CANADA

DEPARTMENT OF MINES AND RESOURCES

MINES AND GEOLOGY BRANCH

BUREAU OF MINES

INVESTIGATIONS IN ORE DRESSING AND METALLURGY

(Testing and Research Laboratories)

July to December, 1938

PAGE

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INVESTIGATIONS IN ORE DRESSING AND METALLURGY, JULY TO DECEMBER, 1938

I

REVIEW OF INVESTIGATIONS

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During the period July to December, 1938, seventy-eight reports of investigations conducted in the laboratories were issued. Thirteen of these reports are printed herewith in Section II, and the remainder are listed by title only in Section III. A review and outline of the special investigations and research completed, in progress, or contemplated, is given in Section IV.

Of the total investigations, twenty-eight concerned ores in which the constituent of major value was gold; eight concerned particular mill problems; twelve concerned other base metal ores; thirteen investigations involved a special microscopic study; and seventeen dealt with steel or other metal products.

In the Research Section, much of the research planned had to be suspended because the staff was almost wholly occupied with problems pertaining to the regular investigations on ore shipments and mill problems.

In the Chemical Laboratories, 4,281 samples were handled, involving 10,868 determinations and 48 different mineral constituents. This is the largest amount of work that has ever gone through the Chemical Laboratories in such a period and reflects to some degree the additional work required in the investigations dealt with.

In the Mineragraphic Laboratory, thirty-eight reports of examinations were completed in connection with investigations of ore treatment, and thirteen special reports were submitted relative to mill problems and ore samples. The number of polished sections prepared and examined amounted to 366. The Hilger spectrograph was utilized to a larger extent, 92 spectra being made of minerals, evaporated solutions, etc., for detection of special and general constituents. Four suites of polished sections totalling 945 sections, selected from those for which there is no further use, were sent to universities for educational use. The universities so supplied were: University of Alberta, Edmonton, Alberta; Nova Scotia Technical College, Halifax, N.S.; Edinburgh University, Scotland; and Benares Hindu University, India.

The Metallurgical Laboratory issued seventeen reports of examination and tests on steel and alloy products, and conducted some special investigational and research work. The requests reaching this laboratory were mainly from steel and alloy producers, Government departments, and mining companies.

Through all sections of the Division requests from numerous sources were received for information and opinions on new processes for ore treatment, production and properties of alloys, plant operating problems, and for references to particular subjects.

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

Summary of Investigations:

Investigations reported	78
Gold-bearing ores	28
Mill problems	8
Molybdenum ores	3
Mercury ores	2
Silver-lead-tungsten ores	1
Silver-cobalt ores	1
Cobalt	2
Chromium	1
Pitchblende	1
Asbestos tailing	1
Microscopic examinations (special)	13
Steel and alloy products	17

Provincially, the above ores originated as follows: Ontario, 26; Quebec, 20; British Columbia, 5; Northwest Territories, 4; Manitoba and Nova Scotia, one each.

Mineragraphic Laboratory:

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

A. Investigations:	
Gold ores	31
Cobalt ores	2
Copper-gold	1
Mercury	1
Copper-zinc	1
Tungsten	1
Antimony	1
Special reports	13
Total	51
B. Spectrographic analyses	92
C. Polished sections prepared:	
For Mineragraphic Laboratory	290
For others	76
Total	366
D. Thin sections prepared:	
For Mineragraphic Laboratory	1

Chemical Laboratories:

During the period July 1 to December 31, 1938, 4,281 samples of ores, minerals, and metal products were analysed by the staff of the Chemical Laboratories and complete records issued thereon. This work included a total of 10,868 chemical and assay determinations, and approximately 48 different mineral constituents were involved.

The samples were made up from the following:

Metallic ore mill products	3,560
Field samples, (Bureau of Geology and Topography)	141
Industrial Minerals Division mill products	213
Pyrometallurgical Laboratory products	85
Fuel Testing Laboratory (coal ash)	14
Customs assays and analyses	268
Total samples	4,281
Total determinations	10,868
Total gold assays	4,032
Total silver assays	683

Staff:

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

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

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

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

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

New Ore Dressing Laboratory:

The new mill laboratory building was completed in October. Equipment from the old mill and some new equipment have been set up and the plant is ready for operating. These new facilities make for a decided improvement over former conditions.

II

INVESTIGATIONS THE RESULTS OF WHICH ARE RECORDED IN DETAIL

Ore Dressing and Metallurgical Investigation No. 748

ARSENICAL GOLD ORE FROM THE GOLD CUP MINING COMPANY, LIMITED, ROSSLAND, BRITISH COLUMBIA

Shipments. Eight sacks of ore, net weight 489 pounds, were received on January 20, 1938, from R. H. Lee Martin, Secretary-Treasurer, Gold Cup Mining Company, Limited, 165 Broadway, New York, N.Y. The shipment comprised four sacks, weighing 232 pounds, from the Mascot claim and four sacks, weighing 257 pounds, from the Georgia claim. A further shipment of 400 pounds net was received on May 25, 1938, consisting of six sacks of ore, of which three were from the Mascot claim and three from the Georgia claim.

The properties of the Gold Cup Mining Company, Limited, from which these present shipments were received are 1 to 1¼ miles from the city of Rossland, in British Columbia.

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

Shipment No. 1:

—	Georgia ore	Mascot ore
Gold, oz./ton	0·225	0·45
Silver, oz./ton	0·05	0·05
Copper, per cent	0·12	0·26
Arsenic, per cent	3·10	8·75
Sulphur, per cent	1·89	5·60

Composite Sample. A composite sample was formed by mixing equal quantities of each lot, and it assayed as follows:

Gold	0·345	oz./ton
Silver	0·05	"
Copper	0·18	per cent
Arsenic	6·00	"
Sulphur	5·60	"
Iron	17·50	"
Lead	Nil	
Zinc	Nil	

The research work was performed on the composite sample.

Shipment No. 2:

—	Georgia ore	Mascot ore
Gold, oz/ton	0·30	0·15
Silver, oz./ton	0·05	0·03
Copper, per cent	0·21	0·11
Arsenic, per cent	3·13	0·41
Sulphur, per cent	4·80	8·56
Pyrrhotite, per cent	7·06	19·63

Characteristics of the Ore:

Shipment No. 1. Twelve polished sections, six from each lot, were prepared and examined microscopically.

The ore from the *Mascot* claim is very heavily mineralized and consists largely of massive sulphides. Pyrrhotite and arsenopyrite predominate, both minerals occurring in massive form. The pyrrhotite is crossed by narrow irregular veinlets which appear to be due to replacement of the pyrrhotite; the material of these veinlets is largely pyrite, a small proportion of which is marcasite. Rare grains of chalcopyrite occur in the pyrrhotite, and a small quantity of "limonite" indicates that this sample has been subjected to surface alteration. The gangue, which forms only a small percentage of the sample, is fine-textured dark silicates or impure quartz with a very small quantity of disseminated carbonate.

The ore from the *Georgia* claim is somewhat less heavily mineralized than that from the Mascot claim. The gangue is grey, fine-textured impure quartz, with probably some silicates and a small quantity of carbonate (calcite). Arsenopyrite and pyrrhotite are the predominant sulphides. The former occurs as masses and disseminated grains, the latter as masses and irregular stringers in the gangue; they are commonly associated with one another. A small quantity of pyrite occurs as rounded irregular grains in pyrrhotite, and locally small masses and grains of chalcopyrite are also present in pyrrhotite. Chalcopyrite occurs to a much less extent as small irregular grains disseminated in the gangue.

One grain of native gold, approximately 60 microns (between 200 and 280 mesh) in size, was observed to occur along a fracture in arsenopyrite.

Shipment No. 2. Twelve polished sections, six from each lot, were prepared and examined microscopically in order to determine the character of the ore.

