse gold in the ore, both the time of agitation and the fineness of grinding can be cut down if cyanide agitation be preceded by traps and blankets, or by some method of concentration such as gold jigs.

If the installation of a cyanide plant be not feasible, a combination of traps, blankets, and flotation will recover 96 per cent of the gold in the form of concentrates. These concentrates can be either sent to a smelter or barrel-amalgamated as economic procedure dictates.

Ore Dressing and Metallurgical Investigation No. 667

GOLD ORE FROM LEITCH GOLD MINES, LTD., STURGEON RIVER AREA, THUNDER BAY DISTRICT, ONTARIO

Shipment. A shipment of 37 bags of ore, weighing 2,490 pounds, was received on December 24, 1935, from W. E. Segsworth, Consulting Engineer, Leitch Gold Mines, Ltd., 67 Yonge Street, Toronto.

It was desired that preliminary tests be conducted for the recovery of gold.

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

Gold	0·85 oz./ton	Iron	1·37 per cent
Silver	0·02 "	Arsenic	0·03 "
		Sulphur	0·16 "
		Copper	Trace

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

The *gangue* consists of white quartz, locally stained brown by iron oxides, which contains numerous narrow sinuous stringers of green chloritic material.

The *metallic minerals,* of which the total quantity is small, are represented mainly by pyrite and arsenopyrite. Pyrite occurs mostly in the chloritic stringers as medium to very fine disseminated cubic crystals or irregular grains. Medium to fine needles and irregular grains of arsenopyrite occur in the same manner. No other minerals were seen, with the exception of two unknown species, which occur in extremely tiny rounded blebs within pyrite. Etching tests failed to reveal the identity of either of these unknown minerals, and the particles are too small to allow of spectrographic or microchemical tests.

No information as to the mode of occurrence of the gold was obtained through the microscopic examination.

Experimental Tests

The test work included cyanidation, hydraulic classification, blanket concentration, and amalgamation. A recovery of 98 per cent was obtained when cyanidation was preceded by traps and blankets.

BARREL AMALGAMATION

Tests Nos. 1 and 2

The ore at −14 mesh was ground in a ball mill to pass 51·1 per cent through 200 mesh in Test No. 1, and 81·4 per cent in Test No. 2. The pulp was amalgamated with mercury for one hour in a jar mill and the tailing assayed for gold.

A screen test showed the grinding to be as follows:

Mesh	Weight, per cent	
	Test No. 1	Test No. 2
− 65+100	6·0	....
−100+150	14·4	2·4
−150+200	28·0	16·1
−200	51·1	81·4

Amalgamation:

Test No.	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
1	0·85	0·24	71·8
2	0·85	0·13	84·7

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

PLATE AMALGAMATION

Test No. 3

The ore at −14 mesh was ground to pass 66·2 per cent through 200 mesh. The pulp was passed over an amalgamation plate and the plate tailing assayed for gold.

Results:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
0·85	0·35	58·8

HYDRAULIC CLASSIFICATION, BLANKET CONCENTRATION, AMALGAMATION, AND CYANIDATION

Test No. 4

The ore at −14 mesh was ground in a ball mill to pass 72·0 per cent through 200 mesh. The pulp was passed through a hydraulic classifier and the heavier particles and coarse gold concentrated. The hydraulic tailing was passed over a corduroy blanket and a blanket concentrate recovered. The combined hydraulic and blanket concentrates were reground and amalgamated with mercury for one hour. The amalgam residue was then added to the blanket tailing and agitated in cyanide solution of a strength of 1·5 pounds of potassium cyanide per ton, with 5 pounds of lime per ton of tailing added to maintain protective alkalinity. The various products were assayed for gold.

A screen test showed the grinding to be as follows:

Mesh	Weight, per cent
− 65+100	1·2
−100+150	6·4
−150+200	20·4
−200	72·0
	100·0

Hydraulic and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·85	100·0	
Combined concentrates	4·1	15·70	75·8	24·4 : 1
Blanket tailing	95·9	0·215	24·2	

The ratio of concentration of the hydraulic concentrate was .... 125 : 1
The ratio of concentration of the blanket concentrate was .... 30 : 1

Amalgamation of the Combined Concentrates:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
15·70	2·56	83·7

Cyanidation of Blanket Tailing and Amalgam Residue:

Agitation, hours	Assay, Au, oz/ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·305	0·02	93·4	0·3	3·5
48	0·305	0·01	96·7	0·4	3·7

Summary of Results, Test No. 4:

Gold recovered in hydraulic and blanket concentrates..............75·8 per cent
Bullion recovered by amalgamation of hydraulic and blanket concentrates..63·4 per cent
Bullion recovered by cyanidation of blanket tailing + amalgam residue (48 hours)..35·4 "

Overall recovery..98·8 "

HYDRAULIC CLASSIFICATION FOLLOWED BY CYANIDATION

Tests Nos. 5 to 8

The ore at −14 mesh was ground to different sizes and passed through a hydraulic classifier. The hydraulic concentrates were reground, amalgamated, and the amalgam residue added to the hydraulic tailing. This product was agitated in cyanide solution, of a strength of 1·5 pounds of potassium cyanide per ton, for 24 hours, 5 pounds of lime being added. The different products were assayed for gold.

A screen test showed the grinding to be as follows:

Mesh	Weight, per cent			
	Test No. 5	Test No. 6	Test No. 7	Test No. 8
−65+100	8·9	1·6	0·2	0·2
−100+150	20·0	8·6	3·8	3·3
−150+200	27·1	28·1	19·5	21·1
−200	43·8	61·7	76·3	75·3

Hydraulic Classification:

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
5	Feed	100·00	0·85	100·0	
	Hydraulic concentrate	1·95	14·93	34·2	51·3 : 1
	Hydraulic tailing	98·05	0·57	65·8	
6	Feed	100·00	0·85	100·0	
	Hydraulic concentrate	2·22	18·02	47·1	45 : 1
	Hydraulic tailing	97·78	0·46	52·9	
7	Feed	100·00	0·85	100·0	
	Hydraulic concentrate	0·73	53·20	45·7	137 : 1
	Hydraulic tailing	99·27	0·465	54·3	
8	Feed	100·00	0·85	100·0	
	Hydraulic concentrate	1·15	33·11	44·8	87 : 1
	Hydraulic tailing	98·85	0·475	55·2	

Amalgamation of Combined Concentrates:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
24·13	0·98	96·9

Cyanidation of Hydraulic Tailing and Amalgam Residue:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
5........	24	0·57	0·04	93·0	0·4	3·60
6........	24	0·46	0·035	92·4	0·4	3·60
7........	24	0·48	0·035	92·7	0·4	3·60
8........	24	0·48	0·035	92·7	0·4	3·60

Summary of Recoveries, Tests Nos. 5 to 8:

——	Test No. 5	Test No. 6	Test No. 7	Test No. 8
Gold recovered in hydraulic concentrates, per cent............	34·2	47·1	45·7	44·8
Bullion recovered from hydraulic concentrates, per cent.......	33·1	45·6	44·3	43·4
Bullion recovered by cyanidation, per cent....................	62·2	50·3	51·6	52·5
Overall recoveries, per cent..................................	95·3	95·9	95·9	95·9

SETTLING TEST

Test No. 9

This test was carried out in a tall glass tube having an inside diameter of $2\frac{1}{8}$ inches. The cyanide tailing from Test No. 7 was transferred to the glass tube and the level of solids (in decimals of feet) was read every five minutes. Readings were made for a two-hour period.

At the end of the test the solution was titrated for cyanide and alkalinity.

The results of the test are recorded in the following table:

Ratio of liquid to solid..............................	2·5 : 1
Lime added per ton solid..............................	5·0 pounds
Potassium cyanide.....................................	1·5 lb./ton solution.
Alkalinity of solution at end of test.................	0·30 CaO, lb./ton solution
Overflow solution.....................................	Clear
Rate of settling......................................	1·18 feet/hour

Time	Settlement of solids, in feet	Cumulative settlement, in feet
0 hr. 5 min....	0·150	0·150
0 " 10 " ...	0·220	0·370
0 " 15 " ...	0·240	0·610
0 " 20 " ...	0·240	0·850
0 " 25 " ...	0·240	1·090
0 " 30 " ...	0·230	1·320
0 " 35 " ...	0·200	1·520
0 " 40 " ...	0·165	1·685
0 " 45 " ...	0·115	1·800
0 " 50 " ...	0·080	1·880
0 " 55 " ...	0·060	1·940
1 " 0 " ...	0·055	1·995
1 " 5 " ...	0·050	2·045
1 " 10 " ...	0·040	2·085
1 " 15 " ...	0·040	2·125
1 " 20 " ...	0·030	2·155
1 " 25 " ...	0·030	2·185
1 " 30 " ...	0·030	2·215
1 " 35 " ...	0·030	2·245
1 " 40 " ...	0·025	2·270
1 " 45 " ...	0·025	2·295
1 " 50 " ...	0·020	2·315
1 " 55 " ...	0·020	2·335
2 " 0 " ...	0·015	2·350

CONCLUSIONS

The ore is readily amenable to cyanidation, and this is the process to be recommended.

Any coarse gold in the ore should be removed by means of traps and blankets prior to cyanidation, the trap and blanket concentrates being barrel-amalgamated to give a recovery of 60 to 65 per cent of the gold in the ore. A further 30 to 35 per cent of the gold should be recovered by cyanidation of the amalgam residue and blanket tailing. An overall recovery of over 97 per cent should be obtained.

Ore Dressing and Metallurgical Investigation No. 668

GOLD ORE FROM THE DARWIN GOLD MINES,* LIMITED,
WAWA, MICHIPICOTEN DISTRICT, ONTARIO

Shipment. Shipments of two separate lots of ore weighing 123 and 500 pounds, were received October 1, 1935 and February 5, 1936, from M. H. Frehburg, Mine Manager, Darwin Gold Mines, Limited, Wawa, Ontario. A further shipment of 40 pounds of mill tailing was received January 13, 1936. The different shipments were forwarded on the advice of H. A. Kee, Consulting Engineer, 304 Bay Street, Toronto, Ontario.

After crushing, cutting and grinding by standard methods, the samples assayed as follows:—

—	Oct. 1, 1935 Ore shipment	Feb. 5, 1936 Ore shipment	Jan. 13, 1936 Tailing shipment
Gold.....oz./ton	0·305	1·235	0·052
Silver.....oz./ton	0·02	0·14	....
Arsenic.....per cent	0·32	0·30	0·38

Characteristics of the Ore. The previous sample from this property reported on in Investigation No. 637 was high grade.

The present examination was carried out on comparatively low-grade ore, and in so far as microscopic information is concerned, is not nearly so satisfactory as the previous examination. The reader is, therefore, referred to the earlier report for details.

The *gangue* of the present shipment is similar to that described in Investigation No. 637 except that it contains more country rock and less quartz. The *metallic minerals* are also similar except for the native gold. Only eight grains of native gold were visible in the present sections studied; they are extremely small, ranging from minus 2300 mesh to nearly 800 mesh, and were found along a narrow fracture zone in quartz. Arsenopyrite is also disseminated as extremely tiny grains along the same zone and some of the gold is intimately associated with it.

Experimental Tests

The test work was undertaken in an endeavour to raise the percentage of recovery of the gold in the ore. Tests Nos. 1 to 7 were on the ore shipment received October 1, 1935; Tests Nos. 8 to 15 were on the mill tailing received January 13, 1936, and Tests Nos. 16 to 20 were on the ore shipment received February 5, 1936.

* Two samples of ore and one of waste rock from this mine formed the subject of Investigation No. 637, in Mines Branch Report No. 771, Investigations in Ore Dressing and Metallurgy, July to December, 1935.

BLANKET CONCENTRATION AND CYANIDATION

Test No. 1

The ore was ground in cyanide solution, passed over blankets and the blanket tailing cyanided for 24 hours. Three different lots were taken, grinding to 68·4, 71·4, and 78·5 per cent through 200 mesh.

The ratio of dilution was 1·2 : 1, 1·5 : 1, and 2 : 1. The strength of solution was 2·5 pounds of potassium cyanide per ton of solution in all cases. Five pounds of lime per ton of tailing was added. The combined blanket concentrates were barrel-amalgamated.

Cyanidation of Blanket Tailing:

Agitation, hours	Ratio of dilution	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton		Grind, per cent through 200 mesh
		Feed	Tailing		KCN	CaO	
24	1·2 : 1	0·085	0·035	58·8	1·1	4·0	68·4
24	1·5 : 1	0·085	0·025	76·5	1·1	3·9	68·4
24	2 : 1	0·085	0·025	76·5	1·1	3·9	68·4
24	1·2 : 1	0·065	0·025	61·5	0·9	4·0	71·4
24	1·5 : 1	0·065	0·025	61·5	0·9	4·0	71·4
24	2 : 1	0·065	0·025	61·5	0·9	3·9	71·4
24	1·2 : 1	0·06	0·0275	54·2	1·0	4·0	78·5
24	1·5 : 1	0·06	0·025	58·3	1·0	4·0	78·5
24	2 : 1	0·06	0·025	58·3	1·0	4·0	78·5

The amalgamation residue from the barrel amalgamation of the blanket concentrates assayed 1·04 ounces of gold per ton.

BARREL AMALGAMATION

Test No. 2

One thousand grammes of the ore was ground to pass 68·4 per cent and 78·4 per cent through 200 mesh and barrel-amalgamated for one hour.

Results:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
0·305	0·075	75·4
0·305	0·06	80·3

BLANKET CONCENTRATION AND CYANIDATION

Tests Nos. 3 and 4

In these tests, the ore was ground in water to pass 77·8 and 82·9 per cent through 200 mesh. The pulp was then passed over a corduroy blanket and the blanket tailing cyanided in different strengths of solution and at different ratios of pulp dilutions. The blanket concentrate was barrel-amalgamated and the amalgam residue cyanided.

Cyanidation of Blanket Tailing:

Results:

Test No.	Agitation, hours	Ratio of dilution	Assay, Au, oz./ton		Extraction, per cent	KCN added, lb./ton solution	Reagents consumed, lb./ton		Grind, per cent through 200 mesh
			Feed	Tailing			KCN	CaO	
3	24	2 : 1	0·08	0·025	68·8	2	0·65	4·5	77·8
3	48	2 : 1	0·08	0·025	68·8	2	0·65	4·3	77·8
3	24	3 : 1	0·08	0·025	68·8	2	0·85	4·2	77·8
3	48	3 : 1	0·08	0·025	68·8	2	1·0	4·1	77·8
4	24	2 : 1	0·07	0·02	71·4	1	0·6	4·4	82·9
4	48	2 : 1	0·07	0·02	71·4	1	0·6	4·3	82·9
4	24	3 : 1	0·07	0·02	71·4	3	0·95	4·3	82·9
4	48	3 : 1	0·07	0·02	71·4	3	1·1	4·2	82·9

The blanket concentrate was barrel-amalgamated and the amalgam residue cyanided for 24 hours. In Test No. 4, the blanket concentrate was reground prior to amalgamation.

Amalgamation:

Test No.	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
3	6·68	0·55	91·7
4	3·25	0·39	88·0

Cyanidation of Amalgamation Residue:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
3	24	0·55	0·36	34·6	2·0	6·0
4	24	0·39	0·18	53·8	2·3	7·2

The ratio of concentration of the blanket concentrate in Test No. 3 was 29·3 : 1 and in Test No. 4, 13·5 : 1.

Summary:

	Test No. 3	Test No. 4
Bullion recovered by cyanidation of blanket tailing	17·5 per cent	15·1 per cent
Bullion recovered by amalgamation of combined concentrates	68·4 "	69·3 "
Bullion recovered by cyanidation of amalgamation residue	2·1 "	5·1 "
Overall recovery	88·0 "	89·5 "

HYDRAULIC CLASSIFICATION, BLANKET CONCENTRATION, AND CYANIDATION

Test No. 5

Two thousand grammes of ore was ground in cyanide solution of 2 pounds of potassium cyanide per ton strength, to pass 70·2 per cent through 200 mesh. The pulp was then passed through a hydraulic classifier and the hydraulic tailing passed over a corduroy blanket. The blanket tailing was reground to pass 89·5 per cent through 200 mesh and portions agitated in cyanide solutions of different strengths for periods of 24 and 48 hours. The ratio of dilution was 3 : 1 and 10 pounds of lime was added per ton of tailing.

Hydraulic Classification:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, of gold per cent	Ratio of concentration
Feed	100·0	0·305	100·0	
Hydraulic concentrate	0·45	32·00	47·2	222 : 1
Hydraulic tailing	99·55	0·105	34·3	
Pregnant solution	......	......	18·5	

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, of gold per cent	Ratio of concentration
Feed	100·0	0·105	100·0	
Blanket concentrate	0·78	5·25	40·8	128 : 1
Blanket tailing	99·22	0·06	59·2	

Cyanidation of Blanket Tailing:

Agitation, hours	Assay, Au, oz./ton		Extraction per cent	KCN added, lb./ton solution	Reagents consumed, lb./ton	
	Feed	Tailing			KCN	CaO
24	0·06	0·025	58·3	1	0·3	7·9
48	0·06	0·025	58·3	1	0·3	8·0
24	0·06	0·02	66·6	2	0·6	7·8
48	0·06	0·025	58·3	2	0·6	8·2
24	0·06	0·025	58·3	3	0·3	7·6
48	0·06	0·025	58·3	3	0·4	7·7

Summary:

Gold recovered in hydraulic concentrate	47·2	per cent
Gold recovered in blanket concentrate	14·0	"
Gold recovered in grinding circuit	18·5	"
Gold recovered by cyanidation of blanket tailing	11·8	"

HYDRAULIC CLASSIFICATION, TABLE CONCENTRATION, AND CYANIDATION

Test No. 6

Four thousand grammes of ore was ground to pass 82·7 per cent through 200 mesh. The pulp was then passed through a hydraulic classifier. The hydraulic tailing was agitated in cyanide solution of 2 pounds per ton strength for 24 hours. The cyanide tailing was then passed over a Wilfley table. The table concentrate was reground to pass 100 per cent through 200 mesh and agitated in cyanide solution of 3 pounds per ton strength for 24 hours.

Hydraulic Classification:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·305	100·0	
Hydraulic concentrate	0·58	34·60	65·8	172 : 1
Hydraulic tailing	99·42	0·105	34·2	

Cyanidation of Hydraulic Tailing:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·105	0·03	71·4	0·3	4·0

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	0·03	100·0	
Table concentrate	1·6	0·93	42·8	62 : 1
Table tailing	98·4	0·02	57·2	

Cyanidation of Table Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·93	0·485	47·8	2·5	6·0

In the table concentration, a large part of the gold was lost in the slime.

HYDRAULIC CLASSIFICATION AND CYANIDATION

Test No. 7

Two thousand grammes of ore was crushed to pass 82·7 per cent through 200 mesh. The pulp was then passed through a hydraulic classifier. The hydraulic tailing was aerated in lime solution in a Denver super-agitator for 16 hours and then agitated for 24 hours in cyanide solution of 2 pounds of potassium cyanide per ton strength.

Hydraulic Classification:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·305	100·0	
Hydraulic concentrate	0·31	60·05	60·8	323 : 1
Hydraulic tailing	99·69	0·12	39·2	

Cyanidation of Hydraulic Tailing:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·12	0·03	75·0	0·4	3·9

Darwin Tailing

A shipment of 40 pounds of tailing was received January 13,1936, which assayed as follows:—

Gold	0·052 oz./ton
Arsenic	0·38 per cent

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

Mesh	Weight, per cent
− 48+65	0·1
− 65+100	1·0
−100+150	11·7
−150+200	22·2
−200	64·9

A screen analysis was made of the tailing as received.

Results:

Mesh	Weight, per cent	Assay, Au, oz./ton
− 65+100	2·4	0·06
−100+150	11·2	0·025
−150+200	20·9	0·04
−200	65·4	0·065

A similar screen analysis was made on the tailing after water-washing.

Results:

Mesh	Weight, per cent	Assay, Au, oz./ton
− 48+ 65	0·2	....
− 65+100	3·0	0·04
−100+150	11·9	0·03
−150+200	21·0	0·035
−200	63·9	0·06

CYANIDATION

Test No. 8

Samples of the tailing were ground to pass 100 per cent through 150- and 200-mesh screens. These samples together with a sample of the original tailing were agitated in cyanide solution of 2 pounds of potassium cyanide per ton strength for periods of 24 hours. Ten pounds of lime per ton of tailing was added.

Results:

Agitation, hours	Mesh grind	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
24	as rec'd.	0·0525	0·045	14·3	0·3	8·05
48	"	0·0525	0·04	23·8	0·3	8·45
24	−150	0·0525	0·05	4·8	0·3	8·20
48	−150	0·0525	0·045	14·3	0·3	8·50
24	−200	0·0525	0·05	4·8	0·3	8·65
48	−200	0·0525	0·05	4·8	0·3	8·80

TABLE CONCENTRATION AND CYANIDATION

Test No. 9

A sample of the tailing was tabled on a Wilfley table. The table concentrate was reground and agitated 24 hours in cyanide solution of 3 pounds of potassium cyanide per ton strength.

Table Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·525	100·0	
Table concentrate	0·30	0·65	37·2	333 : 1
Table tailing	99·70	0·033	62·8	

Cyanidation of Table Concentrates:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24..............	0·65	0·47	27·7	2·6	5·8

A large part of the gold was lost in the slime.

MICROSCOPIC EXAMINATION

Test No. 10

A large sample of the tailing was concentrated in a super-panning machine and the concentrate examined under the binocular microscope. The ratio of concentration was approximately 400 : 1 and the concentrate assayed 1·215 ounces of gold per ton. No free gold was visible under the microscope. The minerals in their order of abundance were: arsenopyrite, pyrite, steel from grinding circuit, magnetite, pyrrhotite, and chalcopyrite.

AMALGAMATION

Test No. 11

In order to discover whether any free gold was present in the tailing, a sample was barrel-amalgamated with mercury for one hour in a jar mill. The amalgamation tailing was assayed for gold.

Results:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
0·0525	0·0500	5·0

FLOTATION

Tests Nos. 12, 13, and 14

Samples of the tailing were floated with different reagents and the flotation concentrate and tailing assayed for gold. In Test No. 12, the tailing was conditioned with 2 pounds of soda ash and 0·05 pound of Aerofloat No. 31 per ton and floated with amyl xanthate and pine oil. In Test No. 13, the tailing was reground prior to conditioning and floated with the same reagents. In Test No. 14, Barrett No. 4 was added in place of Aerofloat No. 31 and 1·0 pound per ton of copper sulphate added in addition to 0·08 pound of amyl xanthate and 0·05 pound of pine oil. The sample was also reground prior to flotation.

Results:

Test No. 12:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·0525	100·0	15·8 : 1
Flotation concentrate	6·32	0·505	62·0	
Flotation tailing	93·68	0·020	38·0	

Test No. 13:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·0525	100·0	
Flotation concentrate	4·75	0·77	65·8	21 : 1
Flotation tailing	95·25	0·02	34·2	

Test No. 14:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·0525	100·0	
Flotation concentrate	2·33	1·05	50·0	43 : 1
Flotation tailing	97·67	0·025	50·0	

Test No. 15

The flotation concentrates from Tests Nos. 13, 14, and 15 were combined and reground to pass 95 per cent through 200 mesh and agitated in cyanide solution of a strength of 3 pounds of potassium cyanide per ton for 24 hours.

Results:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·77	0·52	32·5	2·90	9·55

The remaining test work was conducted on the sample of ore received February 5, 1936, which assayed 1·235 ounces of gold per ton.

BARREL AMALGAMATION

Tests Nos. 16 and 17

The ore was ground to pass 69·2 per cent and 74·6 per cent through 200 mesh and the pulp barrel-amalgamated with mercury for one hour.

A screen test showed the grinding as follows:—

Mesh	Weight, per cent	
	Test No. 16	Test No. 17
− 65 + 100......	1·5	0·6
−100 + 150......	7·4	4·8
−150 + 200......	21·8	19·9
−200	69·2	74·6

Amalgamation:

Test No.	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
16	1·235	0·26	79·0
17	1·235	0·235	81·0

CYANIDATION, FLOTATION, AND ROASTING

Test No. 18

The ore was ground to pass 84·4 per cent through 200 mesh. The pulp was then passed through a hydraulic classifier and the hydraulic tailing passed over a corduroy blanket. The combined hydraulic and blanket concentrates were then barrel-amalgamated with mercury for one hour. The amalgam residue was roasted at a temperature of 600° C. for one hour and the roasted residue reground and agitated in cyanide solution of a strength of 3 pounds per ton for 24 hours. Portions of the blanket tailing were agitated in cyanide solution of different strengths and results noted. A large portion of the blanket tailing was agitated in cyanide solution for 24 hours and the cyanide tailing filtered, washed, and floated. The flotation concentrate produced was roasted at a temperature of 600° C. and the roasted concentrate agitated in cyanide solution of 3 pounds per ton strength.

A screen test of the blanket tailing showed the grinding as follows:—

Mesh	Weight, per cent
−100 + 150	1·6
−150 + 200........	13·7
−200	84·4

Hydraulic Classification and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed..............................	100·0	1·235	100·0	
Hydraulic and blanket concentrates....	2·5	40·62	82·2	40 : 1
Blanket tailing......................	97·5	0·225	17·8	

The ratio of concentration of the hydraulic concentrate was.........270 : 1.
The ratio of concentration of the blanket concentrate was.. .... 47 : 1.

Amalgamation of Combined Concentrates:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
40·62	8·10	80·1

—	Gold, oz./ton	Arsenic, per cent	Sulphur, per cent
Amalgam residue assayed	8·1	3·64	2·66
Roasted amalgam residue assayed	8·82	1·18	0·24

A screen test on the reground roasted residue showed 99·6 per cent passing 200 mesh.

Cyanidation of Roasted Amalgam Residue:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	8·82	0·53	94·0	3·1	6·0

Cyanidation of the Blanket Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
A	24	0·225	0·04	82·2	0·4	4·1
B	48	0·225	0·03	86·7	0·4	4·0
C	24	0·225	0·025	88·9	0·5	4·1
D	48	0·225	0·025	88·9	0·6	4·0
E	24	0·225	0·03	86·7	0·5	3·8
F	48	0·225	0·025	88·9	0·5	3·8
G	24	0·225	0·03	86·7	0·5	4·1
H	48	0·225	0·03	86·7	0·5	4·2
I	24	0·225	0·025	88·9	0·6	4·0
J	48	0·225	0·025	88·9	0·5	4·1

In Test "A" and "B" the tailing was unchanged.
In Test "C" and "D" the tailing was reground to pass 97·6 per cent through 200 mesh.
In Test "E" and "F" the tailing was reground and aerated 16 hours.
In Test "G" and "H" the tailing had 0·12 pound per ton of lead acetate added prior to agitation.
In Test "I" and "J" the tailing had 0·12 pound per ton of sodium peroxide added after 20 hours.

Cyanidation and Flotation:

The remaining portion of the blanket tailing was agitated in cyanide solution of a strength of $1\frac{1}{2}$ pounds of potassium cyanide. The cyanide tailing was filtered, washed, and conditioned with 2 pounds of soda ash, 0·07 pound of Aerofloat No. 31, and 1·0 pound of copper sulphate, and floated with 0·2 pound of amyl xanthate and 0·05 pound of pine oil.

Cyanidation:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24...............	0·225	0·03	86·7	0·4	4·0

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed................................	100·0	0·030	100·0	
Flotation concentrate.................	5·2	0·35	60·7	19·2 : 1
Flotation tailing......................	94·8	0·0125	39·3	

Cyanidation of Roasted Flotation Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24...............	0·38	0·16	57·8	2·5	9·1

Summary:

Gold recovered in hydraulic and blanket concentrates..............82·2 per cent
Bullion recovered by amalgamation from hydraulic and blanket concentrates..65·8 "
Bullion recovered by roasting and cyaniding amalgam residue........15·4 "
Bullion recovered by cyanidation of blanket tailing.................15·4 "
Bullion recovered by flotation, roasting and cyanidation of cyanide tailing.. 1·8 "
Overall recovery...98·4 "

CYANIDATION, FLOTATION, AND ROASTING

Test No. 19

In this test, the ore was ground to pass 87·3 per cent through 200 mesh and the pulp passed through a hydraulic classifier and the hydraulic tailing passed over a blanket similarly to Test No. 18. The combined concentrates were then barrel-amalgamated and the amalgam residue added to the blanket tailing. This product was then agitated in cyanide solution of a strength of 1·5 pounds of potassium cyanide per ton for 24 hours. The cyanide tailing was filtered, washed, and conditioned with 2 pounds of soda ash, 1·00 pound of copper sulphate and 0·07 pound of Aerofloat No. 31 per ton, and floated with 0·2 pound of amyl xanthate and 0·05 pound of pine oil. The flotation concentrate was roasted at a temperature of 600° C. for one hour, reground to pass 98·0 per cent through 200 mesh, and agitated in cyanide solution of 3 pounds of potassium cyanide strength for 24 hours.

A screen test of the blanket tailing showed the grinding as follows:—

Mesh	Weight, per cent
− 65+100	0·1
−100+150	3·3
−150+200	9·2
−200	87·3

Hydraulic Classification and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	1·235	100·0	
Hydraulic and blanket concentrate	2·35	42·18	80·2	42·6 : 1
Blanket tailing	97·65	0·25	19·8	

The ratio of concentration of the hydraulic concentrate was .... 333 : 1
The ratio of concentration of the blanket concentrate was .... 48·8 : 1

Amalgamation of Combined Concentrates:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
42·18	6·74	84·0

Cyanidation of Blanket Tailing plus Amalgam Residue:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·41	0·035	92·5	0·5	3·9

Flotation of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·035	100·0	
Flotation concentrate	2·32	1·14	72·0	43 : 1
Flotation tailing	97·68	0·01	28·0	

The flotation concentrate was roasted at a temperature of 600° C. for one hour, reducing the arsenic content from 11·3 per cent to 4·68 per cent and the sulphur from 4·33 per cent to 0·38 per cent, and raising the gold from 1·14 to 1·25 ounces per ton. The flotation concentrate was then reground to pass 98·5 per cent through 200 mesh and agitated in cyanide solution of 3 pounds of potassium cyanide strength.

Cyanidation of Roasted Flotation Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	1·25	0·59	52·8	3·2	11·0

Summary:

Gold recovered in hydraulic and blanket concentrates..............80·2 per cent
Bullion recovered by amalgamation from hydraulic and blanket concentrates..............67·4 "
Bullion recovered by cyanidation of blanket tailing and amalgam residue..............30·1 "
Bullion recovered by flotation, roasting and cyanidation of cyanide tailing..............1·2 "
Overall recovery..............98·7 "

Test No. 20

In this test, the ore was ground to pass 76·4 per cent through 200 mesh. The pulp was passed through a hydraulic classifier and the hydraulic tailing passed over a blanket. The combined concentrates were amalgamated and the amalgam residue added to the blanket tailing. This product was agitated in cyanide solution of 1·5 pounds of potassium cyanide strength for 24 hours.

The cyanide tailing was reground with flotation reagents, conditioned and floated. The flotation concentrate was roasted at a temperature of 600° C. for one hour. It was then reground in cyanide solution of a strength of 3 pounds of potassium cyanide per ton, to pass 98·1 per cent through 200 mesh and finally agitated in cyanide solution of a strength of 4 pounds of potassium cyanide for 48 hours.

A screen test on the blanket tailing showed the grinding as follows:—

Mesh	Weight, per cent
−100+150	5·8
−150+200	17·8
−200	76·4

Hydraulic Classification and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	1·235	100·0	
Hydraulic and blanket concentrates	1·63	61·58	81·3	61·3 : 1
Blanket tailing	98·37	0·235	18·7	

The ratio of concentration of the hydraulic concentrate was..............192 : 1
The ratio of concentration of the blanket concentrate was..............90 : 1

Amalgamation of Combined Concentrates:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
61·58	9·53	84·5

Cyanidation of Blanket Tailing plus Amalgam Residue:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·395	0·05	87·4	0·6	3·9

The cyanide tailing was then filtered, washed, and reground with 2 pounds of soda ash, 1·0 pound of copper sulphate and 0·05 pound of Barrett No. 4 per ton to pass 90·7 per cent through 200 mesh. It was then floated with 0·2 pound of amyl xanthate and 0·05 pound of pine oil per ton. A screen test of the flotation tailing showed the grinding as follows:—

Mesh	Weight, per cent
−100 + 150	0·6
−150 + 200	8·6
−200	90·7

Flotation of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	0·05	100·0	
Flotation concentrate	3·4	1·40	90·5	29·3 : 1
Flotation tailing	96·6	0·005	9·5	

The flotation concentrate was then roasted at a temperature of 600° C. for one hour, raising the gold content from 1·40 to 1·62 ounces per ton. The roasted concentrate was then ground in cyanide solution of a strength of 3 pounds of potassium cyanide per ton to pass 99 per cent through 200 mesh. The product was then agitated in cyanide solution of a strength of 4 pounds per ton for 48 hours. The cyanide tailing was assayed for gold.

Cyanidation of Roasted Flotation Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
48	1·62	0·33	79·6	2·7	12·5

Summary:

Gold recovered in hydraulic and blanket concentrates..............	81·3 per cent
Bullion recovered by amalgamation of hydraulic and blanket concentrates..............	68·7 "
Bullion recovered by cyanidation of blanket tailing and amalgam residue..............	27·4 "
Bullion recovered by cyanidation of roasted flotation concentrate..	2·7 "
Overall recovery..............	98·8 "

SUMMARY AND CONCLUSIONS

From the test work done on the sample shipments of ore and tailing the following conclusions are made:

1. The cyanide and lime consumptions are low and no advantage is to be gained by increasing the strength of cyanide solution over 1·5 pounds of potassium cyanide per ton.

2. The ratio of dilution, whether 1 of ore to 1·5 of solution or 1 of ore to 3 of solution, does not appreciably affect the extraction.

3. Although the percentage of gold recovery is raised somewhat by finer grinding of the ore, it is questionable whether the added recovery is economical.

4. Owing to the large percentage of loss due to sliming, tabling of the cyanide tailing is not satisfactory.

5. Aeration prior to cyanidation does not aid the gold recovery.

6. Regrinding the amalgam residue before final cyanidation increases the recovery somewhat, as shown in Tests Nos. 3 and 4.

7. Flotation of the cyanide tailing followed by regrinding and cyanidation of the flotation concentrate does not give an economical recovery, as shown in Tests Nos. 12 to 16 inclusive.

8. Flotation of the cyanide tailing and roasting of the flotation concentrate, followed by regrinding and cyanidation, gives an added recovery of the gold, which should be economical to attempt.

The overall recovery of 98·8 per cent shown in Test No. 20 was based on the shipment of mine ore running 1·235 ounces of gold per ton. Assuming the gold in the run-of-mine ore to be 0·40 ounce per ton, which it is understood is the present grade, the overall recovery would be 97·5 per cent. This result being figured on the final combined tailing of 0·01 ounce of gold per ton.

The test work shows that an added recovery of from 7 to 8 per cent should be obtained on ore running 0·40 ounce of gold per ton by the use of flotation, roasting, regrinding, and cyanidation of the flotation concentrate from the cyanide tailing.

Ore Dressing and Metallurgical Investigation No. 669

GOLD ORE FROM THE A-X SYNDICATE, YELLOWKNIFE RIVER, GREAT SLAVE LAKE, N.W.T.

Shipment. A shipment of gold ore was received on December 11, 1935, from the property of the A-X. Syndicate, Yellowknife river, Great Slave Lake, N.W.T. The shipment weighed 200 pounds and was submitted by Col. C. D. H. MacAlpine, 1006 Concourse Building, Toronto, Ontario.

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

The *gangue* is dark grey, fine-textured smoky quartz.