The sections of the *Georgia* claim are largely massive sulphides in which pyrrhotite and arsenopyrite predominate together with a lesser amount of chalcopyrite. These three minerals are very closely associated. Each contains inclusions of the others and all contain inclusions of gangue. Most of the pyrrhotite and chalcopyrite occur as granular masses, the arsenopyrite as coarse to fine crystals disseminated in gangue and in the other two sulphides. A considerable amount of pyrite is present largely as coarse crystals in gangue. It contains inclusions of gangue, pyrrhotite, and chalcopyrite, and in places is fractured and veined with gangue. The latter consists of impure quartz with a few small patches and stringers of carbonate.

A dozen grains of gold were visible in arsenopyrite and in gangue, 53·6 per cent in the former and 46·4 per cent in the latter. In size the largest grain is 48 microns (−280 mesh) and the smallest is 4 microns (−2300 mesh).

The sections of the *Mascot* claim were not so heavily mineralized as were those of the Georgia sample. The gangue is impure quartz with some rock-forming silicates and finely disseminated carbonate with local rust stains. The metallic minerals, in their order of abundance, are: pyrrhotite, pyrite, chalcopyrite, arsenopyrite, and "limonite". Pyrrhotite is by far the most prominent and occurs as small granular masses with numerous inclusions of gangue. Pyrite is present as coarse crystals and small masses in gangue. Much of the latter presents the corroded, banded appearance of a colloid mineral, and the former is fractured and veined with gangue, chalcopyrite, and pyrrhotite. Irregular grains of chalcopyrite also occur in gangue and in pyrrhotite as do small, occasional grains of arsenopyrite. A small quantity of "limonite" together with the appearance of the massive pyrite indicates that this ore has undergone some oxidization.

No gold was visible.

Experimental Tests

The research procedure consisted of concentration of the gold, arsenic, and copper by flotation, tables, or blankets. On the shipment received on May 25, 1938, a number of cyanidation tests were also made.

From ore similar to the shipment received on January 20, 1938, the mine operators wished to produce either a gold concentrate carrying one ounce of gold per ton and not over 2 per cent of arsenic, for shipment to Trail, British Columbia, or an arsenical gold-copper concentrate having a value of at least $50.00 per ton and carrying over 8 per cent of arsenic, for shipment to Tacoma.

On the shipment received on May 25, 1938, it was desired that cyanidation tests be made both before and after flotation concentration.

On the first shipment the work showed that owing to a large proportion of the gold being in the arsenopyrite it was not possible to produce a gold concentrate free from arsenic. On the other hand a bulk concentrate can be made assaying 1 to 1·25 ounces of gold per ton and 25 per cent of arsenic. Owing to the badly oxidized and weathered condition of the Mascot ore sample, the recovery by flotation was possibly lower than would be the case in mill practice. Tests Nos. 1 to 6 inclusive consisted of the work on the composite sample of the first shipment received on January 20, and Tests Nos. 7 to 15 inclusive were performed on the second shipment received on May 25.

Shipment No. 1

FLOTATION AND BLANKET CONCENTRATION

Test No. 1

The ore at −14 mesh was ground in a ball mill with 8 pounds of soda ash and 0·07 pound of Aerofloat No. 31 per ton. The pulp was floated with the addition of 0·2 pound of amyl xanthate and 0·07 pound of pine oil per ton. The flotation concentrate was cleaned and the flotation tailing passed over a corduroy blanket set at a slope of 2½ inches to the foot. The different products were assayed for gold, copper, and arsenic. A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65+100	0·2
−100+150	2·6
−150+200	15·8
−200	81·4
	100·0

Results:

Flotation:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	Cu	As	Au	Cu	As	
Feed	100·00	0·36*	0·19*	6·51*	100·0	100·0	100·0	
Flotation concentrate	28·28	1·01	0·57	19·31	77·9	83·0	83·8	3·5 : 1
Flotation middling	13·75	0·155	0·16	2·86	5·8	11·0	6·0	7·3 : 1
Flotation tailing	57·97	0·10	0·02	0·42	16·3	6·0	10·2	

Blanket Concentration of Flotation Tailing:

Feed	100·00	0·10	....	....	100·0	....	....	
Blanket concentrate	2·20	1·66	....	....	36·5	....	....	45·5 : 1
Final tailing	97·80	0·065	....	....	63·5	....	....	

* Calculated.

Summary:

	Au, per cent	As, per cent
Recovered in flotation concentrate	77·9	83·1
Recovered in blanket concentrate	5·9	....
Total recoveries	83·8	83·1

FLOTATION AND TABLE CONCENTRATION

Test No. 2

The amount of soda ash added to the grind was increased to 13 pounds per ton, raising the pH from 8·0 in Test No. 1 to 9·0. Conditions of the flotation were otherwise similar to Test No. 1, the abnormal amount of soda ash necessary being due to the oxidized and weathered condition of the ore. After the removal and cleaning of the flotation concentrate as in Test No. 1, the flotation tailing was passed over a Wilfley table and a table concentrate recovered. A screen analysis showed the grinding as follows:

Mesh	Weight, per cent
−100+150	0·3
−150+200	9·8
−200	89·9
	100·0

Results:

Flotation:

Product	Weight, per cent	Assay Oz./ton Au	Assay Per cent Cu	Assay Per cent As	Distribution, per cent Au	Distribution, per cent Cu	Distribution, per cent As	Ratio of concentration
Feed	100·00	0·33*	0·20*	6·17*	100·0	100·0	100·0	
Flotation concentrate	15·25	1·33	0·97	26·30	61·5	73·0	65·0	6·5 : 1
Flotation middling	20·80	0·395	0·17	9·10	24·9	17·5	30·7	4·8 : 1
Flotation tailing	63·95	0·07	0·03	0·42	13·6	9·5	4·3	

Table Concentration of Flotation Tailing:

Product	Weight, per cent	Assay Oz./ton Au	Assay Per cent Cu	Assay Per cent As	Distribution, per cent Au	Distribution, per cent Cu	Distribution, per cent As	Ratio of concentration
Feed	100·00	0·07	....	0·42	100·0	....	....	
Table concentrate	1·55	0·25	....	....	5·4	....	....	64·5 : 1
Table middling	17·80	0·10	....	....	25·4	....	....	5·6 : 1
Final tailing	80·65	0·06	....	0·32	69·2	....	....	

* Calculated.

Summary:

	Au, per cent	As, per cent
Recovered in flotation concentrate	61·5	65·0
Recovered in table concentrate	0·7	....
Totals	62·2	65·0

FLOTATION AND TABLE CONCENTRATION

Test No. 3

Barrett No. 4 oil, 0·17 pound per ton, was added in place of the Aerofloat No. 31. Conditions of flotation were otherwise similar to Test No. 2. The grinding was 89·9 per cent −200 mesh and the flotation tailing was passed over a Wilfley table. A portion of the table tailing was panned on a Haultain superpanner and a portion of the panner tailing was treated with aqua regia solution.

Results:

Flotation:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	Cu	As	Au	Cu	As	
Feed	100·00	0·335*	0·175*	6·13*	100·0	100·0	100·0	
Flotation concentrate	14·75	1·26	0·48	28·00	55·4	40·0	67·4	6·8 : 1
Flotation middling	17·32	0·55	0·22	10·85	28·4	21·6	30·6	5·8 : 1
Flotation tailing	67·93	0·08	0·10	0·18	16·2	38·4	2·0	

Table Concentration of the Flotation Tailing:

Feed	100·00	0·08	....	0·18	100·0	....	....	
Table concentrate	0·61	0·28	....	....	2·1	....	....	164 : 1
Table middling	11·00	0·15	....	....	20·6	....	....	9·1 : 1
Table tailing	88·39	0·07	....	0·16	77·3	....	....	

* Calculated.

Haultain Superpanning of Table Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	0·07	100·0
Panner concentrate	1·88	0·59	16·0
Tailing	98·12	0·06	84·0

A portion of the panner tailing was treated with aqua regia and this resulted in an aqua regia residue of 0·045 ounce of gold per ton.

Summary:

	Au, per cent	As, per cent
Recovered in flotation concentrate	55·4	67·4
Recovered in table concentrate	0·3	....
Totals	55·7	67·4

In the foregoing tests a large percentage of the gold and arsenic is contained in the flotation middlings and would eventually be recovered in mill practice.

In Tests Nos. 1, 2, and 3 an attempt was made to make a bulk concentrate of the gold and arsenic in the ore having a value of at least $50.00 per ton and an arsenical content of at least 8 per cent.

In the remaining tests a copper concentrate was made having an arsenical content as low as possible and at the same time having a high gold content.

FLOTATION

Test No. 4

The ore was ground with 9 pounds of lime per ton to pass 75·4 per cent −200 mesh. The copper was floated with the addition of 0·06 pound of butyl xanthate and 0·05 pound of cresylic acid per ton. A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 65+100	0·7
−100+150	6·7
−150+200	17·1
−200	75·5
	100·0

Results:

Product	Weight, per cent	Assay Oz./ton Au	Assay Per cent Cu	Assay Per cent As	Distribution, per cent Au	Distribution, per cent Cu	Distribution, per cent As	Ratio of concentration
Feed	100·00	0·325*	0·16*	6·02*	100·0	100·0	100·0	
Copper concentrate	2·80	4·32	4·60	15·30	37·2	81·6	7·1	35·7 : 1
Tailing	97·20	0·21	0·03	5·76	62·8	18·4	92·9	

* Calculated.

Test No. 5

The ore was ground with 10 pounds of lime per ton to pass 89·9 per cent −200 mesh. The pulp was floated with the addition of 0·02 pound of butyl xanthate per ton and 0·03 pound of pine oil per ton. The resulting copper concentrate was cleaned.

Results:

Product	Weight, per cent	Assay Oz./ton Au	Assay Per cent Cu	Assay Per cent As	Distribution, per cent Au	Distribution, per cent Cu	Distribution, per cent As	Ratio of concentration
Feed	100·00	0·32*	0·17*	5·85*	100·0	100·0	100·0	
Copper concentrate	1·14	7·74	10·33	5·20	27·7	69·5	1·0	88 : 1
Copper middling	0·89	2·16	1·39	4·22	6·0	7·3	0·6	112 : 1
Tailing	97·97	0·215	0·04	5·88	66·3	23·2	98·4	

* Calculated.

The pH of the pulp was 9·0.

Test No. 6

The ore was ground with 10·0 pounds of lime per ton to pass 89·9 per cent −200 mesh. The pulp was floated with 0·02 pound of butyl xanthate and 0·02 pound of pine oil per ton. A copper concentrate was removed, cleaned, and the copper middling product added to the main body of the pulp. The pulp was reactivated with 1·0 pound of copper sulphate ($CuSO_4$) per ton, and a pyrite concentrate removed after the addition of 0·1 pound of amyl xanthate and 0·05 pound of pine oil per ton was cleaned, giving a middling product. The flotation tailing was concentrated on a Haultain superpanning machine and a portion of the panner tailing was treated with aqua regia solution. The different products were assayed for gold, copper, and arsenic.

Results:

Flotation:

Product	Weight, per cent	Assay			Distribution, per cent			Ratio of concentration
		Oz./ton	Per cent					
		Au	Cu	As	Au	Cu	As	
Feed	100·00	0·35*	0·17*	6·01*	100·0	100·0	100·0	
Copper concentrate	1·30	9·50	9·38	4·36	35·1	70·0	0·9	77 : 1
Pyrite concentrate	16·10	0·84	0·28	29·64	38·4	25·9	79·4	6·2 : 1
Pyrite middling	10·36	0·38	Nil	8·71	11·2	....	15·0	9·7 : 1
Tailing	72·24	0·075	0·01	0·39	15·3	4·1	4·7	

* Calculated.

The panning of the flotation tailing gave a panner tailing assaying 0·07 ounce of gold per ton.

This product was subjected to an overnight extraction with aqua regia solution, the residue assaying 0·06 ounce of gold per ton.

SUMMARY AND CONCLUSIONS, SHIPMENT No. 1

The results of the research work on this ore sample show that it is possible to make a bulk flotation concentrate containing 80 to 85 per cent of the gold and 75 to 80 per cent of the arsenic and assaying 1 to 1·25 ounces of gold per ton and 25 per cent of arsenic. In the copper concentration a flotation concentrate was made containing 70 to 75 per cent of the copper, 35 to 40 per cent of the gold, and 1·0 per cent of the arsenic; this copper concentrate assaying 10·0 per cent of copper, 7 to 9 ounces of gold per ton, and 4·5 per cent of arsenic.

The sample was badly oxidized and weathered and on this account the amounts of reagents used in the flotation tests were largely in excess of normal. The use of large quantities of lime also tended to depress the gold-bearing sulphides.

The aqua regia tests showed, however, that part of the gold was locked up in the quartz gangue in submicroscopic form and was therefore not amenable to flotation.

It was not found possible to produce a copper concentrate containing the minimum amount of arsenical content, namely 2·00 per cent. It was also found that in the flotation tests the major portion of the gold was contained in the arsenopyrite, this result being observed also in the microscopic work.

The small amount of copper (0·18 per cent) in the ore also precludes its recovery on an economic basis.

This concludes the work performed on the composite shipment received on January 20, 1938, the remaining tests being made on the No. 2 shipment received on May 25, 1938.

Shipment No. 2

BARREL AMALGAMATION

Test No. 7

The ore at −14 mesh was ground to pass 65 per cent −200 mesh and the pulp was amalgamated with mercury in a jar mill. The amalgam tailing was assayed for gold.

Results:

Ore sample	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
Georgia	0·30	0·255	15·0
Mascot	0·15	0·085	43·4

On separation, the mercury floured and showed some sickening. It is apparent that amalgamation is not a suitable metallurgical treatment for this ore.

CYANIDATION

Test No. 8

The ores at −14 mesh were ground in cyanide solution of 1 pound of potassium cyanide per ton strength to pass 86·9 per cent −200 mesh for the Georgia ore and 88·1 per cent for the Mascot ore. The pulps were agitated for a 24-hour period.

Results:

Ore sample	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
Georgia	24	0·30	0·08	73·3	0·92	0·16	2·00	10·7
Mascot	24	0·15	0·04	73·3	0·80	0·14	3·88	10·7

Grinding in cyanide solution therefore results in a high tailing and a high reagent consumption.

AERATION AND CYANIDATION

Test No. 9

The ore at −14 mesh was ground in a ball mill to pass 86·9 per cent −200 mesh for the Georgia ore and 88·1 per cent for the Mascot ore. Five pounds of lime per ton of ore was added during the grind. The pulps were aerated, with 10 pounds of lime added, for a 16-hour period. Portions of the pulp were then agitated for 24 or 48 hours in cyanide solutions as noted at the end of the test.