The *metallic minerals* present in the sections are: arsenopyrite, "limonite", pyrite, and native gold. The total quantity of sulphides is extremely small; tiny scattered grains of arsenopyrite and pyrite occur in the quartz, and the pyrite has in some cases been partly replaced by limonite.

Native gold is present in the quartz, where it occurs as small, disseminated, irregular to rounded grains. The grain size of the gold is shown in the following table:

Mesh	Gold, per cent	Cumulative, per cent
+ 150	9·1	9·1
− 150 + 200	13·2	22·3
− 200 + 280	17·3	39·6
− 280 + 400	15·4	55·0
− 400 + 560	15·0	70·0
− 560 + 800	13·1	83·1
− 800 +1100	9·0	92·1
−1100 +1600	3·9	96·0
−1600 +2300	3·3	99·3
−2300	0·7	100·0
	100·0	

The gold is free in the quartz and quite finely divided. No interfering minerals are present, with the exception of limonite due to surface weathering.

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

Gold	2·434 oz./ton
~ilver	0·57 "

Experimental Tests

Test work carried out on the ore comprised amalgamation, hydraulic classification, blanket concentration, and cyanidation.

Test No. 1

This was a barrel-amalgamation test on the raw ore. A sample of ore was ground in a water pulp and then amalgamated with 100 grammes of mercury in an Abbé jar for 1 hour. The mercury was removed in a hydraulic classifier and the tailing was assayed for gold.

Gold in feed	2·434 oz./ton
Gold in tailing	0·510 "
Recovery	79·04 per cent

Screen Test of Amalgamation Tailing:

Mesh	Weight, per cent
+100	3·7
−100 +150	8·2
−150 +200	16·5
−200	71·6
	100·0

Test No. 2

In this test the ore was ground in an Abbé pebble jar and the pulp then fed to a hydraulic classifier. The oversize was panned to remove excess gangue. Considerable fine free gold was visible in the panned product.

The results of the test are as follows:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	2·29	100·00	
Oversize	0·04	1,592·54	27·86	2,500 : 1
Overflow	99·96	1·65	72·14	

Screen Test of Classifier Overflow:

Mesh	Weight, per cent
+100	1·5
−100 +150	5·7
−150 +200	14·7
−200	78·1
	100·0

Test No. 3

This was a blanket concentration test. A sample of ore was ground to a fineness of approximately 71 per cent −200 mesh and the pulp was then fed to a corduroy blanket. The concentrate was panned to remove excess gangue.

The results are as follows:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	2·12	100·00	
Concentrate	0·15	846·52	59·80	666·6 : 1
Tailing	99·85	0·855	40·20	

Test No. 4

In this test a sample of ore, weight 3,000 grammes, was ground to a fineness of approximately 71 per cent −200 mesh. The pulp was then fed to a corduroy blanket and the concentrate further concentrated by panning to remove excess gangue. The panned concentrate was amalgamated with mercury and the tailing from the amalgamation was added to the blanket tailing.

Two portions of the combined tailings were cyanided for 24 and 48 hours, respectively, in a solution having a strength equivalent to 1 pound of potassium cyanide per ton at a pulp dilution of 2·5 : 1. Five pounds of lime per ton was added at the start as protective alkalinity.

The results of the test are as follows:

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	2·434	100·00	
Concentrate	0·10	1,499·935	61·62	1,000 : 1
Tailing	99·90	0·935	38·38	

Recovery of gold by blanket concentration and amalgamation.................. $\frac{2{\cdot}434 - 1{\cdot}58}{2{\cdot}434} \times 100 = 35{\cdot}09$ per cent

Cyanidation of Blanket and Amalgamation Tailings:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton tailing	
	Feed	Tailing		KCN	CaO
24	1·58	0·56	64·56	0·10	3·75
48	1·58	0·04	97·47	0·25	4·13

Summary:

Gold recovery by blankets and amalgamation	35·09 per cent
Gold recovery by cyanidation of tailings (48 hours): 97·47 per cent of (100 − 35·09)	63·27 "
Overall gold recovery	98·36 "

Test No. 5

This was a duplicate of Test No. 4.

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	2·434	100·00	
Blanket concentrate	0·15	1,116·76	68·82	666·67 : 1
Tailing	99·85	0·76	31·18	

Recovery of gold by blanket concentration and amalgamation: $\frac{2{\cdot}434 - 1{\cdot}20}{2{\cdot}434} \times 100$ .... 50·70 per cent

Cyanidation of Blanket and Amalgamation Tailings:

Agitation, hours	Assay, Au, oz./ton		Extraction, of gold, per cent	Reagents consumed, lb./ton tailing	
	Feed	Tailing		KCN	CaO
24	1·20	0·045	96·25	0·25	5·50
48	1·20	0·04	96·67	0·50	4·75

Summary of Test No. 5:

Gold recovery by blankets and amalgamation	50·70 per cent
Gold recovery by cyanidation of tailings (48 hours): 96·67 per cent of (100 − 50·70)	47·66 "
Overall gold recovery	98·36 "

Test No. 6

In this test, two lots of ore were ground to a fineness of approximately 71 per cent −200 mesh. Each lot was then fed to a hydraulic classifier and the oversize products amalgamated. After separating the mercury, the oversize products were added to the respective classifier overflow products and treated by cyanidation, one lot for 24 hours and the other for 48 hours.

The strength of the cyanide solution was equivalent to 1 pound of potassium cyanide per ton. Lime, 5 pounds per ton, was added as protective alkalinity. The pulp dilution was 2·5 : 1. The results of the test are as follows:

Recovery of gold by amalgamation of hydraulic classifier oversize:

Lot No. 1	30·98 per cent
Lot No. 2	15·78 "

Cyanidation of Classifier Overflow and Amalgamation Tailing of Classifier Oversize:

Lot No.	Agitation, hours	Assay, Au, oz./ton		Extraction, of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
1	24	1·68	0·02	98·81	0·10	4·25
2	48	2·05	0·045	97·80	0·10	4·37

Summary:

Lot No. 1 (24-hour agitation)—

Gold recovery by hydraulic classifier and amalgamation	30·98 per cent
Gold recovery by cyanidation of classifier overflow and amalgamation tailing: 98·81 per cent of (100 − 30·98)	68·20 "
Overall gold recovery	99·18 "

Lot No. 2 (48-hour agitation)—

Gold recovery by hydraulic classifier and amalgamation	15·78 per cent
Gold recovery by cyanidation of classifier overflow and amalgamation tailing: 97·80 per cent of (100 − 15·78)	82·37 "
Overall gold recovery	98·15 "

Test No. 7

This was a duplicate of Test No. 6.

Recovery of gold by amalgamation of hydraulic classifier oversize:

Lot No. 1	28·92 per cent
Lot No. 2	41·25 "

Cyanidation of Classifier Overflow and Amalgamation Tailing of Classifier Oversize:

Lot No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
1	24	1·73	0·82	52·60	0·125	4·13
2	48	1·43	0·275	80·76	0·25	4·50

Summary:

Lot No. 1 (24-hour agitation)—

Gold recovery by hydraulic classifier and amalgamation	28·92 per cent
Gold recovery by cyanidation of classifier overflow and amalgamation tailing: 52·60 per cent of 71·08 per cent	37·39 "
Overall gold recovery	66·31 "

Lot No. 2 (48-hour agitation)—

Gold recovery by hydraulic classifier and amalgamation.........	41·25 per cent
Gold recovery by cyanidation of classifier overflow and amalgamation tailing: 80·76 per cent of 58·75 per cent............	47·45 "
Overall gold recovery..................................	88·70 "

The varying results obtained on the cyanide tailings indicate the presence of coarse gold in the cyanide circuit.

In a high-grade ore, such as the one under investigation, this condition is likely to occur, and prolonged cyanidation would be necessary to dissolve the coarser particles of gold. It is essential that this free coarse gold be removed by traps and blankets prior to cyanide treatment, in order to keep the period of contact within reasonable limits.

Test No. 8

This test was a duplicate of Tests Nos. 6 and 7. A small amount of mercury was ground with the ore.

The recovery of gold by amalgamation of hydraulic classifier oversize was:

$$\frac{2{\cdot}434 - 0{\cdot}75}{2{\cdot}434} \times 100 \ldots\ldots\ldots\ldots \quad 69{\cdot}18 \text{ per cent}$$

Cyanidation of Classifier Overflow and Amalgamation Tailing of Classifier Oversize:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton		Pulp dilution
	Feed	Tailing		KCN	CaO	
48..................	0·75	0·065	91·33	0·50	4·82	2·5 : 1

Gold recovery by hydraulic classifier and amalgamation.........	69·18 per cent
Gold recovery by cyanidation of classifier overflow and amalgamation tailing: 91·33 per cent of (100 − 69·18)............	28·15 "
Overall gold recovery..................................	97·33 "

SUMMARY AND CONCLUSIONS

The ore is amenable to cyanide treatment, and the tests indicate high recoveries with low consumption of reagents. The small amount of sulphides present in the sample of ore under investigation is favourable for its metallurgical treatment.

Moderately fine grinding is necessary to free the gold, which occurs largely as fine grains. The results of the tests indicate the presence of some coarse grains of gold.

The use of traps and blankets is recommended to remove free gold from the grinding mill—classifier circuit.

The results of tests by removal of gold on blankets indicated an overall recovery of 98·36 per cent. (*See* Tests Nos. 4 and 5.)

The results by using a hydraulic classifier indicated an overall gold recovery of 99·18 per cent. (*See* Test No. 6.)

Ore Dressing and Metallurgical Investigation No. 670

GOLD-SILVER LEAD-ZINC ORE FROM YMIR DUNDEE MINE AT YMIR, BRITISH COLUMBIA

Shipment. A shipment of three sacks of ore, net weight 202 pounds, received on December 9, 1935, was submitted by B. N. Sharp, Manager, Ymir Dundee Gold Mining Company, Limited, Ymir, British Columbia.

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

The *gangue* is light grey quartz containing light grey carbonate, and grey carbonate schist.

The *metallic minerals* are abundant and comprise about one-third the volume of the ore. They are: pyrite, sphalerite, galena, arsenopyrite, chalcopyrite, and native gold. Their abundance is as follows:

Pyrite	Most abundant
Sphalerite	Moderately abundant
Galena	" "
Arsenopyrite	Small quantity
Chalcopyrite	Very small quantity
Native gold	Rare

Pyrite occurs as medium to fine disseminated grains, concentrated along wavy bands; it is commonly included in later sulphide aggregates, and shows veining and replacement by sphalerite and galena.

Sphalerite and galena are intimately associated as small masses and small disseminated grains; small grains of each are present in masses of the other, and they are apparently contemporaneous. The small quantity of arsenopyrite occurs as small diamond-shaped crystals, usually within pyrite; as the pyrite has enclosed the well-formed crystals of the mineral, arsenopyrite is apparently older than the pyrite. Rare small grains of chalcopyrite occur in both sphalerite and galena, and in the quartz.

Four grains of native gold were seen in the sections. They ranged from 3 to 10 microns in size, and show four dissimilar modes of occurrence. The first occurs in a small grain of pyrite that is enclosed by sphalerite; the second is in pyrite that occurs in gangue; the third is associated with a small grain of galena enclosed by sphalerite; and the fourth occurs in a small grain of galena in gangue.

Paragenesis. The relations of the metallic minerals indicate that the order of deposition was as follows:

Arsenopyrite
Pyrite
Sphalerite-galena-chalcopyrite

Although no great age difference is displayed between the early sulphides (arsenopyrite and pyrite) and the later sulphides (sphalerite, galena, and chalcopyrite), the later group has veined and to some extent replaced and enclosed the earlier. The position of gold in the paragenesis may have some bearing on the deductions; all of the gold grains seen are tiny and well-rounded blebs, and have the appearance of being contemporaneous with the minerals in which they occur. It, therefore, follows that gold deposition accompanied that of both groups, and it is evident that the mode of occurrence of the gold is relatively complex. Although gold was not seen in the quartz, it may be present in such an association.

Sampling and Analysis. The ore was crushed and sampled through a Jones sampler. The sample thus obtained assayed as follows:

Gold	0·23 oz./ton
Silver	2·45 "
Copper	Trace
Lead	4·42 per cent
Zinc	4·35 "
Iron	10·32 "
Sulphur	11·44 "

Experimental Tests

Tests by cyanidation, amalgamation, and concentration, both alone and in combination were made.

By straight cyanidation, 93·5 per cent of the gold can be extracted in 48 hours with the ore ground 71·5 per cent through 200 mesh. Approximately 25 per cent of the gold is recoverable by barrel amalgamation at the same grinding. By flotation in lime pulp, with the ore ground 88 per cent through 200 mesh, a lead concentrate can be produced containing about 80 per cent of the gold and lead and assaying 1·7 ounces of gold per ton and 31·7 per cent of lead.

The tests are described in detail as follows:

CYANIDATION

Tests Nos. 1 to 4

Samples of the ore were ground 71·5 and 87·7 per cent through 200 mesh in ball mills and portions of each agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for periods of 24 and 48 hours. The cyanide tailings were assayed for gold.

Summary:

Feed sample: gold, 0·23 oz./ton.

Test No.	Grinding, per cent −200 mesh	Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
					KCN	CaO
1	71·5	24	0·02	91·4	0·80	3·5
2	71·5	48	0·015	93·5	1·09	4·0
3	87·7	24	0·02	91·4	1·07	4·2
4	87·7	48	0·015	93·5	1·35	4·2

FLOTATION

Test No. 5

A sample of the ore was ground 71·5 per cent through 200 mesh in a ball mill and then floated selectively. The products were assayed for gold, silver, lead, and zinc.

Charge to Ball Mill:

Ore	2,000 grms. −14 mesh
Soda ash	3·0 lb./ton
Sodium cyanide	0·10 "

Reagents to Cell:

Lead Flotation:

Cresylic acid	0·14 lb./ton

Zinc Flotation:

Copper sulphate	1·0 lb./ton
Sodium ethyl xanthate	0·10 "
Pine oil	0·05 "

Pyrite Flotation:

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

Summary:

Product	Weight, per cent	Assay				Distribution, per cent			
		Oz./ton		Per cent					
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Lead concentrate	8·2	0·69	12·30	37·4	8·45	33·3	53·6	69·9	15·3
Zinc "	22·3	0·21	2·74	4·75	14·00	27·5	30·6	24·1	69·5
Pyrite "	13·5	0·43	1·96	1·53	4·70	34·2	13·3	4·7	15·2
Tailing	56·0	0·015	0·09	0·10	Nil	5·0	2·5	1·3	......
Feed (cal.)	100·0	0·17	1·99	4·39	4·52	100·0	100·0	100·0	100·0

Test No. 6

A sample of the ore was ground 87·7 per cent through 200 mesh in a ball mill and floated selectively. The products were assayed for gold, silver, lead, and zinc.

Charge to Ball Mill:

Ore	2,000 grms. −14 mesh
Soda ash	1·0 lb./ton
Zinc sulphate	1·0 "

Reagents to Cell:

Lead Flotation:

Butyl xanthate	0·05 lb./ton
Cresylic acid	0·14 "

Zinc-Iron Flotation:

Copper sulphate	1·0 lb./ton
Potassium amyl xanthate	0·10 "
Pine oil	0·05 "

Summary:

Product	Weight, per cent	Assay				Distribution, per cent			
		Oz./ton		Per cent					
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Lead concentrate	6·9	1·23	19·13	40 12	14·10	36·0	53·4	66·0	22·7
Zinc-iron concentrate	39·7	0·37	2·85	3·48	8·30	61·8	45·4	32·9	77·1
Tailing	53·4	0·01	0·06	0·08	0·05	2·2	1·2	1·1	0·2
Feed (cal.)	100·0	0·237	2·49	4·20	4·27	100·0	100·0	100·0	100·0

By a comparison of Tests Nos. 5 and 6 it will be seen that the calculated gold and silver feed sample assays in Test No. 5 are considerably lower than those of Test No. 6. This is no doubt due to the dissolving action of the sodium cyanide used in Test No. 5 as a flotation reagent.

BARREL AMALGAMATION AND CYANIDATION

Tests Nos. 7 and 8

Samples of the ore were ground 71·5 and 87·7 per cent through 200 mesh in ball mills and the pulp amalgamated with mercury in jar mills for one hour. The amalgamation tailings were sampled and assayed, and portions of each were agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 24 hours. The cyanide tailings were assayed for gold.

Summary:

Feed sample: gold, 0·23 oz./ton.

Test No.	Grinding, per cent −200 mesh	Amal. tailing assay, Au, oz./ton	Extraction by amalgamation, per cent	Cyanide tailing assay, Au, oz./ton	Extraction by cyanidation, per cent	Reagents consumed, lb./ton	
						KCN	CaO
7	71·5	0·17	26·1	0·02	65·2	0·70	4·50
8	87·7	0·175	23·9	0·015	69·6	1·0	4·80

FLOTATION AND CYANIDATION

Test No. 9

A sample of the ore was ground 87·7 per cent through 200 mesh in a ball mill and a lead concentrate floated. The concentrate was cleaned and the cleaner tailing reunited with the flotation tailing. The flotation tailing was sampled and assayed and a portion of it agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 24 hours.

Charge to Ball Mill:

Ore	2,000 grms. −14 mesh
Lime	0·50 lb./ton

Reagents to Cell:

Butyl xanthate	0·05 lb./ton
Cresylic acid	0·14 "

Reagents to Cleaner Cell:

Zinc sulphate	1·0 lb./ton
Cresylic acid	0·07 "

Summary:

Product	Weight, per cent	Assay				Distribution, per cent			
		Oz./ton		Per cent					
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Lead concentrate	15·1	1·26	13·98	23·79	16·76	81·7	81·9	82·3	56·8
Flotation tailing	84·9	0·05	0·55	0·91	2·27	18·3	18·1	17·7	43·2
Feed (cal.)	100·0	0·233	2·58	4·36	4·46	100·0	100·0	100·0	100·0
Flotation tailing cyanided	....	0·0175	......	......	......	......	......	......	......

Extraction by cyanidation .... 14·0 per cent total gold

Reagents Consumed:

KCN .... 1·20 lb./ton ore
CaO .... 4·70 " "

FLOTATION AND CYANIDATION

Test No. 10

A sample of the ore was ground 87·7 per cent through 200 mesh in a ball mill and a lead concentrate floated. The concentrate was cleaned and the cleaner tailing reunited with the flotation tailing. The flotation tailing was sampled and assayed and a portion of it agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 24 hours.

Charge to Ball Mill:

Ore .... 2,000 grms. −14 mesh
Lime .... 0·50 lb./ton
Zinc sulphate .... 1·0 "

Reagents to Cell:

Butyl xanthate .... 0·05 lb./ton
Cresylic acid .... 0·14 "

No additional reagents were added to the cleaning cell.

Summary:

Product	Weight, per cent	Assay				Distribution, per cent			
		Oz./ton		Per cent					
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Lead concentrate	10·6	1·72	17·28	31·76	14·82	77·3	74·0	78·7	36·0
Flotation tailing	89·4	0·06	0·72	1·02	3·13	22·7	26·0	21·3	64·0
Feed (cal.)	100·0	0·236	2·48	4·29	4·37	100·0	100·0	100·0	100·0
Flotation tailing cyanided	....	0·0125	......	......	......	......	......	......	......

Extraction by cyanidation .... 20·1 per cent total gold

Reagents Consumed:

KCN .... 1·25 lb./ton ore
CaO .... 4·4 " "

CYANIDATION AND TABLE CONCENTRATION

Test No. 11

This test was conducted for the purpose of determining whether an economic proportion of the lead could be recovered from a cyanide tailing by table concentration.

A sample of the ore was ground 65 per cent through 200 mesh in a ball mill and then agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 24 hours. The cyanide tailing was then treated on a small laboratory-size concentrating table where a lead concentrate was taken off. The products were assayed for gold, silver, lead, and zinc.

Summary:

Product	Weight, per cent	Assay				Distribution, per cent			
		Oz./ton		Per cent					
		Au	Ag	Pb	Zn	Au	Ag	Pb	Zn
Table concentrate	9·7	0·13	5·98	17·00	6·50	41·1	39·1	37·8	16·6
Table tailing	90·3	0·02	1·00	3·00	4·20	58·9	60·9	62·2	83·4
Cyanide tailing (cal.)	100·0	0·031	1·48	4·36	4·42	100·0	100·0	100·0	100·0

Extraction by cyanidation	86·5	per cent total gold
Recovered in table concentrate	5·5	" "
Total recovery	92·0	" "

The small amount of lead recovered in the table concentrate was disappointing but this result cannot be taken as conclusive, as the test was conducted on a small table. Better results might be obtained on a large table such as would be used in a mill.

CYCLE CYANIDATION TESTS

Tests Nos. 12 to 15

A sample of ore was ground dry to pass through a 100-mesh screen, and part of it was agitated in fresh cyanide solution for 24 hours. The solution was then filtered off, made up to original strength, and then used to treat a fresh batch of ore. The filtered pulp was washed and assayed for gold, the washings being discarded.

This operation was repeated four times, to see if any decrease in extraction would be caused by fouling due to repeated use of the same solution. An increase in the cyanide tailing assays was noted.

Cyanide solutions were kept at one pound of potassium cyanide per ton and the dilution ratio was kept at 2 : 1.

Summary:

Test No.	Assay, Au, oz./ton	Extraction, per cent
12	0·02	91·3
13	0·03	86·9
14	0·035	84·8
15	0·035	84·8

Examination of the final solution showed it to contain the following:

Iron	1·76 grm./litre
Copper	0·054 "
Potassium thiocyanate	1·36 "
Reducing power	912 c.c. $\frac{N}{10}$ $KMnO_4$/litre

CONCLUSIONS

The process recommended for treatment of this ore follows along the lines of Test No. 10, with the added feature of aeration before contact with cyanide solution.

The chief reasons for these recommendations are:

(1) To recover the lead;

(2) To remove chalcopyrite if more of this mineral should occur at depth; and

(3) To counteract fouling constituents of the ore, particularly pyrrhotite if any should occur at depth.

The potential difficulties referred to under (2) and (3) may be reasonably expected to materialize, in the light of available information about the district in which this ore is found. Furthermore, in the results of Tests Nos. 12 to 15 there is definite evidence that fouling and re-precipitation of gold actually occur after repeated use of the same cyanide solution.

An outline of a proposed flow-sheet intended to overcome the abovementioned difficulties follows:

The ore would be ground with water in a ball mill, with lime and zinc sulphate added. The ball mill would discharge into a unit flotation cell where a lead concentrate would be taken off and any coarse gold trapped in the bottom of the cell. The unit cell tailing would go to a classifier, which would return the oversize to the ball mill and overflow into a thickener. (Provision should be made for the possible installation of a battery of flotation cells to treat the classifier overflow before thickening.) The thickener overflow would go to waste and the underflow would be filtered and washed. The filter cake would be repulped in lime water and aerated for 8 hours, after which the cyanide would be added and agitation carried on for 24 hours.

Ore Dressing and Metallurgical Investigation No. 671

ARSENICAL GOLD ORE FROM WHITEWATER MINE, TAKU RIVER DISTRICT, ATLIN MINING DIVISION, TULSEQUAH, BRITISH COLUMBIA

Shipment. Thirty-four sacks of gold ore, weighing 4,120 pounds, were received November 5, 1935 from D. C. Sharpstone, Duluth, Minnesota, U.S.A. The ore was taken from the Whitewater mine, Taku River district, 6 miles north of Tulsequah, British Columbia. The shipment consisted of two samples, the larger one weighed approximately two tons and consisted of sixteen smaller samples, numbered 500 to 515 inclusive. Each of these was from a different part of the ore-body or from a different vein. The weights of these were roughly proportionate to the tonnage of ore in that particular part. The second sample was numbered 5200 and was taken from a different part of the property.

Two earlier shipments of ore from this property had been received and examined. The first was received in August 1932 from the N.A. Timmins Corporation and the second in March 1935 from D. C. Sharpstone. Results of this latter investigation are shown in Investigation No. 632 of the Ore Dressing and Metallurgical Laboratories.

Tests were desired to see if the results were conformable to those obtained on the rather different ore of the earlier investigations and to gain further information concerning the effect of re-treating the cleaner tailing. Sample No. 5200 which contained antimony was sent for preliminary concentration tests to determine the gold recovery.

Character of the Ore: Twelve representative specimens were selected from Sample No. 1, and polished sections prepared and examined microscopically. Six similar sections of Sample No. 5200 also were examined.

Sample No. 1 is similar microscopically to the samples studied in previous investigations. The gangue is fine-textured, dark to light green carbonate rock, which is locally schistose. It contains stringers and veinlets of calcite and patches of rusty to white quartz.

The metallic minerals present are arsenopyrite, pyrite, an unknown grey mineral, and pyrrhotite. Arsenopyrite and pyrite are the only abundant sulphides. The former occurs disseminated as extremely fine to medium crystals, which show a distinct tendency to elongation to the needle-like form. The pyrite occurs as medium to fine irregular grains usually, but not always, associated with arsenopyrite. A very small amount of an unknown grey mineral occurs as rare tiny grains in pyrite. Extremely rare tiny grains of pyrrhotite are also present in pyrite.

A grain analysis of the arsenopyrite and pyrite is shown in the following table. The percentages are by volume on a basis of 100 per cent of the two minerals.

Mesh	Arsenopyrite, per cent	Pyrite, per cent	Total, per cent
+ 65	4·4	4·8	9·2
− 65 +100	6·4	5·1	11·5
−100 +150	7·8	5·1	12·9
−150 +200	9·0	6·3	15·3
−200 +280	7·1	5·0	12·1
−280 +400	5·7	3·7	9·4
−400 +560	6·6	2·3	8·9
−560	16·7	4·0	20·7
Totals	63·7	36·3	100·0

Calculated approximately by weight, the proportions of arsenopyrite and pyrite are:

Arsenopyrite	68·3 per cent
Pyrite	31·7 "

Sample No. 5200 is somewhat different from Sample No. 1. The gangue is mottled grey to white fine-textured carbonate rock with patches of white quartz, which is stained yellowish brown in some places by iron oxides. It is marked by the absence of the light green material that is so prominent in the other sample.

The metallic minerals present are arsenopyrite, pyrite, and stibnite. All three minerals occur in considerable quantities. Arsenopyrite is typically very finely divided and pyrite occurs as irregular grains and is often associated with the arsenopyrite. Stibnite, on the other hand, is typically coarse and occurs as irregular stringers, patches, and small grains, all of which appear to lie along incipient fractures in the gangue.

In all of the sections examined no native gold is visible. The evidence is strongly in favour of the supposition that the gold is contained in the sulphides in submicroscopic form.

Experimental Tests

Samples Nos. 500 to 515 inclusive, comprising large Sample No. 1, were crushed, ground, and sampled individually; all these samples were then mixed and again sampled. Sample No. 5200 was handled separately. The analyses of the samples are as follows:

Sample No.	Weight, pounds	Assay				
		Au, oz./ton	As, per cent	Sb, per cent	Fe, per cent	S per cent
500	221	0·26	0·44	None	2·90	0·60
501	216	0·25	0·44	"	2·70	0·69
502	119	0·53	1·64	"	5·70	1·72
503	341	0·555	1·39	"	4·17	1·51
504	116	0·19	0·58	"	4·68	0·96
505	236	0·485	1·59	"	3·96	1·16
506	260	0·32	0·84	"	4·43	1·48
507	230	0·195	0·58	"	4·94	1·47
508	243	0·255	0·65	"	5·00	1·57
509	175	0·47	1·26	"	4·01	1·33
510	155	0·39	0·84	"	4·12	1·10
511	179	0·38	1·46	"	5·51	2·88
512	321	0·46	1·20	"	4·22	1·90
513	419	0·48	1·10	"	4·43	1·28
514	154	0·41	1·36	"	5·15	1·95
515	530	0·44	1·56	"	5·10	2·00
500 to 515	3915	0·405	1·10	"	4·39	1·69
5200	87	0·78	2·17	1·16	5·82	2·48

Sample No. 1

The experimental tests conducted on Sample No. 1 were a continuation of those reported in March 1935. Grinding to 98 per cent −200 mesh was adopted as necessary for maximum recovery. A small addition of copper sulphate to the flotation cells was found to increase the flotability of the sulphides. The details and results of these preliminary tests are practically the same as those in the former investigation and are, therefore, not included in this later work.

CYCLE TEST

Test No. 1

To determine the effect of returning the cleaner tailing to the circuit, a locked test was carried out.

A sample of the ore was ground wet to pass 98·8 per cent through 200 mesh. Soda ash, 5 pounds per ton, and 0·20 pound of Barrett No. 4 were added to the grinding mill. The pulp was then conditioned for ten minutes with 1·0 pound of copper sulphate and 0·2 pound of potassium amyl xanthate per ton. A rougher concentrate was removed by adding 0·15 pound of pine oil. This concentrate was cleaned once and the resulting cleaner tailing added to the flotation cell together with a second batch of ore ground in the same manner as was batch No. 1. The mixture then was floated and the cleaner tailing from this second concentrate returned to the cell with the third lot of ore. In all, five cycles were made.

Results:

Cycle No.	Concentrate		Tailing
	Weight, per cent	Assay, Au, oz./ton	Au, oz./ton
1	5·4	6·02	0·055
2	6·4	5·83	0·045
3	5·9	5·94	0·04
4	7·0	5·24	0·045
5	7·1	5·20	0·04
Final cleaner tailing	7·2	0·37	....

These results indicate that the return of the cleaner-tailing to the feed of the flotation circuit does not lower the recovery.

Mill Run No. 1

To determine on a larger scale the results obtainable on Sample No. 1, the ore crushed to quarter inch was fed at the rate of 135 pounds per hour to a ball mill in closed circuit with a classifier. Extremely fine grinding was not obtainable as the classifier overflow was maintained at 35 per cent solids for flotation. This overflow passed to a conditioning tank and from there to the second cell of a ten-cell flotation machine. A rougher concentrate was taken from cells Nos. 2 and 3 and a scavenger concentrate from cells Nos. 4 to 10. This last concentrate was returned to cell No. 2 with the feed. The concentrate from cells Nos. 2 and 3 was not cleaned.

Screen Test of Classifier Overflow:

Mesh	Weight, per cent
+100	5·5
−100 +150	9·1
−150 +200	14·8
−200	70·3

Reagents:

To Ball Mill:

	Lb./ton ore
Soda ash	5·0
Barrett No. 4	0·09

To Conditioning Tank:

Copper sulphate	1·0
Potassium amyl xanthate	0·15
Cresylic acid	0·15

Results:

Feed	0·405 Au, oz./ton
Classifier overflow	0·35 " "
Flotation concentrate	5·60 " "
Flotation tailing	0·035 " "
Recovery	90·5 per cent
Ratio of concentration	17·7 : 1

Screen Analysis of Flotation Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton
+ 100	8·0	0·08
−100 + 150	8·3	0·055
−150 + 200	15·0	0·045
−200	68·7	0·03

Mill Run No. 2

For this run, the feed was cut to 100 pounds per hour. The quantities of reagents added were the same as those in the preceding run. The flow-sheet was the same also.

Screen Test of Classifier Overflow:

Mesh	Weight, per cent
+ 65	0·8
− 65 +100	5·5
−100 +150	8·4
−150 +200	15·2
−200	70·1

Results:

Feed	0·405	Au, oz./ton
Classifier overflow	0·30	" "
Flotation concentrate	5·84	" "
Flotation tailing	0·03	" "
Recovery	93·3	per cent
Ratio of concentration	16·1 : 1	

Screen Analysis of Flotation Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton
+ 100	7·1	0·05
−100 + 150	13·3	0·04
−150 + 200	9·1	0·035
−200	70·5	0·025

Mill Run No. 3

This run is similar to Run No. 2, with the exception that the concentrate from cells Nos. 2 and 3 was cleaned in cell No. 1. The cleaner-tailing was returned to cell No. 2 with the feed.

Screen Test of Classifier Overflow:

Mesh	Weight, per cent
+ 65	0·2
− 65 + 100	4·4
−100 + 150	9·7
−150 + 200	14·5
−200	71·2

Results:

Feed	0·405 Au, oz./ton
Classifier overflow	0·395 " "
Flotation concentrate	6·64 " "
Flotation tailing	0·04 " "
Recovery	90·7 per cent
Ratio of concentration	18·1 : 1

Screen Analysis of Flotation Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton
+ 100	4·5	0·09
−100 + 150	14·8	0·055
−150 + 200	9·6	0·04
−200	71·1	0·035

Mill Run No. 4

Endeavouring to make a higher grade concentrate, sodium silicate at the rate of one pound per ton, together with four pounds of soda ash was fed to the ball mill. All other conditions were the same as those in Run No. 3.

Screen Test of Classifier Overflow:

Mesh	Weight, per cent
+ 100	1·6
−100 + 150	12·2
−150 + 200	8·7
−200	77·5

Results:

Feed	0·40 Au, oz./ton
Classifier overflow	0·395 " "
Flotation concentrate	5·72 " "
Flotation tailing	0·03 " "
Recovery	94·5 per cent
Ratio of concentration	15·6 : 1

Screen Analysis of Flotation Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton
+ 100	3·8	0·06
−100 + 150	10·9	0·045
−150 + 200	9·5	0·04
−200	75·8	0·025

ROASTING AND CYANIDATION OF FLOTATION CONCENTRATE

To determine what recovery of gold could be obtained from the flotation concentrate on the property, a sample of concentrate from Mill Run No. 3 was dead-roasted, washed, and cyanided.

Results:

Before roasting	5·82 Au, oz./ton
After roasting	8·40 " "
Loss in weight	33·2 per cent

Cyanidation:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent
	Feed	Tailing	
48	8·40	2·44	70·9
72	8·40	2·45	70·9
96	8·40	2·65	68·5

It is apparent that a high extraction of gold, by cyaniding the roasted concentrate, is not to be expected.

Sample No. 5200

Selective flotation tests were run on this sample to determine if an antimony concentrate low in gold could be made and a gold-arsenopyrite concentrate recovered. Bulk flotation tests also were carried out.

Analysis showed the sample to contain:—

Gold	0·78 oz./ton	Antimony	1·16 per cent
Silver	0·07 "	Iron	5·82 "
Arsenic	2·17 per cent	Sulphur	2·48 "

Test No. 1

A sample of the ore was ground wet in a natural (non-alkaline) circuit until 92·8 per cent passed 200 mesh; 0·06 pound of pine oil per ton was then added and a concentrate removed. The pulp was then conditioned with 1·0 pound of copper sulphate and 0·20 pound of amyl xanthate and a second concentrate removed with 0·06 pound pine oil.

Results:

Product	Weight, per cent	Assay			Distribution, per cent		
		Au, oz./ton	Sb, per cent	As, per cent	Au	Sb	As
Feed (cal.)	100·0	0·78	1·04	2·14	100·0	100·0	100·0
Concentrate No. 1	3·7	2·26	24·27	11·02	10·7	85·9	19·1
Concentrate No. 2	4·2	3·74	3·28	5·06	20·2	13·2	9·9
Tailing	92·1	0·585	0·01	1·64	69·1	0·9	71·0

A recovery of 85·9 per cent of the antimony in concentrate No. 1 is made, with a ratio of concentration of 27 : 1. This concentrate contains 2·26 ounces of gold per ton.

Test No. 2

In this test, the ore was ground in an alkaline circuit containing six pounds of soda ash per ton of ore. One concentrate was then taken off by adding 0·06 pound of pine oil to the flotation cell.

Results:

Product	Weight, per cent	Assay			Distribution, per cent		
		Au, oz./ton	Sb, per cent	As, per cent	Au	Sb	As
Feed (cal.)	100·0	0·79	1·17	2·17	100·0	100·0	100·0
Concentrate	10·0	3·84	10·06	10·56	48·7	86·0	48·6
Tailing	90·0	0·45	0·18	1·24	51·3	14·0	51·4

The antimony concentrate carries 48·7 per cent of the gold.