A screen test showed the grinding as follows:

Mesh	Weight, per cent	
	Georgia ore	Mascot ore
−100+150	4·2	3·2
−150+200	8·9	8·7
−200	86·9	88·1
	100·0	100·0

Results of Cyanidation:

Test No.	Ore sample	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
			Feed	Tailing		KCN	CaO	KCN	CaO
9A	Mascot	24	0·15	0·035	76·7	0·7	0·25	2·30	6·5
B	"	24	0·15	0·015	90·0	1·4	0·25	2·33	6·5
C	"	24	0·15	0·01	93·3	0·6	0·25	2·13	6·5
D	"	24	0·15	0·01	93·3	0·5	0·20	2·08	6·5
E	Georgia	24	0·30	0·09	70·0	1·0	0·50	2·10	9·0
F	"	24	0·30	0·08	73·4	2·0	0·55	3·05	8·9
G	"	24	0·30	0·06	80·0	1·0	0·50	2·36	9·0
H	"	24	0·30	0·05	83·3	2·0	0·60	3·05	8·8
I	"	48	0·30	0·07	76·7	1·0	0·45	2·87	9·1
J	"	48	0·30	0·06	80·0	2·0	0·50	4·33	9·0
K	"	48	0·30	0·05	83·3	1·0	0·45	2·62	9·1
L	"	48	0·30	0·04	86·7	2·0	0·45	3·05	9·1

The following reagents were added during agitation:

Test No.	Ore sample	Reagents added, lb./ton			
		KCN	CaO	Litharge	Lead acetate
9A	Mascot	1·0	10·0		
B	"	2·0	10·0		
C	"	1·0	10·0	0·5	
D	"	2·0	10·0		0·5
E	Georgia	1·0	8·0		
F	"	2·0	8·0		
G	"	1·0	8·0		0·5
H	"	2·0	8·0		0·5
I	"	1·0	8·0		
J	"	2·0	8·0		
K	"	1·0	8·0		0·5
L	"	2·0	8·0		0·5

Test No. 9 shows the beneficial effects of aeration of the pulp prior to cyanidation and also that the addition of a lead salt during the agitation period results in a higher extraction of the gold. The consumption of cyanide is still high.

FLOTATION AND CYANIDATION

Test No. 10

The ore at −200 mesh was ground with 3 pounds of lime per ton to a similar fineness to Test No. 9. The pulp was floated with 0·03 pound of butyl xanthate and 0·03 pound of pine oil per ton and a concentrate was removed. The flotation tailing was aerated in a lime pulp for a 16-hour period and agitated in cyanide solution of 1 pound per ton strength for the Mascot ore and 2 pounds per ton for the Georgia ore; 0·05 pound of lead acetate per ton was added during agitation.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Flotation (Mascot ore):				
Feed	100·00	0·15	100·0	
Flotation concentrate	1·77	5·70	67·3	56·5 : 1
Flotation tailing	98·23	0·05	32·7	
Flotation (Georgia ore):				
Feed	100·00	0·30	100·0	
Flotation concentrate	2·82	5·20	48·5	35·5 : 1
Flotation tailing	97·18	0·16	51·5	

Cyanidation of Flotation Tailing:

Ore sample	Agitation, hours	Assay, Au, oz./ton		Extraction, of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
Mascot	24	0·05	0·01	80·0	0·84	0·65	0·47	4·7
Georgia	48	0·16	0·025	84·4	1·96	0·56	0·39	3·9

Summary:

	Per cent Mascot ore	Per cent Georgia ore
Gold recovered in flotation concentrate	67·3	48·5
Gold extracted from flotation tailing	26·1	43·5
Total recoveries	93·4	92·0

The flotation concentrate from the Mascot claim assayed 0·97 per cent of arsenic and 4·72 per cent of copper, whereas that from the Georgia claim assayed 2·87 per cent of arsenic and 5·51 per cent of copper.

A saving in the cyanide consumption therefore results from flotation prior to cyanidation.

Test No. 10 concluded the work on the Mascot sample.

FLOTATION AND MAGNETIC CONCENTRATION

Test No. 11—Georgia Ore

In an endeavour to determine the association of the gold with the different minerals contained in the ore, the following method was employed:

The ore was ground in a ball mill to pass 89·7 per cent −200 mesh, 3·5 pounds of soda ash and 0·07 pound Aerofloat No. 31 per ton being added during the grind. The pulp was transferred to a flotation machine and a bulk concentrate made with the addition of 0·15 pound of amyl xanthate, 0·10 pound of pine oil and 0·5 pound of copper sulphate per ton. This concentrate was passed through a wet magnetic separator and a pyrrhotite concentrate recovered. The remainder was reconditioned with 3·0 pounds of lime per ton and an arsenical concentrate recovered with the addition of 0·03 pound butyl xanthate and 0·05 pound pine oil per ton. The remaining pulp was reactivated with the addition of 1·0 pound of copper sulphate per ton and a pyrite concentrate was floated off with the addition of 0·05 pound of amyl xanthate and 0·05 pound of pine oil per ton.

Results:

Bulk Flotation:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Au, oz./ton	As, per cent	Au	As	
Feed	100·00	0·30	3·13	100·0	100·0	
Flotation concentrate	20·14	1·31	15·22	88·0	97·9	4·9 : 1
Tailing	79·86	0·045	0·08	12·0	2·1	

The magnetic separation and succeeding selective flotation concentration of the bulk concentrate resulted as follows:

Product	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	As, per cent	Au	As
Feed (bulk concentrate)	20·14	1·31	15·22	88·0	97·9
Pyrrhotite concentrate	6·57	1·12	6·86	22·9	13·5
Arsenical concentrate	7·54	2·30	33·61	53·8	76·2
Pyrite concentrate	1·85	1·22	10·86	6·9	6·1
Middling	4·18	0·35	1·77	4·4	2·1
Original tailing	79·86	0·045	0·08	12·0	2·1

The pyrrhotite concentrate assayed 55·15 per cent of $Fe_{11}S_{12}$.

The results, showing 53·8 per cent of the gold contained in the arsenical concentrate, check the microscopic determination fairly closely.

CYANIDATION AND FLOTATION

Tests Nos. 12 and 13—Georgia Ore

The ore at −14 mesh was ground to pass 97·3 per cent −200 mesh and aerated in a lime pulp for 16 hours. The pulp was agitated in cyanide solution of 2 pounds of potassium cyanide per ton strength for a 48-hour period. The cyanide residue was washed and reactivated with 1·0 pound of copper sulphate per ton and floated with 0·2 pound of amyl xanthate and 0·10 pound of pine oil per ton. The resulting flotation concentrate was reground in cyanide solution of 3 pounds per ton strength to pass 99 per cent −325 mesh and agitated for a 48-hour period. Test No. 13 was a duplication of Test No. 12, the flotation concentrate being roasted prior to regrinding and cyanidation. A screen test on the cyanide tailing showed the grinding as follows:

Mesh	Weight, per cent
−100+150	0·2
−150+200	2·3
−200	97·3
	99·8

Results:

Cyanidation:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
12	48	0·30	0·055	81·7	1·48	0·46	1·45	10·1
13	48	0·30	0·05	83·3	1·36	0·40	1·69	10·2

Flotation of Cyanide Tailing, Test No. 12:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·055	100·0	
Flotation concentrate	13·38	0·25	60·7	7·5 : 1
Tailing	86·62	0·025	39·3	

Flotation of Cyanide Tailing, Test No. 13:

Feed	100·00	0·05	100·0	
Flotation concentrate	16·14	0·18	58·0	6·2 : 1
Tailing	83·86	0·025	42·0	

The concentrate from Test No. 12 was reground and agitated and the concentrate from Test No. 13 was roasted prior to regrinding and agitation.

Results of Cyanidation:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
12......	48	0·25	0·08	68·0	3·0	0·25	7·80	17·5
13......	48	0·21	0·025	88·1	2·9	0·30	13·30	25·5

Summary of Tests Nos. 12 and 13:

	Per cent	
	Test No. 12	Test No. 13
Gold extracted by cyanidation of aerated pulp...........	81·7	83·3
Gold extracted by cyanidation of flotation concentrate...	7·5	8·3
Total recoveries................................	89·2	91·6

The above results show that roasting of the flotation concentrate, from the cyanide residue, gives an improved extraction but at the same time increases materially the amount of cyanide consumed. Owing to the large amount of sulphides in the ore the ratio of concentration was necessarily low, the flotation concentrate assaying 0·25 ounce of gold per ton.