Test No. 3

This is a selective flotation test in which the arsenopyrite was depressed by cyanide and an antimony concentrate removed. The arsenopyrite was then activated and floated.

The sample was ground wet with 6·0 pounds of soda ash and 0·20 pound of sodium cyanide per ton. A screen test showed the grind to be 90 per cent −200 mesh; 0·08 pound of pine oil was added to the pulp in the flotation cell and an antimony concentrate removed; 1·0 pound of copper sulphate and 0·20 pound of amyl xanthate then were added and after conditioning, a second concentrate was taken.

Results:

Product	Weight, per cent	Assay			Distribution, per cent		
		Au, oz./ton	Sb, per cent	As, per cent	Au	Sb	As
Feed (cal.)	100·0	0·80	1·14	2·10	100·0	100·0	100·0
Antimony concentrate	3·7	1·90	27·44	3·21	8·8	89·3	5·6
Arsenic concentrate	12·0	5·38	0·66	14·21	80·7	7·0	81·1
Tailing	84·3	0·10	0·05	0·33	10·5	3·7	13·3

A concentrate containing 89·3 per cent of the antimony, and assaying 27·44 per cent antimony with a ratio of concentration of 27: 1, is obtained. A second concentrate, which assays 5·38 ounces of gold per ton, recovers 80·7 per cent of the gold.

Test No. 4

In this test, three parts of Sample No. 1 and one part of Sample No. 5200 were mixed and ground in a ball mill with 6 pounds of soda ash per ton; 0·06 pound of cresylic acid was added to the flotation cell and an antimony concentrate removed; 1·0 pound of copper sulphate, 0·20 pound of amyl xanthate and 0·12 pound of a mixture of coal-tar creosote, coal tar, and cresylic acid were added and a second concentrate removed.

Results:

Product	Weight, per cent	Assay			Distribution, per cent		
		Au, oz./ton	Sb, per cent	As, per cent	Au	Sb	As
Feed (cal.)	100·0	0·50	0·41	1·47	100·0	100·0	100·0
Antimony concentrate	5·0	3·82	5·86	10·40	37·9	70·6	35·4
Arsenic concentrate	5·3	4·72	0·44	14·38	49·6	5·6	51·8
Tailing	89·7	0·07	0·11	0·21	12·5	23·8	12·8

SUMMARY AND CONCLUSIONS

The results obtained in continuous mill runs on Sample No. 1 show that recoveries of from 90 to 94 per cent of the gold can be expected. The grade of concentrate should be from 5·7 to 6·0 ounces of gold per ton. Ratio of concentration will be approximately 16 : 1.

Fine grinding to at least 90 per cent −200 mesh is indicated. An examination of the screen analyses of the flotation tailings of the various mill runs confirms this. The return of the cleaner tailing to the circuit does not lower the recovery of gold.

Sample No. 5200, which contains antimony, should be treated in a separate unit. Selective flotation producing an antimony concentrate and a gold-arsenopyrite concentrate is the most suitable method to apply to this type of ore.

Ore Dressing and Metallurgical Investigation No. 672

GOLD ORE FROM THE HEDLUND PROPERTY OF THE ERIE CANADIAN MINES, LIMITED, MATACHEWAN DISTRICT, NORTHERN ONTARIO

Shipment. Thirty bags of ore, weighing 2,020 pounds, were received December 24, 1935, from C. E. Rodgers, Manager, Erie Canadian Mines Limited, P.O. Box EX, Kirkland Lake, Ontario.

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

Gold.............	0·17 oz./ton	Sulphur...........	2·95 per cent
Silver.............	0·05 "	Copper............	Trace
Iron..............	6·63 per cent	Arsenic............	Nil.

Characteristics of the Ore. The *gangue* is rather fine-textured, green to grey mottled chloritic rock containing patches and grains of carbonate. The larger carbonate patches are white calcite.

The *metallic minerals* present in the sections are: pyrite, magnetite (and probably ilmenite?), chalcopyrite, and pyrrhotite. Pyrite is quite abundantly disseminated as coarse to fine cubes and irregular crystalline aggregates. A small quantity of magnetite (and ilmenite?) occurs as small disseminated grains. Chalcopyrite and pyrrhotite are extremely rare and occur as tiny inclusions in pyrite. No native gold was seen.

Experimental Tests

The test work included amalgamation, flotation, table concentration, and cyanidation. Cyanidation followed by flotation of the cyanide tailing and regrinding and cyanidation of the flotation concentrate produced the highest overall recovery, namely 92·4 per cent of the gold.

AMALGAMATION

Test Nos. 1 and 2

In these tests 1,000-gramme samples of ore at −14 mesh were ground 25 and 35 minutes in a ball mill to pass 66·4 and 89·9 per cent through 200 mesh respectively. The pulp was then amalgamated with mercury for one hour in a jar mill. The amalgamation tailing was assayed for gold.

Results:

Test No.	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
1	0·17	0·15	11·8
2	0·17	0·15	11·8

The above tests indicate that the amount of free-milling gold in the ore is very small at the degrees of comminution involved.

HYDRAULIC CLASSIFICATION AND FLOTATION

Test No. 3

The ore at −14 mesh was ground to pass 78·8 per cent through 200 mesh. The pulp was then passed through a hydraulic classifier and any coarse gold and heavier mineral particles removed. The hydraulic tailing was then conditioned with 2 pounds of soda ash per ton and floated with 0·2 pound of amyl xanthate and 0·05 pound of pine oil per ton. The different products were assayed for gold.

A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65 +100	2·3
−100 +150	5·3
−150 +200	13·5
−200	78·8

Hydraulic Classification:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·17	100·0	
Hydraulic concentrate	0·73	2·20	9·4	137 : 1
Hydraulic tailing	99·27	0·155	90·6	

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·155	100·0	
Flotation concentrate	16·9	0·75	81·9	5·9 : 1
Flotation tailing	83·1	0·03	18·1	

Summary:

	Per cent
Gold recovered in hydraulic concentrate	9·4
Gold recovered in flotation concentrate	74·2

FLOTATION

Test No. 4

The ore at −14 mesh was ground with 2 pounds of soda ash and 0·12 pound of Barrett No. 4 per ton, for 35 minutes in a ball mill. The pulp was then floated with 0·2 pound of amyl xanthate and 0·05 pound of pine oil and a flotation concentrate removed. A screen test showed the grinding as follows:

Mesh	Weight, per cent
−100 +150	0·7
−150 +200	9·0
−200	89·9

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·16*	100·0	
Flotation concentrate	9·55	1·44	86·0	10·5 : 1
Flotation tailing	90·45	0·025	14·0	

*Calculated.

FLOTATION AND BLANKET CONCENTRATION

Test No. 5

The ore at −14 mesh was ground in a ball mill with 2 pounds of soda ash and 0·1 pound of Barrett No. 4 per ton for 25 minutes. The pulp was then floated with 0·2 pound of amyl xanthate and 0·05 pound of pine oil. A flotation concentrate was removed. The flotation tailing was then passed over a corduroy blanket set at a slope of 2·5 inches per foot, and a blanket concentrate recovered. The different products were assayed for gold. A screen test showed the grinding as follows:—

Mesh	Weight, per cent
− 65 +100	3·6
−100 +150	11·1
−150 +200	18·6
−200	66·4

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·17	100·0	
Flotation concentrate	22·8	0·57	77·1	4·4 : 1
Flotation tailing	77·2	0·05	22·9	

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·05	100·0	
Blanket concentrate	1·0	0·21	4·4	100 : 1
Blanket tailing	99·0	0·045	95·6	

Summary:

	Per cent
Gold recovered in flotation concentrate	77·1
Gold recovered in blanket concentrate	1·0

CYANIDATION

Tests Nos. 6, 7, 8, and 9

In these tests the ore at −14 mesh was ground in a ball mill to different degrees of fineness. The pulp was then made up to 3 : 1 dilution and 1 pound of potassium cyanide per ton of solution and 10 pounds of lime per ton of ore added. Agitation was conducted for periods of 24 and 48 hours. The cyanide tailing was assayed for gold. Screen tests showed the different grinds as follows:—

Mesh	Weight, per cent			
	Test No. 6	Test No. 7	Test No. 8	Test No. 9
+ 65	0·1	....	....	....
− 65 +100	6·8	0·5	0·5	....
−100 +150	11·6	4·1	3·1	0·7
−150 +200	21·5	14·0	12·0	6·7
−200	59·9	81·2	84·3	92·4

Cyanidation:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
6	24	0·17	0·04	76·5	0·3	8·35
6	48	0·17	0·035	79·4	0·4	8·65
7	24	0·17	0·03	82·4	0·4	8·50
7	48	0·17	0·03	82·4	0·4	8·80
8	24	0·17	0·02	88·2	0·4	8·50
8	48	0·17	0·03	82·4	0·4	8·80
9	24	0·17	0·02	88·2	0·4	8·35
9	48	0·17	0·025	85·3	0·4	8·45

Maximum extraction is apparent at a grinding of 84·3 per cent through 200 mesh and 24 hours' agitation. There is a tendency towards re-precipitation of the gold in the 48-hour agitations.

Tests Nos. 10, 11, 12, and 13

In these tests, the ore was ground to pass 90 per cent through 200 mesh and portions agitated for 24 hours in cyanide solutions as described below. The cyanide tailings were assayed for gold.

In Test No. 10, 2 pounds of potassium cyanide per ton of solution and 5 pounds of lime per ton of ore were added.

In Test No. 11, aeration was conducted prior to cyanidation for a period of 16 hours.

In Test No. 12, 0·6 pound of lead acetate per ton of ore was added.

In Test No. 13, 0·6 pound of sodium peroxide per ton of ore was added after 20 hours' agitation.

In all these tests, 2 pounds of potassium cyanide per ton of solution and 5 pounds of lime per ton of ore were added and the solution kept at those strengths. A pulp dilution of 2·5 : 1 was used.

Results:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
10	24	0·17	0·025	85·3	0·4	8·2
11	24	0·17	0·025	85·3	0·4	8·5
12	24	0·17	0·025	85·3	0·4	8·8
13	24	0·17	0·025	85·3	0·4	8·5

The smaller quantity of lime added at the start of agitation, namely 5 pounds per ton of ore, is not so effective as the 10 pounds added in Tests Nos. 8 and 9.

CYANIDATION AND TABLE CONCENTRATION

Test No. 14

The ore at −14 mesh was ground to pass 57·9 per cent through 200 mesh. The pulp was agitated in cyanide solution of 1 pound of potassium cyanide per ton strength, and 5 pounds of lime per ton of ore added. The cyanide tailing was passed over a Wilfley table and the table concentrate reground and agitated in cyanide solution of 5 pounds per ton strength for 24 hours. A screen analysis on the table tailing showed the grinding and assay values as follows:—

Mesh	Weight, per cent	Assay, Au, oz./ton
− 35 + 48	0·3	....
− 48 + 65	2·3	0·03
− 65 +100	10·5	0·025
−100 +150	11·2	0·025
−150 +200	17·5	0·02
−200	57·9	0·02

Cyanidation:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·17	0·05	70·6	0·4	8·0

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·05	100·0	
Table concentrate	8·8	0·35	62·0	11·4 : 1
Table middling	6·1	0·035	4·0	
Table tailing	85·1	0·02	34·0	

Cyanidation of Table Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·35	0·065	81·4	1·1	11·2

Summary:

Gold recovered by cyanidation of raw ore	70·6 per cent
Gold recovered by cyanidation of table concentrate	14·8 "
Overall recovery	85·4 "

CYANIDATION AND TABLE CONCENTRATION

Test No. 15

The ore at −14 mesh was ground to pass 77·5 per cent through 200 mesh and agitated with 1 pound of potassium cyanide per ton of solution and 10 pounds of lime per ton of ore for 24 hours. The ratio of pulp dilution was 2·5 : 1. The cyanide tailing was sampled and passed over a Wilfley table. The table concentrate was reground to pass 95 per cent through 200 mesh and agitated in cyanide solution of 4 pounds per ton strength for 24 hours. The various products were assayed for gold.

A screen test of the cyanide tailing showed the grinding as follows:—

Mesh	Weight, per cent
− 65 +100	0·9
−100 +150	5·6
−150 +200	15·9
−200	77·5

Cyanidation:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·17	0·025	85·3	0·3	8·0

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·03*	100·0	
Table concentrate	9·3	0·165	50·8	10·7 : 1
Table middling	12·1	0·025	10·0	
Table tailing	78·6	0·015	39·2	

*Calculated.

Cyanidation of Table Concentrates:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·165	0·04	75·8	1·3	14·1

Summary:

Gold recovered by cyanidation of raw ore........ 85·3 per cent
Gold recovered by cyanidation of table concentrate........ 5·7 "
Overall recovery........ 91·0 "

CYANIDATION AND FLOTATION

Tests Nos. 16 and 17

The ore at −14 mesh was ground in water to pass 70·0 per cent through 200 mesh. In Test No. 16, 1 pound of potassium cyanide per ton of solution was used and in Test No. 17, 2 pounds of potassium cyanide. Ten pounds of lime per ton of ore was added in both tests. The ratio of dilution was 2·5 : 1. After agitation, the cyanide tailing was washed and conditioned with 2 pounds of soda ash and 0·15 pound of Barrett No. 4 per ton and floated with 0·2 pound of amyl xanthate and 0·05 pound of pine oil per ton. The flotation concentrates were removed, reground to pass 95 per cent through 200 mesh; combined and agitated in cyanide solution strength of 5 pounds of potassium cyanide per ton for 24 hours. The various products were assayed for gold.

Cyanidation:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
16	24	0·17	0·035	79·4	0·4	8·2
17	24	0·17	0·025	85·3	0·4	8·3

Flotation of Cyanide Tailing:

Test No. 16

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·035	100·0	
Flotation concentrate	8·7	0·21	66·8	11·5 : 1
Flotation tailing	91·3	0·01	33·2	

Test No. 17

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·025	100·0	
Flotation concentrate	9·7	0·19	67·2	10·3 : 1
Flotation tailing	90·3	0·01	32·8	

Cyanidation of Combined Concentrates:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents, consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·20	0·06	70·0	1·4	13·9

Summaries:

Test No. 16

Gold recovered by cyanidation of raw ore	79·4 per cent
Gold recovered by cyanidation of flotation concentrate	9·6 "
Overall recovery	89·0 "

Test No. 17

Gold recovered by cyanidation of raw ore	85·3 per cent
Gold recovered by cyanidation of flotation concentrate	6·9 "
Overall recovery	92·2 "

Tests Nos. 18, 19, 20, and 21

The procedure in these tests was similar to that in Tests Nos. 16 and 17, namely cyanidation, flotation of cyanide tailing, and regrinding and cyanidation of the flotation concentrate. The initial grinds were 90·8 through 200 mesh in Tests Nos. 18 and 19 and 97·8 per cent in Tests Nos. 20 and 21. One pound of potassium cyanide per ton of solution was used in Tests Nos. 18 and 20, and 2 pounds of potassium cyanide in Tests Nos. 19 and 21. Ten pounds of lime per ton of ore was added in each case.

The ratio of pulp dilution was 2·5 : 1. Screen tests showed the grinding as follows:—

Mesh	Weight, per cent	
	Test Nos. 18 and 19	Test Nos. 20 and 21
−100 +150	0·7	0·1
−150 +200	7·8	1·8
−200	90·8	97·8

Cyanidation:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
18	24	0·17	0·02	88·8	0·4	8·3
19	24	0·17	0·02	88·8	0·5	8·4
20	24	0·17	0·02	88·8	0·4	8·6
21	24	0·17	0·02	88·8	0·5	8·6

FLOTATION OF CYANIDE TAILING

The cyanide tailing of Tests Nos. 18 and 19 were combined and floated, and similarly the cyanide tailing of Tests Nos. 20 and 21.

Tests Nos. 18 and 19:

Product	Weight, per cent	Assay, Au, oz./ton	Distri-bution, per cent	Ratio of concen-tration
Feed	100·0	0·02	100·0	
Flotation concentrate	9·0	0·12	55·0	11·1 : 1
Flotation tailing	91·0	0·01	45·0	

Tests Nos. 20 and 21:

Feed	100·0	0·02	100·0	
Flotation concentrate	8·6	0·12	55·0	11·6 : 1
Flotation tailing	91·4	0·01	45·0	

CYANIDATION OF FLOTATION CONCENTRATES

The flotation concentrates of the four tests were combined, reground and agitated in cyanide solution strength of 5 pounds of potassium cyanide per ton for 24 hours.

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·12	0·05	58·4	1·4	13·5

Summary:

Gold recovered by cyanidation of raw ore	88·8 per cent
Gold recovered by cyanidation of flotation concentrates	3·6 "
Overall recovery	92·4 "

SETTLING TESTS

Tests Nos. 22 and 23

These tests were carried out in a tall glass tube having an inside diameter of 2⅛ inches. The ore was ground to pass 75·8 per cent through 200 mesh in Test No. 22, and 96·6 per cent in Test No. 23. Lime was added, prior to grinding, at the rate of 10 pounds per ton of ore in both cases. The ratio of pulp dilution was 2·5 : 1. Readings were made for a one-hour period. At the end of the test, the solution was titrated for lime.

Test No. 22

Ratio of liquid to solid	2·5 : 1
Lime added per ton solid	10·0 pounds
Alkalinity of solution at end of test	0·4 CaO, lb./ton solution
Overflow	Clear
Rate of settling	0·52 foot/hour

A screen test showed the grinding as follows:—

Mesh	Weight, per cent
− 65 +100	1·4
−100 +150	5·9
−150 +200	16·6
−200	75·8

Time	Settlement of solids in feet	Cumulative settlement
0 hours 5 minutes	0·050	0·050
0 " 10 "	0·050	0·100
0 " 15 "	0·040	0·140
0 " 20 "	0·040	0·180
0 " 25 "	0·045	0·225
0 " 30 "	0·045	0·270
0 " 35 "	0·040	0·310
0 " 40 "	0·045	0·355
0 " 45 "	0·040	0·395
0 " 50 "	0·045	0·440
0 " 55 "	0·040	0·480
1 " 0 "	0·040	0·520

Test No. 23

Ratio of liquid to solid	2·5 : 1
Lime added per ton solid	10·0 pounds
Alkalinity of solution at end of test	0·35 CaO, lb. /ton solution
Overflow	Clear
Rate of settling	0·32 foot/hour

A screen test showed the grinding as follows:—

Mesh	Weight, per cent
−100 +150	0·2
−150 +200	3·1
−200	96·6

Time	Settlement of solids in feet	Cumulative settlement
0 hours 5 minutes	0·030	0·030
0 " 10 "	0·035	0·065
0 " 15 "	0·030	0·095
0 " 20 "	0·030	0·125
0 " 25 "	0·030	0·155
0 " 30 "	0·025	0·180
0 " 35 "	0·020	0·200
0 " 40 "	0·030	0·230
0 " 45 "	0·025	0·255
0 " 50 "	0·020	0·275
0 " 55 "	0·025	0·300
1 " 0 "	0·020	0·320

The tests show that the time of settling of the pulp is longer than normal. Fine grinding, as in Test No. 23, accentuates this condition.

CONCLUSIONS

Cyanidation followed by either flotation or table concentration of the cyanide tailing and regrinding and cyanidation of the resulting concentrate, appears to be the proper procedure to follow on this ore.

Ore Dressing and Metallurgical Investigation No. 673

BARITE-BEARING MILL TAILING FROM THE KAMLOOPS HOMESTAKE MINES, LIMITED, JAMIESON CREEK, B.C.

Shipment. A shipment of mill tailing was received on April 1, 1936, from the Kamloops Homestake Mines, Limited, Jamieson Creek, Kamloops mining division, B.C. The shipment weighed 200 pounds and it was submitted by Robert T. Colquhon, 475 Howe Street, Vancouver, B.C.

Sampling and Analysis: The tailing was sampled by standard methods and assayed as follows:

	Per cent
Zinc	1·00
Iron	0·96
Barium sulphate	73·25
Sulphur	1·39

A screen test on the tailing gave the following results:

Mesh	Weight, per cent
+ 65	2·0
− 65 +100	4·4
−100 +150	8·1
−150 +200	18·5
−200	67·0
	100·0

Experimental Tests

The object of the experimental tests was to produce a barite concentrate of commercial grade.

Preliminary flotation tests were carried out to determine the best reagents for cleaning the tailing of zinc. These tests were followed by table and flotation concentration of the zinc flotation tailing to produce a barite concentrate. A satisfactory grade of concentrate was made as regards barium sulphate content. The colour of the barite, however, is somewhat stained. For the manufacture of lithopone and barium chemicals an iron-stained product is reported as being used extensively.

The results of the tests follow in detail:

FLOTATION

Test No. 1

Charge to Flotation Cell: (Denver Sub-A 1,000-gramme unit cell)

Tailing	1,000 grms.
Soda ash	2·0 lb./ton
Potassium xanthate	0·2 "
Pine oil	0·05 "

Product	Weight, per cent	Assay, per cent			Distribution, per cent			Ratio of concentration
		Zn	Fe	$BaSO_4$	Zn	Fe	$BaSO_4$	
Feed	100·00	0·91	0·96	72·90	100·00	100·00	100·00	
Concentrate	9·14	10·00	5·45	65·90	100·00	51·81	8·26	10·94 : 1
Tailing	90·86	Nil	0·51	73·60	Nil	48·19	91·74	

Test No. 2

This test was similar to Test No. 1, with the addition of 0·5 pound of sodium silicate to the pulp.

Product	Weight, per cent	Assay, per cent			Distribution, per cent			Ratio of concentration
		Zn	Fe	$BaSO_4$	Zn	Fe	$BaSO_4$	
Feed	100·00	0·92	0·98	75·64	100·00	100·00	100·00	
Concentrate	5·21	17·65	9·50	40·90	100·00	50·59	2·82	19·19 : 1
Tailing	94·79	Nil	0·51	76·70	Nil	49·41	97·18	

The addition of sodium silicate is beneficial, tending to depress the barite and thus increase the barite content of the flotation tailing.

Test No. 3

In this test, 1·0 pound of copper sulphate per ton was added to the pulp. No sodium silicate was added and the other reagents were:

Soda ash	2·0 lb./ton
Potassium xanthate	0·2 "
Pine oil	0·10 "

Product	Weight, per cent	Assay, per cent			Distribution, per cent			Ratio of concentration
		Zn	Fe	$BaSO_4$	Zn	Fe	$BaSO_4$	
Feed	100·00	0·96	1·00	74·67	100·00	100·00	100·00	
Concentrate	5·56	17·25	6·41	46·90	100·00	35·53	3·49	17·9 : 1
Tailing	94·44	Nil	0·69	76·30	Nil	64·47	96·51	

The results of these flotation tests would indicate that the zinc floats satisfactorily without additions of copper sulphate, which appears to have a tendency to increase the iron content in the tailings.

Test No. 4

In this test a sample of tailing was first floated to remove the zinc, and the flotation tailing was then run over a laboratory Wilfley table.

The tabling test gave two products of a satisfactory grade of barite. The first cut (concentrate) contained over 1·5 per cent of iron, while the second cut (middling) contained less than 1·0 per cent.

The results of the test are as follows:

Reagents Added to Cell:

Soda ash	2·0 lb./ton
Copper sulphate	1·0 "
Potassium xanthate	0·2 "
Pine oil	0·05 "

Product	Weight, per cent	Assay, per cent			Distribution, per cent			Ratio of concentration
		Zn	Fe	$BaSO_4$	Zn	Fe	$BaSO_4$	
Feed	100·00	1·01	1·01	73·49	100·00	100·00	100·00	
Zinc concentrate	4·63	21·90	3·31	50·48	100·00	15·15	3·18	21·6 : 1
Tailing	95·37	Nil	0·90	74·61	Nil	84·85	96·82	

As in Test No. 3, the copper sulphate appears to have a tendency to depress the iron.

Tabling Test on Flotation Tailing:

Product	Weight, per cent	Assay, per cent			Distribution, per cent			Ratio of concentration
		Zn	Fe	$BaSO_4$	Zn	Fe	$BaSO_4$	
Feed	100·00	Nil	0·90	75·61	Nil	100·00	100·00	
First cut	15·03	"	1·60	92·30	"	26·71	18·59	6·65 : 1
Second cut	29·66	"	0·80	95·28	"	24·14	37·88	
Tailing	55·31	"	0·80	58·72	"	49·15	43·53	

Overall recovery of barite:
(18·59 + 37·88) per cent of 96·82 per cent .... 54·67 per cent

This percentage recovery includes the high iron product in the first cut or concentrate.

Test No. 5

In this test the zinc was removed from the tailing by flotation, sodium silicate being used instead of copper sulphate. The flotation tailing was then tabled and a larger, second cut or middling product taken off. The analysis indicated that this second cut was too large, as the grade was lowered appreciably.

The results of the test are as follows:

Reagents Added to Cell:

Soda ash	2·0 lb./ton
Sodium silicate	0·25 "
Potassium xanthate	0·20 "
Pine oil	0·05 "

Product	Weight, per cent	Assay, per cent			Distribution, per cent			Ratio of concentration
		Zn	Fe	$BaSO_4$	Zn	Fe	$BaSO_4$	
Feed	100·00	0·97	0·97	72·94	100·00	100·00	100·00	
Zinc concentrate	6·51	14·85	6·51	55·86	100·00	43·45	4·99	13·56 : 1
Flotation tailing	93·49	Nil[1]	0·59	74·13	Nil	56·55	95·01	

Tabling Test on Flotation Tailing:

Product	Weight, per cent	Assay, per cent		Distribution, per cent		Ratio of concentration
		Fe	$BaSO_4$	Fe	$BaSO_4$	
Feed	100·00	0·59	74·13	100·00	100·00	
First cut, concentrate	16·02	0·74	94·46	19·65	20·41	6·24 : 1
Second cut, middling	48·37	0·43	85·20	35·44	55·59	2·06 : 1
Tailing, includes slime	35·61	0·74	49·96	44·91	24·00	

Test No. 6

In this test the zinc was removed from the tailing by flotation as in the preceding test and the flotation tailing was tabled. A first cut, small in bulk, was made in order to account for the small amount of iron sulphides remaining after flotation. The middling or second cut was controlled to give a high-grade barite product.

Zinc Flotation:

Product	Weight, per cent	Assay, per cent			Distribution, per cent			Ratio of concentration
		Zn	Fe	$BaSO_4$	Zn	Fe	$BaSO_4$	
Feed	100·00	0·91	1·09	71·67	100·00	100·00	100·00	
Zinc concentrate	5·26	17·25	5·39	55·12	100·00	25·81	4·05	19 : 1
Tailing	94·74	Nil	0·86	72·59	Nil	74·19	95·95	

Tabling Test on Flotation Tailing:

Product	Weight, per cent	Assay, per cent		Distribution, per cent		Ratio of concentration
		Fe	$BaSO_4$	Fe	$BaSO_4$	
Feed	100·00	0·86	72·59	100·00	100·00	
First cut	6·78	1·66	91·38	13·08	8·53	14·75 : 1
Second cut	27·81	0·69	93·86	22·30	35·96	3·6 : 1
Table tailing	65·41	0·85	61·60	64·62	55·51	

Test No. 7

In this test, 0·5 pound of sodium silicate per ton was added to the flotation circuit instead af the 0·25 pound previously used. A higher grade of zinc concentrate resulted and the barite in the zinc concentrate was lowered. In the tabling test the slime was recovered separately from the sand.

Zinc Flotation:

Product	Weight, per cent	Assay, per cent			Distribution, per cent			Ratio of concentration
		Zn	Fe	$BaSO_4$	Zn	Fe	$BaSO_4$	
Feed	100·00	0·90	0·87	71·95	100·00	100·00	100·00	
Zinc concentrate	3·47	26·10	4·75	43·98	100·00	18·96	2·12	28·82 : 1
Tailing	96·53	Nil	0·73	72·96	Nil	81·04	97·88	

Tabling Test on Flotation Tailing:

Product	Weight, per cent	Assay, per cent		Distribution, per cent		Ratio of concentration
		Fe	$BaSO_4$	Fe	$BaSO_4$	
Feed	100·00	0·73	72·96	100·00	100·00	
First cut	6·89	1·17	93·94	11·02	8·87	14·51 : 1
Second cut	27·46	0·85	94·32	31·92	35·50	3·64 : 1
Sand tailing	41·90	0·48	59·22	27·50	34·01	
Slime	23·75	0·91	66·42	29·56	21·62	

Test No. 8

In this test the barite was floated. A high-grade concentrate was made, which was lower in iron and was also of a better colour than the barite product obtained from table concentration.

The zinc was first floated off in the usual manner and a rougher concentrate of barite made. This rougher concentrate was then cleaned to give a high-grade barite product.

The reagents used were as follows:

Zinc Float:

Soda ash	2·0 lb./ton
Sodium silicate	0·25 "
Potassium xanthate	0·20 "
Pine oil	0·05 "

Rougher Barite Float:

Sodium silicate	1·3 lb./ton
Oleic acid	0·48 "

Cleaner Barite Float:

Sodium silicate	0·5 lb./ton
Oleic acid	0·16 "

Product	Weight, per cent	Assay, per cent			Distribution, per cent			Ratio of concentration
		Zn	Fe	$BaSO_4$	Zn	Fe	$BaSO_4$	
Feed	100·00	1·02	1·07	72·16	100·00	100·00	100·00	
Zinc concentrate	5·78	17·70	7·64	45·40	100·00	40·96	3·64	
Barite "	46·02	Nil	0·43	97·28	Nil	18·36	62·04	2·17 : 1
Middling	22·02	"	0·85	85·06	"	17·36	25·95	
Rougher tailing	26·18	"	0·96	23·08	"	23·32	8·37	

Test No. 9

The reagents used were as follows:

Zinc Float:

Soda ash	2·0 lb./ton
Sodium silicate	0·5 "
Potassium xanthate	0·2 "
Pine oil	0·05 "

Rougher Barite Float:

Sodium silicate	1·3 lb./ton
Oleic acid	0·416 "

Cleaner Barite Float:

Sodium silicate	1·0 lb./ton
Oleic acid	0·16 "

Product	Weight, per cent	Assay, per cent			Distribution, per cent			Ratio of concentration
		Zn	Fe	$BaSO_4$	Zn	Fe	$BaSO_4$	
Feed	100·00	0·94	1·05	72·31	100·00	100·00	100·00	
Zinc concentrate	5·05	18·70	9·67	43·88	100·00	46·39	3·06	
Barite concentrate	43·25	Nil	0·43	96·90	Nil	17·67	57·95	2·31 : 1
Middling	28·80	"	0·59	85·72	"	16·14	34·14	
Rougher tailing	22·90	"	0·91	15·30	"	19·80	4·85	

Test No. 10

In this test, crude corn oil was used to replace oleic acid to float the barite. The results were unsatisfactory, as the corn oil was a poor collector and failed to produce a froth.

The results of the flotation tests indicate a higher recovery of barite than by table concentration and a better grade of concentrate as regards higher barite content, lower iron content, and better colour.

BLEACHING TESTS

Several bleaching tests were carried out on both the table product and the flotation product. The products were boiled in a 5 per cent (by volume) solution of sulphuric acid for 2 hours. The pulp dilution was 2 : 1.

A slight improvement in the colour was obtained, the bleached material having a greyish tinge rather than the brownish grey tinge of the unbleached products.

SUMMARY AND CONCLUSIONS

The results of the tests indicate that a concentrate, satisfactory as to barium sulphate content, can be made from the mill tailing submitted.

The zinc was entirely removed by flotation from the tailing, with a very small loss of barite. The reagents recommended are: sodium silicate (water glass) 0·25 to 0·50 pound per ton; soda ash, 2 pounds per ton; and potassium xanthate, 0·20 pound per ton.

No difficulty was experienced in concentrating the barite, which can be carried out by tabling or by flotation. The table product is satisfactory as regards barium sulphate content, but the recovery is not much over 50 per cent.

The flotation product, however, is of higher grade, lower in iron, and better in colour, and accounts for a recovery of from 60 to 75 per cent.

The reagents recommended in flotation are sodium silicate (water glass) and oleic acid, the amount of the latter ranging from 0·50 to 0·75 pound per ton.

The faint brownish tint of the barite was removed by bleaching, but a pure white colour was not obtained. As colour is reported as not being a requirement in the lithopone or chemical trade, bleaching of the product may not be necessary.

Ore Dressing and Metallurgical Investigation No. 674

LEAD ORE FROM CROWN KING MINE, SUMMERVILLE TOWNSHIP, VICTORIA COUNTY, ONTARIO

Shipment. A shipment of lead ore was received on March 30, 1936, from the Consolidated Lead Mines Syndicate, Toronto, Ontario.

Location of Property. The sample was taken from the Crown King mine, located in lot 1, concession VII, Victoria county (north), Ontario.

Character and Analysis of Ore Sample. The sample was already crushed to ¼ inch when received. The principal sulphides present are galena and pyrite, and the gangue is calcite. Because the sample had been previously crushed, it is impossible to determine how large the galena masses are and at what size the first jigging operation could be attempted.

Analysis:

Lead	2·85 per cent
Iron	3·31 "
Copper	Nil

Experimental Tests

A sample weighing 56 pounds was taken from the shipment, and without further crushing was screened on the following size screens:—

Mesh	Weight, pounds
− ¼ inch + 14	22·5
− 14 + 28	4·5
− 28 + 65	12·0
− 65 +100	3·5
−100	13·5
	56·0

The −¼-inch +14-mesh material was jigged, and all the remaining sizes were run separately over a laboratory-size Wilfley table with the exception of the −100-mesh material, which was concentrated by flotation.

In recording the results the jig products are shown separately, but all products from the table were respectively combined giving one table concentrate, a middling, and a tailing. The results obtained by the flotation of the −100-mesh product are also shown separately.

Results of Jigging −¼-inch +14-mesh Material:

Product	Weight		Assay, Pb, per cent	Units	Distribution of lead, per cent
	Grammes	Per cent			
Concentrate	217·3	2·3	83·8	192·7	96·8
Middling	269·0	2·8	1·3	3·6	1·8
Tailing	8,984·0	94·9	0·03	2·8	1·4
Totals	9,470·3	100·0	1·99	199·1	100·0

Results of Tabling −14+28; −28+65; −65+100-mesh Material:

Products	Weight		Assay, Pb, per cent	Units	Distribution, Pb, per cent
	Grammes	Per cent			
Concentrate	364·0	4·3	68·10	292·8	78·6
Middling	354·0	4·1	17·60	72·2	19·4
Tailing	7,907·0	91·6	0·08	7·3	2·0
Totals	8,625·0	100·0	3·72	372·3	100·0

Results of Flotation of −100-mesh Material:

Products	Weight		Assay, Pb, per cent	Units	Distribution, Pb, per cent
	Grammes	Per cent			
Concentrate	302·5	5·1	74·7	381·0	95·6
Middling	218·6	3·7	3·5	13·0	3·3
Tailing	5,380·0	91·2	0·05	4·6	1·1
Totals	5,901·1	100·0	3·99	398·6	100·0

Summary:

Products	Weight		Assay, Pb per cent,	Distribution, Pb, per cent
	Grammes	Per cent		
Feed, calculated	23,996·4	100·0	3·10	100·0
Jig concentrate	217·3	0·9	83·8	24·5
Table concentrate	364·0	1·5	68·1	32·9
Flotation concentrate	302·5	1·3	74·7	31·3
Jig middling	269·0	1·1	1·3	0·5
Table middling	354·0	1·5	17·6	8·4
Flotation middling	218·6	0·9	3·5	1·0
Jig tailing	8,984·0	37·4	0·03	0·3
Table tailing	7,907·0	33·0	0·08	0·8
Flotation tailing	5,380·0	22·4	0·05	0·3

These tables show that out of every 100 tons of ore there would be produced:—

Jig concentrate, 0·9 ton assaying 83·8 per cent lead.
Table concentrate, 1·5 tons assaying 68·1 per cent lead.
Flotation concentrate, 1·3 tons assaying 74·7 per cent lead.
Total concentrate=3·7 tons assaying 74·5 per cent lead.

This gives a recovery of 88·7 per cent of the lead as a concentrate, but there is 3·5 tons of middling containing 9·9 per cent of the total lead. It can safely be assumed that half this lead could be recovered from the middling by regrinding and flotation, which would raise the total recovery of lead to about 93·5 per cent.

CONCLUSIONS

The ore responds readily to concentration by jigs, and there is little doubt that jigging could commence at a much coarser size than $\frac{1}{4}$ inch. This point, however, could not be definitely determined because as already pointed out the sample when received was crushed to $\frac{1}{4}$ inch.

The ore also responds readily to table concentration, and in this connexion particular attention is drawn to the following statement: There is no doubt that at least an 80 per cent lead concentrate can be produced on the tables. The reason it was not produced in the test work is due to the difficulty of obtaining a clean-cut on the small laboratory table.

The flotation of the ore is also extremely simple, and there is no question whatever that a concentrate carrying more than 75 per cent lead could be produced.

It is, therefore, estimated that a concentrate containing 80 per cent lead can be readily produced by a mill using a flow-sheet similar to the one described briefly as follows: Ore to be crushed in jaw crusher and rolls to $\frac{1}{2}$ inch, then screened on impact screens $-\frac{1}{2}$ $+\frac{1}{4}$; $-\frac{1}{4}$ $+14$ mesh for jigging. The -14-mesh product should be classified by hydraulic classification into No. 1 product for jigging; No. 2 and No. 3, product for tabling and the slime product, should be thickened for flotation. All middling products should go to a small ball mill for regrinding and direct to flotation.

Attention is also directed to the possibility of producing pig lead at the property. With a concentrate running 80 per cent lead a Scotch hearth could be built and successfully operated on the jig and coarse table concentrate.

Ore Dressing and Metallurgical Investigation No. 675

ARSENICAL GOLD ORE FROM THE FLIN FLON MINING SYNDICATE, FLINFLON, MANITOBA

Shipment. Sixty-two bags of gold ore, weighing 6,710 pounds, were received on November 1, 1935, from the Flin Flon Mining Syndicate, at Douglas Lake, three miles west of Flinflon, Manitoba, by Dr. J. F. Wright. The shipment was made to carry out an investigation to determine the most economical method of milling the ore. Before the completion of the test work, the company decided to roast the ore and ship the calcine to the copper smelter at Flinflon. Tests were then carried out to determine the most economical size of feed for a roaster.

Microscopic Examination. Twelve polished sections of selected specimens of the ore were prepared and examined microscopically.

The *gangue* is chiefly white vein quartz of fine texture. This contains small inclusions of green chloritic material and rusty brown patches of what are regarded as altered chloritic inclusions.

The *metallic minerals,* in their order of abundance, are arsenopyrite, pyrite, chalcopyrite, sphalerite, limonite, pyrrhotite, magnetite, covellite, and native gold.

Arsenopyrite is abundant in most of the sections, forming from 25 per cent to 75 per cent of the volume. It is coarse-textured and often quite massive, contains numerous tiny inclusions of gangue and is traversed by a network of fine fractures. Many of the fractures are filled with gangue. Some are filled with chalcopyrite and/or sphalerite and rarely contain pyrrhotite.

A considerable amount of pyrite occurs as medium to small grains, usually associated with arsenopyrite. In one place it was seen to vein the arsenopyrite, but the pyrite itself does not appear to be fractured or veined by the other sulphides.

Chalcopyrite and sphalerite are present in small amounts and usually occur together, although both are present as grains in the gangue. Small grains and veinlets of each occur in arsenopyrite.

Pyrrhotite is present only in very small amount as tiny irregular grains in both gangue and arsenopyrite and very rarely as fine veinlets in arsenopyrite.

Only a very small amount of native gold was visible in the sections. With one exception it occurs as tiny grains enclosed in dense arsenopyrite and has no apparent genetic relation to the fracturing. Thirty-six grains were observed to occur in this manner, and only two were seen to be present in a narrow veinlet of gangue cutting across arsenopyrite. The percentages of gold of the above modes of occurrence are as follows:

	Per cent
Gold present as tiny irregular grains within dense arsenopyrite..........	96
Gold present as tiny grains in veinlets of gangue cutting arsenopyrite.....	4

Grain Size of the Native Gold. A quantitative microscopic analysis was carried out to determine the grain size of the gold. From the paucity of visible gold and from its extremely finely divided condition in the arsenopyrite, it is possible that a certain unknown quantity may be present in the arsenopyrite in sub-microscopic form.

Mesh	Gold in dense arsenopyrite, per cent	Gold in gangue veinlets in arsenopyrite, per cent	Total gold, per cent
+ 800	7·4	....	7·4
− 800 +1100	17·0	....	17·0
−1100 +1600	36·9	....	36·9
−1600 +2300	13·2	2·6	15·8
−2300	21·5	1·4	22·9
Totals	96·0	4·0	100·0

The results of the microscopic examination indicate very definitely that:

(*a*) The gold occurs chiefly, if not wholly, in the arsenopyrite;
(*b*) The visible gold is very finely divided;
(*c*) The arsenopyrite is coarse;
(*d*) Small amounts of chalcopyrite and sphalerite are intimately associated with arsenopyrite; and
(*e*) The presence of sub-microscopic gold, though not proved, is strongly suspected.

Experimental Tests

The entire shipment was crushed and sampled.

Analysis showed it to contain:

Gold	0·625	oz./ton
Silver	0·88	"
Zinc	4·10	per cent
Arsenic	22·65	"
Copper	0·40	"
Iron	22·35	"
Sulphur	13·45	"

The results obtained in the experimental tests indicate that 73·6 per cent of the gold could be extracted by cyanidation.

Flotation recovered a concentrate containing 97·6 per cent of the gold, with a ratio of concentration of 2·1 : 1.

Roasting of the raw ore crushed to ¼-inch reduced the arsenic in the calcine to 0·48 per cent. When the ore was crushed to −14 mesh the arsenic in the roasted product was 0·22 per cent.

The roasted flotation concentrate contained 2·00 ounces of gold and 2·68 ounces of silver per ton.

The tests in detail follow:

AMALGAMATION

Test No. 1

To obtain some idea of the amount of free gold in the ore, a sample was ground wet to pass 89 per cent −200 mesh, and amalgamated. The mercury was badly floured.

Results:

Feed	0·625 oz. Au/ton
Amalgamation tailing	0·355 " "
Recovery	43·2 per cent

HYDRAULIC CONCENTRATION

Test No. 2

A sample ground as in Test No. 1 was passed through a hydraulic concentrator, and a gold concentrate recovered.

Results:

Product	Weight, per cent	Assay, oz./ton		Distribution, per cent	
		Au	Ag	Au	Ag
Feed (cal.)	100·00	0·67	1·12	100·0	100·0
Hydraulic concentrate	1·05	11·71	11·60	18·4	10·9
Hydraulic tailing	98·95	0·55	1·01	81·6	89·1

CYANIDATION

Test No. 3

A sample of the ore was ground with a solution having 2·0 pounds of potassium cyanide per ton, and 8·0 pounds of lime per ton, to pass 89 per cent −200 mesh. The pulp then was diluted to 1 : 2·5 with a solution having 2·0 pounds of potassium cyanide per ton and agitated for 24 hours.

Results:

Feed	0·625 oz. Au/ton,	0·88 oz. Ag/ton
Tailing	0·165 " "	0·35 " "
Extraction—Au		73·6 per cent
Ag		60·2 "

Reagents Consumed:

KCN	2·0 lb./ton
CaO	9·1 "

FLOTATION

Test No. 4

To note the results obtainable by flotation, a selective flotation test was made to produce a copper concentrate, a zinc concentrate, and a pyrite concentrate.

The ore was ground wet in a ball mill with 9·0 pounds of soda ash and 0·10 pound of sodium cyanide per ton. A screen analysis showed the pulp to be 93·8 per cent −200 mesh. The pulp was then floated.

Reagents:

		Lb./ton
Copper cell—	Butyl xanthate	0·06
	Pine oil	0·05
Zinc cell—	Copper sulphate	0·2
	Potassium amyl xanthate	0·04
	Pine oil	0·08
Pyrite cell—	Copper sulphate	1·0
	Amyl xanthate	0·30
	Pine oil	0·40

Results:

Product	Weight, per cent	Assay					Distribution, per cent				
		Oz./ton		Per cent							
		Au	Ag	Cu	Zn	As	Au	Ag	Cu	Zn	As
Feed (cal.)	100·0	0·63	1·32	0·47	4·23	22·11	100·0	100·0	100·0	100·0	100·0
Copper concentrate	5·6	6·66	9·30	4·44	8·75	25·66	58·9	39·2	52·5	11·6	6·5
Zinc "	12·9	0·49	1·62	0·76	16·50	24·95	10·0	15·7	20·7	50·3	14·6
Pyrite "	47·5	0·39	1·10	0·16	2·50	35·32	29·3	39·4	16·0	28·1	75·9
Tailing	34·0	0·035	0·22	0·15	1·25	1·98	1·8	5·7	10·8	10·0	3·0

Test No. 5

In this test a bulk flotation concentrate was made. The ore, ground with 9·0 pounds of soda ash, 0·24 pound of Barrett No. 4, and 0·10 pound of amyl xanthate, was floated with 0·25 pound of pine oil per ton. Copper sulphate, 1·0 pound per ton, was added to the flotation cell. A screen test made on the flotation tailing showed 71·6 per cent −200 mesh.

Results:

Product	Weight, per cent	Assay					Distribution, per cent				
		Oz./ton		Per cent							
		Au	Ag	Cu	Zn	As	Au	Ag	Cu	Zn	As
Feed (cal.)	100·0	0·64	1·50	0·55	4·48	21·55	100·0	100·0	100·0	100·0	100·0
Flotation concentrate	61·4	1·02	1·80	0·82	6·80	33·05	97·6	73·5	91·6	93·1	94·1
Flotation tailing	38·6	0·04	1·03	0·12	0·80	3·27	2·4	26·5	8·4	6·9	5·9

Test No. 6

This test is the same as Test No. 5, with the exception that the pulp, after grinding, was aerated for 8 hours and then floated.

Results:

Product	Weight, per cent	Assay					Distribution, per cent				
		Oz./ton		Per cent							
		Au	Ag	Cu	Zn	As	Au	Ag	Cu	Zn	As
Feed (cal.)...........	100·0	0·57	0·85	0·41	3·73	22·16	100·0	100·0	100·0	100·0	100·0
Flotation concentrate.	27·3	1·40	2·18	1·36	12·20	26·57	66·9	70·1	89·5	89·3	32·7
Flotation tailing......	72·7	0·26	0·35	0·06	0·55	20·50	33·1	29·9	10·5	10·7	67·3

Aeration improves the flotability of the copper and zinc minerals but retards that of the arsenopyrite. As this mineral carries gold, the tailing contains a large proportion of the gold.

Test No. 7

This is a selective flotation test to remove copper, followed by cyanidation of the flotation tailing.

A sample of the ore was ground wet to pass 98·1 per cent through 200 mesh. Reagents added to the grinding mill were 4·0 pounds of lime and 1·0 pound of zinc sulphate per ton. Butyl xanthate, 0·10 pound per ton, and pine oil, 0·06 pound per ton, were added to the flotation cell and a concentrate removed.

Results:

Product	Weight, per cent	Assay					Distribution, per cent				
		Oz./ton		Per cent							
		Au	Ag	Cu	Zn	As	Au	Ag	Cu	Zn	As
Feed (cal.)...........	100·00	0·64	0·73	0·35	4·04	22·51	100·0	100·0	100·0	100·0	100·0
Concentrate..........	15·22	2·98	4·34	2·10	6·50	30·94	70·9	90·6	90·4	24·4	20·9
Tailing..............	84·78	0·22	0·08	0·04	3·60	21·00	29·1	9·4	9·6	75·6	79·1

The concentrate was dead-roasted and analysed and found to contain:

Gold..	4·84 oz./ton
Silver..	6·58 "
Copper..	3·40 per cent
Zinc..	8·80 "
Arsenic...	1·35 "
Loss in weight on ignition....................................	38·0 "

Cyanidation of the flotation tailing reduced the gold content from 0·22 ounce to 0·12 ounce per ton, an extraction of 45·5 per cent or a total recovery of 70·9 per cent of the gold as a roasted concentrate and 13·2 per cent by cyanidation.

Test No. 8

This test was carried out along the same lines as was Test No. 7. However, in this case, a bulk flotation concentrate was made and cleaned. This concentrate was roasted and the flotation tailing cyanided.

Results of Flotation:

Product	Weight, per cent	Assay					Distribution, per cent				
		Oz./ton		Per cent							
		Au	Ag	Cu	Zn	As	Au	Ag	Cu	Zn	As
Feed (cal.)...........	100·0	0·59	0·88	0·35	4·03	22·50	100·0	100·0	100·0	100·0	100·0
Concentrate..........	47·8	1·06	1·64	0·66	7·60	34·14	84·7	88·7	89·4	90·2	72·5
Middling............	15·4	0·50	0·60	0·10	1·60	31·52	12·9	10·5	4·4	6·1	21·6
Tailing..............	36·8	0·04	0·02	0·06	0·40	3·60	2·4	0·8	6·2	3·7	5·9

The flotation concentrate was roasted and analysed and found to contain:

Gold..	2·00 oz./ton
Silver..	2·68 "
Copper..	1·27 per cent
Zinc..	12·18 "
Arsenic..	0·30 "

Cyanidation of the flotation tailing reduced the gold content from 0·04 ounce to 0·02 ounce per ton within 24 hours.

ROASTING

Test No. 9

It was desired to determine to what size the ore should be crushed to roast effectively, so as to reduce the arsenic content of the calcine to approximately 1 per cent.

Samples of the raw ore were crushed to pass 3, 4, 8, 14, 28, 35, 48, and 65 mesh. These were roasted in a gas-fired, hand-rabbled muffle furnace until white fumes of arsenic ceased to be given off. The temperature was kept at about 450° C. for this period, and then raised to about 750° C. It was observed that the coarser sizes roasted more rapidly than the finer sizes. The 3-, 4-, and 8-mesh samples required approximately 30 minutes to drive off arsenic, while the finer sizes required from 60 to 90 minutes.

Analysis of Calcines:

Element	Grind—Mesh							
	−3	−4	−8	−14	−28	−35	−48	−65
Gold..............oz./ton	0·985	1·03	0·84	0·94	0·85	0·90	0·92	0·88
Silver............. "	1·34	1·29	1·72	1·62	1·25	1·28	1·28	1·22
Copper............per cent	0·48	0·45	0·45	0·51	0·47	0·47	0·48	0·47
Zinc............... "	4·35	4·50	4·30	4·65	4·45	4·60	4·45	4·30
Arsenic............ "	0·48	0·38	0·38	0·22	0·23	0·24	0·25	0·63
Iron............... "	29·49	30·08	30·06	....	....	....	....	....
Sulphur........... "	2·79	3·04	1·90	....	....	....	....	....
Silica............. "	43·90	42·75	....	....	....	....	....	....
Lime.............. "	0·33	0·20	....	....	....	....	....	....
Magnesia.......... "	0·32	0·27	....	....	....	....	....	....
Loss on ignition.... "	26·0	27·4	26·2	28·3	26·3	28·1	28·5	27·5

It is apparent that crushing to −3 mesh or 0·026-inch diameter is quite fine enough to obtain a calcine under 1 per cent arsenic.

SUMMARY AND CONCLUSIONS

It is apparent that this ore contains gold that is not readily recovered by cyanidation. Microscopic examination leads to the supposition that a certain unknown quantity may be present in sub-microscopic form.

Flotation recovers 97·6 per cent of the gold in the form of a concentrate, with a concentration ratio of 2·1 : 1. This product can be roasted to a high-grade product low in arsenic.

The raw ore also can be roasted to a low arsenic calcine.

The most suitable treatment for this ore is roasting and smelting. The property is located quite close to a copper smelter, which is equipped to handle the roasted ore. This arrangement will prove to be the most satisfactory for the recovery of gold from this refractory ore.

Ore Dressing and Metallurgical Investigation No. 676

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

Shipment. A shipment of ore, net weight 140 pounds, was received on February 11, 1936. It was submitted by W. A. Gamble of the Granada Gold Mines, Limited, Rouyn, Quebec.

The purpose of the tests was to determine the best method of treatment.

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

The *gangue* is white to grey vein quartz. It is locally stained brown by what appears to be iron oxide.

The *metallic minerals*, in their order of abundance in the polished sections, are as follows:

Pyrrhotite	Major
Arsenopyrite, Galena	Minor
Chalcopyrite, Sphalerite, Native gold	Accessory

Pyrrhotite is moderately abundant, as irregular stringers, patches, and grains; it is associated intimately with galena, and to some extent also with arsenopyrite and a little chalcopyrite. Arsenopyrite is disseminated as coarse to medium crystals. Galena occurs in the same manner as pyrrhotite and is closely associated with it. A small amount of chalcopyrite is present as irregular patches and grains and contains rare small grains of sphalerite.

Native gold occurs chiefly in the gangue, where it is largely free, but a minor quantity is associated with arsenopyrite and pyrrhotite. A small quantity is enclosed in arsenopyrite. The grains visible in the sections are too few to allow of a grain analysis, but the grain sizes range from about 200 mesh down to −2300 mesh, with the largest percentage in the −200 +325-mesh size. Measurements give the following percentages of the various modes of occurrence:

	Per cent
In Gangue:	
Free	58·3
Associated with arsenopyrite	17·2
Associated with pyrrhotite	14·4
	89·9
In Arsenopyrite:	
Enclosed	10·1
	100·0

Sampling and Analysis: The sample of ore was crushed and sampled by standard methods and a representative portion showed the following values:

Gold	1·05 oz./ton
Silver	1·04 "
Copper	0·41 per cent
Arsenic	0·80 "

Experimental Tests

The tests made on this sample of ore consisted of amalgamation, straight cyanidation and concentration using traps, blankets and flotation. Combinations of concentration and cyanidation were made on ore crushed from 65 to 80 per cent −200 mesh.

Amalgamation of −48-mesh ore gave a recovery of 73 per cent of the gold. On −100-mesh ore, amalgamation gave a recovery of 65 per cent.

Straight cyanidation of the ore gave a maximum extraction of 95 per cent of the gold in 24 hours on −150-mesh ore. Owing to the presence of free gold, the results in this test were not uniform.

Trap and blanket concentration gave a recovery of 66 per cent of the gold on ore crushed 80 to 85 per cent −200 mesh. Flotation of the blanket tailing gave an additional recovery of 31 per cent of the gold.

Concentration by blankets followed by flotation gave an overall recovery of 91 per cent.

Trap concentration followed by cyanidation gave an extraction of 99 per cent of the gold on 80 per cent −200-mesh ore. In this test the same solution was used on four charges of pulp without any appreciable fouling of the solution, with high extraction of gold.

Further tests using trap and blanket concentration and cyanidation gave a similar high recovery on ore crushed to 65 per cent −200 mesh, showing that fine grinding is unnecessary.

AMALGAMATION

Tests Nos. 1 and 2

Representative samples of −14-mesh ore were crushed to pass 48- and 100-mesh screens and then amalgamated with 10 per cent by weight of mercury at a dilution of one part ore to one part water.

The mercury and amalgam were separated and the tailing assayed for gold. A screen test was made on each tailing to show the grind.

Screen Test:

Mesh	Weight, per cent	
	−48-mesh ore	−100-mesh ore
− 48 + 65	8·75	....
− 65 +100	17·70	....
−100 +150	13·85	6·9
−150 +200	15·80	21·6
−200	43·90	71·5
	100·00	100·0

Amalgamation:

Test No.	Mesh	Assay, Au, oz./ton		Recovery of gold, per cent
		Feed	Tailing	
1	− 48	1·05	0·285	72·9
2	−100	1·05	0·37	64·8

STRAIGHT CYANIDATION

Tests Nos. 3 to 10

Representative samples of −14-mesh ore were crushed to pass 48-, 100-, 150-, and 200-mesh screens. Portions of each were agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, and lime at the rate of 5 pounds per ton of ore, for periods of 24 and 48 hours. The cyanide tailings were assayed for gold.

Summary:

Test No.	Mesh	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton ore	
			Feed	Tailing		KCN	CaO
3	− 48	24	1·05	0·17	83·8	1·17	5·35
4	−100	24	1·05	0·09	91·4	2·10	6·55
5	−150	24	1·05	0·055	94·8	2·85	8·50
6	−200	24	1·05	0·125	88·1	2·85	9·35
7	− 48	48	1·05	0·18	82·9	1·47	5·40
8	−100	48	1·05	0·07	93·3	2·55	6·70
9	−150	48	1·05	0·07	93·3	3·80	9·40
10	−200	48	1·05	0·09	92·4	4·11	9·70

Subsequent tests show that after the removal of free gold by means of a trap the cyanidation gives a very low tailing. Straight cyanidation without removing particles of free gold is unsatisfactory.

CONCENTRATION TESTS

Test No. 11

A representative sample of −14-mesh ore was ground in ball mills to give a product 83 per cent −200 mesh. The pulp was concentrated in a hydraulic trap. The trap tailing was passed over a corduroy blanket on a table sloping 2·5 inches in one foot.

The blanket tailing was treated by flotation. The pulp was ground in a ball mill. One portion, A, was ground to give a product 89 per cent −200 mesh, the other, B, 92 per cent −200 mesh. The following reagents were added to the ball mill:

	Lb./ton
Soda ash	3·0
Potassium amyl xanthate	0·1

The pulp dilution in the ball mill was approximately four parts pulp to three parts water.

The reagent added to the flotation cell was:

Pine oil.. 0·05 lb./ton

Summary:

Trap Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Extraction of gold, per cent
Feed..	100·00	1·05	100·00
Trap concentrate...................................	0·02	1828·60	34·83
Trap tailing...	99·98	0·68*	65·17

* Calculated values from products of test. Feed assay of ore, Au, 1·05 oz./ton.

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Extraction of gold, per cent
Feed..	100·00	0·51*	100·00
Blanket concentrate................................	0·36	67·31	47·39
Blanket tailing......................................	99·64	0·27	52·61

* Calculated values from products of test. Feed assay of ore, Au, 1·05 oz./ton.

Gold recovered in trap concentrate.............................. 34·83 per cent
Gold recovered in blanket concentrate, 47·39 × 65·17.......... 30·88 "

Total gold recovered in concentrate..................... 65·71 "

Flotation of Blanket Tailing A:

Product	Weight, per cent	Assay					Recovery of gold, per cent
		Oz./ton		Per cent			
		Au	Ag	Pb	Cu	As	
Feed........................	100·00	0·25*	....	....	......	....	100·00
Flotation concentrate........	9·65	2·32	7·0	10·66	3·80	5·36	90·83
Flotation tailing............	90·35	0·025	....	....	......	....	9·17

Flotation of Blanket Tailing B:

Product	Weight, per cent	Assay					Recovery of gold, per cent
		Oz./ton		Per cent			
		Au	Ag	Pb	Cu	As	
Feed........................	100·00	0·29*	....	....	....	....	100·00
Flotation concentrate........	8·58	3·15	8·85	12·82	4·32	5·71	92·20
Flotation tailing............	91·42	0·025	....	....	....	....	7·80

* Calculated values from products of test. Blanket tailing assay, Au, 0·27 oz./ton.

Per cent gold left in blanket tailing,
65·17 × 52·61...................... 34·29 per cent
Gold extracted by flotation in A....... 90·83 per cent of gold in blanket tailing
Gold extracted by flotation in B........ 92·20 " " "

Test No. 11—A:

Gold recovered in trap and blanket concentrate..................	65·71 per cent
Gold recovered in flotation concentrate, 34·29 × 90·83...........	31·15 "
Total gold recovered in concentrates.....................	96·86 "

Test No. 11—B:

Gold recovered in trap and blanket concentrate..................	65·71 per cent
Gold recovered in flotation concentrate, 34·29 × 92·20...........	31·62 "
Total gold recovered in concentrates.....................	97·33 "

BLANKET CONCENTRATION FOLLOWED BY FLOTATION

Test No. 12

A representative sample of −14-mesh ore was ground in ball mills to give a product 87 per cent −200 mesh. The pulp was passed over a corduroy blanket, slope 2·5 inches in 1 foot. The blanket tailing was treated by flotation without regrinding. The following reagents were used:

Soda ash..	2·0 lb./ton
Potassium amyl xanthate....................................	0·1 "
Pine oil..	0·05 "

Summary:

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Recovery of gold, per cent
Feed..	100·00	0·86*	100·00
Blanket concentrate....................................	0·45	104·92	54·88
Blanket tailing..	99·55	0·39	45·12

*Calculated values from products of test.

Flotation:

Product	Weight, per cent	Assay					Recovery of gold, per cent
		Oz./ton		Per cent			
		Au	Ag	Cu	Pb	As	
Feed........................	100·00	0·34*	....	....	....	....	100·0
Flotation concentrate........	4·57	5·90	13·94	7·38	22·3	3·85	80·1
Flotation tailing............	95·43	0·07	....	....	....	....	19·9

*Calculated values from products of test.

Gold recovered in blanket concentrate...........................	54·88 per cent
Gold recovered in flotation concentrate, 80·1 × 45·12............	36·16 "
Total gold recovered in concentrates.....................	91·04 "

CONCENTRATION BY TRAP AND BLANKETS

Test No. 13

A representative sample of −14-mesh ore was ground in ball mills to give a product 85 per cent −200 mesh. The pulp was concentrated, first in a hydraulic trap, and then by means of a corduroy blanket sloping 2·5 inches in 1 foot. The two concentrates were panned to remove as much gangue as possible and then dried, weighed, and assayed. A screen test was made on the tailing and a sample of the tailing was assayed for gold.

Summary:

Trap Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	0·98*	100·00
Trap concentrate	1·43	47·28	68·87
Trap tailing	98·57	0·31	31·13

*Calculated feed assay from products of test.

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	0·31	100·00
Blanket concentrate	0·36	25·02	29·13
Blanket tailing	99·64	0·22	70·87

Gold recovered in trap concentrate	68·87	per cent
Gold recovered in blanket concentrate, 29·13 × 31·13	9·07	"
Total gold recovered in concentrates	77·94	"

CONCENTRATION FOLLOWED BY CYANIDATION

Test No. 14

A representative sample of −14-mesh ore was ground in ball mills to give a product 80 per cent −200 mesh. The ground pulp was passed through a hydraulic trap and the trap tailing was agitated for 24 hours in a solution containing 1·0 pound of potassium cyanide per ton and lime at the rate of 4·0 pounds per ton of pulp. The cyanide tailing was assayed for gold and the gold in the solution was precipitated with zinc. A portion of the barrel solution was used in the succeeding test. The solution was made up to strength by additions of cyanide and lime. Four cycle tests were made in this manner, using the same solution. The assay of the cyanide tailing shows an extraction of 97·4 to 98·2 per cent, with a low consumption of reagents and no appreciable fouling of the solution.

Summary:

Trap Concentration:

Test No.	Weight of concentrate, per cent	Assay, Au, oz./ton		Distribution of gold, per cent
		Trap conc.	Trap tailing	
14-A	0·62	93·29	0·47	55·1
14-B	1·20	47·32	0·49	54·1
14-C	1·78	37·81	0·38	64·1
14-D	2·97	17·71	0·54	50·1

Cyanidation of Trap Tailing:

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton dry pulp	
	Feed	Tailing		KCN	CaO
14-A	0·47	0·01	97·9	1·80	3·28
14-B	0·49	0·01	98·0	1·41	2·24
14-C	0·48	0·01	97·4	0·91	1·80
14-D	0·54	0·01	98·2	0·83	2·68

CONCENTRATION FOLLOWED BY CYANIDATION

Test No. 15

To determine the recovery of gold obtained when using coarser grinding, two representative samples of ore were crushed to give products 65 per cent −200 mesh and 77 per cent −200 mesh, respectively. Each was concentrated by a trap and blanket. The blanket tailing was agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, and lime at the rate of 4·0 pounds per ton of pulp, for 24 hours. The cyanide tailing was assayed for gold.

Summary:

Trap Concentration, Test No. 15-A (65 per cent −200 mesh):

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	1·05	100·00
Trap concentrate	0·93	78·18	69·25
Trap tailing	99·07	0·33	30·75

Blanket Concentration, Test No. 15-A:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	0·33	100·0
Blanket concentrate	0·40	28·13	34·1
Blanket tailing	99·60	0·22	65·9

Cyanidation of Blanket Tailing, Test No. 15-A:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton of dry pulp	
Feed	Tailing		KCN	CaO
0·22	0·01	95·45	1·15	3·40

Gold recovered in trap concentrate..............................	69·25 per cent
Gold recovered in blanket concentrate, 34·1 × 30·75............	10·48 "
Gold remaining in blanket tailing, 65·9 × 30·75=20·27 per cent.	
Gold recovered by cyanidation of blanket tailing, 20·27 × 95·45.	19·35 "
Total gold recovered in this test........................	99·08 "

Trap Concentration, Test No. 15-B (77 per cent −200 mesh):

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed..	100·00	1·05	100·00
Trap concentrate..	1·24	54·60	64·48
Trap tailing..	98·76	0·38	35·52

Blanket Concentration, Test No. 15-B:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed..	100·00	0·38	100·0
Blanket concentrate.....................................	0·43	22·10	25·0
Blanket tailing...	99·57	0·29	75·0

Cyanidation of Blanket Tailing, Test No. 15-B:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton of dry pulp	
Feed	Tailing		KCN	CaO
0·29	0·015	94·83	1·46	3·35

Gold recovered in trap concentrate..............................	64·48 per cent
Gold recovered in blanket concentrate, 35·52 × 25·0............	8·88 "
Gold remaining in blanket tailing, 35·52 × 75·0=26·64 per cent.	
Gold recovered by cyanidation of blanket tailing, 26·64 × 94·83.	25·26 "
Total gold recovered in this test........................	98·62 "

CONCLUSIONS

The results of the experimental tests indicate that the ore should be treated by cyanidation, and traps and blankets should be used in the cyanide circuit to remove the coarse gold.

The flow-sheet recommended would follow the general principles outlined briefly as follows:

If the ore is to be crushed in a Hadsel mill or a ball mill, a jig should be installed between the classifier and the mill. The classifier overflow should pass over blankets and to a standard type of cyanide plant. Grinding can be done in cyanide solution, because the cycle tests Nos. 14-A to 14-D show no effect from fouling of the solutions.

The test work also shows that grinding need not be finer than 65 per cent through 200 mesh.

Ore Dressing and Metallurgical Investigation No. 677

GOLD ORE FROM THE BIDGOOD MINE, BIDGOOD KIRKLAND GOLD MINES, LIMITED, LEBEL TOWNSHIP, TIMISKAMING COUNTY, ONTARIO

Shipment. Two shipments of ore from the newly-found 520 vein of the Bidgood mine, in the east Kirkland Lake area, were received from O. L. Knutson, General Manager, Bidgood Kirkland Gold Mines, Limited, Kirkland Lake, Ontario. The first, a shipment of 28 sacks of ore, net weight 2,310 pounds, was received on January 15, 1936, and the second, a shipment of 4 sacks of ore, weighing 355 pounds, was received on March 4, 1936.

Characteristics of the Ore. A microscopic study was carried out for the purpose of determining the general character of the ore, and in particular the mode of occurrence of the gold. Hand specimens and unmounted mill products were examined under the binocular microscope, and 57 polished sections were examined under the reflecting microscope.

Samples. The ore sections, 42 in number, were prepared from specimens selected from Shipments Nos. 1 and 2, and from a special lot of five samples which had been forwarded for microscopic study.

The sections of mill products, 15 of which were prepared from products obtained during the various tests, added no further information to the nature of the ore.

The *gangue* is fine-textured and greenish grey in colour. It is composed largely of chlorite, and commonly shows rather pronounced schistosity indicating its highly altered condition. Narrow veinlets of white calcite are present, along which may occur narrow borders of serpentine; a considerable amount of calcite is also disseminated throughout the gangue.

The *metallic minerals*, of which pyrite is by far the most abundant, are disseminated in the chloritic gangue.

These minerals in their order of abundance, are:

Pyrite	Abundant	Major ore mineral
Chalcopyrite	Considerable quantity	Minor ore minerals
Magnetite	Small quantity	
Unknown No. 1, blue-grey	Small quantity	
Tetradymite	Small quantity	Accessory ore minerals
Altaite	Small quantity	
Galena	Small quantity	
Native gold	Very small quantity	
Coloradoite	Very small quantity	
Calaverite	Very small quantity	
Unknown No. 2	Traces	

Pyrite (*See* Plates IA to IIIB) is disseminated as imperfect cubes and irregular grains in the chloride gangue. It is sometimes so abundant as to form massive granular aggregates, usually along narrow bands roughly parallel to the schistosity. The pyrite contains various other minor and accessory minerals. The grain size of the pyrite in Shipment No. 1 is shown in Table I, and this is quite representative of all samples examined.

TABLE I

Grain Size of Pyrite in Shipment No. 1

(Quantitative microscopic analysis)

Mesh	Per cent	Cumulative per cent
+ 100	71·26	71·26
− 100 + 150	9·11	80·37
− 150 + 200	7·38	87·75
− 200 + 280	3·75	91·50
− 280 + 400	3·28	94·78
− 400 + 560	2·20	96·98
− 560 + 800	1·70	98·68
− 800 +1100	0·70	99·38
−1100 +1600	0·39	99·77
−1600 +2300	0·18	99·95
−2300	0·05	100·00
Total	100·00	

Chalcopyrite (*See* Plates IIA, B, C, and IIIA) occurs as disseminated irregular grains in both chloritic gangue and in calcite veinlets; a small quantity also occurs as veinlets and round blebs in pyrite. The grain size of the chalcopyrite, with the amounts of free and combined mineral, are shown in Table II.

TABLE II

Grain Size of Chalcopyrite, and Amounts of Free and Combined Chalcopyrite in Shipment No. 1

(Quantitative microscopic analysis)

Mesh	Combined,* per cent	Free, per cent	Total, per cent	Cumulative per cent
+ 100	6·37	....	6·37	6·37
− 100 + 150	9·47	8·74	18·21	24·58
− 150 + 200	11·84	2·73	14·57	39·15
− 200 + 280	8·75	4·37	13·12	52·27
− 280 + 400	4·73	5·46	10·19	62·46
− 400 + 560	6·01	5·10	11·11	73·57
− 560 + 800	8·19	4·37	12·56	86·13
− 800 +1100	6·34	0·58	6·92	93·05
−1100 +1600	4·88	....	4·88	97·93
−1600 +2300	1·09	....	1·09	99·02
−2300	0·98	....	0·98	100·00
Total	68·65	31·35	100·00	

*Chiefly with pyrite.

Magnetite (*See* Plate IIA) occurs sparingly in the chloritic gangue as small irregular grains. A light grey mineral, translucent and relatively soft, is widespread, and may represent magnetite which has been altered

PLATE II

A

B

C

D

A. Gold associated with pyrite, chalcopyrite, and magnetite.
Gold—white (**Au**); pyrite—light grey (**Py**); chalcopyrite—light grey (**Cp**); magnetite—dark grey (**Mag**); gangue—black. Magnification ×75.