FLOTATION AND CYANIDATION (CYCLE TESTS)

Test No. 14—Georgia Ore

In order to discover the amount of fouling in the cyanide solution during agitation, a sample of the ore was ground in a ball mill with 3 pounds of lime per ton to pass 95·0 per cent −200 mesh and a flotation concentrate was removed. The flotation tailing was aerated for 16 hours and portions of the pulp were agitated for a 24-hour period. After agitation, the same cyanide solution was used to agitate a fresh portion of the pulp. In all, three portions of pulp were used. The cyanide tailing was assayed for gold, and the final cyanide solution for reducing power, KCNS, and copper.

Results:

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed..	100·00	0·30	100·0	
Flotation concentrate.......................	2·12	6·40	45·7	47 : 1
Tailing.....................................	97·88	0·165	54·3	

Cyanidation of Flotation Tailing:

Cycle No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO	KCN	CaO
1.......	24	0·165	0·055	66·7	1·0	0·25	0·69	4·5
2.......	24	0·165	0·055	66·7	0·9	0·25	0·58	4·0
3.......	24	0·165	0·055	66·7	1·0	0·25	0·44	4·0

The final cyanide solution assayed:

Reducing power...........................	448 millilitres $\frac{N}{10}$ $KMnO_4$/litre
KCNS................................	0·44 grm./litre
Copper................................	0·021 "

Although there is a certain amount of fouling, as shown by the reducing power of the final solution, yet the final cyanide residue has a value as low as in the first cycle of cyanidation and the cyanide consumption remains at a comparatively low level.

SETTLING TEST

Test No. 15 A and B—Georgia Ore

The ore at −14 mesh was ground in cyanide solution of 1 pound of potassium cyanide per ton strength to pass 84·9 per cent −200 mesh. Five pounds of lime per ton of ore was added to the grind. The pulp was made up to a 1·5 : 1 ratio of dilution and transferred to a tall glass tube of 2-inch diameter and allowed to settle for one hour. Readings were taken at 5-minute intervals in decimals of feet. The test was repeated at a 2 : 1 dilution ratio.

Results:

	Test No. 15A	Test No. 15B
Ratio of solid to liquid..................................	1·5 : 1	2 : 1
Lime added per ton solid, in pounds.....................	5	5
Alkalinity of solution at end of test, in pounds per ton of solution..	0·32	0·20
Overflow solution..	Clear	Clear
Rate of settling, ft./hour................................	0·50	1·05

The pulp settles at a normal rate of speed.

SUMMARY AND CONCLUSIONS, SHIPMENT No. 2

As deduced from the investigative work on these two lots of ore, two methods of possible milling procedure take precedence.

By the first method cyanidation of the ore is followed by flotation of the cyanide tailing and regrinding and re-agitation of the flotation concentrate.

In the second method, flotation of a high-grade shipping concentrate is followed by cyanidation of the flotation tailing.

Owing to the large amount of pyrrhotite in the ore, aeration in a lime pulp should precede cyanidation in each case.

The first method offers the advantage of producing all the gold on the property and therefore involves no shipping charge. Cyanide consumption would be high, however, and the amount of sulphides in the ore necessitates a bulky low-grade flotation concentrate from the cyanide tailing. This concentrate would have to be reground and re-agitated and would add to the grinding and cyanide costs. The extraction by this method totalled 89·2 per cent of the gold in the Georgia shipment. When the flotation concentrate was roasted prior to regrinding and agitation the total extraction was raised to 91·6 per cent. On the Mascot shipment cyanidation of the ore when preceded by aeration in a lime pulp gave an extraction of 93·3 per cent of the gold and a cyanide residue of 0·01 ounce of gold per ton.

By the second method, 40 to 50 per cent of the gold was obtained in a flotation concentrate assaying about 5 ounces of gold per ton, 5 to 10 per cent of copper, and 2 to 5 per cent of arsenic. From 40 to 45 per cent of the gold remaining in the flotation tailing can be extracted by cyanidation. The total recovery of the gold was 92·0 per cent from the shipment of Georgia ore and 93·4 per cent from the Mascot shipment. Cyanide consumption was lowered to about 0·5 pound of potassium cyanide per ton. If the amount of arsenic in the ore be high, the marketing of the flotation concentrate may prove difficult.

Ore Dressing and Metallurgical Investigation No. 749

CINNABAR ORE FROM THE MANITOU MINING COMPANY, LIMITED, MUD CREEK PROPERTY, BRIDGE RIVER, BRITISH COLUMBIA

Shipment. A 300-pound shipment of cinnabar ore from the Manitou Mining Company, Limited, Mud Creek property in Bridge River, B.C., was received on May 5, 1938. The material was submitted by C. P. Riel, President, Manitou Mining Company, Limited, 1021 Hall Building, Vancouver, British Columbia.

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

In the polished sections impure quartz and calcite predominate in the *gangue*, but in hand specimens dark grey rock is the principal constituent.

In their order of abundance the *metallic minerals* in the ore are: pyrite, cinnabar, sphalerite, and arsenopyrite. Pyrite predominates as coarse to fine crystals and small, fine-grained masses in gangue. It is locally fractured and contains numerous veins and inclusions of gangue. Arsenopyrite is present as fine-grained, tenuous masses with pyrite in gangue. In only one section does cinnabar appear in appreciable quantity, as a fairly large granular mass in calcite containing numerous inclusions and veins of gangue. It occurs also as small masses and films throughout the gangue, and as fine, earthy streaks along irregular calcite stringers. In some hand specimens native mercury is seen as tiny globules. Occasional tiny grains of sphalerite are visible in the gangue.

Sampling and Assaying. After crushing, cutting, and grinding by the standard method, a sample was obtained, which assayed as follows:

Gold	Trace
Silver	Nil
Mercury	0·83 per cent
Zinc	0·13 "
Iron as pyrite	2·45 "
Total iron (acid soluble)	6·15 "
Sulphur	0·93 "
CaO	7·03 "
Insoluble	66·16 "
Arsenic	Trace
Lead	Trace

Results of the Test Work. The results of the Wilfley table concentration were not satisfactory. The mercury minerals are very fine and passed over the table with the tailing, resulting in high mercury loss.

Flotation tests showed that the ore is amenable to concentration. The tests were essentially to determine what reagents gave the best results. The following information was obtained:

(1) A flotation tailing assaying as low as 0·05 per cent mercury can be obtained when no depressant for pyrite is used. Some depressants increased the mercury in the tailing.

(2) A depressant for pyrite is necessary in order to obtain a concentrate low in iron. Corn starch showed a depressing effect on pyrite, the concentrate assaying 8·98 per cent iron, but the ratio of concentration was low, 9·0 : 1. Sodium cyanide and Bear Brand Depressant depressed the pyrite, the concentrates assaying as low as 10 per cent iron; but these reagents have a depressing effect on the mercury minerals, the tailing assaying over 0·07 per cent of mercury.

(3) A better ratio of concentration was obtained when a dispersing reagent was used.

(4) Excessive sliming lowered the ratio of concentration.

In Test No. 20 a bulk float was made, the tailing assaying 0·05 per cent of mercury. The bulk concentrate was treated by selective flotation using sodium cyanide and lime to depress the pyrite. The cleaner concentrate assayed 27·20 per cent of mercury, a ratio of concentration of 35·8 : 1. The iron in the cleaner concentrate was high, namely, 18·64 per cent. The recovery was 90·84 per cent, not including the mercury in the middling products and the cleaner tailing, which would probably increase it by about 2 per cent.