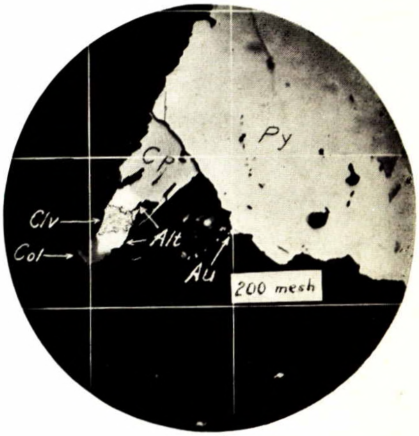
B. Gold associated with chalcopyrite.
Gold—white (**Au**); chalcopyrite—grey (**Cp**); pyrite—light grey (**Py**); gangue—black. Magnification ×275.

C. Gold in dense pyrite.
Gold—white (**Au**); pyrite—light grey (**Py**); chalcopyrite—grey (**Cp**); galena—black, etched with HNO_3 (**Ga**); gangue—black. Magnification ×75.

D. Tellurides and gold along cracks in pyrite.
Gold—white (**Au**); coloradoite—grey (**Col**); tetradymite—white (**Td**); pyrite—light grey (**Py**); gangue—black. Magnification ×75.

A

B



A. Tetradymite and chalcopyrite in pyrite.
Tetradymite—white (**Td**); chalcopyrite—grey (**Cp**); pyrite—light grey (**Py**); gangue—black. Magnification ×75.

B. Coloradoite, altaite, and calaverite, and a particle of gold in contact with pyrite.
Coloradoite—dark grey (**Col**); altaite—white (**Alt**); calaverite—stippled (**Clv**); chalcopyrite—grey (**Cp**); gold—white (**Au**); pyrite—light grey (**Py**); gangue—black. Magnification ×275.

to leucoxene (?). The unknown blue-grey mineral (No. 1) is comparatively rare and occurs as grains and elongated flake-like particles usually associated with magnetite. It is hard, strongly anisotropic, and shows red internal reflections; it may be an iron mineral, or may contain manganese or tungsten.

Tetradymite ($Bi_2(Te,S)_3$) occurs typically as small grains in pyrite (*See* Plates IID and IIIA) and as grains along fractures in pyrite, sometimes associated with coloradoite and/or native gold (*See* Plates IC, ID, and IID). A very small quantity of *galena*, which is not as yet positively differentiated from possible altaite (PbTe), occurs as small grains in both gangue and pyrite, in the latter case being present as either small rounded blebs in the pyrite or along fractures (*See* Plates IA, ID, and IIC). One grain identified as *altaite* occurs in association with chalcopyrite and calaverite (*See* Plate III B).

A large number of grains of *native gold* were observed. The gold is consistently very finely divided. It occurs as follows:

A.—In gangue; mostly commonly free, but rarely associated with the tellurides or galena (*See* Plate IA and B).

B.—Along pyrite-gangue boundaries. It may occur either against the pyrite, or may penetrate into the pyrite, as shown in Plates ID and IIA.

C.—Within pyrite. The native gold within pyrite shows two modes of occurrence: (1) It is present in the form of interstitial films between pyrite grains (*See* Plate IC), or along fractures in the pyrite. (2) It is present as small irregular grains in dense pyrite (*See* Plate IIC). In all of these occurrences it may be associated with chalcopyrite, galena, or tellurides.

The grain size of the gold and the modes of occurrence are shown in Table III. All of the 42 sections were carefully examined and all of the grains seen were measured.

TABLE III

Grain Size of the Native Gold in All Ore Sections

(Quantitative microscopic analysis; 1,008 grains of gold)

Mesh	A, free in gangue, per cent	B, against pyrite, per cent	C—in pyrite		Totals, per cent	Cumulative, per cent
			Along fractures, etc., per cent	In dense pyrite, per cent		
− 150 + 200........	1·7	1·2	....	....	2·9	2·9
− 200 + 280........	2·3	1·5	0·8	0·4	5·0	7·9
− 280 + 400........	3·4	2·3	1·2	0·6	7·5	15·4
− 400 + 560........	4·7	3·2	1·5	0·7	10·1	25·5
− 560 + 800........	7·9	4·5	2·2	1·3	15·9	41·4
− 800 +1100........	12·7	2·5	1·2	0·4	16·8	58·2
−1100 +1600........	12·9	1·4	1·1	0·3	15·7	73·9
−1600 +2300........	12·9	0·8	0·4	0·3	14·4	88·3
−2300	11·1	0·2	0·2	0·2	11·7	100·0
Total........	69·6	17·6	8·6	4·2	100·0	

Coloradoite (HgTe) is rare; it occurs as shown in Plate IID along fractures in pyrite and as shown in Plate IIIB where it is associated with calaverite and altaite. *Calaverite* ($(Au, Ag)Te_2$) is also rare; it was observed in certain concentrates and in the ore sections as shown in Plate IIIB.

Unknown No. 2, which is white in colour and hard, was seen only in the heavy fraction panned from the copper concentrate from Mill Run No. 1. It is anisotropic and is tarnished by HNO_3; it may be arsenopyrite.

Conclusions from Microscopic Study of the Ore and Mill Products. The microscopic study of polished sections of the ore and mill products from the newly discovered ore-body at Bidgood Kirkland Gold Mines has led to certain conclusions regarding its character in so far as it affects treatment. Referring to Table III, most of the gold is seen to be free in the gangue, although a small percentage occurs in dense pyrite; the remainder occurs along the boundaries of pyrite grains. The gold in the gangue is very finely divided, that in the pyrite is slightly less so, a relationship in sizes which is extremely rare. Gold telluride is present, and it is estimated that the amount of gold occurring in this form is quite considerable; the grain size of the tellurides, however, is somewhat coarser than that of the native gold. Chalcopyrite is present in considerable quantity, and is the only cyanicide known to occur in the ore in sufficient quantity to cause trouble.

These characteristics will, no doubt, largely determine the behaviour of the ore in the mill.

Sampling and Analysis. Samples were cut from the shipments by standard methods and assayed as follows:

—	Shipment No. 1	Shipment No. 2
Gold........oz./ton	1·07	0·69
Silver........"	0·50	0·50
Copper........per cent	0·20	0·04
Iron........"	16·73	11·23
Sulphur........"	15·01	3·96
Arsenic........"	Nil	....

These two shipments, therefore, represent a higher grade and more refractory type of ore than any previously found at this property, one of which formed the subject of Investigation No. 535.[1]

Experimental Tests

The test work consisted of small-scale tests by cyanidation and flotation both alone and in combination, and cyanidation tests on roasted ore.

On the first shipment, a mill run was conducted in which the ore was treated by cyanidation in a small Pachuca tank, and in a second mill run on the same shipment the copper was floated off in lime pulp and the tailing was cyanided in bottles in small-batch lots.

Work on the second shipment was confined to small-scale cyanidation tests following along the lines of the most satisfactory tests conducted on the first shipment.

Maximum recovery in any of these tests was 90 per cent of the gold.

[1] Mines Branch, Dept. of Mines, Canada, Invest. Ore Dress. and Met., July to December, 1933, p. 107.

The tests are described in detail as follows:

Shipment No. 1

CYANIDATION OF THE ORE

Tests Nos. 1 to 16

Samples of the ore were crushed dry to pass through 48-, 100-, 150-, and 200-mesh screens. Portions of each lot were agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, in bottles for periods of 24 and 48 hours. The cyanide tailings were assayed for gold. In these tests the lime was kept at 0·30 to 0·40 pound per ton of solution.

Summary:

Feed sample: gold, 1·07 oz./ton

Test No.	Grinding, mesh	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton	
					KCN	CaO
1	− 48	24	0·36	66·3	1·34	5·8
2	− 48	24	0·355	66·8	.....	.....
3	−100	24	0·315	70·6	2·54	7·1
4	−100	24	0·325	69·6	.....	.....
5	−150	24	0·37	65·4	2·92	8·1
6	−150	24	0·36	66·4	.....	.....
7	−200	24	0·355	66·8	3·07	9·1
8	−200	24	0·37	65·4	.....	.....
9	− 48	48	0·29	72·9	2·28	6·8
10	− 48	48	0·28	73·8	.....	.....
11	−100	48	0·255	76·1	3·67	8·2
12	−100	48	0·26	76·6	.....	.....
13	−150	48	0·24	77·5	4·31	9·2
14	−150	48	0·235	78·0	.....	.....
15	−200	48	0·225	79·0	4·31	10·1
16	−200	48	0·225	79·0	.....	.....

BARREL AMALGAMATION FOLLOWED BY CYANIDATION

Tests Nos. 17 to 20

Portions of the same four lots of ore as were used in Tests Nos. 1 to 16 were amalgamated with mercury in jar mills for a period of one hour. The amalgamation tailings were sampled and assayed, and portions of each were agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 24 hours. The cyanide tailings were assayed for gold.

Summary:

Feed sample: gold, 1·07 oz./ton

Test No.	Grinding, mesh	Amalgamation tailing assay, Au, oz./ton	Extraction by amalgamation, per cent	Cyanide tailing assay, Au, oz./ton	Extraction by cyanidation, per cent	Reagents consumed, lb./ton	
						KCN	CaO
17	− 48	0·97	9·35	0·38	55·1	1·10	5·2
18	−100	0·91	14·95	0·335	53·7	1·64	6·2
19	−150	0·85	20·55	0·285	52·8	2·28	5·5
20	−200	0·89	16·8	0·255	59·3	2·59	5·8

CYANIDATION WITH RED LEAD AND HIGH LIME IN DENVER SUPER-AGITATORS

Tests Nos. 21 and 22

Two samples of ore were ground 84 per cent and 98·7 per cent through 200 mesh in ball mills, with 1·0 pound of red lead per ton added. The pulps were agitated 48 hours in cyanide solution, 1·0 pound of potassium cyanide per ton, in Denver super-agitators. Lime was kept up to 1·0 pound per ton of solution during the agitation period. The cyanide tailings were assayed for gold.

Summary:

Feed sample: gold, 1·07 oz./ton

Test No.	Grinding, per cent −200 mesh	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton	
				KCN	CaO
21	84·0	0·165	84·5	1·88	12·40
22	98·7	0·125	88·3	3·80	12·40

CYANIDATION WITH HIGH LIME IN DENVER SUPER-AGITATORS

Tests Nos. 23 and 24

Samples of the ore were ground 84 per cent and 98·7 per cent through 200 mesh in ball mills and then agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 48 hours. Lime was kept at 1·0 pound per ton of solution during the agitation period. The cyanide tailings were assayed for gold.

Summary:

Feed sample: gold 1·07 oz./ton

Test No.	Grinding, per cent −200 mesh	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton	
				KCN	CaO
23	84·0	0·165	84·6	3·0	18·4
24	98·7	0·155	85·5	4·95	19·8

ROASTING AND CYANIDATION

Tests Nos. 25 to 27

Samples of the ore at −14 mesh were roasted in a muffle furnace at maximum temperatures of 600, 650, and 750 degrees Centigrade. The charges were placed in the cold furnace and heated up with it. Maximum temperature was maintained for one hour. The calcines were ground 90 per cent through 200 mesh, water-washed, and then agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 48 hours. The cyanide tailings were assayed for gold.

Summary:

Test No.	Maximum roasting temperature, degrees C.	Calcine assay, Au, oz./ton	Tailing assay, Au, oz./ton	Extraction, of gold, per cent	Reagents consumed, lb./ton	
					KCN	CaO
25	650	1·12	0·125	88·9	9·10	21·8
26	600	1·155	0·125	89·2	10·0	25·7
27	750	1·18	0·155	89·2	8·3	20·9

FLOTATION AND CYANIDATION

Test No. 28

A sample of the ore was ground 84 per cent through 200 mesh in a ball mill and a copper concentrate was floated from it in lime pulp. The concentrate was recleaned and the cleaner tailing added to the flotation tailing. The flotation tailing was agitated in cyanide solution for periods of 24 and 48 hours. The cyanide tailings were assayed for gold.

Charge to Ball Mill:

Ore	2,000 grms. −14 mesh
Lime	5·0 lb./ton

Reagents to Cell:

Sodium ethyl xanthate	0·10 lb./ton
Cresylic acid	0·10 "

Summary:

Product	Weight, per cent	Assay		Distribution, per cent		Reagents consumed, lb./ton tailing	
		Au, oz./ton	Cu, per cent	Au	Cu	KCN	CaO
Copper concentrate	1·7	27·20	9·94	43·1	62·1	....	....
Flotation tailing	98·3	0·62	0·105	56·9	37·9	....	....
Feed (cal.)	100·0	1·07	0·27	100·0	100·0	....	....
Flotation tailing cyanided—							
24 hours	....	0·155	....	....	....	1·20	5·7
48 hours	....	0·145	....	....	....	1·85	6·3

Extraction by Cyanidation of Flotation Tailing:

(1) 24 hours	42·7 per cent total gold
(2) 48 hours	43·6 " "

Loss in Cyanide Tailing:

(1) 24 hours	14·2 per cent total gold
(2) 48 hours	13·3 " "

FLOTATION AND CYANIDATION

Test No. 29

This test is a duplication of Test No. 28, with the ore ground 98·7 per cent through 200 mesh.

Summary:

Product	Weight, per cent	Assay		Distribution, per cent		Reagents consumed, lb./ton tailing	
		Au, oz./ton	Cu, per cent	Au	Cu	KCN	CaO
Copper concentrate*	1·25	48·52	14·54	60·5	69·7	....	....
Flotation tailing	98·75	0·40	0·08	39·5	30·3	....	....
Feed (cal.)	100·00	1·00	0·26	100·0	100·0	....	....
Flotation tailing cyanided							
24 hours	....	0·115	....	....	....	1·32	5·8
48 hours	....	0·10	....	....	....	1·71	6·0

*This concentrate was found, on analysis, to contain 0·112 per cent of tellurium.

COPPER FLOTATION WITH CYANIDATION FOLLOWED BY PYRITE FLOTATION

Test No. 30

A sample of the ore was ground 84 per cent through 200 mesh in a ball mill and the copper floated in a lime pulp. The copper concentrate was cleaned, but the cleaner tailing was not put back with the copper flotation tailing as was done in previous tests. The copper flotation tailing was agitated in cyanide solution for 24 hours and then reconditioned with soda ash and the pyrite floated off. The products were assayed for gold.

Charge to Ball Mill:

Ore	2,000 grms. −14 mesh
Lime	5·0 lb./ton

Reagents to Cell:

	Lb./ton
Sodium ethyl xanthate	0·10
Cresylic acid	0·10

Summary:

First (Copper) Flotation:

Product	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	Cu, per cent	Au	Cu
Copper concentrate	2·9	19·12	5·84	53·9	86·4
Copper cleaner tailing	1·8	4·08	0·42	7·2	3·9
Copper flotation tailing	95·3	0·42	0·02	38·9	9·7
Feed (cal.)	100·0	1·03	0·20	100·0	100·0

Flotation of Cyanide Tailing:

Product	Weight, per cent, of cyanide tailing	Assay, Au, oz./ton	Distribution, per cent	
			Gold in cyanide tailing	Total gold in ore
Pyrite concentrate from cyanide tailing	26·8	0·24	46·8	6·0
Flotation tailing from cyanide tailing	73·2	0·10	53·2	6·8
Cyanide tailing (cal.)	100·0	0·138	100·0	12·8

Extraction by cyanidation........................... 26·1 per cent total gold
Reagents Consumed:—KCN........................... 0·68 lb./ton tailing
CaO........................... 5·74 " "

SELECTIVE FLOTATION AND REGRINDING OF PYRITE; CYANIDATION OF PYRITE AND FLOTATION TAILING

Test No. 31

A sample of the ore was ground 84 per cent through 200 mesh in a ball mill and the chalcopyrite and pyrite floated from it selectively. The pyrite concentrate was reground all through 325 mesh and agitated in cyanide solution for a period of 48 hours. The flotation tailing was filtered and washed and agitated in cyanide solution, without further grinding, for a similar period of time. The products were assayed for gold and copper.

Charge to Ball Mill:

Ore.. 2,000 grms. −14 mesh
Soda ash.. 2·0 lb./ton
Sodium cyanide.. 0·20 "

Reagents to Cell:

Copper Flotation—
Sodium ethyl xanthate............................... 0·10 lb./ton
Cresylic acid............................... 0·10 "

Pyrite Flotation—
Copper sulphate............................... 1·0 lb./ton
Potassium amyl xanthate............................... 0·10 "
Pine oil............................... 0·05 "

Summary:

Product	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	Cu, per cent	Au	Cu
Copper concentrate	2·4	23·12	6·62	54·7	68·2
Copper cleaner tailing	1·8	3·88	0·99	6·9	7·6
Pyrite concentrate	31·2	0·73	0·18	22·5	24·2
Flotation tailing	64·6	0·25	Nil.	15·9	....
Feed (cal.)	100·0	1·014	0·233	100·0	100·0
Pyrite concentrate cyanided	....	0·10	....	....	....
Flotation tailing cyanided	....	0·11	....	....	....

Pyrite Concentrate:

Extraction by cyanidation	19·4 per cent total gold
Reagents Consumed:—KCN	16·9 lb./ton concentrate
CaO	22·1 " "

Flotation Tailing:

Extraction by cyanidation	8·4 per cent total gold
Reagents Consumed:—KCN	0·82 lb./ton tailing
CaO	7·52 " "

FLOTATION AND CYANIDATION

Test No. 32

A sample of the ore was ground 84 per cent through 200 mesh in a ball mill and a copper concentrate floated off in lime pulp. The flotation tailing was agitated in cyanide solution, 2·0 pounds of potassium cyanide per ton, for 48 hours. The products were assayed for gold and copper.

Charge to Ball Mill:

Ore	2,000 grms. at −14 mesh
Lime	5·0 lb./ton

Reagents to Cell:

Sodium ethyl xanthate	0·10 lb./ton
Cresylic acid	0·10 "

Summary:

Product	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	Cu, per cent	Au	Cu
Copper concentrate	0·80	44·94	19·56	36·2	74·2
Copper cleaner tailing	1·5	7·10	1·67	10·7	12·0
Copper flotation tailing	97·7	0·54	0·03	53·1	13·8
Feed (cal.)	100·0	0·994	0·211	100·0	100·0
Copper flotation tailing cyanided	....	0·14	....	....	....

Extraction by cyanidation of copper flotation tailing	39·4 per cent total gold
Reagents Consumed:—KCN	1·25 lb./ton tailing
CaO	5·35 " "

Test No. 33

A sample of the ore was ground all through 325 mesh with lime in a ball mill. The copper was floated off and cleaned four times, each of the cleaner tailings being kept separate. The object of four cleaning operations was to produce a high-grade copper concentrate for microscopic examination. The copper flotation tailing was then divided into two parts. One part was cyanided and the other part was washed with water and reconditioned with soda ash and the pyrite floated out of it. The pyrite concentrate and the pyrite flotation tailing were cyanided separately. The products were assayed for gold and copper.

Charge to Ball Mill:

Ore	4,000 grms. −14 mesh
Lime	5·0 lb./ton

Reagents to Cell:

Copper Flotation:

Sodium ethyl xanthate	0·10 lb./ton
Cresylic acid	0·10 "

1st Cleaning:

Cresylic acid	0·05 lb./ton

2nd Cleaning.

No additional reagents.

3rd Cleaning:

Cresylic acid	0·02 lb./ton

4th Cleaning:

Sodium cyanide	0·005 lb./ton

Pyrite Flotation:

Soda ash	2·0 lb./ton
Copper sulphate	1·5 "
Potassium amyl xanthate	0·10 "
Pine oil	0·10 "

Summary:

Copper Flotation:

Product	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	Cu, per cent	Au	Cu
Copper concentrate	0·43	71·36	31·92	29·9	68·0
First cleaner tailing	1·97	2·74	0·19	5·3	1·9
Second cleaner tailing	0·43	17·80	1·68	7·5	3·6
Third cleaner tailing	0·10	28·05	3·69	2·7	1·8
Fourth cleaner tailing	0·15	76·75	13·90	11·2	10·3
Copper flotation tailing	96·92	0·46	0·03	43·4	14·4
Feed (cal.)	100·00	1·026	0·202	100·0	100·0

Pyrite Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	
			Gold in copper flotation tailing	Total gold in ore
Pyrite concentrate	40·5	0·73	78·6	34·1
Pyrite flotation tailing	59·5	0·135	21·4	9·3
Copper flotation tailing (cal.)	100·0	0·38	100·0	43·4

Cyanidation of Products:

Product	Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent		Reagents consumed, lb./ton product	
			Gold in product	Total gold in ore	KCN	CaO
Copper flotation tailing..........	24	0·075	83·7	36·3	1·65	5·95
Copper flotation tailing..........	48	0·070	84·8	36·8	2·90	7·00
Pyrite concentrate from copper flotation tailing..............	48	0·130	82·2	28·0	3·24	14·70
Pyrite flotation tailing from copper flotation tailing.......	48	0·055	59·3	5·5	1·55	7·00

Test No. 34

A sample of the ore was ground 84 per cent through 200 mesh with lime in a ball mill. The copper was floated off with sodium ethyl xanthate and cresylic acid, as in previous tests, and the tailing was agitated in cyanide solution, 5·0 pounds of potassium cyanide per ton, for periods of 24 and 48 hours. This test was made to determine whether or not a better extraction could be obtained by using the stronger solution. The copper concentrate was not cleaned.

Summary:

Product	Weight, per cent	Assay		Distribution, per cent		Reagents consumed, lb./ton tailing	
		Au, oz./ton	Cu, per cent	Au	Cu	KCN	CaO
Copper concentrate..................	2·14	24·90	8·18	49·7	81·7	....	....
Flotation tailing....................	97·86	0·55	0·04	50·3	18·3	....	....
Feed (cal.)..........................	100·00	1·07	0·214	100·0	100·0	....	....
Tailing cyanided—							
24 hours........................	....	0·145	....	....	....	1·45	5·20
48 hours........................	....	0·140	....	....	....	1·75	5·20

Extraction by Cyanidation:

24 hours..	37·0 per cent total gold
48 hours..	37·5 " "

Test No. 35

This is a duplication of Test No. 34, with the ore ground 98·7 per cent through 200 mesh.

Summary:

Product	Weight, per cent	Assay		Distribution, per cent		Reagents consumed, lb./ton tailing	
		Au, oz./ton	Cu, per cent	Au	Cu	KCN	CaO
Copper concentrate	2·07	22·86	8·38	45·0	81·6	....	....
Flotation tailing	97·93	0·59	0·04	55·0	18·4	....	....
Feed (cal.)	100·00	1·05	0·212	100·0	100·0	....	....
Tailing cyanided—							
24 hours	....	0·11	....	....	....	1·75	6·20
48 hours	....	0·11	....	....	....	2·80	6·47

Extraction by Cyanidation:

24 hours.. 44·8 per cent total gold
48 hours.. 44·8 " "

Test No. 36

A sample of the ore was ground 84 per cent through 200 mesh with lime in a ball mill and the copper was floated with sodium ethyl xanthate and cresylic acid as before. The tailing was agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, in Denver super-agitators for periods of 24 and 48 hours. This test was made to determine whether or not better extraction would be obtained by more thorough aeration.

Summary:

Product	Weight, per cent	Assay		Distribution, per cent		Reagents consumed, lb./ton tailing	
		Au, oz./ton	Cu, per cent	Au	Cu	KCN	CaO
Copper concentrate	2·20	23·36	7·56	49·8	82·9	....	....
Flotation tailing	97·80	0·53	0·035	50·2	17·1	....	....
Feed (cal.)	100·00	1·03	0·20	100·0	100·0	....	....
Tailing cyanided—							
24 hours	....	0·17	....	....	....	0·75	7·35
48 hours	....	0·17	....	....	....	1·37	10·60

Extraction by Cyanidation:

24 hours.. 34·1 per cent total gold
48 hours.. 34·1 " "

CYANIDATION OF THE ORE

Mill Run No. 1

The ore, crushed dry to pass through a 14-mesh screen and mixed with lime, was fed at the rate of 100 pounds per hour to a ball mill in closed circuit with a classifier. The classifier overflowed at 63·5 per cent through 200 mesh and 40 per cent solids to a Pachuca tank. In this tank the lime was brought up to approximately 1 pound per ton of solution and the pulp was aerated for 6 hours. Then the cyanide was added and kept at approximately 1 pound per ton of solution for the 60-hour period of agitation which followed.

Grab samples were taken for assay at regular intervals, and at the end of the run, while the cyanide tailing was being discharged to the tailing launder, a final sample was cut from the pulp stream at regular intervals.

Results:

—	Au, oz./ton	Cu, per cent
Feed to ball mill	1·075	0·22
Classifier overflow	0·96	0·20
Tailing cyanided:—		
12 hours	0·21	
18 "	0·185	
24 "	0·175	
30 "	0·18	
36 "	0·175	
42 "	0·175	
48 "	0·15	
54 "	0·15	
Final sample	0·175	

Reagents Consumed:

KCN	2·3 lb./ton ore
CaO	18·5 " "

COPPER FLOTATION IN LIME PULP WITH CYANIDATION OF TAILING

Mill Run No. 2

The ore, mixed with lime and crushed dry to pass through a 14-mesh screen, was fed at the rate of 100 pounds per hour to a ball mill in closed circuit with a classifier. The classifier overflowed into a conditioning tank where sodium ethyl xanthate was added and where the pulp was also aerated before it went to the cells. The conditioning tank overflowed to the fourth cell in a battery of ten. A rougher concentrate from Cells Nos. 4 to 10 was returned to Cells Nos. 2 and 3 for cleaning, and the concentrate from Cells Nos. 2 and 3 was returned to Cell No. 1 for recleaning. The final concentrate was taken from Cell No. 1.

Samples of the flotation tailing were agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, in bottles for 24 and 48 hours. Flotation products were assayed for gold and copper.

Results:

—	Au, oz./ton	Cu, per cent
Feed to ball mill	1·07	0·22
Classifier overflow	0·975	0·19
Flotation concentrate	44·78	3·58
Flotation tailing	0·46	0·09
Tailing cyanided:—		
24 hours	0·175	
48 hours	0·115	

Gold recovered in flotation concentrate	53·4 per cent total gold
Gold in tailing, by difference	46·6 " "
Extraction by cyanidation of flotation tailing (48 hours)	34·9 " "

Reagents Consumed: (lb./ton tailing)

	24 hours	48 hours
KCN	1·15	1·80
CaO	7·60	9·30

Shipment No. 2

CYANIDATION OF THE ORE

Tests Nos. 1 to 8

Samples of the ore were crushed dry to pass through 48-, 100-, 150-, and 200-mesh screens and portions of each were agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for periods of 24 and 48 hours. The tailings were filtered, washed, and assayed for gold.

Summary:

Feed sample: gold, 0·69 oz./ton

Test No.	Grinding, mesh	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton	
					KCN	CaO
1	− 48	24	0·25	66·7	0·74	5·1
2	−100	24	0·215	68·8	1·02	6·8
3	−150	24	0·215	68·8	1·19	7·0
4	−200	24	0·25	63·8	1·19	7·1
5	− 48	48	0·19	72·5	0·85	5·2
6	−100	48	0·17	75·4	1·10	7·0
7	−150	48	0·16	76·8	1·23	7·0
8	−200	48	0·185	73·2	1·53	7·1

Tests Nos. 9 to 12

Samples of the ore were ground 86·6, 92·7, 98·1, and 98·9 per cent through 200 mesh in ball mills, with red lead added in the proportion of 2·0 pounds per ton of ore. The pulps were then agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, in bottles. Agitation was carried on for 48 hours at 2·5 : 1 dilution, and during this period the lime was kept up to 1·0 pound per ton of solution.

Summary:

Feed sample: gold, 0·69 oz./ton

Test No.	Grinding, per cent −200 mesh	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton	
				KCN	CaO
9	86·6	0·12	82·5	0·31	8·80
10	92·7	0·10	85·5	0·44	9·60
11	98·1	0·09	86·9	0·58	9·90
12	98·9	0·09	86·9	0·71	10·30

Test No. 13

A sample of the ore was ground 98·1 per cent through 200 mesh in a ball mill with red lead added in the proportion of 2·0 pounds per ton of ore and lime added in the proportion of 10 pounds per ton of ore. The pulp was agitated in a Denver super-agitator with cyanide solution, 1·0 pound of potassium cyanide per ton, for 48 hours. Lime was kept up to 1·0 pound per ton of solution during this period. The dilution ratio was 2·5 : 1.

Summary:

Feed sample	0·69 oz. Au/ton
Tailing assay	0·09 " "
Extraction	86·9 per cent total gold
Reagents Consumed:—KCN	0·56 lb./ton
CaO	20·6 "

CYANIDATION AND FLOTATION

Test No. 14

A sample of the ore was ground 98·1 per cent through 200 mesh with red lead and lime added in the same proportions as in Test No. 13. The pulp was agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, in bottles for 48 hours, with the lime kept up to 1·0 pound per ton of solution. The cyanide tailing was filtered, washed, and a small amount of concentrate floated off (similar to Wright Hargreaves' practice) with the following reagents:

Soda ash	0·25 lb./ton
Copper sulphate	0·25 "
Potassium xanthate	0·05 "
Cresylic acid	0·10 "

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	5·3	0·38	21·0
Tailing	94·7	0·08	79·0
Cyanide tailing (cal.)	100·0	0·096	100·0

CYANIDATION WITH TABLE CONCENTRATION

Test No. 15

A sample of the ore was ground 98·1 per cent through 200 mesh with red lead and lime and then agitated in cyanide solution, as was done in Test No. 14.

The cyanide tailing was then passed over a small concentrating table where as much coarse sulphide and sand as possible was concentrated off. The concentrate was reground and recyanided. Table tailing and cyanide tailing from reground table concentrate were assayed for gold.

Summary:

Feed sample: gold, 0·69 oz./ton.

Product	Weight, per cent	Assay, Au, oz./ton	Extraction of gold, per cent
Table tailing	8·4	0·065	
Cyanide tailing from reground table concentrate	91·6	0·095	
Cyanide tailing (cal.)	100·0	0·092	86·7

Reagents Consumed:

KCN	0·84 lb./ton ore
CaO	13·50 "

CONCLUSIONS

The test work has shown that the gold is very fine and is intimately associated with both the gangue and the sulphide minerals. It will be very difficult, if at all possible, to recover more than 85 to 90 per cent of the gold by any process other than smelting.

Assuming that the second shipment represents closely the grade and character of ore that will be sent to the mill for treatment, there will be no excessive consumption of reagents in the operation of a cyanide plant and this process is the one to be recommended.

The points to be kept in mind in laying out such a plant are that the ore will require extremely fine grinding, the finer the better within economic limits, and that it will require a long period of agitation with high lime and the addition of lead salts to precipitate sulphides taken into solution by the high lime.

The ore would be ground in cyanide solution and all operations carried out according to standard cyanidation practice, with the above-mentioned features added. In this way, 85 to 90 per cent of the gold should be extracted.

If the copper content of the ore should increase for any reason, it may become necessary to float the copper off before cyaniding in order to keep cyanide consumption down to a reasonable figure and also to keep the solution from fouling, as the test work on the first shipment has shown that the copper mineral is a very active cyanicide that would create a serious problem if present in any quantity. In this case, of course, the ore should be ground with water and lime and the flotation tailing cyanided. The copper concentrate would be sold to a smelter.

Ore Dressing and Metallurgical Investigation No. 678

GOLD ORE FROM DELNITE MINES, DELORO TOWNSHIP, COCHRANE DISTRICT, PORCUPINE AREA, NORTHERN ONTARIO

Shipment. A shipment of 42 bags of ore, net weight 2,500 pounds, was received on January 23, 1936, from the Delnite Mines, Limited, Deloro township, Cochrane district, Porcupine area, northern Ontario. The shipment was submitted by W. S. Maguire, General Manager, P.O. Box EX, Kirkland Lake, Ontario.

Sampling and Analysis. After crushing, cutting and grinding by standard methods, the sample obtained assayed as follows:—

Gold	0·53 oz./ton	Arsenic	0·15 per cent
Silver	0·10 "	Sulphur	2·13 "
Copper	0·05 per cent		

Characteristics of the Ore. The *gangue* consists of mottled light green to brown chloritic rock of fine texture and white vein quartz. A small quantity of light grey dolomite is present.

The *metallic minerals* present, in their order of abundance, are pyrite, chalcopyrite, and arsenopyrite. Pyrite is rather sparingly disseminated as coarse cubic crystals and irregular grains, though in some portions of the ore it is quite abundant. A small quantity of chalcopyrite occurs as grains and patches in the chloritic gangue, occasionally associated with pyrite. Arsenopyrite commonly occurs as small disseminated crystals either associated with pyrite or in the chloritic gangue.

No native gold was seen in the polished sections, and its mode of occurrence is not known.

Experimental Tests

The test work included amalgamation, cyanidation, flotation and table concentration. Flotation, followed by regrinding and cyanidation of the flotation concentrate, produced the highest recovery of the gold.

BARREL AMALGAMATION

Tests Nos. 1 and 2

One thousand grammes of the ore at −14 mesh was ground in a ball mill to pass, 79·8 per cent in Test No. 1 and 87·9 per cent in Test No. 2, through a 200-mesh screen. The pulp was then amalgamated with mercury for one hour in a jar mill. The amalgamation tailing was assayed for gold.

Screen tests showed the grindings as follows:—

Mesh	Weight, per cent	
	Test No. 1	Test No. 2
− 65 +100	1·0	0·5
−100 +150	4·8	1·9
−150 +200	14·4	9·7
−200	79·8	87·9

Amalgamation:

Test No.	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
1	0·53	0·40	24·5
2	0·53	0·35	34·0

The above tests were for the purpose of determining the total amounts of gold set free by these particular degrees of comminution and the result is not comparable to the amount of gold that could be recovered by either plates or blankets.

CYANIDATION

Tests Nos. 3, 4, and 5

In these tests, the ore at −14 mesh was ground in a ball mill to different degrees of fineness. The resulting pulps were then filtered and portions agitated in cyanide solutions of a strength of 1·5 pounds of potassium cyanide. Five pounds of lime was added per ton of ore and agitation was conducted for periods of 24 and 48 hours. The ratio of dilution was 3 of solution to 1 of ore. The cyanide tailings were assayed for gold.

Screen tests showed the grindings as follows:—

Mesh	Weight, per cent		
	Test No. 3	Test No. 4	Test No. 5
− 35 + 48	0·3	....	....
− 48 + 65	1·3	0·2	....
− 65 +100	6·3	2·1	....
−100 +150	10·7	7·1	1·2
−150 +200	18·2	17·0	8·0
−200	63·2	73·6	90·8

Cyanidation:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
3	24	0·53	0·03	94·3	0·3	4·0
3	48	0·53	0·03	94·3	0·6	4·1
4	24	0·53	0·03	94·3	0·3	4·0
4	48	0·53	0·03	94·3	0·9	4·2
5	24	0·53	0·03	94·3	0·7	6·2
5	48	0·53	0·03	94·3	0·9	6·4

CYANIDATION

Tests Nos. 6 and 7

In these tests, the ore was ground in cyanide solutions at strengths of 2 pounds per ton, to a fineness of 85·0 per cent in Test No. 6 and 89·5 per cent in Test No. 7, through 200 mesh. The pulps were then agitated for 24 hours in cyanide solutions of the same strength. Ten pounds of lime per ton of ore was added to maintain protective alkalinity.

Screen tests showed the grindings as follows:—

Mesh	Weight, per cent	
	Test No. 6	Test No. 7
− 65 +100	0·3	0·1
−100 +150	3·2	1·6
−150 +200	11·5	8·6
−200	85·0	89·5

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
6	24	0·53	0·03	94·3	0·5	8·70
7	24	0·53	0·025	95·3	0·6	8·60

AERATION AND CYANIDATION

Test No. 8

In this test, the pulp was aerated sixteen hours prior to cyanidation. One and a half pounds of potassium cyanide per ton of solution and 10 pounds of lime per ton of ore were used during 24 hours' agitation. The ore was ground to pass 89·0 per cent through 200 mesh.