In Test No. 24, an attempt was made to obtain a high-grade concentrate by floating for a short period, followed by a middling float, which was returned to the feed of the succeeding flotation test. The concentrates of the first, second, third, and fourth runs assayed 19·30, 17·28, 16·66, and 15·40 per cent of mercury, respectively, indicating that returning the middling to the feed of the circuit lowers the grade of concentrate.

In Test No. 30, a cleaner concentrate was obtained assaying 27·30 per cent of mercury and 13·10 per cent of iron, a ratio of concentration of 43·7 : 1. The recovery was 81·10 per cent. Some of the mercury in the middling products and the cleaner tailing could be recovered, which would raise the recovery appreciably. The rougher tailing was high, assaying 0·075 per cent of mercury; this is due to sodium cyanide being used.

Experimental Tests

The test work consisted of hydraulic concentration and flotation.

HYDRAULIC CONCENTRATION

A representative sample of − 14-mesh material was ground in a ball mill to 63 per cent −200 mesh. The pulp was classified in the hydraulic classifier, giving three products, namely, overflow, sand, and native mercury concentrate.

The classifier overflow and the sand were concentrated by tabling. The results of the test were as follows:

Wilfley Table Concentration of Classifier Sands:

Product	Weight, per cent	Mercury, per cent		Ratio of concentration
		Assay	Distribution	
Feed (sand)	100·00	0·85	100·00	8·0 : 1
Table concentrate	12·51	3·90	57·54	
Table middling	6·76	0·55	4·39	
Table tailing	80·73	0·40	38·07	

Wilfley Table Concentration of Classifier Overflow:

Product	Weight, per cent	Mercury, per cent		Ratio of concentration
		Assay	Distribution	
Feed (overflow)	100·00	0·55	100·00	26·7 : 1
Table concentrate	3·74	1·50	10·24	
Table middling	1·24	1·35	3·05	
Table tailing	95·02	0·50	86·71	

The results were not satisfactory. The mercury minerals are very fine and passed over the Wilfley table with the gangue minerals, resulting in high mercury in the tailing.

CONCENTRATION BY FLOTATION

A number of flotation tests were conducted to determine the most suitable reagents for flotation concentration.

Test Series No. 1

Representative samples of ore (2,000 grammes −14-mesh) were ground in the ball mill for a period of 10 minutes and one of 15 minutes. The reagents used and the results obtained are tabulated in Table I.

Screen Tests on the Flotation Tailings:

Mesh	Weight, per cent	
	10-minute grind	15-minute grind
+ 48	0·6	....
+ 65	4·5	0·5
+100	12·7	5·0
+150	19·6	17·3
+200	13·0	14·3
−200	49·6	62·9
	100·0	100·0

TABLE I

Test No.	Period of grinding, minutes	Reagents, lb./ton of ore: Ball mill: Reagent	Ball mill: Amount	Flotation cell: Concentrate float: Reagent	Concentrate float: Amount	Middling float: Reagent	Middling float: Amount	Floating period, minutes: Concentrate	Floating period, minutes: Middling	Product	Weight, per cent	Mercury, per cent: Assay	Mercury, per cent: Distribution	Ratio of concentration	Remarks
2	10	Aerofloat No. 25....	0·20	Aerofloat No. 25.... Copper sulphate..... Cresylic acid...... Pine oil.....	0·10 0·25 0·30 0·09					Feed........ Concentrate Tailing.....	100·0 9·1 90·9	0·80 8·51 0·025	100·0 97·2 2·8	11·0 : 1	
3	10	Soda ash.... Aerofloat No. 243...	0·50 0·50	Copper sulphate..... Coal-tar creosote.. Pine oil.....	0·50 0·40 0·18					Feed........ Concentrate Tailing.....	100·0 9·5 90·5	0·81 7·80 0·076	100·0 91·5 8·5	10·5 : 1	
4	10	Soda ash... Aerofloat No. 243... Copper sulphate.....	0·50 0·50 0·50	Coal-tar creosote.. Pine oil.....	0·40 0·18	Copper sulphate..... Aerofloat No. 243...	0·05 0·10	8	7	Feed........ Concentrate Middling... Tailing.....	100·0 9·6 1·0 89·4	0·79 7·60 1·37 0·05	100·0 92·6 1·7 5·7	10·4 : 1	
5	15	Soda ash... Aerofloat No.243.... Copper sulphate.....	0·50 0·50 0·50	Coal-tar creosote.. Pine oil.....	0·40 0·18			8		Feed........ Concentrate Tailing.....	100·0 6·5 93·5	0·82 11·15 0·10	100·0 88·6 11·4	15·4 : 1	
7	10	Lime.......	2·0	Copper sulphate..... KAmX*.... Cresylic acid...... Pine oil.....	0·10 0·10 0·20 0·20 0·18			12		Feed........ Concentrate Tailing.....	100·0 6·2 93·8	0·77 12·00 0·025	100·0 96·9 3·1	16·1 : 1	Concentrate assayed 17·02 per cent iron.
9	10	Soda ash...	0·5	Copper sulphate..... KAmX*.... Cresylic acid...... Pine oil.....	0·10 0·10 0·20 0·18			12		Feed........ Concentrate Tailing.....	100·0 4·9 95·1	0·71 13·30 0·062	100·0 91·7 8·3	20·4 : 1	Concentrate assayed 15·24 per cent iron.
10	10	Lime.......	2·0	Aerofloat No. 25.... Cresylic acid......	0·20 0·30			12		Feed........ Concentrate Tailing.....	100·00 6·82 93·13	0·80 11·00 0·055	100·0 93·6 6·4	14·7 : 1	Concentrate assayed 13·86 per cent iron.

* KAmX = Potassium amyl xanthate.

Test Series No. 2

All the native mercury was not recovered by flotation in Series No. 1. In these tests, therefore, the samples were deslimed by decantation to decrease sliming. The deslimed sand (about 80 per cent of the feed) was ground in the ball mill for periods of 10 and 15 minutes. Native mercury in the sand was recovered by hydraulic classification. The sand and the slime were combined for flotation. The reagents and the results are tabulated in Table II, page 26.

Screen Tests on the Flotation Tailings:

Mesh	Weight, per cent	
	10-minute grind	15-minute grind
+ 65	1·7	....
+100	8·1	1·7
+150	16·8	10·3
+200	14·3	14·0
−200	59·1	74·0
	100·0	100·0

Summary of Test Series Nos. 1 and 2

The tests indicate that a flotation tailing assaying as low as 0·05 per cent mercury can be obtained when no depressant for pyrite is used, the concentrate assaying over 15 per cent of iron. The depressants increased the mercury in the tailing.

Various depressants for pyrite have been tried. Sodium cyanide and the Bear Brand Depressant showed a marked lowering of the iron in the concentrate, but the mercury in the tailing increased to above 0·07 per cent. Corn starch showed a depressing effect on the pyrite but the ratio of concentration was low, 9·0 : 1.

Various collectors were tried. Aerofloat No. 25 and Sodium Aerofloat gave encouraging results; neither floats iron sulphides as readily as do xanthates. Sliming lowered the ratio of concentration. Dispersing reagents increased the ratio of concentration.

Test No. 20

An attempt was made to make a bulk float, which would give low mercury in the tailing, and a differential flotation of the bulk concentrate.

Representative samples were deslimed by decantation, and the sand was ground in the ball mill for 10 minutes. Native mercury in the sand was recovered by hydraulic classification. The slime and the sand were combined for flotation.

Reagents to Flotation Cell:

Conditioning:

	Lb./ton
Soda ash	0·5

Conditioning period, 10 minutes

Bulk Float:

Copper sulphate	0·10
Potassium ethyl xanthate	0·10
Cresylic acid	0·50

Floating period, 12 minutes.