Results:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
8	24	0·53	0·025	95·3	0·6	8·6

HYDRAULIC CLASSIFICATION, AMALGAMATION, AND CYANIDATION

Tests Nos. 9, 10, and 11

In these tests, the ore at −14 mesh was ground in a ball mill to pass 68·7, 79·8, and 86·0 per cent through 200 mesh. The pulps were then passed through a hydraulic classifier and the heavier mineral particles removed. These hydraulic concentrates were then combined and amalgamated and the amalgam residues added to the hydraulic tailing. These products were agitated in cyanide solutions of 1·5 pounds of potassium cyanide strength for 24 hours, 10 pounds of lime per ton of tailing being added.

Screen tests showed the different grinds as follows:—

Mesh	Weight, per cent		
	Test No. 9	Test No. 10	Test No. 11
− 48 + 65	0·2	....	....
− 65 +100	3·2	0·8	0·4
−100 +150	9·4	5·0	2·2
−150 +200	18·5	14·4	11·4
−200	68·7	79·8	86·0

Hydraulic Classifications:

Test No. 9:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·53	100·0	
Hydraulic concentrate	1·82	5·92	20·3	55 : 1
Hydraulic tailing	98·18	0·43	79·7	

Test No. 10:

Feed	100·0	0·53	100·0	
Hydraulic concentrate	1·1	11·32	23·5	91 : 1
Hydraulic tailing	98·9	0·41	76·5	

Test No. 11:

Feed	100·0	0·53	100·0	
Hydraulic concentrate	1·1	9·52	19·8	91 : 1
Hydraulic tailing	98·9	0·43	80·2	

Amalgamation of Combined Concentrates:

Tests Nos.	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
9, 10, 11	8·38	2·45	70·7

Cyanidation of Hydraulic Tailing Plus Amalgam Residue:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
9	24	0·465	0·025	94·6	0·3	7·6
10	24	0·43	0·025	94·2	0·3	7·7
11	24	0·045	0·025	94·5	0·3	7·8

Summary:

—	Test No. 9, per cent	Test No. 10, per cent	Test No. 11, per cent
Bullion recovered by amalgamation	14·3	16·6	14·0
Bullion recovered by cyanidation	81·0	78·6	81·3
Overall recovery	95·3	95·2	95·3

HYDRAULIC CLASSIFICATION AND FLOTATION

Test No. 12

The ore at −14 mesh was ground to pass 66·2 per cent through 200 mesh. The pulp was then passed through a hydraulic classifier and the heavier particles removed. The hydraulic tailing was reground with 3 pounds of soda ash and 0·2 pound of Barrett No. 4 per ton and floated with 0·2 pound of amyl xanthate and 0·05 pound of pine oil. The combined hydraulic and flotation concentrates were reground to pass 100 per cent through 200 mesh and agitated in cyanide solution of a strength of 5 pounds of potassium cyanide per ton. Agitation was conducted for a 24-hour period.

A screen analysis showed the grinding to be as follows:—

Mesh	Weight, per cent	
	Hydraulic tailing	Flotation tailing
− 48 + 65	0·2	....
− 65 +100	3·7	0·5
−100 +150	10·4	1·1
−150 +200	19·4	7·2
−200	66·2	91·2

Hydraulic Classification:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·53	100·0	
Hydraulic concentrate	0·79	9·90	14·8	126 : 1
Hydraulic tailing	99·21	0·455	85·2	

Flotation of Hydraulic Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·455	100·0	
Flotation concentrate	10·3	4·29	97·0	9·7 : 1
Flotation tailing	89·7	0·015	3·0	

Cyanidation of Flotation Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	4·75	0·09	98·1	2·5	19·25

Summary:

Gold recovered in hydraulic concentrate .... 14·8 per cent
Gold recovered in flotation concentrate .... 82·6 "
Bullion recovered by cyanidation .... 95·5 "

FLOTATION AND CYANIDATION

Test No. 13

The ore at −14 mesh was ground in a ball mill with 3 pounds of soda ash and 0·2 pound of Barrett No. 4 per ton. The fineness of grinding being 90·0 per cent through 200 mesh. The pulp was then floated with 0·2 pound of amyl xanthate and 0·05 pound of pine oil per ton. The flotation concentrate was reground to pass 100 per cent through 200 mesh, and agitated in cyanide solution of a strength of 5 pounds per ton for 24 hours.

A screen test showed the grinding to be as follows:

Mesh	Weight, per cent
−100 +150	1·2
−150 +200	8·8
−200	90·0

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·0	0·53	100·0	
Flotation concentrate	13·5	3·90	99·2	7·4 : 1
Flotation tailing	86·5	0·005	0·8	

Cyanidation of Flotation Concentrates:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	3·90	0·075	98·1	2·3	18·50

Summary:

	Per cent
Gold recovered in flotation concentrate	99·2
Bullion recovered by cyanidation	97·3

FLOTATION AND CYANIDATION

Test No. 14

In this test, the ore was treated exactly similar to Test No. 13, with the exception that the grind was somewhat coarser. A screen test showed the grinding as follows:—

Mesh	Weight, per cent
− 48 + 65	0·1
− 65 +100	2·3
−100 +150	7·7
−150 +200	16·7
−200	73·2

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·53	100·0	
Flotation concentrate	10·25	5·04	97·5	9·7 : 1
Flotation tailing	89·75	0·015	2·5	

Cyanidation of Flotation Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	5·04	0·09	98·2	2·80	18·00

Summary:

Gold recovered in flotation concentrate	97·5 per cent
Bullion recovered by cyanidation	95·75 "

CYANIDATION AND TABLE CONCENTRATION

Test No. 15

The ore at −14 mesh was ground to pass 76·4 per cent through 200 mesh. The pulp was agitated in cyanide solution of a strength of 1·5 pounds of potassium cyanide per ton. The cyanide tailing was passed over a Wilfley table and a table concentrate recovered. This concentrate was reground to pass 95 per cent through 200 mesh and agitated in cyanide solution of 5 pounds per ton strength. The different products were assayed for gold.

A screen test on the cyanide tailing showed the grinding to be as follows:

Mesh	Weight, per cent
− 65 +100	1·4
−100 +150	5·9
−150 +200	16·3
−200	76·4

Cyanidation:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·53	0·035	93·4	0·4	8·5

Table Concentration of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·035	100·0	
Table concentrate	5·38	0·475	72·9	18·6 : 1
Table tailing	94·62	0·01	27·1	

Cyanidation of Table Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·475	0·11	76·8	2·0	18·65

Summary:

Bullion recovered by cyanidation of raw ore	93·4 per cent
Bullion recovered by cyanidation of table concentrate	3·7 "
Overall recovery	97·1 "

SETTLING TESTS

Tests Nos. 16 and 17

These tests were carried out in a tall glass tube having an inside diameter of 2 inches. The pulp, having been previously ground with 5 pounds of lime per ton of ore, was transferred to the glass tube and the level of solids in decimals of feet read every five minutes. Readings were made for a one-hour period. At the end of the test the solution was titrated for alkalinity. A screen test showed the grinding in both tests to be as follows:—

Mesh	Weight, per cent
− 65 +100	0·1
−100 +150	2·0
−150 +200	9·1
−200	88·8

The results of the tests are recorded in the following tables.

Test No. 16:

Ratio of solid to liquid	1 : 2·5
Lime added per ton solid	10·0 pounds
Alkalinity of solution at end of test	0·50 CaO, lb./ton of solution
Overflow solution	Clear
Rate of settling	1·33 foot/hour

Time	Settlement of solids in feet	Cumulative settlement
0 hour 5 minutes	0·07	0·07
0 " 10 "	0·10	0·17
0 " 15 "	0·115	0·285
0 " 20 "	0·105	0·390
0 " 25 "	0·120	0·510
0 " 30 "	0·130	0·640
0 " 35 "	0·120	0·760
0 " 40 "	0·110	0·870
0 " 45 "	0·12	0·99
0 " 50 "	0·12	1·11
0 " 55 "	0·12	1·23
1 " 0 "	0·10	1·33

Test No. 17:

Ratio of solid to liquid	1 : 2
Lime added per ton solid	10·0 pounds
Alkalinity of solution at end of test	0·40 CaO, lb./ton of solution
Overflow solution	Clear
Rate of settling	0·97 foot/hour

Time	Settlement of solids in feet	Cumulative settlement
0 hour 5 minutes	0·085	0·085
0 " 10 "	0·075	0·160
0 " 15 "	0·080	0·240
0 " 20 "	0·080	0·320
0 " 25 "	0·070	0·390
0 " 30 "	0·090	0·480
0 " 35 "	0·08	0·56
0 " 40 "	0·08	0·64
0 " 45 "	0·08	0·72
0 " 50 "	0·09	0·81
0 " 55 "	0·08	0·89
1 " 0 "	0·08	0·97

The rate of settling in these tests is normal and no difficulty should be experienced from this source in the treatment of the ore.

SUMMARY AND CONCLUSIONS

The ore grinds easily and the consumption of cyanide and lime is normal.

Straight cyanidation of the ore gives a maximum extraction of 95·3 per cent of the gold; when followed by table concentration of the cyanide tailing, and regrinding and cyanidation of the resulting concentrate, the overall recovery is raised to 97·1 per cent.

Flotation of the ore followed by regrinding and cyanidation of the flotation concentrate resulted in an overall recovery of 95·75 per cent with a fineness of grinding approximating the cyanidation and table concentration test, namely 73 to 76 per cent through 200 mesh. When the fineness of grinding was raised to 90 per cent through 200 mesh, the overall recovery of the gold was 97·3 per cent.

The sample of ore submitted, as evidenced by the barrel amalgamation, hydraulic classification, and microscopic examination, contains little free and no coarse gold.

Ore Dressing and Metallurgical Investigation No. 679

GOLD ORE FROM THE MORRIS KIRKLAND MINE IN THE EAST KIRKLAND AREA, LEBEL TOWNSHIP, TIMISKAMING COUNTY, ONTARIO

Shipment. A shipment of 13 sacks of ore, net weight 1,040 pounds, was received on March 30, 1936. The sample was submitted by T. C. Fawcett, Mine Superintendent, Morris Kirkland Gold Mines, Limited, King Kirkland, Ontario.

Character of the Ore. Eleven polished sections were prepared and examined microscopically for the purpose of determining the character of the ore.

The *gangue* is fine-textured and siliceous, and contains a small quantity of carbonate as disseminated grains and narrow veinlets.

Pyrite is the only abundant metallic mineral. It is disseminated as medium to fine grains and poorly formed cubes. A small quantity of chalcopyrite is present as finely disseminated grains and more rarely as veinlets in pyrite. Rare small grains of galena occur in both gangue and pyrite, but it is possible that some of these may be altaite. A light grey mineral resembling tetrahedrite is also of rare occurrence in the gangue.

Only one grain of native gold was seen. This is approximately 1600 mesh in size and occurs in dense pyrite.

Sampling and Analysis. The sample received was crushed and sampled through a Jones sampler. The feed sample thus obtained assayed as follows:

Gold	0·25 oz./ton
Copper	0·04 per cent

Experimental Tests

Previous tests have been conducted on ore from this property and were reported in Investigation No. 529, July 1933.

The object of the present investigation was to determine whether the ore represented by the last shipment could be treated in the same way as the first one.

The work done on this ore consisted of cyanidation tests with various grindings and periods of agitation and also cyanidation coupled with table concentration and regrinding of the sulphides.

By straight cyanidation, 92·1 per cent of the gold was extracted in 48 hours when the ore was ground 97·8 per cent through 200 mesh. With the same initial grind plus table concentration and regrinding of the sulphides, 92·4 per cent of the gold was extracted.

The tests are described in detail as follows:

CYANIDATION

Tests Nos. 1, 2, and 3

Samples of the ore were ground 87·4, 94·9, and 97·8 per cent through 200 mesh in ball mills and agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 24 hours. The tailings were assayed for gold. Lime in solution was kept up to 1·0 pound per ton during the agitation period.

Summary: Feed sample: gold, 0·25 oz./ton

Test No.	Grinding, per cent −200 mesh	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton	
					KCN	CaO
1..................	87·4	24	0·035	86·0	0·71	8·7
2..................	94·9	24	0·03	87·9	0·71	8·9
3..................	97·8	24	0·03	87·9	0·71	8·9

Tests Nos. 4, 5, and 6

Samples of the ore were ground in ball mills for the same periods of time as in Tests Nos. 1 to 3, but red lead was added to the charge to the ball mill in the proportion of 2·0 pounds per ton of ore. The pulps were agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 24 hours. As in Tests Nos. 1 to 3, lime was kept up to 1 pound per ton of solution during the agitation period.

Summary: Feed sample: gold, 0·25 oz./ton

Test No.	Grinding, per cent −200 mesh	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton	
					KCN	CaO
4..................	87·4	24	0·03	87·9	0·37	8·50
5..................	94·9	24	0·03	87·9	0·47	8·70
6..................	97·8	24	0·025	90·0	0·57	8·80

Tests Nos. 7, 8, and 9

Samples of the ore were ground for the same periods of time as in the first two series of tests, with red lead and lime added to the mill in the proportions of 2·0 and 6·0 pounds per ton respectively. The pulps were agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 48 hours. Lime was kept up to 1·0 pound per ton of solution during the agitation period.

Summary: Feed sample: gold, 0·25 oz./ton

Test No.	Grinding, per cent −200 mesh	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton	
					KCN	CaO
7	87·4	48	0·03	87·9	0·31	10·25
8	94·8	48	0·025	90·0	0·31	10·50
9	97·8	48	0·02	92·1	0·31	10·60

CYANIDATION WITH TABLE CONCENTRATION AND REGRINDING OF THE SULPHIDES

Test No. 10

A sample of the ore was ground 97·8 per cent through 200 mesh in a ball mill, with red lead and lime added in the proportions of 2·0 and 6·0 pounds per ton respectively. The pulp was agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 24 hours. Lime was kept up to 1·0 pound per ton of solution during the agitation period. The primary cyanide tailing was treated on a small concentrating table where a sulphide concentrate was produced. This concentrate was reground all through 325 mesh and re-agitated in cyanide solution for a further period of 24 hours. The table tailing and the cyanide tailing from the reground concentrate were assayed for gold.

Summary: Feed sample: gold, 0·25 oz./ton

Product	Weight, per cent	Assay, Au, oz./ton	Extraction, of gold, per cent	Reagents consumed, lb./ton	
				KCN	CaO
Table tailing from primary cyanide tailing	87·5	0·015			
Table concentrate cyanided	12·5	0·045			
Average tailing (cal.)	100·0	0·019	92·4	0·47	11·0

Test No. 11

This test was conducted in exactly the same way as was Test No. 10 except that the initial grind was only 75 per cent through 200 mesh.

Summary: Feed sample: gold, 0·25 oz./ton

Product	Weight, per cent	Assay, Au, oz./ton	Extraction, of gold, per cent	Reagents consumed, lb./ton	
				KCN	CaO
Table tailing from primary cyanide tailing	80·7	0·02			
Table concentrate cyanided	19·3	0·06			
Average tailing (cal.)	100·0	0·028	88·9	0·57	11·8

CONCLUSIONS

There appears to be no difference between this shipment and the previous one. The gold is very fine and the ore will require fine grinding and a long period of agitation. Some benefit will be derived from the use of red lead and high lime, the function of the lead being to precipitate sulphides taken into solution by the high lime.

Extraction will be slightly increased by table concentration and regrinding of the sulphides.

Ore Dressing and Metallurgical Investigation No. 680

GOLD ORE FROM THE JOWSEY ISLAND GOLD MINES, LIMITED, GOD'S LAKE, MANITOBA

Shipment. A shipment of gold ore, weight 3,000 pounds, was received on March 23, 1936, from the Jowsey Island Gold Mines, Limited, God's Lake area, 130 miles from Ilford on the Hudson Bay railway, Manitoba.

Characteristics of the Ore. The *gangue* consists chiefly of greenish grey silicified chlorite schist, in some portions of which the alteration has been so marked that very slight trace of the schistosity remains. This contains veins of dark grey smoky quartz.

The *metallic mineral* content of the ore is small, and is chiefly pyrrhotite, pyrite, and arsenopyrite. Magnetite, chalcopyrite, and galena are present in exceedingly small quantities, and eight grains of native gold were seen. The sulphides occur mostly in the schist, though rarely; pyrite and arsenopyrite are present along the borders of and within the quartz veins. Galena and native gold were seen only in the quartz near the borders of the veins. Pyrrhotite is finely disseminated, and pyrite and arsenopyrite are somewhat more coarsely disseminated in the schist.

All of the native gold seen occurs as rounded particles ranging from about 1100 mesh to 560 mesh in size. One grain is enclosed in galena, which in turn is enclosed in pyrite; three grains lie along the boundaries of arsenopyrite crystals; and four grains are free in the quartz.

Sampling and Analysis. The shipment was crushed and sampled by standard methods, and the analysis of the feed sample is as follows:

Gold	0·34 oz./ton
Silver	0·07 "
Arsenic	0·20 per cent
Copper	Trace
Sulphur	0·62 per cent

Experimental Tests

Test work on the ore comprised hydraulic classification at different finenesses of grinding; standard cyanidation tests; barrel amalgamation tests to determine the amount of free-milling gold at different grinds; blanket concentration to remove free gold, which was amalgamated, and subsequent cyanidation of combined amalgamation tailing and blanket tailing; and a cycle test on −100-mesh ore to determine to what extent fouling of the cyanide solution occurs.

Grinding tests show that the ore is comparatively hard.

The results of the tests follow in detail:

HYDRAULIC CLASSIFICATION

Test No. 1

A sample of −14-mesh ore was ground in an Abbé grinding jar and the pulp then fed to a hydraulic classifier. The oversize was panned to remove excess gangue.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·32	100·00	
Oversize	0·42	18·34	24·37	238·1 : 1
Overflow	99·58	0·24	75·63	

Screen Test on Overflow:

Mesh	Weight, per cent
+ 28	3·3
− 28 + 35	8·0
− 35 + 48	11·9
− 48 + 65	15·2
− 65 +100	13·0
−100 +150	9·7
−150 +200	9·5
−200	29·4
	100·0

Test No. 2

In this test the grinding was increased.

Visible gold was observed in the panned oversize.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·31	100·00	
Oversize	0·03	499·30	48·36	3333·3 : 1
Overflow	99·97	0·16	51·64	

Screen Test on Overflow:

Mesh	Weight, per cent
+ 48	2·8
− 48 + 65	7·9
− 65 +100	15·3
−100 +150	14·0
−150 +200	16·6
−200	43·4
	100·0

Test No. 3

In this test still finer grinding was carried out. The higher gold assay of the overflow product would indicate the liberation of fine gold, which was carried into the overflow product.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·35	100·00	
Oversize	0·03	420·45	35·42	3333·3 : 1
Overflow	99·97	0·23	64·58	

Screen Test on Overflow:

Mesh	Weight, per cent
+ 65	0·9
− 65 +100	6·2
−100 +150	13·4
−150 +200	20·2
−200	59·3
	100·0

CYANIDATION

Test No. 4

This was a standard cyanidation test on samples of ore ground to different degrees of fineness. Agitation was carried out in bottles for 24 hours at a pulp dilution of 3 : 1 in solution strength equivalent to 1 pound of potassium cyanide per ton, with 5 pounds of lime per ton added as protective alkalinity.

Grinding, mesh	Assay, Au, oz./ton		Extraction, of gold, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
− 48	0·34	0·01	97·06	0·66	4·10
−100	0·34	0·015	95·59	0·66	4·25
−150	0·34	0·01	97·06	0·96	5·25
−200	0·34	0·01	97·06	0·96	6·25

Screen Tests on Respective Samples:

−48 Mesh	Weight, per cent
+ 65	14·0
− 65 +100	21·5
−100 +150	14·8
−150 +200	12·7
−200	37·0
	100·0

−100 Mesh	Weight, per cent
+100	0·4
−100 +150	15·1
−150 +200	20·3
−200	64·2
	100·0

−150 Mesh	Weight, per cent
+200	21·7
−200	78·3
	100·0

Test No. 5

This test was similar to Test No. 4, but agitation was carried out for 48 hours.

Grinding, mesh	Assay, Au, oz./ton		Extraction, of gold, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
− 48	0·34	0·015	95·59	0·96	4·75
−100	0·34	0·01	97·06	0·96	4·75
−150	0·34	0·01	97·06	1·11	5·40
−200	0·34	0·01	97·06	0·96	6·40

BARREL AMALGAMATION

Test No. 6

In this test a sample of ore was ground to a fineness of 35·5 per cent −200 mesh and then barrel-amalgamated with mercury for 1 hour in order to determine the free-milling gold at the fineness of grinding indicated.

The tailing assayed 0·085 ounce gold per ton, indicating a gold recovery of 75·00 per cent.

Test No. 7

In this test the ore sample was ground to a fineness of 59·3 per cent −200 mesh and then barrel-amalgamated. The tailing assayed 0·075 ounce gold per ton, indicating a gold recovery of 77·94 per cent.

BLANKET CONCENTRATION, AMALGAMATION, AND CYANIDATION

Test No. 8

From the results of the foregoing tests it is apparent that a fairly large proportion of the gold is freed by moderately fine grinding and will easily amalgamate. In view of this, the following test was carried out:

A sample of ore was ground in a 4 : 3 pulp to a fineness of 71·8 per cent −200 mesh and then fed to a corduroy blanket. The blanket concentrate was then amalgamated with mercury in a mortar for half an hour, after which the amalgamation tailing was combined with the blanket tailing and treated by cyanide in two lots for 24 and 48 hours respectively.

The strength of the cyanide solution was equivalent to 1 pound of potassium cyanide per ton and the pulp dilution was 2·5 : 1.

The results are as follows:

Agitation, hours	Assay of final tailing, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton	
			KCN	CaO
24	0·005	98·53	0·50	4·50
48	0·005	98·53	0·775	4·88

Test No. 9

This test was similar to Test No. 8, with the exception that the ore was ground to a fineness of 87·4 per cent −200 mesh. The results indicate that this increased grinding is not necessary.

The results are as follows:

Agitation, hours	Assay of final tailing, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton	
			KCN	CaO
24	0·005	98·53	0·50	4·50
48	0·005	98·53	0·775	5·00

CYCLE TESTS

Test No. 10

This was a cycle test consisting of five 24-hour cycles. Five bottles were used in the first cycle, four in the second, three in the third, two in the fourth, and one in the fifth. The solution strength was 1 pound of potassium cyanide per ton, dilution 3 : 1. Tests were carried out in standard bottles—200 grammes of ore and 600 c.c. of solution.

The ore was ground dry in a Braun pulverizer to a fineness of approximately 64 per cent −200 mesh.

Cycle No. 1:

Five bottles containing 200 grammes of ore and 600 c.c. of solution having a strength of 1 pound potassium cyanide per ton.

Results:

Bottle	Reagent consumption, lb./ton		Assay, Au, oz./ton		Extraction of gold, per cent
	KCN	CaO	Feed	Tailing	
A	0·3	5·25	0·34	0·01	
B	0·3	5·25	0·34	0·005	
C	0·3	5·25	0·34	0·01	Average 97·65
D	0·3	5·25	0·34	0·005	
E	0·3	5·25	0·34	0·01	

NOTE.—No cyanide was added during the period of the cycle. One pound of lime per ton was added to each bottle. Final titration at 24 hours was:

KCN	0·90 lb./ton.
CaO	0·25 "

Cycle No. 2:

Four bottles were made up with the solution from the five bottles used in the previous cycle. No wash water was added to dilute the solutions. Cyanide and lime were added to bring the solution strength back to 1 pound of potassium cyanide, and 5 pounds of lime per ton.

Results:

Bottle	Reagent consumption, lb./ton		Assay, Au, oz./ton		Extraction of gold, per cent
	KCN	CaO	Feed	Tailing	
A	0·90	4·10	0·34	0·005	Average 97·79
B	0·90	4·10	0·34	0·01	
C	0·90	4·10	0·34	0·005	
D	0·90	4·10	0·34	0·01	

NOTE.—No additions of cyanide or lime made during run. Final titration at 24 hours:

KCN.. 1·0 lb./ton
CaO.. 0·30 "

Cycle No. 3:

Three bottles were made up with the solution from four bottles used in the previous cycle.

Results:

Bottle	Reagent consumption, lb./ton		Assay, Au, oz./ton		Extraction of gold, per cent
	KCN	CaO	Feed	Tailing	
A	0·15	4·27	0·34	0·01	Average 97·06
B	0·15	4·27	0·34	0·01	
C	0·15	4·27	0·34	0·01	

NOTE.—No cyanide was added during run. Five pounds of lime per ton was added. Solution was treated with zinc dust at the end of this cycle. Final titration at 24 hours:

KCN.. 0·95 lb./ton
CaO.. 0·275 "

Cycle No. 4:

Two bottles were made up with the solution from the previous cycle. Addition of cyanide was made to bring the solution strength back to 1 pound of potassium cyanide per ton.

Results:

Bottle	Reagent consumption, lb./ton		Assay, Au, oz./ton		Extraction of gold, per cent
	KCN	CaO	Feed	Tailing	
A	0·30	5·32	0·34	0·01	Average 97·06
B	0·30	5·32	0·34	0·01	

NOTE.—No cyanide was added during run. One pound of lime per ton was added. Final titration at 24 hours:

KCN.. 0·90 lb./ton
CaO.. 0·225 "

Cycle No. 5—Final:

Addition of cyanide to bring solution back to 1 pound of potassium cyanide per ton.

Results:

Bottle	Reagent consumption, lb./ton		Assay, Au, oz./ton		Extraction of gold, per cent
	KCN	CaO	Feed	Tailing	
A..............	0·30	4·25	0·34	0·01	97·06

NOTE.—No cyanide or lime was added during the final cycle. Final titration at 24 hours:
KCN.. 1·0 lb./ton
CaO.. 0·25 "
Final solution precipitated by zinc dust.

CYANIDE SOLUTION

Test No. 10

Reducing power: 192 c.c. standard $\frac{N}{10}$ $KMnO_4$/litre.

KCNS.. 0·09 grm./litre
Fe.. 0·0034 "
Cu.. 0·04 "

Summary of Test No. 10:

Cycle No.	Reagent consumption, lb./ton		Average assay, Au, oz./ton		Average extraction of gold, per cent
	KCN	CaO	Feed	Tailing	
1..............................	0·30	5·25	0·34	0·008	97·65
2..............................	0·90	4·10	0·34	0·0075	97·79
3..............................	0·15	4·27	0·34	0·01	97·06
4..............................	0·30	5·32	0·34	0·01	97·06
5..............................	0·30	4·25	0·34	0·01	97·06

CYCLE TESTS

Test No. 11

The possibility that a thicker pulp might lower the extraction suggested itself.

A cycle test was therefore run using a pulp dilution of 1·5 : 1—200 grammes of ore and 300 c.c. of solution.

The cycles were run in a similar manner to the previous test. No zinc dust was added to the solutions for precipitation. The ore was ground to the same degree as in Test No. 10. The agitation period was 24 hours in all cycles.

Cycle No. 1:

Five bottles containing 200 grammes of ore and 300 c.c. of solution having a strength of 1 pound potassium cyanide and 5 pounds lime per ton

Results:

Bottle	Reagent consumption, lb./ton		Assay, Au, oz./ton		Extraction of gold, per cent
	KCN	CaO	Feed	Tailing	
A..............	0·225	4·70	0·34	0·01	
B..............	0·225	4·70	0·34	0·005	
C..............	0·225	4·70	0·34	0·005	Average 98·24
D..............	0·225	4·70	0·34	0·005	
E..............	0·225	4·70	0·34	0·005	

NOTE.—No cyanide or lime was added during the run. Final titration at 24 hours:
KCN.. 0·85 lb./ton
CaO.. 0·20 "

Cycle No. 2:

Four bottles were made up with solution from the previous cycle. Owing to the high pulp density of this test there was not sufficient solution for the four bottles, so that it was necessary to add 17·5 c.c. of fresh water to each bottle to bring the solution volume up to 300 c.c.

After adding fresh ore to the bottles, the solution on titration showed potassium cyanide 0·05 pound per ton and lime, nil. Cyanide was added to bring the solution to 1 pound of potassium cyanide per ton. Five pounds of lime per ton was also added.

Results:

Bottle	Reagent consumption, lb./ton		Assay, Au, oz./ton		Extraction of gold, per cent
	KCN	CaO	Feed	Tailing	
A..............	1·20	4·70	0·34	0·005	
B..............	1·20	4·70	0·34	0·005	Average 98·16
C..............	1·20	4·70	0·34	0·01	
D..............	1·20	4·70	0·34	0·005	

NOTE.—No additions of cyanide or lime were made during the run. Final titration at 24 hours:
KCN.. 1·0 lb./ton
CaO.. 0·2 "

Cycle No. 3:

Three bottles were made up with solution from the previous run plus 10 c.c. fresh water to each bottle to give necessary volume of 300 c.c. Cyanide was added to give strength of 1 pound of potassium cyanide per ton.

Results:

Bottle	Reagent consumption, lb./ton		Assay, Au, oz./ton		Extraction of gold, per cent
	KCN	CaO	Feed	Tailing	
A..............	0·30	4·70	0·34	0·01	Average 97·55
B..............	0·30	4·70	0·34	0·01	
C..............	0·30	4·70	0·34	0·005	

NOTE.—No addition of cyanide or lime was made during the run. Titration at 24 hours:
KCN.. 1·0 lb./ton
CaO.. 0·2 "

Cycle No. 4:

Two bottles were made up with solution from the previous run. No additional water was added. Cyanide and lime were added to bring solution to 1 pound of potassium cyanide per ton and 5 pounds of lime per ton.

Results:

Bottle	Reagent consumption, lb./ton		Assay, Au, oz./ton		Extraction of gold, per cent
	KCN	CaO	Feed	Tailing	
A	0·53	4·70	0·34	0·005	Average 98·53
B	0·53	4·70	0·34	0·005	

NOTE.—No addition of cyanide or lime was made during the run. Final titration at 24 hours:
KCN.. 1·0 lb./ton
CaO.. 0·2 "

Cycle No. 5—Final Cycle:

No cyanide was added to the solution for the final cycle.

Results:

Bottle	Reagent consumption, lb./ton		Assay, Au, oz./ton		Extraction of gold, per cent
	KCN	CaO	Feed	Tailing	
A	0·15	4·70	0·34	0·01	97·06

NOTE.—One pound of lime per ton was added during the run. Final titration at 24 hours:
KCN.. 0·9 lb./ton
CaO.. 0·2 "

CYANIDE SOLUTION

Test No. 11

Reducing power: 400 c.c. standard $\frac{N}{10}$ $KMnO_4$/litre

KCNS.. 0·58 grm./litre
Cu.. 0·080 "

The results of this cycle test, compared with Test No. 10, indicate that agitation in a denser pulp does not lower the extraction. Results from the analysis of the final solution would indicate that a greater fouling of the solution occurs in the denser pulp.

CONCLUSIONS

The results of the tests conducted on the ore submitted indicate that the ore is somewhat hard, although extremely fine grinding is not necessary to free the gold. At a fineness of 59·3 per cent −200 mesh, 77·94 per cent of the gold was free-milling, as indicated in Test No. 7. At a grinding giving a fineness of 71·8 per cent −200 mesh, and with treatment by amalgamation and cyanidation, tests indicated a recovery of 98·53 per cent of the gold. (Test No. 8).

Two cycle tests at pulp densities of 3 : 1 and 1·5 : 1, consisting of five 24-hour cycles, showed no serious fouling of the solution or lowered extraction.

The results of the tests indicate that the ore is amenable to treatment by cyanidation and that in spite of the presence of a small amount of pyrrhotite no fouling took place in the cycle tests to lower extraction. It is recommended that the coarser gold particles freed by grinding should be recovered by jigs in the classifier–mill circuit.

Ore Dressing and Metallurgical Investigation No. 681

GOLD ORE AND TAILING FROM THE LONG LAKE PROPERTY OF THE LEBEL ORO MINES, LIMITED, SUDBURY MINING DISTRICT, NORTHERN ONTARIO

Shipment. Twenty-two bags of ore weighing 1,200 pounds and sixteen bags of tailing weighing 2,750 pounds were received March 2, 1936, from D. W. M. Ross, Superintendent, Lebel Oro Mines, Limited, P.O. Box 156, Sudbury, Ontario. The ore and tailing samples were from the old Long Lake mine, situated near Naughton, in the Sudbury mining district, Ontario.

Characteristics of the Ore. The *gangue* is medium- to fine-textured granular quartz, varying from light grey to dark grey in colour.

The *metallic minerals* present in the polished sections are pyrite, arsenopyrite, chalcopyrite, and ilmenite. Pyrite occurs as irregular masses and grains, and is abundant locally, though the quantity in the ore as a whole is probably small. A small quantity of arsenopyrite is present as coarse crystals or clusters of crystals; it is rarely associated with pyrite. Corroded grains of ilmenite are sparsely disseminated in the gangue, and a very small quantity of chalcopyrite occurs as irregular grains and fine stringers in the quartz.

Sampling and Assaying. After crushing, cutting, and grinding by standard methods, samples of the ore and tailing were obtained which assayed as follows:—

	Ore	Tailing	
Gold	0·58	0·185	oz./ton
Silver	0·05	0·03	"
Copper	0·07	0·06	per cent
Arsenic	0·07	5·57	"
Sulphur	6·76	6·04	"
Iron	7·16	9·61	"

EXPERIMENTAL TESTS

The test work included amalgamation, hydraulic classification, blanket concentration, and cyanidation.

From the ore sample, about 98 per cent of the gold was recovered by cyanidation and from the tailing sample, 80 per cent can be recovered by the same method. A mixed sample of approximately 50 per cent ore and 50 per cent tailing gave a recovery of 85 per cent.

Tests Nos. 1 to 5 were made on the ore sample. Tests Nos. 5 to 13 were made on the tailing sample and Tests Nos. 14 to 16 were made on the combined sample.

Previously, test work had been conducted by the Department on a sample of tailing received on July 17, 1926, and is covered by Report No. 252 of that year.

AMALGAMATION

Test Nos. 1 and 2

One thousand grammes of ore at −14 mesh was ground in a ball mill to pass 70·0 per cent in Test No. 1 and 86·2 per cent in Test No. 2, through 200 mesh. The pulp was then amalgamated with mercury in a jar mill for one hour. The amalgamation tailings were assayed for gold. Screen tests showed the grindings as follows:—

Mesh	Weight, per cent	
	Test No. 1	Test No. 2
− 65 +100	0·8	....
−100 +150	6·0	1·3
−150 +200	23·2	12·5
−200	70·0	86·2

Test No.	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
1	0·58	0·13	77·6
2	0·58	0·105	81·9

The above tests were for the purpose of determining the total amounts of gold set free by these particular degrees of comminution and the result is not comparable to the amount of gold that could be recovered by either plates or blankets.

CYANIDATION

Test No. 3

In this test, the ore at −14 mesh was ground to different degrees of fineness and the pulps agitated in cyanide solutions of a strength of one pound of potassium cyanide per ton. Ten pounds of lime were added per ton of ore. Agitation was conducted for 24- and 48-hour periods and the ratio of dilution was three of solution to one of ore.

Screen tests showed the grindings as follows:—

Mesh	Weight, per cent		
	Test No. 3-A	Test No. 3-B	Test No. 3-C
− 65 +100	0·7	0·2	....
−100 +150	5·0	2·9	1·4
−150 +200	19·1	16·3	11·3
−200	75·2	80·6	87·3

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
3-A	24	0·58	0·23	60·3	0·3	8·8
3-A	48	0·58	0·02	96·6	0·3	9·2
3-B	24	0·58	0·18	69·0	0·3	9·1
3-B	48	0·58	0·02	96·6	0·3	9·2
3-C	24	0·58	0·12	79·3	0·3	9·1
3-C	48	0·58	0·02	96·6	0·45	9·2

HYDRAULIC CLASSIFICATION, BLANKET CONCENTRATION, CYANIDATION

Test No. 4

The ore at −14 mesh was ground in a ball mill to pass 88·5 per cent through 200 mesh. The pulp was then passed through a hydraulic classifier and a separation made of the coarse gold and heavier particles. The hydraulic tailing was then passed over a corduroy blanket, set at a slope of 2½ inches per foot and a blanket concentrate recovered. The combined hydraulic and blanket concentrates were then amalgamated with mercury and the amalgamation tailing added to the blanket tailing. This product was divided into two parts and agitated at 2·5 : 1 dilution in cyanide solution of a strength of 1·5 pounds of potassium cyanide per ton for periods of 24 and 48 hours, 10 pounds of lime per ton of ore being added.

A screen test showed the grinding as follows:—

Mesh	Weight, per cent
−100 +150	1·2
−150 +200	10·3
−200	88·5

Hydraulic and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·58	100·0	
Combined concentrate	1·60	23·35	64·4	62·5 : 1
Blanket tailing	98·40	0·21	35·6	

Amalgamation of Combined Concentrates:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
23·35	0·42	98·2

Cyanidation of Amalgamation Residue Plus Blanket Tailing:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·215	0·06	72·1	1·85	9·25
48	0·215	0·01	95·4	2·15	9·10

Summary:

	Per cent
Bullion recovered by amalgamation	63·2
" " " cyanidation (48 hours)	35·1
Overall recovery	98·3

CYANIDATION

(Cycle Test)

Test No. 5

The ore at −14 mesh was ground in a ball mill to pass 85·0 per cent through 200 mesh. Four portions were then agitated in cyanide solutions of a strength of 1·5 pounds per ton. The ratio of dilution was three of solution to one of ore. Ten pounds of lime per ton of ore was added. After 48 hours' agitation, the pulp was filtered and a fresh cycle on three portions of ore started with the same solutions as were used in Cycle No. 1. The procedure was repeated in Cycles Nos. 3 and 4. The cyanide tailings were assayed for gold and the consumption of cyanide and lime noted. A screen test showed the grinding as follows:—

Mesh	Weight, per cent
−100 +150	1·2
−150 +200	13·8
−200	85·0

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
Cycle No. 1:					
48	0·58	0·01	98·3	1·80	9·1
48	0·58	0·01	98·3	1·80	9·1
48	0·58	0·01	98·3	1·80	9·1
48	0·58	0·01	98·3	1·80	9·1
Cycle No. 2:					
48	0·58	0·01	98·3	0·55	7·75
48	0·58	0·01	98·3	0·55	7·75
48	0·58	0·01	98·3	0·55	7·75
Cycle No. 3:					
48	0·58	0·025	95·7	0·45	4·75
48	0·58	0·015	97·4	0·45	4·75
Cycle No. 4:					
48	0·58	0·01	98·3	0·35	4·60

Assays showed the final solution to contain the following:—

KCNS	0·39 grm./litre
Cu	0·109 "
Fe	0·0019 "
Reducing power	305 c.c. $\frac{N}{10}$ $KMnO_4$/litre

Test No. 5 concluded the work on the ore sample. Tests Nos. 6 to 13 were conducted on the tailing sample.

SCREEN ANALYSIS OF THE TAILING

Test No. 6

A screen test and assay of the different mesh products of the tailing sample resulted as follows:—

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
− 20 + 28	4·5	0·125	3·2
− 28 + 35	4·7	0·14	3·7
− 35 + 48	3·8	0·16	3·4
− 48 + 65	4·4	0·135	3·3
− 65 +100	16·2	0·125	11·3
−100 +150	18·8	0·115	12·0
−150 +200	23·0	0·15	19·2
−200	24·5	0·32	43·7

AMALGAMATION

Tests Nos. 7 and 8

One thousand grammes of tailing was ground in a ball mill to pass 77·0 per cent and 90·6 per cent through 200 mesh. The pulp was then amalgamated with mercury in a jar mill for one hour. The amalgamation tailings were assayed for gold. Screen tests showed the grinding as follows:–

Mesh	Weight, per cent	
	Test No. 7	Test No. 8
− 48 + 65	0·2	....
− 65 +100	0·9	....
−100 +150	4·9	1·2
−150 +200	17·0	8·2
−200	77·0	90·6

Amalgamation:

Test No.	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
7	0·185	0·13	29·7
8	0·185	0·12	35·1

The results of these tests show that 29·7 and 35·1 per cent of the gold was free at these particular degrees of comminution.

Test No. 9

The tailing was ground to different sizes and the different pulps agitated in cyanide solution having a ratio of dilution of 3 : 1 and a strength of 1 pound of potassium cyanide, for 24 and 48 hours. Ten pounds of lime per ton of tailing was added. Frequent additions of cyanide and lime were necessary.

Screen tests showed the grindings as follows. In Test No. 9-A, the tailing was agitated without regrinding.

Mesh	Weight, per cent		
	Test No. 9-B	Test No. 9-C	Test No. 9-D
− 20 + 28	1·4	....	....
− 28 + 35	1·5	....	....
− 35 + 48	1·0	....	....
− 48 + 65	4·0	....	....
− 65 +100	12·0	1·3	....
−100 +150	17·5	6·0	0·6
−150 +200	19·5	18·0	7·8
−200	43·0	74·4	91·5

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
9-A	24	0·185	0·095	48·7	4·30	30·0
9-A	48	0·185	0·07	62·2	6·60	59·7
9-B	24	0·185	0·095	48·7	4·90	30·0
9-B	48	0·185	0·055	70·3	6·90	59·4
9-C	24	0·185	0·085	54·1	5·20	30·0
9-C	48	0·185	0·055	70·3	8·30	59·7
9-D	24	0·185	0·09	51·4	4·90	30·0
9-D	48	0·185	0·065	64·9	9·60	59·7

Test No. 10

This test was a duplicate of Tests Nos. 7, 8, 9, and 10, Report No. 252, 1926, Investigations in Ore Dressing and Metallurgy.

No. 10-A. Tailing 750 grammes at −24·5 mesh, ground three hours, in a pebble mill using 1 : 1 pulp, 0·025 per cent potassium cyanide solution and lime 25 pounds per ton. Cyanided 48 hours 1 : 2 pulp using 0·05 pound of potassium cyanide.

No. 10-B. Tailing 600 grammes at −24·5 mesh, ground three hours in a pebble mill using 1 : 1·5 pulp, 0·025 per cent potassium cyanide solution and lime 25 pounds per ton. Cyanided 48 hours 1 : 3 pulp, 0·05 pound of potassium cyanide.

No. 10-C. Tailing 600 grammes at −24·5 mesh, ground three hours in a pebble mill using 1 : 1·5 pulp with lime, 10 pounds per ton. Cyanided 48 hours 1 : 3 pulp, 0·075 pound of potassium cyanide.

No. 10-D. Tailing 600 grammes at −24·5 mesh, ground three hours in 1 : 1·5 pulp, then dewatered and filtered. Cyanided 48 hours 1 : 3 pulp, 0·075 per cent of potassium cyanide.

Results:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
10-A	48	0·185	0·04	78·4	2·15	48·8
10-B	48	0·185	0·02	89·2	1·55	48·6
10-C	48	0·185	0·04	78·4	2·95	16·2+
10-D	48	0·185	0·02	89·2	3·40	38·4

Test Nos. 11 and 12

The tailing was ground to pass 77·3 per cent and 94·7 per cent through 200 mesh. The pulp was then filtered and washed thoroughly. The products were then repulped and aerated for sixteen hours and finally agitated in cyanide solution of a strength of 1·5 pounds per ton for 24 hours. The ratio of dilution was 2·5 : 1. Twenty pounds of lime per ton of tailing was added.

Screen tests showed the grinding as follows:—

Mesh	Weight, per cent	
	Test No. 11	Test No. 12
− 65 +100	0·3	....
−100 +150	2·4	0·2
−150 +200	19·8	5·0
−200	77·3	94·7

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
11	24	0·185	0·065	64·9	3·85	19·7
12	24	0·185	0·06	67·4	5·40	19·7

CYANIDATION

(Cycle Test)

Test No. 13

This test on the tailing was conducted similarly to Test No. 5, on the ore.

The tailing was ground to pass 84·1 per cent through 200 mesh and agitated in cyanide solution of 1·5 pounds per ton strength with 25 pounds of lime per ton of tailing added.

A screen test of the grinding was as follows:—

Mesh	Weight, per cent
−100 +150	1·5
−150 +200	14·3
−200	84·1

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
Cycle No. 1:					
48	0·185	0·05	73·0	6·90	38·8
48	0·185	0·05	73·0	6·90	38·8
48	0·185	0·04	78·4	6·90	38·8
48	0·185	0·04	78·4	6·90	38·8
Cycle No. 2:					
48	0·185	0·025	86·5	7·65	31·9
48	0·185	0·03	83·8	7·65	31·9
48	0·185	0·03	83·8	7·65	31·9
Cycle No. 3:					
48	0·185	0·025	86·5	7·80	41·7
48	0·185	0·03	83·8	7·80	41·7
Cycle No. 4:					
48	0·185	0·035	81·1	9·55	53·4

Assays on the final solution resulted as follows:—

KCNS	6·8	grm./litre
Cu	0·058	"
Fe	0·0031	"
Reducing power	3,921	c.c. $\frac{N}{10}$ $KMnO_4$/litre

Test No. 13 concluded the test work on the tailing sample.

Tests Nos. 14 to 16 were conducted on a mixed sample of the ore and tailing. Fifty per cent of each product was used, and the composite sample assayed 0·385 ounce of gold per ton.

Tests Nos. 14 and 15

In these tests the composite sample was ground to pass 84·6 per cent through 200 mesh. The pulp was filtered, washed, and aerated 16 hours.

In Test No. 14, one pound of potassium cyanide was used per ton of solution and in Test No. 15, 1·5 pounds. Twenty pounds of lime per ton of product was used in both tests. The ratio of dilution was 2·5 : 1 and the time of agitation 48 hours.

A screen test showed the grinding as follows:—

Mesh	Weight, per cent
− 65 +100	0·3
−100 +150	1·4
−150 +200	13·7
−200	84·6

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
14	48	0·385	0·085	77·9	3·90	21·75
15	48	0·385	0·055	85·7	5·20	21·75

CYANIDATION

(Cycle Test)

Test No. 16

In this test the composite sample was ground to pass 84·6 per cent through 200 mesh as in Test No. 15. The pulp was then washed, filtered, and aerated 16 hours. Cyanidation was conducted similarly to Tests Nos. 5 and 13. The cyanide solutions having a strength of 1·5 pounds of potassium cyanide, and 20 pounds of lime per ton of sample being added.

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
Cycle No. 1:					
48	0·385	0·045	88·3	8·95	34·25
48	0·385	0·115	70·9	8·95	34·25
48	0·385	0·055	85·7	8·95	34·25
48	0·385	0·08	79·2	8·95	34·25
Cycle No 2:					
48	0·385	0·015	96·1	5·00	34·70
48	0·385	0·015	96·1	5·00	34·70
48	0·385	0·015	96·1	5·00	34·70
Cycle No. 3:					
48	0·385	0·01	97·4	5·60	32·00
48	0·385	0·015	96·1	5·60	32·00
Cycle No. 4:					
48	0·385	0·015	96·1	5·40	32·00

Assays showed the final solution as follows:—

KCNS	3·89 grm./litre
Reducing power	2,471 c.c. $\frac{N}{10}$ $KMnO_4$/litre

SUMMARY AND CONCLUSIONS

The ore shipment assaying 0·58 ounce of gold per ton and 0·07 per cent of arsenic presents no metallurgical problem and will cyanide satisfactorily after the removal of the coarse gold by traps or a gold jig. Over 60 per cent of the gold can be recovered by amalgamation, and of the remaining 40 per cent, over 35 per cent can be extracted by cyanidation.

The tailing shipment, assaying 0·185 ounce of gold per ton and 5·57 per cent of arsenic, requires to be well washed, reground, and aerated to produce the maximum extraction of 80 per cent of the gold. The consumption of lime and cyanide still remains very high after washing and aeration.

In treating a mixed sample of the ore and tailing, an extraction of 85·7 per cent was obtained by cyanidation. A high consumption of lime and cyanide was entailed.

The partly oxidized condition of the sulphides in the tailing makes the problem of satisfactory extraction, with economical amounts of lime and cyanide, a difficult one. In a mill feed composed of ore and tailing, it is recommended that washing and aeration be adopted prior to cyanidation. Any coarse gold in the ore should be removed by traps or a gold jig; this product being barrel-amalgamated and the amalgam residue returned to the grinding circuit. The consumption of lime and cyanide can be reduced considerably by washing and aeration of the tailing. It can also be reduced by having the ratio of ore to tailing in the mill feed, 3 : 1, or even 4 : 1, if economically possible.

Ore Dressing and Metallurgical Investigation No. 682

GOLD ORE FROM BILMAC GOLD MINES, MACMURCHY TOWNSHIP, WEST SHININGTREE AREA, SUDBURY MINING DIVISION, NORTHERN ONTARIO

Shipment. Shipments of two separate bags of material, No. 1, weighing 77 pounds and consisting of vein material, and No. 2 of 54 pounds containing hanging-wall material were received May 23, 1936, from L. F. Hogarth, President, Bilmac Gold Mines, Limited, 33 Temperance Street, Toronto.

After crushing, cutting, and grinding by standard methods, a sample of each lot was obtained which assayed as follows:—

	Lot No. 1	Lot No. 2
Gold	0·49	0·02 oz./ton
Silver	0·14	0·01 "
Arsenic	0·04	0·05 per cent
Copper	Trace	Trace "
Sulphur	0·48	0·83 "

Characteristics of the Ore. Only Sample No. 1, consisting of the vein material, was investigated mineragraphically. Six polished sections were prepared and examined microscopically for the purpose of providing a short description of the sample.

The *gangue* is medium- to fine-textured white quartz with small patches of rusty impure carbonate.

The *metallic minerals* present are pyrite, arsenopyrite, "limonite", chalcopyrite, pyrrhotite, and native gold.

Pyrite and arsenopyrite are sparingly disseminated as medium to fine grains, and surface oxidation has resulted in their partial alteration to "limonite"; the amount of "limonite" is very small, and is present chiefly as stains. Small grains of chalcopyrite occur rarely in both gangue and pyrite, and very rare tiny grains of pyrrhotite are present in the pyrite. Only two grains of native gold were seen; both are between 800 and 1600 mesh in size, and occur in quartz.

Preparation of Sample for Testing:

To allow for dilution of the vein material when mining, a composite sample of the two lots was made by mixing 20 per cent of the hanging-wall material of Lot No. 2 with 100 per cent of the vein material of Lot No. 1.

The composite sample assayed 0·375 ounce of gold per ton. The experimental test work was performed on the composite sample.

Experimental Tests

The test work included, amalgamation, hydraulic classification, blanket concentration, table concentration, and cyanidation. Cyanidation combined with amalgamation produced a recovery of 97 per cent of the gold.

BARREL AMALGAMATION

Test Nos. 1 and 2

One thousand grammes of the ore at −14 mesh was ground in a ball mill to pass 77 per cent in Test No. 1 and 87 per cent in Test No. 2 through a 200-mesh screen. The pulps were then amalgamated with mercury for one hour in a jar mill. The amalgamation tailing was assayed for gold.

A screen test showed the grinding as follows:—

Mesh	Weight, per cent	
	Test No. 1	Test No. 2
− 65 +100	1·2	0·1
−100 +150	7·0	2·5
−150 +200	14·8	10·4
−200	77·0	87·0

Test	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
1	0·375	0·075	80·0
2	0·375	0·05	86·7

The above tests were for determining the total amounts of gold set free by these particular degrees of comminution and the result is not comparable to the amount of gold that could be recovered by either plates or blankets.

PLATE AMALGAMATION, TABLE CONCENTRATION

Test No. 3

The ore at −14 mesh was ground to pass 89·3 per cent through 200 mesh. The pulp was then passed over an amalgamation plate and the plate tailing concentrated on a Wilfley table. A screen test showed the grinding as follows:—

Mesh	Weight, per cent
− 65 +100	0·2
−100 +150	2·7
−150 +200	7·7
−200	89·3

Plate Amalgamation:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
0·375	0·09	76·0

Table Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·09	100·0	26 : 1
Table concentrate	3·84	0·83	34·1	
Table middling	18·87	0·10	20·3	
Table tailing	77·29	0·055	45·6	

Summary:

Bullion recovered by amalgamation.............................. 76·0 per cent
Gold recovered in table concentrate.............................. 8·2 "

PLATE AMALGAMATION, BLANKET CONCENTRATION

Test No. 4

The ore at −14 mesh was ground to pass 79·0 per cent through a 200-mesh screen. The pulp was then passed over an amalgamation plate and the plate tailing passed over a corduroy blanket. A screen test showed the grinding as follows:—

Mesh	Weight, per cent
− 65 +100	0·6
−100 +150	5·6
−150 +200	14·7
−200	79·0

Plate Amalgamation:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
0·375	0·10	73·4

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·10	100·0	95 : 1
Blanket concentrate	1·05	2·74	31·0	
Blanket tailing	98·95	0·075	69·0	

Summary:

Bullion recovered by amalgamation.............................. 73·4 per cent
Gold recovered in blanket concentrate............................ 8·2 "

HYDRAULIC CLASSIFICATION, CYANIDATION

Tests Nos. 5, 6, 7, 8

In these tests the ore was ground to different degrees of fineness and the pulps passed through a hydraulic classifier. The hydraulic concentrates were combined and amalgamated and the amalgam residues added to the hydraulic tailings. These products were then agitated in cyanide solution of a strength of 1·0 pound of potassium cyanide per ton for 24 hours with 5·0 pounds of lime per ton of ore added to maintain protective alkalinity. The ratio of dilution was one of ore to two of solution. Screen tests showed the different grinds as follows:—

Mesh	Weight, per cent			
	Test No. 5	Test No. 6	Test No. 7	Test No. 8
− 65 +100	4·5	1·6	0·4	0·1
−100 +150	11·9	9·3	4·3	1·6
−150 +200	19·3	18·3	14·1	9·3
−200	64·3	70·8	81·1	89·0

Hydraulic Classification:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Test No. 5:				
Feed	100·00	0·375	100·0	
Hydraulic concentrate	1·37	9·02	32·9	73 : 1
Hydraulic tailing	98·63	0·255	67·1	
Test No. 6:				
Feed	100·00	0·375	100·0	
Hydraulic concentrate	1·41	12·26	46·1	71·1 : 1
Hydraulic tailing	98·59	0·205	53·9	
Test No. 7:				
Feed	100·00	0·375	100·0	
Hydraulic concentrate	1·11	12·40	36·7	90 : 1
Hydraulic tailing	98·89	0·24	63·3	
Test No. 8:				
Feed	100·00	0·375	100·0	
Hydraulic concentrate	0·86	15·36	35·2	116 : 1
Hydraulic tailing	99·14	0·245	64·8	

Cyanidation of Hydraulic Tailings and Amalgam Residues:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·265	0·02	92·5	0·3	3·95
24	0·22	0·02	90·9	0·3	3·95
24	0·25	0·01	96·0	0·3	4·10
24	0·26	0·01	96·2	0·3	4·25

Summary:

Test No.	Bullion recovered by		Overall recovery
	Amalgamation	Cyanidation	
5	19·9	74·1	93·0
6	27·9	65·5	93·4
7	22·2	74·7	96·9
8	21·3	75·7	97·0

HYDRAULIC CLASSIFICATION, BLANKET CONCENTRATION, CYANIDATION

Test No. 9

In this test, the ore was ground to pass 80·4 per cent through a 200-mesh screen. The pulp was then passed through a hydraulic classifier and a hydraulic or trap concentrate removed. The hydraulic tailing was then passed over a corduroy blanket and a blanket concentrate recovered. The combined concentrates were then amalgamated in a jar mill and the amalgam residue added to the blanket tailing. This product was divided into two parts and agitated in cyanide solutions of 1·0 pound of potassium cyanide per ton strength for 24 and 48 hours, 5 pounds of lime per ton of tailing being added. The various products were assayed for gold.

A screen test showed the grinding as follows:—

Mesh	Weight, per cent
− 65 +100	0·4
−100 +150	4·9
−150 +200	14·3
−200	80·4

Hydraulic Classification:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·375	100·0	
Hydraulic concentrate	0·57	20·20	30·7	175 : 1
Hydraulic tailing	99·43	0·26	69·3	

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·26	100·0	
Blanket concentrate	0·97	13·40	50·0	103 : 1
Blanket tailing	99·03	0·13	50·0	

Amalgamation of Combined Concentrates:

Assay, Au, oz./ton		Recovery, per cent
Feed	Tailing	
16·05	0·405	97·5

Cyanidation of Blanket Tailing and Amalgam Residue:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
	Feed	Tailing		KCN	CaO
24	0·135	0·01	92·6	0·3	4·1
48	0·135	0·01	92·6	0·4	4·4

Summary:

Gold recovered in hydraulic concentrate	30·7 per cent
Gold recovered in blanket concentrate	34·6 "
Bullion recovered by amalgamation	63·65 "
Bullion recovered by cyanidation	33·65 "
Overall recovery	97·3 "

SETTLING TEST

Test No. 10

This test was carried out in a tall glass tube having an inside diameter of 2 inches. The ore was ground with 7 pounds of lime per ton to pass 83·6 per cent through a 200-mesh screen. The pulp was then transferred to the glass tube and the level of solids in decimals of a foot read every five minutes. Readings were made for a one-hour period. At the end of the test the solution was titrated for alkalinity. The results of the test are recorded in the following table.

Ratio of liquid to solid	2 : 1
Lime added per ton solid	7·0 pounds
Alkalinity of solution at end of test	0·40 lb./ton of solution
Overflow	Clear
Rate of settling	0·50 foot/hour

Time	Settlement of solids in feet	Cumulative settlement
0 hour 5 minutes	0·05	0·05
0 " 10 "	0·05	0·10
0 " 15 "	0·04	0·14
0 " 20 "	0·04	0·18
0 " 25 "	0·04	0·22
0 " 30 "	0·04	0·26
0 " 35 "	0·04	0·30
0 " 40 "	0·04	0·34
0 " 45 "	0·04	0·38
0 " 50 "	0·04	0·42
0 " 55 "	0·04	0·46
1 " 0 "	0·04	0·50

SUMMARY AND CONCLUSIONS

Over 60 per cent of the gold in the ore can be recovered by amalgamation. Of the remaining 40 per cent, over 35 per cent can be extracted by cyanidation provided the ore is ground to 75 to 80 per cent through 200 mesh.

Cyanidation is the method indicated for the treatment of this ore. Traps and blankets to remove coarse gold from the grinding circuit should be installed. This would necessitate handling the concentrate produced, barrel amalgamation, and retorting the amalgam.

No difficulty should be encountered in making a 96 to 97 per cent recovery of the gold from ore similar to that sent for test purposes.

Ore Dressing and Metallurgical Investigation No. 683

SILVER-BEARING MILL TAILING AND CONCENTRATE FROM THE ELDORADO GOLD MINES, LIMITED, ECHO BAY, GREAT BEAR LAKE, N.W.T.

A. Mill Tailing

Shipment. Two shipments of mill tailing were received on January 3 and March 25, 1936, respectively. The first shipment weighed 89 pounds and the second 80 pounds. The shipments were made by the Eldorado Gold Mines, Limited, from its mill at Echo Bay, Great Bear Lake, N.W.T.

Characteristics of Tailing. These shipments represent the tailing product from gravity concentration of the ore, and the silver contained therein is largely of mineral types other than native silver. Appreciable amounts of copper are also present.

The pitchblende and the greater amount of the native silver are recovered by gravity concentration in a high-grade product, which is shipped to the radium refinery at Port Hope, Ontario.

Purpose of Test Work. Tests were made on the mill tailing with the object of making a silver flotation concentrate that was sufficiently high in grade to allow for economic shipment to a smelter. Differential concentration of the silver and copper was also tried, but did not prove successful.

Sampling and Assaying. The assays of the shipments were as follows:

—	Ag, oz./ton	Cu, per cent	U_3O_8, per cent
Shipment No. 1	38·08	1·27	0·20
Shipment No. 2	43·20	1·30	0·12

Experimental Tests

Test work was carried out on the material of Shipment No. 1 only, as the second shipment analysed very closely to the first and was considered to be similar.

FLOTATION

Test No. 1

A sample of the mill tailing was given a short grind in a pulp containing 4 pounds of soda ash per ton. The pulp was then conditioned with 2 pounds of sodium silicate per ton, 0·2 pound of potassium ethyl xanthate per ton, and 0·15 pound of pine oil per ton.

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Ag, oz./ton	Cu, per cent	Ag	Cu	
Feed	100·00	24·03	1·27	100·00	100·00	
Concentrate	4·48	200·80	16·26	37·43	57·23	22·32 : 1
Tailing	95·52	15·74	0·57	62·57	42·77	

Screen Test on Flotation Tailing:

Mesh	Weight, per cent
+ 20	0·7
− 20 + 28	3·1
− 28 + 35	3·3
− 35 + 48	11·4
− 48 + 65	14·6
− 65 +100	20·1
−100 +150	13·8
−150 +200	13·0
−200	20·0
	100·0

The results indicate that finer grinding of the mill tailing is necessary.

Test No. 2

In this test the mill tailing was given a finer grind. The reagents used were the same as in Test No. 1.

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Ag, oz./ton	Cu, per cent	Ag	Cu	
Feed	100·00	38·22	1·24	100·00	100·00	
Concentrate	6·29	462·00	15·24	76·02	77·32	15·9 : 1
Tailing	93·71	9·78	0·30	23·98	22·68	

A screen test on the tailing indicated a fineness of grinding of 35·4 per cent −200 mesh.

Test No. 3

The grinding in this test was still further increased to give an indicated fineness of 58·4 per cent −200 mesh.

The reagents added were:

	Lb./ton
Soda ash	4·0
Sodium silicate	1·0
Potassium ethyl xanthate	0·20
Pine oil	0·10

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Ag, oz./ton	Cu, per cent	Ag	Cu	
Feed	100·00	38·46	1·27	100·00	100·00	
Concentrate	6·46	493·40	17·00	82·88	86·07	15·48 : 1
Tailing	93·54	7·04	0·19	17·12	13·93	

AMALGAMATION

Test No. 4

This test was carried out to determine if any free native silver were still present in the mill tailing.

A sample of the tailing, 1,000 grammes in weight, was ground in a pebble jar with water, 2 grammes of soda ash, and 0·5 gramme of sodium cyanide.

The pulp was then barrel-amalgamated with 100 grammes of mercury for 1 hour.

Silver in feed	38·08 oz./ton
Silver in amalgamation tailing	29·74 "
Recovery by amalgamation	21·90 per cent

Test No. 5

In this test the mill tailing was ground with sodium sulphide and coal-tar creosote. The results are not encouraging, as shown below:

Reagents Added:	Lb./ton
Soda ash	4·0
Sodium sulphide	4·0
Coal-tar creosote	0·128
Potassium amyl xanthate	0·20
Pine oil	0·10

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Ag, oz./ton	Cu, per cent	Ag	Cu	
Feed	100·00	33·39	1·23	100·00	100·00	
Concentrate	8·24	341·8	12·76	84·34	85·78	12·13 : 1
Tailing	91·76	5·7	0·19	15·66	14·22	

Test No. 6

In this test, 2 pounds of copper sulphate per ton was added as well as the reagents used in Test No. 5.

A slightly higher grade of concentrate was made.

Test No. 7

In this test the tailing was first tabled on a small Wilfley table and a concentrate carrying 85·04 ounces of silver per ton was obtained. This concentrate was then reground in a soda ash pulp with a drop of coal-tar creosote and then conditioned with sodium silicate and potassium amyl xanthate. The flotation concentrate assayed 587·80 ounces of silver per ton and carried 16·52 per cent of copper.

The results did not indicate that any advantage would be gained by this method.

The following tests consisted in making a bulk concentrate and then cleaning the concentrate. Attempts were made to depress either the silver or the copper in the cleaner circuit, but these met with no appreciable success as regards differential flotation of the silver and copper.

Test No. 8

Reagents Added to Bulk Flotations:	Lb./ton
Soda ash	3·0
Barrett No. 4	0·176
Potassium amyl xanthate	0·4
Pine oil	0·05

The bulk concentrate was conditioned with lime, 28 pounds per ton.

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Ag, oz./ton	Cu, per cent	Ag	Cu	
Feed	100·00	38·37	1·17	100·00	100·00	
Cleaner concentrate	5·00	481·4	17·30	62·8	73·8	20 : 1
Middling	3·77	231·6	5·52	22·8	17·7	
Bulk tailing	91·23	6·1	0·11	14·4	8·5	

Test No. 9

In this test the bulk concentrate was conditioned with cyanide in order to depress the copper sulphide and make a low copper, high silver cleaner concentrate. The results indicated no selective action.

A bulk float was made as in Test No. 8, and the rougher concentrate conditioned with sodium cyanide, 7·3 pounds per ton of concentrate.

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Ag, oz./ton	Cu, per cent	Ag	Cu	
Feed	100·00	35·81	1·24	100·00	100·00	
Cleaner concentrate	4·26	515·80	20·11	61·35	68·9	23·47 : 1
Middling	2·60	321·00	8·42	23·30	17·6	
Bulk tailing	93·14	5·90	0·18	15·35	13·5	

The grinding was to a fineness of 72 per cent −200 mesh.

Test No. 10

In this test the fineness of grinding was 48 per cent −200 mesh and the bulk concentrate was conditioned with 80 pounds of lime per ton of concentrate for 10 minutes. No depressant effect on the silver was apparent.

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Ag, oz./ton	Cu, per cent	Ag	Cu	
Feed	100·00	37·35	1·18	100·00	100·00	
Cleaner concentrate	4·11	549·30	18·32	60·45	63·71	24·33 : 1
Middling	3·33	237·84	7·32	21·21	20·63	
Bulk tailing	92·56	7·40	0·20	18·34	15·66	

Test No. 11

The following test was carried out similarly to the three preceding tests. Grinding was to a fineness of 61·6 per cent −200 mesh. The rougher concentrate was conditioned with bleaching powder, 34·5 pounds per ton of concentrate. A small lowering only of the silver content of the cleaner concentrate was effected.

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Ag, oz./ton	Cu, per cent	Ag	Cu	
Feed	100·00	37·34	1·24	100·00	100·00	
Cleaner concentrate	4·15	514·90	18·82	57·23	63·13	24·09 : 1
Middling	3·10	305·70	9·61	25·38	24·11	
Bulk tailing	92·75	7·00	0·17	17·39	12·76	

CONCLUSIONS

The results of the tests indicate that a cleaned silver-copper concentrate carrying 550 ounces of silver per ton and 18·00 per cent copper can be made. Attempts to float selectively the silver and copper met with no success.

This grade of concentrate is probably not sufficiently high to meet the high freight charges incidental to its shipment to a smelter.

B. Silver-Copper Flotation Concentrate

Shipment. A shipment of flotation concentrate, weight 80 pounds, was received on January 28, 1935, from the Eldorado Gold Mines, Limited, Port Hope, Ontario. The sample, which originated from the mill at Echo Bay, Great Bear Lake, N.W.T., was submitted by M. Pochon, Port Hope, Ontario.

Characteristics of Concentrate. The concentrate is a flotation product of the table tailing from the pitchblende concentrating tables. It carries the bulk of the base metal sulphides associated with the ore and also an appreciable amount of silver. It contains less than 1 per cent of uranium oxide.

Sampling and Assaying. The material was sampled and assayed as follows:

Silver	1,068·4 oz./ton
Copper	11·90 per cent
Iron	15·85 "
Uranium oxide (U_3O_8)	0·90 "
Sulphur	14·10 "
Insoluble	22·50 "

Screen Test:

Mesh	Weight, per cent
+ 48	0·3
+ 65	0·6
+100	15·9
+150	15·9
+200	19·3
−200	48·0
	100·0

Object of Experimental Tests. Experimental work is covered in two parts.

In Part I, tests were carried out with the object of recovering the silver by amalgamation. The results were not encouraging and poor recoveries of silver were made.

Part II deals with the extraction of silver by chemical methods and a process for its treatment is suggested.

Experimental Tests

Part I

AMALGAMATION

Test No. 1

A sample, 1,000 grammes, was barrel-amalgamated for 1 hour with 400 grammes of mercury and 500 c.c. of water.

The tailing assayed 900·0 ounces of silver per ton, indicating a silver recovery of 15·76 per cent.

The mercury separated from the tailing was clean. A large spongy mass, apparently a mixture of amalgam and silver, was also separated from the tailing.

Test No. 2

In this test, 4 pounds of soda ash per ton was added to the mixture of concentrate and mercury. The tailing assayed 816·2 ounces silver per ton, indicating a recovery of 23·61 per cent.

Test No. 3

In this test, 0·4 pound sodium cyanide was added to the concentrate pulp and mercury. The results indicated a very low recovery of only 5·44 per cent.

From the low recoveries obtained it was apparent that the larger amount of the silver was in a condition that prevented its amalgamation with the mercury.

Test No. 4

A sample, 1,000 grammes, of concentrate was barrel-amalgamated with 800 grammes of mercury and 6 pounds of soda ash per ton in a water pulp for 1 hour.

The tailing assayed 732·2 ounces silver per ton, indicating a recovery of silver of 31·47 per cent.

Test No. 5

In this test a few quartz pebbles were added to the jar containing the concentrate and mercury, the object being to keep the surface of the mercury layer broken. The results, however, showed a lower recovery than the previous test.

Test No. 6

In order to determine whether grease or oil was coating the fine silver grains, a sample was given a wash with a 4 per cent sodium hydroxide solution before amalgamation. A silver recovery of 33·63 per cent indicated little change from previous tests.

Test No. 7

In this test 10 pounds of ferric sulphate per ton was added to the amalgamation charge and agitation continued for two hours. Silver recovery was 33·32 per cent.

The results obtained from the amalgamation tests carried out would indicate that without regrinding it is not possible to amalgamate more than 33 or 34 per cent of the silver. This leads to the probability that the remaining 66 to 67 per cent of the silver is present as mineral silver, or combined metallic silver.

A microscopic examination of the amalgamation tailing indicated a considerable amount of metallic silver associated with the gangue material of the concentrate.

Test No. 8

The amalgamation tailing from Test No. 1 was floated, with the object of obtaining a higher-grade copper concentrate.

Reagents Added:

	Lb./ton
Soda ash	4·0
Potassium amyl xanthate	0·4
Pine oil	0·05

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Ag, oz./ton	Cu, per cent	Ag	Cu	
Feed	100·00	900·0	12·20	100·00	100·00	
Concentrate	54·31	1,306·5	20·20	79·51	89·62	1·84 : 1
Tailing	45·69	400·3	2·78	20·49	10·38	

Considering the silver still present in the feed, there does not appear to be any advantage gained in further concentration by flotation of this grade of concentrate. In the case of a concentrate carrying less silver, say 200 to 500 ounces, re-concentrating by flotation might be advantageous.

Test No. 9

A blanket test carried out on the concentrate showed practically no concentration of the silver.