Results:

Product	Weight, per cent	Mercury, per cent		Fe, per cent	Insoluble, per cent	Ratio of concentration
		Assay	Distribution			
Feed	100·00	0·86	100·00			
Native Hg concentrate	0·05	29·41	1·71			
Bulk concentrate	4·56	17·31	92·76	16·78	31·17	21·9 : 1
Bulk tailing	95·39	0·05	5·53			

Cleaning Test:

The bulk concentrate was refloated at 4·4 per cent solids.

Reagents to Flotation Cell:

Conditioning:

	Lb./ton
Lime	5·5
Sodium cyanide	0·28

Conditioning period, 10 minutes

Concentrate Float:

Cresylic acid	0·55

Floating period, 6 minutes

Middling Float:

Copper sulphate	0·28
Potassium amyl xanthate	0·28
Cresylic acid	0·55

Floating period, 5 minutes

NOTE.—The amounts of the reagents are high, owing to high pulp dilution. In mill practice, where flotation pulp density is high, the consumptions of reagents per ton of pulp would be much lower.

Results:

Product	Weight, per cent	Mercury, per cent		Fe, per cent	Insoluble, per cent
		Assay	Distribution		
Bulk concentrate	100·00	17·31	100·00	16·78	31·17
Cleaner concentrate	61·15	27·20	96·09	18·64	18·10
Cleaner middling	12·04	3·40	2·36	20·46	39·36
Cleaner tailing	26·81	1·00	1·55	10·90	57·30

A cleaner concentrate was obtained assaying 27·20 per cent mercury, and a ratio of concentration of 35·8 : 1, but the iron was 18·64 per cent.

	Per cent
Recovery of native mercury	1·71
Recovery by flotation (cleaner concentrate)	89·13
Total	90·84

Some of the mercury in the middling and the cleaner tailing would be recovered in mill practice. This would raise the total recovery.

TABLE II

Test No.	Period of grinding, min.	Reagents, lb./ton of ore: Ball mill or conditioning in cell — Reagent	Ball mill or conditioning in cell — Amount	Flotation cell: Concentrate float — Reagent	Concentrate float — Amount	Middling float — Reagent	Middling float — Amount	Floating period, minutes: Concentrate	Floating period, minutes: Middling	Product	Weight, per cent	Mercury, per cent: Assay	Mercury, per cent: Distribution	Iron, per cent	Ratio of concentration
11	15	*Ball Mill* Lime........	2·5	Aerofloat No. 25..........	0·20			12		Feed..........	100·00	0·83	100·0	15·46	14·6 : 1
		Conditioning Lime.........	2·0	Cresylic acid.	0·30					Native Hg concentrate.	0·02	45·94	1·1		
										Flotation concentrate.....	6·94	11·20	93·3		
										Flotation tailing..........	93·04	0·05	5·6		
12	10	*Ball Mill* Lime.........	2·5	Aerofloat No. 25..........	0·20			12		Feed..........	100·00	0·79	100·0	14·25	17·1 : 1
		Conditioning Lime.........	2·0	Cresylic acid.	0·30					Native Hg concentrate.	0·03	38·30	1·5		
										Flotation concentrate.....	5·85	12·20	90·2		
										Flotation tailing..........	94·12	0·07	8·3		
14	10	*Ball Mill* Lime.........	1·2	Aerofloat No. 25..........	0·20			12		Feed..........	100·00	0·77	100·0	7·36	28·8 : 1
		Conditioning Lime.........	4·0	Cresylic acid.	0·30					Native Hg concentrate.	0·08	14·74	1·5		
		Sodium cyanide.........	0·20							Flotation contrate.......	3·47	16·40	73·6		
										Flotation tailing..........	96·45	0·20	24·9		
15	10	*Ball Mill* Lime.........	1·2	Aerofloat No. 25..........	0·20			12		Feed..........	100·00	0·81	100·0	8·14	21·0 : 1
		Conditioning Lime.........	4·0	Cresylic acid.	0·30					Native Hg concentrate.	0·04	32·72	1·6		
		Sodium cyanide.........	0·10							Flotation concentrate.....	4·76	13·40	78·5		
										Flotation tailing..........	95·20	0·17	19·9		
16	10	*Conditioning* Sodium silicate........	3·0	Copper sulphate.......	0·10			12		Feed..........	100·00	0·86	100·0	12·80	18·6 : 1
		Sodium cyanide.........	0·10	KEtX*.......	0·10					Native Hg concentrate.	0·07	25·82	2·1		
				Pine oil.......	0·27					Flotation concentrate.....	5·36	14·50	90·2		
										Flotation tailing..........	94·57	0·07	7·7		

17	10	*Conditioning* Sodium hydroxide..... Sodium cyanide.........	 3·0 0·10	Copper sulphate....... KEtX*....... Pine oil.......	0·10 0·10 0·27			12		Feed.......... Native Hg concentrate. Flotation concentrate..... Flotation tailing..........	100·00 0·07 4·74 95·19	0·80 26·15 15·00 0·07	100·0 2·3 89·4 8·3	10·60	21·1 : 1
18	10	*Conditioning* Lime......... Corn starch... Sodium hydroxide.....	 4·0 0·50 0·20	Copper sulphate....... Sodium Aerofloat........ Pine oil.......	0·05 0·10 0·27			5	7	Feed.......... Native Hg concentrate. Flotation concentrate..... Flotation middling........ Flotation tailing..........	100·00 0·03 11·10 1·18 87·69	0·80 30·42 6·10 3·55 0·07	100·0 1·1 85·8 5·3 7·8	9·87	9·0 : 1
21	10	*Conditioning* Sodium silicate........ Sodium cyanide......... Aerofloat No. 25..........	 3·0 0·07 0·10	Copper sulphate....... Cresylic acid.	0·10 0·30			12		Feed.......... Native Hg concentrate. Flotation concentrate..... Flotation tailing..........	100·00 0·06 3·22 96·72	0·82 24·36 22·00 0·10	100·00 1·78 86·42 11·80	10·66	31·1 : 1
22	10	*Conditioning* Sodium silicate........ Sodium cyanide.........	 3·0 0·07	Copper sulphate....... Sodium Aerofloat........ Cresylic acid.	0·10 0·10 0·40			12		Feed.......... Native Hg concentrate. Flotation concentrate..... Flotation tailing..........	100·00 0·05 2·86 97·09	0·85 22·75 26·04 0·10	100·00 1·34 87·28 11·38	11·50	33·9 : 1
23	15	*Conditioning* Sodium silicate........ Sodium cyanide.........	 3·0 0·07	Copper sulphate....... Sodium Aerofloat........ Cresylic acid.	0·10 0·10 0·40			12		Feed.......... Native Hg concentrate. Flotation concentrate..... Flotation tailing..........	100·00 0·04 3·0 96·96	0·76 23·62 25·64 0·07	100·00 1·11 90·86 8·03	7·40	33·3 : 1
25	10	*Conditioning* Sodium silicate........ Bear Brand Depressant. Aerofloat No. 25..........	 1·5 0·25 0·10	Cresylic acid.	0·30			4	7	Feed.......... Native Hg concentrate. Flotation concentrate..... Flotation middling........ Flotation tailing..........	100·00 0·03 3·71 1·00 95·26	0·77 30·44 18·20 1·92 0·07	100·00 1·18 87·67 2·49 8·66	9·60	26·9 : 1

* KEtX = Potassium ethyl xanthate.