Product	Weight, per cent	Assay, Ag, oz./ton	Distribution of silver, per cent	Ratio of concentration
Feed	100·00	1,068·4	100·00	
Concentrate	9·57	1,442·13	12·96	10·45 : 1
Tailing	90·43	1,025·0	87·04	

Test No. 10

In this test an amalgamation of the calcine from a salt roast of the concentrate was carried out. Brine solution and iron filings were used. In brief, the action of the test is as follows:

The soluble copper salts formed during the roast will deposit metallic copper on the iron filings. The silver dissolved by the salt solution will be precipitated in turn by the metallic copper as metallic silver, the silver thus being in a condition to amalgamate with the mercury.

The amalgamation was carried out in a mortar on the following charge:

Roasted concentrate	100	grammes
Mercury	100	"
Saturated brine solution	50	c.c.
Iron filings	10	grammes
Weight of amalgamation tailing	63·7	grammes
Silver in roasted concentrate	852	oz./ton
Silver in amalgamation tailing	681·52	"
Silver recovery	20·0	per cent

Test No. 11

In this test the concentrate was given a 15-minute grind prior to amalgamation, and in the amalgamation 900 grammes of mercury and 4 pounds of soda ash per ton were used.

Silver in feed	1,068·4	oz./ton
Silver in tailing	599·2	"
Recovery of silver by amalgamation	43·92	per cent

This indicates a better extraction than the previous tests.

A screen test showed the fineness of the material to be 89·0 per cent −200 mesh.

Test No. 12

A sample of concentrate, 500 grammes, was ground for 30 minutes and then barrel-amalgamated for 1 hour with 500 grammes of mercury and 10 grammes of salt (NaCl).

Silver in concentrate	1,068·4	oz./ton
Silver in amalgamation tailing	561·5	"
Recovery of silver	47·44	per cent

A screen test indicated that the reground concentrate was 99·9 per cent −200-mesh material. Finer grinding increased the recovery by a few per cent.

CONCLUSIONS FROM PART I

The recovery of the silver by amalgamation in the flotation concentrate under examination, is not satisfactory.

By regrinding to have practically all −200 mesh, the indicated recovery was only 47·44 per cent.

Whether the remaining silver was present as mineral silver or whether the metallic particles were, in some way, rendered inactive to the mercury, it is difficult to say.

A chemical treatment might offer a solution to the problem, and in Part II of this report tests of this nature are covered.

In a concentrate containing lower silver, say from 200 to 500 ounces per ton, it might be possible to re-float and further concentrate the silver in a copper-silver concentrate suitable for shipment to a smelter.

Part II

EXTRACTION OF SILVER FROM FLOTATION CONCENTRATE BY CHEMICAL METHODS

This report summarizes the test work conducted on chemical methods of treatment for recovery of silver from flotation concentrate. Although the present sample shipment carries a silver content of over 1,000 ounces per ton, the fact that other concentrates of much lower silver content are produced and are available for treatment has been considered in the experimental work conducted.

The experimental results indicate that a process involving chloridizing roasting followed by water leaching and hypo leaching is best suited to this type of material in securing maximum extraction of silver content.

Approximate composition of flotation concentrate:

	Per cent
Chalcopyrite	34·60
Pyrite	3·75
Siderite	7·35
Rhodocrosite	10·00
Calcite	7·40
Magnesite	7·50
Uranium oxide	0·90
Insoluble	22·50
Alumina	Not determined
Silver	3·66

From the relatively poor results obtained in the amalgamation tests (Part I) it is evident that much of the silver is in a form other than metallic.

Direct treatment by cyanide leaching is out of the question owing to the presence of high percentages of siderite and rhodocrosite and small amounts of soluble copper compounds which act as cyanicides.

Preferential chloridizing of the silver by means of hydrochloric acid gives unsatisfactory results. Chlorination by chlorine gas attacks the sulphides, causing a high consumption of chlorine, recovery of which is not considered economic.

It would seem, therefore, that the only wet process treatment applicable would be one involving roasting.

Two methods of roasting are available, namely, (1) chloridizing roasting and (2) sulphating roasting.

(1) Chloridizing Roasting

This comprises roasting the concentrate with the addition of 5 to 10 per cent of common salt. This type of roasting usually involves a loss of silver through volatilization but converts the silver and silver compounds to chloride which is readily leachable with a solution of hyposulphite of soda or calcium. The pregnant solution can then be treated with sodium or calcium sulphide to precipitate almost all the silver, excess of the precipitant being avoided and the hypo thus regenerated.

From the tests conducted a relatively low silver tailing is obtained. The results with outline of the various steps are given below.

Some provision for recovery of the volatilized silver by condensing or by precipitating would result in a very favourable overall recovery of silver.

It would appear that this process would be equally applicable to low-grade silver concentrate (200 ounces per ton) and to high-grade silver concentrate (1,000 ounces per ton, or over).

Provided the tonnage to be treated is not too great, the process might be carried out at the mine. The chemicals required are not expensive and amount to less than 300 pounds per ton of 1,000-ounce silver concentrate. The resultant saving in freight would appear sufficient to offset capital and operating expenses and leave a favourable margin, as against shipping the concentrate to a smelter.

The flow-sheet of the process may be outlined as follows:

Concentrate
|
Roast with salt
|
Leach with water, etc.,
remove soluble copper salt
|
Leach with hypo
|
Filter

Residue to waste

Filtrate—Na_2S
|
Ag_2S Cu_2S
to smelter
|
Hypo regenerated.

Test No. 1

Five hundred grammes of concentrate and 50 grammes of salt were roasted in an electric muffle furnace.

Starting temperature	450 °C.
After 1 hour	600 °C.
After 2 hours	630 °C.

Weight	563·0 grammes
Calculated silver content	948·4 oz./ton
Or calculated silver content, per cent	3·26 per cent

Assay:

Silver	Unreliable on account of volatility of AgCl
SO_2	19·55 per cent
Cl	19·80 "
S	Nil

Leaching:

One hundred grammes was leached with 300 c.c. water to remove excess salt and soluble copper salts.

Residue was leached with 20 grammes hypo in 300 c.c. water—agitated 3 hours. Filtered. Residue was leached with 5 grammes hypo in 300 c.c. water for 2 hours. Filtered.

Excess sulphide was added to the combined filtrates.

Sulphide precipitate	4·5	grammes
Silver in precipitate	3·11	"
Copper in precipitate	0·80	"
Silver recovered	95·6	per cent
Tailing weight	63·5	grammes
Assay	22·15	Ag, oz./ton
Extraction of silver from roast	98·5	per cent
Silver loss by volatilization	2·90	"
Silver recovered	95·6	per cent
Silver in tailing	1·5	"
Silver volatilized	2·9	"
	100·0	

Test No. 2

(Duplicate of Test No. 1)

Five hundred grammes of concentrate and 50 grammes of salt were roasted in an electric muffle furnace.

Starting temperature	420 °C.
Finishing temperature	680 °C.
Leached with water—filtered and dried.	
Calcine leached	395 grammes
Calculated silver content	1,352 oz./ton
Or calculated silver content, per cent	4·63 per cent

The determination of the silver content by assay is unreliable on account of volatility of the silver chloride, and chemical determination is likewise difficult.

Leaching:

Half of the calcine, 197·5 grammes, was leached with 25 grammes of hypo in 500 c.c. water for 3 hours and filtered.

The residue was re-leached with 10 grammes of hypo in 400 c.c. water for 2 hours and filtered. The residue was again leached with 10 grammes of hypo in 400 c.c. water and filtered.

Excess sodium sulphide was added to each filtrate.

1st filtrate gave a precipitate of	8·85	grammes
2nd " " "	2·12	"
3rd " " "	0·91	"
Total precipitate	11·88	"
Silver in precipitate	8·65	grammes
Copper "	0·852	"
Silver recovered	94·5	per cent
Tailing weight	173·0	grammes
Assay	23·62	Ag, oz./ton
Extraction of silver from roast	98·4	per cent
Silver loss by volatilization	3·9	"
Silver recovered	94·5	per cent
Silver in tailing	1·6	"
Silver volatilized	3·9	"
	100·0	

Remarks. Impurities in the hypo solution consist mainly of manganese, calcium, and copper. The copper is removed with the silver and a small amount of manganese is occluded, but the main portion of the manganese remains in the hypo solution. Just what effect this manganese would have in building up has not been determined. It seems reasonable, however, that 90 per cent of the hypo can be regenerated. The presence of copper in the hypo solution is considered to be beneficial as it attacks any sulphide silver present. The sulphide precipitate could be smelted in a small furnace instead of being shipped as precipitate.

Double leaching, that is, re-leaching the residue after the first leach, appears to be necessary with high-grade material. This assumption may be incorrect, as the leaching process has not been investigated in this respect.

In roasting, a temperature of 630° C. in presence of the salt is apparently sufficient to oxidize the sulphides completely.

(2) Sulphating Roast and Leaching

In oxidizing roasting in the presence of excess sulphur, the silver tends to form silver sulphate which is soluble in water and recoverable by suitable precipitation. The presence of manganese, however, is not favourable to high extraction, as, in roasting, some manganese combines with silver forming an insoluble silver compound.

Some such reaction apparently occurs in the present instance, as complete sulphatization is not obtained and a high silver tailing results, as the tests below show.

Roasting under special control conditions, such as control of SO_3 gas pressure to permit preferential decomposition of the sulphates other than silver, has some possibility, but no equipment is available for this method of roasting in this laboratory.

Various schemes were tried, such as elimination of manganese prior to roasting, chloridizing after roasting, and acid leaching, but the results were quite disappointing, the tailing running some 300 ounces silver per ton. Sulphatizing roasting apparently, therefore, is not applicable to this kind of material.

A brief description of the tests conducted is shown below.

Roasting:

Five hundred grammes of concentrate was roasted in an electric muffle furnace.

Starting temperature	480 °C.
After 1 hour	590 °C.
After 2 hours	670 °C.
After 2½ hours	720 °C.
Weight	509 grammes
Calculated silver in the calcine	1,049 oz./ton or 3·6 per cent

Water Leach:

(a) One hundred grammes of calcine was agitated with 600 c.c. hot water for 2 hours. Filtered and washed.

Silver recovered from solution	2·265 grammes
Extraction	63 per cent

(b) Residue was re-treated with 10 c.c. of sulphuric acid and 500 c.c. of water. Filtered and washed.

Silver recovered from solution	0·395 grammes
Extraction	11 per cent

(c) Residue in 200 c.c. water was treated with chlorine gas and left for 1 hour (warm), filtered and washed. Washed residue was leached with hypo solution.

Silver recovered	0·375	grammes
Extraction	10·4	per cent
Total extraction	84·3	"
Final tailing weight	62·0	grammes
Assay:		
Silver	0·91	per cent, or 266 oz./ton

These results indicate that only 63 per cent of the silver is sulphated and that about 20 per cent is combined with some element in the ore that is refractory to chemical treatment.

Acid Leach:

One hundred grammes of the roast was treated with 10 grammes of bleaching powder and 10 c.c. of sulphuric acid in 500 c.c. of water and then bottle-agitated overnight. After filtration and washing, the residue was leached with hypo solution. The residue weighed 76 grammes and assayed 329·6 ounces of silver per ton, 9·60 per cent of copper, 2·75 per cent of manganese, and no sulphur.

The recovery of silver was 76·2 per cent.

Leaching the roasted ore with 12·5 grammes of sulphuric acid and 2·5 grammes of nitric acid per 100 grammes, gave a tailing assay of 349·6 ounces of silver per ton and a recovery of 82 per cent of the silver.

Roasting with Addition of Bi-Sulphate:

Five hundred grammes of concentrate was roasted with 30 grammes of sodium bi-sulphate for 4 hours to a maximum temperature of 750° C.

A 100-gramme sample was leached with water acidulated with sulphuric acid. A tailing, assaying 288 ounces silver per ton and 1·50 per cent manganese, was obtained and 82 per cent of the silver was recovered.

A 100-gramme sample was leached first with water, then with hypo solution. The residue showed 238 ounces of silver per ton. The recovery of silver was 80·2 per cent.

Removal of Manganese prior to Roasting:

A 100-gramme sample was leached with 700 c.c. of 4 per cent sulphur dioxide solution overnight. The washed residue was dried and roasted. The calcine was leached with water.

A tailing carrying 356 ounces of silver per ton and 1·18 per cent of manganese was obtained.

The recovery of silver was 78·35 per cent.

A similar test using sulphuric acid for leaching prior to roasting resulted in a tailing of 324 ounces silver per ton and a recovery of 80 per cent of the silver.

CONCLUSIONS FROM PART II

The results obtained in this investigation would warrant the consideration of the Patera or Russel process, namely, chloridizing roasting followed by water and hypo leaching, for extraction of the silver.

Approximately 95 to 98 per cent of the silver can be recovered as high-grade precipitate. The process is equally applicable to lower-grade silver concentrate than the sample submitted.

Compared with shipment of the concentrate from mine to smelter, this process would appear to be economically more favourable, owing to the difference in freight costs involved.

Compared with a direct smelting method at the mine the process would still appear to have advantages, considering the high copper content of the concentrate and its effect on lowering the grade of bullion.

III

INVESTIGATIONS THE RESULTS OF WHICH ARE SYNOPSIZED

Gold Ore from the Dominion Mine, Lake Thomas Syndicate, Ltd., Waverley, Halifax County, Nova Scotia. Five drums of ore, weight 550 pounds, were received on December 27, 1935, containing quartz gangue and the metallic minerals arsenopyrite, pyrrhotite, chalcopyrite, pyrite, sphalerite, and galena. Arsenopyrite is the most abundant. No native gold was seen in the sections examined under the microscope. Assays gave gold 0·385 ounce per ton, silver 0·045 ounce per ton, and arsenic 0·29 per cent.

The ore is admirably suited for treatment with cyanide and at a grinding of 62 per cent through 200 mesh an extraction of 98·70 per cent of the gold was effected. Amalgamation accounted for a recovery of 89·6 per cent and in conjunction with table concentration and cyanidation gave a recovery of over 91 per cent of the gold.

Silver-lead-zinc Ore from Invermay Annex Mining Company, in Unsurveyed Territory on the Skagit River, about 24 miles East of Hope, B.C. One box containing 12 pounds of ore was received on December 11, 1935. The sample contained galena and sphalerite in a soft argillaceous gangue and assayed as follows: 0·01 ounce gold, 35·5 ounces silver, 3·04 per cent lead, and 4·30 per cent zinc.

By flotation, more than 90 per cent of the silver can be concentrated with the galena, with a 9 : 1 ratio of concentration.

Flotation Gold-pyrite Concentrate from Gillies Lake-Porcupine Gold Mines, Ltd., Porcupine, Tisdale Township, Ontario. A shipment of 110 pounds of concentrate was received on February 14, 1936.

Flotation practice at the mill was satisfactory, but cyanidation of the concentrate was not giving the highest extraction. The concentrate assayed, gold 3·77, and silver 2·44 ounces per ton. A screen test showed the concentrate to be 49·9 per cent −200 mesh, with 7·4 per cent +65 mesh.

Test work showed that finer grinding was required. Lead oxide was found to be advantageous, materially decreasing the consumption of cyanide; 98 per cent extraction was obtained by the above methods. Time of agitation also could be decreased without lowering extraction.

Gold Ore from Claim E 237, Little Turtle Lake, Fort Francis Mining District, Northern Ontario. On January 3, 1936, a shipment of 200 pounds of ore was received, which assayed 0·62 ounce gold per ton.

The gangue consisted of vein quartz and chloritic schist, and the metallic minerals were: pyrite, pyrrhotite, marcasite, limonite, chalcopyrite, arsenopyrite, galena, and native gold. The native gold is commonly isolated in the quartz but occasionally in contact with pyrite or pyrrhotite. The gold was found to be largely free-milling and amenable to cyanidation.

A recovery of 97 to 98 per cent was obtained when the coarse gold was amalgamated in the form of a hydraulic concentrate and the hydraulic tailing subjected to cyanidation. A grind of 80 per cent through 200 mesh was found to be the most economical.

Zinc Ore from Enterprise, County of Lennox and Addington, Ontario. Four bags of ore, weighing 540 pounds, were received on January 11, 1936. The ore contained large disseminated grains of sphalerite in a calcite gangue and assayed 11·65 per cent zinc.

By flotation, 97·5 per cent of the zinc can be recovered in a concentrate assaying 57 per cent zinc, with a ratio of concentration of 5·5 : 1.

Gold-bearing Gravel from "Winfield Placers", near Kelowna, British Columbia. A shipment of 209 pounds was received December 11, 1935. A recovery of 64 per cent of the gold can be made in a sluice equipped with undercurrents, but it is necessary to use some kind of puddling apparatus to disintegrate the cemented material. Flotation recovered 16 per cent of the total gold from the blanket tailing and is, therefore, not economically justified.

Neither the quartz pebbles nor schist carried much gold, although the +1-inch material assayed 0·023 ounce per ton, which must be due to gold contained in the cemented material, a fact emphasizing the need for careful puddling.

Thickener Underflow from Macassa Mines, Limited, Kirkland Lake, Ontario. On March 17, 1936, two cans of thickener underflow were received. The material was ground 90 per cent through 200 mesh and assayed 0·14 ounce gold per ton.

The object was to determine the conditions under which maximum extraction could be obtained.

This was obtained by 48 hours' agitation with high lime content in solution and red lead added at the rate of one pound per ton of solids.

Gold Ore from Sachigo River Exploration Co., Ltd., Sachigo River Area, in the Northwestern part of Patricia District, Ontario. On December 11, 1935, one bag of ore, weighing 30 pounds, was received. The ore contained pyrite, chalcopyrite, tetradymite, sphalerite, and native gold in a gangue of quartz and chloritic schist. Limonite and covellite were also present, due to surface weathering.

The sample assayed 0·425 ounce gold, 0·05 ounce silver, 0·43 per cent copper, 1·77 per cent iron, and 0·70 per cent sulphur.

By taking out the coarse gold and amalgamating it and cyaniding the residue in a high lime pulp, more than 90 per cent of the gold can be extracted.

Gold Ore from the Neswaba Gold Syndicate, Walls Township, Oba District, Northern Ontario. On January 30, 1936, a shipment of 2,000 pounds was received, which assayed 0·46 ounce gold per ton.

The gangue consisted of silicified chloritic schist and translucent to glassy vein quartz, and the metallic minerals were: pyrite, pyrrhotite, chalcopyrite, sphalerite, and native gold. The native gold occurs as large angular grains and smaller well rounded blebs in quartz. A small percentage is also associated with pyrrhotite.

The ore was found to be amenable to amalgamation and cyanidation. After grinding to 75 per cent through 200 mesh, 61 per cent of the gold was recovered by amalgamation from a trap concentrate, and over 36 per cent was recovered by cyanidation of the trap tailing, giving an overall recovery of 97 per cent of the gold.

Mill Products from the Wendigo Gold Mines, Ltd., Kenora, Ontario. Samples of jig concentrate, blanket concentrate, and amalgam barrel residues were received May 5, 1936, which assayed as follows:

Jig concentrate	27·06	Au, oz./ton
Blanket concentrate	149·37	" "
Amalgam barrel tailing	6·04	" "

Poor results were being obtained by barrel amalgamation at the mill owing to sickening and flouring of the mercury. Test work showed that the difficulty could be overcome by amalgamating the concentrates without regrinding in the amalgamation barrel, and returning the residues to the grinding circuit for further reduction. A recovery of 96 to 97 per cent was obtained with no excessive loss of mercury.

Gold Ore from the Salmon River Gold District, Halifax County, Nova Scotia. Three shipments of ore, comprising 559 pounds from Bluenose Ventures Syndicate, 511 pounds Lead 1 and 102 pounds Lead 2 from Crown Reserve Gold Mines, Ltd., were received March 4, 1936. The ore of each lot was similar in appearance, consisting of translucent quartz with stringers of green chloritic material and small amounts of arsenopyrite and pyrite. Some coarse gold was seen under the microscope in the arsenopyrite. The assays were 0·015, 0·115, and 0·22 ounce of gold per ton respectively. No tests were made on the low-grade sample.

When the ore was ground to −25·5 per cent through 200 mesh amalgamation recovered 88 per cent of the gold. Amalgamation followed by cyanidation gave a recovery of over 98 per cent of the gold.

Gold Ore from the Diana Gold Mines, Ltd., Rice Lake District, Manitoba. A shipment of ore weighing 150 pounds was received on March 16, 1936. It contained quartz and silicified chlorite schist and the metallic minerals pyrite, chalcopyrite, pyrrhotite, galena, sphalerite, and native gold. Pyrite is sparsely disseminated and native gold was found only in the quartz, commonly as round particles, comparatively finely divided. A little over 3 per cent of the gold is indicated as being associated with the sulphides.

The assay was 1·50 ounces of gold per ton and 0·25 ounce of silver.

Amalgamation gave a recovery of 91·67 per cent of the gold at moderately fine grinding.

Titaniferous Iron Sand, Fort William, Thunder Bay District, Ontario. A shipment, 40 pounds of titaniferous sand, was received on May 11, 1936, and analysed as follows: iron 38·9 per cent, TiO_2 13·3 per cent.

The purpose of the test was to determine: (1) whether a clean magnetite iron concentrate could be produced; (2) whether a concentrate containing over 40 per cent titanium oxide (TiO_2) could be made.

A magnetite concentrate low in titanium cannot be made which is acceptable for making pig iron.

The titanium oxide product obtained as a tailing from the reground concentrate ran only 32·9 per cent. Re-treatment might possibly raise the grade to 40 per cent TiO_2, but as only 7·7 tons of 32·9 per cent product is obtainable from each 100 tons of sand treated, the cost of production would be so high that the product could not be sold in competition with other titanium ores.

See Report No. 446 in the 1932 Report of Investigations in Ore Dressing and Metallurgy issued by the Mines Branch, which covers the results of tests conducted on a sample of similar sand received from J. S. Dobie, Port Arthur, Ontario.

Gold Ore from the Darwin Gold Mines, Ltd., Michipicoten District, Northern Ontario. On October 1, 1935, a shipment of 123 pounds was received which assayed 1·235 ounces gold per ton. It was required to show the effects of the addition of different amounts of lime on the extraction of the gold by cyanide solutions.

Amalgamation of the hydraulic and blanket concentrates recovered 83 per cent of the gold. The tailing was agitated in cyanide solutions containing from 5 to 20 pounds of lime per ton of tailing. Litharge was also added to the solutions containing the higher amounts of lime.

The results showed that the additions of larger amounts of lime did not improve the extraction of the gold in the ore.

Arsenical Gold Ore from Wisconsin Claims, Hennessey Mountain, Nelson Mining Division, B.C. A shipment of 52 pounds was received January 7, 1936, consisting of white quartz and a little grey carbonate, with pyrite, arsenopyrite, sphalerite, chalcopyrite, galena, chalcocite, and native gold. It assayed 0·23 ounce of gold per ton and 1·42 ounces of silver; 0·59 per cent of copper, 0·03 of lead, 2·40 of zinc, 27·03 of iron, 10·22 of arsenic, 24·08 of sulphur, and 0·16 of antimony.

By flotation about 90 per cent of the gold was recovered in a low-grade concentrate, and by roasting this and cyaniding from 65 to 70 per cent. A recovery of 50 per cent can be made by straight cyanidation. It would, therefore, be necessary to float the copper from the ore before cyanidation, the concentrate perhaps being marketable to a smelter.

Jig Concentrate from the Perron Gold Mines, Limited, Perron, Abitibi County, Quebec. A shipment of 20 pounds of jig concentrate was received on March 11, 1936. The feed assay was gold, 28·18 ounces per ton.

Because of the difficulty experienced at the mill in amalgamating the jig products, test work was requested to find a method whereby the maximum recovery of gold could be obtained by amalgamation.

Several of the tests in which the mercury was added directly to the ball mill showed low recovery with the mercury badly floured and scattered through the filtered pulp.

The best recovery was obtained by grinding the concentrate separately in a strongly alkaline pulp with lime, removing the balls, and adding 20 per cent by weight of mercury and sodium cyanide at the rate of approximately 1 pound KCN per ton. The recovery in this test was 98·89 per cent of the gold.

Gold-bearing Tungsten Ore from Indian Path Mines, Ltd., Indian Path, Lunenburg County, Nova Scotia. A shipment of 1,300 pounds of ore was received on December 11, 1935, consisting of white quartz and dark grey to black schistose material. Pyrite and arsenopyrite are locally fairly abundant. Coarse, irregular masses of scheelite are present. The assay was 0·145 ounce gold per ton, 0·60 per cent tungstic oxide, 0·07 per cent arsenic, and 1·88 per cent sulphur.

Amalgamation recovered about 66 per cent of the gold. Results of tests indicated a somewhat lengthy treatment necessary to obtain a satisfactory scheelite concentrate. Cleaning the table concentrate by flotation and roasting gave a product of the following grade:

Tungstic oxide (WO_3)	65·08 per cent
Arsenic	0·35 "
Sulphur	0·31 "

This indicates, roughly, a recovery of only 6·84 pounds of marketable tungsten oxide per ton of ore.

Gold Ore from Claim 2230, South Shore of Vermilion Lake, Vermilion Township, Kenora District, Ontario. A shipment of 760 pounds of gold ore was received on March 30, 1936.

Feed Assay:

Gold	0·68 oz./ton
Silver	0·19 "
Copper	0·06 per cent
Arsenic	0·74 "
Lead	None
Zinc	"

Concentration by jigs and blankets followed by amalgamation of the concentrates recovered 85 per cent of the gold. Straight cyanidation at −100 mesh extracted 98 per cent of the gold within 24 hours.

For a mill treating a small tonnage, single-stage grinding with a jig receiving the ball mill discharge is recommended; the classifier overflow to pass over blankets and the jig and blanket concentrates to be barrel-amalgamated.

Gold-silver Tailing from the Kamloops District, B.C. Two separate lots of tailing were received on May 18, 1936, weighing 1 pound each, and a further small shipment, weighing 10 pounds, was received on June 23, 1936. The average assay of the shipments ran 0·04 ounce gold per ton, 1·0 ounce silver per ton, 1·6 per cent zinc, 0·4 per cent lead, and 4·5 per cent sulphur.

Amalgamation did not recover any appreciable amounts of either gold or silver.

Flotation recovered 75 per cent of the gold, 85 per cent of the silver, 98 per cent of the lead, and 83 per cent of the zinc, in a bulk flotation concentrate having a ratio of concentration of 7·7 : 1.

Unless the prices of the different metals improve greatly, it is questionable whether an economic recovery can be obtained.

Gold Ore from Bayonne Consolidated Mines, Ltd., Bayonne, B.C. A shipment of 300 pounds of ore was received on May 4, 1936. The purpose was to check cyanidation results obtained on a previous shipment reported in Investigation No. 629, Investigations in Ore Dressing and Metallurgy, Report No. 763.

Assay:

Au	0·83 oz./ton
Ag	0·83 "

The test checked previous work. Coarse gold must be removed prior to agitation; by so doing the recovery is increased from 27 to 96 per cent.

For a small mill, single-stage grinding in cyanide solution with a jig between ball mill and classifier is recommended. Jig concentrate should be ground and amalgamated, amalgamation residue returned to the classifier; the classifier overflow thickened, the underflow repulped 1 : 1·5 and agitated; thickener overflow precipitated.

Sand from Consolidated Sand and Gravel, Limited, Toronto, Ontario. Four samples of sand were received on February 5 and 6, 1936, designated: No. 1, Durham, 135 pounds; No. 2, Victoria Park, 64 pounds; No. 3, Swansea, 61 pounds; and No. 4, Leaside, 62 pounds. Grinding tests were desired to determine how the other three samples compared with the Durham sample, as the company has a plant at Durham and is thinking of starting one at Toronto.

The Durham sample was 16·79 per cent acid insoluble and about 4 mesh, the Victoria Park sample was 78·17 per cent insoluble and about 10 mesh, the Swansea sample was 74·02 per cent insoluble and about 28 mesh with a great deal of fines, and the Leaside sample was 76·97 per cent insoluble and about 35 mesh with a fair amount of fines.

In the following table the samples are listed according to their grindability to different sizes. The higher the order of sample in a column the easier it is to grind to the mesh size at the head of the column.

150 Mesh	*200 Mesh*	*325 Mesh*
Swansea	Swansea	Durham
Leaside	Durham	Swansea
Durham	Leaside	Leaside
Victoria Park	Victoria Park	Victoria Park

Gypsum from Island Point, Boularderie Island, Cape Breton County, Nova Scotia. Two samples of gypsum rock, net weights 32 and 86 pounds, were received on April 29, 1936, from Boularderie island, Cape Breton county, Nova Scotia.

The crude rock was massive, compact, and creamy-white, mottled with light brown. The calcined product after set was white with a very slight ivory tint.

The analyses showed the gypsum content of the samples to be 98·15 and 97·19 per cent, and the anhydrite 0·38 and 1·31 per cent.

The samples were mixed, crushed, calcined, and tested for time of setting, and for tensile and compressive strengths. The tests showed that a very good plaster may be made, suitable for finishing and for the manufacture of all materials having a gypsum base.

Slate from Kingsbury and Ste. Hénédine, Quebec. Six bags of slate, weight 1,000 pounds, were received on June 8, 1936, from Pulverized Products, Ltd., Montreal, Quebec. Three were of grey slate from Kingsbury, Quebec, and three of red slate from Ste. Hénédine, Quebec.

Pulverized Products, Ltd. have a jaw crusher and a gyratory crusher capable of crushing 6 tons an hour to ½ inch. Crushing tests, using rolls and hammer mill, were desired to determine what results could be obtained in making roofing granules from the ½-inch product.

The best way to crush the grey and red slate from about ½ inch to make slate granules would be to use a set of rolls in closed circuit with an 8-mesh vibrating screen, the ½-inch material going first to the screen.

Two screens should be used so as to give the −8 mesh a better opportunity to get out of the crushing circuit. These screens should be two-surface so that the −26 mesh can be screened out of the granules at the same time.

The rolls should be operated with the faces just touching, for even if this gives less −8-mesh per hour it gives a slightly better recovery.

Metallographic Study of the Longitudinal Wires from Two Screens of a Foudrinier Machine. Parts of two screens, one of which had worn twice as long as the other, were sent in by a local paper company for an examination of the longitudinal wires to determine whether structural differences would account for the superior wearing properties of one. The longitudinal bronze wires were untarnished and the wear, as estimated by the area of the flats on the wire, was directly proportional to the time the screens had been in service. The mesh of the two screens was identical, but the diameter of the wires in the screen with the longer service was a half-thousandth of an inch greater than that of the wires in the other. The smaller wires would have a tensile strength approximately 10 per cent less than the larger. Under the microscope no fundamental difference was noted, and the premature failure of one screen is believed to be due to the fact that it was made of a finer wire, and probably to faulty alignment in the machine.

Investigation of the Material Used in the Main Shaft, Intermediate Gear Thrust Automotive Bearing. A sample part, which had failed prematurely in service, was sent in by a local automobile firm and was examined microscopically. The steel was found to be in a normalized condition. A heat treatment was carried out and this was followed by hardness tests and microscopic examination. Heat treatment would probably improve the serviceability of the article.

Investigation of Two Galvanized Iron Sheets from Tole Gaufree "Ideale" Enrg., St. Hyacinthe, Quebec. The weight of zinc per square foot of sheet surface was determined chemically. The uniformity of the coating thickness was checked by means of the Preece test. Both coatings were found to be uniform in thickness, but one sheet had 0·78 ounce of zinc per square foot of sheet surface while the other had only 0·48 ounce.

Testing of a Galvanized Sheeting, submitted by Tole Gaufree "Ideale" Enrg., St. Hyacinthe, Quebec. The weight of zinc on a square foot of sheet surface was found to be 0·89 ounce. The Preece test showed that this coating was uniform.

Probable Resistance to Corrosion of Two Types of Galvanized Sheeting, submitted by Wilfrid Simard, Grande Baie, Chicoutimi, Quebec. The amount of copper in the sheet material and the weight of the zinc coating was determined by chemical means. The uniformity of the zinc coating thickness was checked by means of the Preece test and was found to be satisfactory for both sheets. The steel in one of the sheets was of the corrosion resisting type and contained 0·24 per cent copper. The other steel contained 0·09 per cent copper, probably present only as an impurity. The low copper sheet was coated with 0·66 ounce of zinc per square foot of sheet surface. The copper-bearing sheet had 0·53 ounce on a similar area. It was considered that the sheet with the heavier coating would wear better in spite of the fact that its base material was less resistant to corrosion.

The Effect of Temperature on the Austenitic Grain Size of Four Tool Steels, submitted by the Canadian Atlas Steels, Limited, Welland, Ontario. The grain size was determined at 1400° F. and for every 50° F. increment until the coarsening temperature was reached, and their critical points were determined in a Rockwell dilatometer. Similar curves were obtained for all steels.

Austenitic grain size in hypoeutectoid steels is usually determined by carburizing the material, the excess carbide present in the outer hypereutectoid case revealing the grain. This method, however, is not practical below 1700° F. as satisfactory carburization will not take place. Grain boundary ferrite will reveal the grain size in medium carbon steels at all temperatures. The four steels examined were, however, eutectoid in composition and to determine grain size at temperatures below 1700° F. one-inch round samples were quenched in water, making the outer portion of the sample martensitic while the inner remained pearlitic. The intermediate areas consisted of martensite with fine pearlite (troostite) at the grain boundaries. This grain boundary material effectively revealed the austenitic grain size.

The depth of the outer martensitic zone, which is affected by the grain size of the steel, was measured and used to check the accuracy of the grain size determinations.

One steel was found to coarsen at 1650° F., one was mixed-grained and did not fully coarsen until 1750° F. The other two steels coarsened at 1850° F. and 1900° F. respectively. Above 1700° F. the carburizing method of determining the grain size was employed. At 1700° F. the results obtained by the quenching method were checked against those obtained by the carburizing method. The agreement was satisfactory.

Test Work on Steel Bars from Naval Service, Department of National Defence. Six $\frac{7}{8}$-inch hexagonal bars, two $\frac{1}{2}$-inch square bars, two standard three-notch Izod impact test pieces, and two tensile test pieces of 4-inch gauge length and $\frac{1}{2}$-inch diameter were sent in by the Department of National Defence. Tensile, bend, impact, and hardness tests were requested.

The Brinell hardness of the steel was found to be 143. The $\frac{7}{8}$-inch round bars machined from the $\frac{7}{8}$-inch hexagonal bars all bent satisfactorily through 180 degrees. The results obtained from the impact tests were as follows:

Specimen No.	Notch 1	Notch 2	Notch 3	Remarks
1..........	91·0	97·0	97·0	Received machined.
2..........	87·0	92·0	93·0	Received machined.
3..........	30·5	36·5	26·5	Machined from ½-inch square bar.
4..........	42·5	28·0	34·0	Machined from ½-inch square bar.

The results obtained from the tensile tests were as follows, the gauge length being 4 inches in all cases:—

Specimen No.	Ultimate strength, p.s.i.	Ultimate strength, long tons	Yield point, p.s.i.	Elongation, per cent	Reduction of area, per cent	Remarks
1..........	85,550	38·2	66,200	20	57	Received machined.
2..........	85,550	38·2	66,200	19	56·5	Received machined.
3..........	66,750	29·8	52,500	22	64	Machined from $\frac{7}{8}$-inch hexagonal bar, diameter 0·505 inch.
4..........	65,500	29·2	50,200	20	64	Machined from $\frac{7}{8}$-inch hexagonal bar, diameter 0·505 inch.
5..........	66,100	29·5	51,250	21	64	Machined from $\frac{7}{8}$-inch hexagonal bar, diameter 0·505 inch.

Neither the steel received in the machined condition nor the steel from which the test specimens were machined met the specifications, the former having too low an elongation value and the latter too low a tensile strength. The deviation from specification values was, however, quite small.

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Canada, mines branch reports.
774, ore dressing investigations, Jan-June, 1936, c. 4.

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