TABLE II—Concluded

Test No.	Period of grinding, min.	Reagents, lb./ton of ore: Ball mill or conditioning in cell: Reagent	Amount	Flotation cell: Concentrate float: Reagent	Amount	Flotation cell: Middling float: Reagent	Amount	Floating period, minutes: Concentrate	Middling	Product	Weight, per cent	Mercury, per cent: Assay	Distribution	Iron, per cent	Ratio of concentration
26	10	*Conditioning*		Cresylic acid.	0·30			4	7	Feed..........	100·00	0·76	100·00		
		Sodium silicate........	1·5							Native Hg concentrate.	0·03	39·16	1·59		
		Aerofloat No. 25..........	0·05							Flotation concentrate.....	3·71	18·00	87·54	11·81	26·9 : 1
										Flotation middling........	0·84	1·92	2·11		
										Flotation tailing..........	95·42	0·07	8·76		
28	10	*Conditioning*		Sodium Aerofloat........	0·06	Sodium Aerofloat	0·04	4	7	Feed..........	100·00	0·83	100·00		
		Sodium silicate........	1·5	Cresylic acid.	0·30					Native Hg concentrate.	0·03	30·23	1·09		
		Potassium bichromate...	0·5	Pine oil.......	0·09					Flotation concentrate.....	3·72	20·50	91·76	18·00	26·9 : 1
										Flotation middling........	0·63	1·84	1·40		
										Flotation tailing..........	95·62	0·05	5·75		
29	10	*Conditioning*		Copper sulphate.......	0·10	Sodium Aerofloat	0·07	4	7	Feed..........	100·00	0·80	100·00		
		Sodium silicate........	1·5	Sodium Aerofloat........	0·08					Native Hg concentrate.	0·04	31·36	1·56		
		Sodium cyanide.........	0·07	Cresylic acid.	0·30					Flotation concentrate.....	2·40	26·80	80·32	12·30	41·7 : 1
				Pine oil.......	0·09					Flotation middling........	1·16	4·20	6·08	11·00	
										Flotation tailing..........	96·40	0·10	12·04		

* KEtX = Potassium ethyl xanthate.

Test No. 24

An attempt was made to obtain a high-grade concentrate by floating for a short period of time followed by a middling float, which was returned to the feed of the succeeding flotation test.

Representative samples of ore were deslimed and the sand was ground for 10 minutes. The native mercury was recovered by hydraulic classification. The sand and the slime were combined for flotation.

Reagents to Flotation Cell:

Conditioning:

	Lb./ton
Sodium silicate	1·5
Aerofloat No. 25	0·10
Conditioning period, 10 minutes	

Concentrate Float:

Cresylic acid	0·30
Floating period, 4 minutes	

Middling Float:

Aerofloat No. 25	0·05
Floating period, 7 minutes	
pH of tailing solution	8·9

Results:

	Product	Assay, per cent		Ratio of concentration
		Mercury	Iron	
	Feed	0·81		
1st run	Flotation concentrate	19·30	13·00	24·6 : 1
	Flotation tailing	0·025		
2nd run	Flotation concentrate	17·28	8·80	22·7 : 1
	Flotation tailing	0·05		
3rd run	Flotation concentrate	16·66	....	21·9 : 1
	Flotation tailing	0·05		
4th run	Flotation concentrate	15·40	9·60	20·2 : 1
	Flotation middling	1·92	8·20	
	Flotation tailing	0·05		

Returning the middling to the feed of the circuit lowered the grade of concentrate and the ratio of concentration.

Test No. 30

Twelve rougher flotation runs were made to obtain sufficient amounts of concentrate for subsequent cleaning tests.

The deslimed sand was ground for 10 minutes and the native mercury recovered by hydraulic classification. The slime and sand were combined for flotation.

Rougher Flotation:

Reagents to Flotation Cell:

Conditioning:

	Lb./ton
Sodium silicate	1·5
Sodium cyanide	0·07
Conditioning period, 10 minutes	

Concentrate Float:

Copper sulphate	0·10
Sodium Aerofloat	0·08
Cresylic acid	0·30
Pine oil	0·14
Floating period, 4 minutes	

Middling Float:

Sodium Aerofloat	0·07
Floating period, 7 minutes	
pH of the tailing solution	8·4

Results:

Product	Weight, per cent	Mercury, per cent		Iron, per cent	Insoluble, per cent	Ratio of concentration
		Assay	Distribution			
Feed	100·00	0·78	100·00	....	....	....
Native Hg concentrate	0·03	24·44	0·93	....	....	....
Rougher concentrate	3·09	21·32	84·52	11·83	34·43	32·4 : 1
Rougher middling	0·91	4·55	5·31	9·86	51·08	....
Rougher tailing	95·97	0·075	9·24	....	....	....

Cleaning Test:

The rougher concentrate was refloated at 8·5 per cent solids.

Reagents to Flotation Cell:

Cleaner Concentrate: No additional reagents.

Floating period, 7 minutes

Cleaner Middling:

	Lb./ton
Cresylic acid	0·27
Floating period, 7 minutes	

Results:

Product	Weight, per cent	Mercury, per cent		Iron, per cent	Insoluble, per cent
		Assay	Distribution		
Rougher concentrate	100·00	21·32	100·00	11·83	34·43
Cleaner concentrate	74·18	27·30	94·98	13·10	27·52
Cleaner middling	6·01	10·70	3·02	11·30	47·44
Cleaner tailing	19·81	2·15	2·00	7·26	56·40

The cleaner concentrate assayed 27·30 per cent of mercury, a ratio of concentration of 43·7 : 1, and a recovery of 80·28 per cent. Recovery of the mercury in the middling products and the cleaner tailing would increase the overall recovery.

Ore Dressing and Metallurgical Investigation No. 750

GOLD-SILVER-LEAD ORE FROM THE CONSOLIDATED NICOLA GOLDFIELDS, LIMITED, STUMP LAKE, NEAR KAMLOOPS, BRITISH COLUMBIA

Shipment. A sample of ore, weight 103 pounds, was received on June 4, 1938, from the Consolidated Nicola Goldfields, Limited, Stump Lake, British Columbia. The shipment was made by John F. Coats, Mining Engineer, Consolidated Nicola Goldfields, Limited, 406-407 Bank of Nova Scotia Building, Vancouver, B.C.

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

The *gangue* consists of milky-white quartz with some finely disseminated carbonate. The latter mineral also fills a few fine, sinuous fissures in the quartz.

Pyrite and *galena* predominate; *sphalerite* is also abundant with a much less amount of chalcopyrite. Pyrite is present as irregular grains and small masses in gangue. In places it is fractured and veined with gangue and galena, and contains numerous inclusions of the other sulphides. Galena occurs in the same manner as the pyrite and contains the same kind of inclusions. Sphalerite is prevalent in association with the other sulphides in gangue as small masses and irregular grains. It encloses numerous tiny rounded dots of chalcopyrite; small irregular grains of chalcopyrite, galena, pyrite, and gangue are also included. Chalcopyrite is present also as occasional, small, irregular grains and veins in gangue. No gold is visible in the sections.

Sampling and Analysis. The sample was crushed and sampled and analysed as follows:

Gold	0·18 oz./ton
Silver	9·16 "
Copper	0·38 per cent
Lead	2·41 "
Zinc	2·02 "
Sulphur	5·62 "

Results of Experimental Research. Numerous flotation methods were tried on the ore.

By making a bulk concentrate a gold recovery of 92·4 per cent, a silver recovery of 93·8 per cent and a lead recovery of 97·3 per cent was shown. The ratio of concentration was 6·52 : 1 (see Test No. 4).

Several tests were run with the object of making a high-grade lead-gold-silver concentrate by depressing the zinc in the tailing. The tailing was cyanided. The results showed a concentrate carrying gold 1·82 ounces per ton, silver 103·38 ounces per ton, lead 29·66 per cent, zinc 5·56 per cent. The ratio of concentration was 13 to 1. The cyanidation tailing as shown in Tests Nos. 8 and 9 was 0·01 ounce gold per ton. The gold and silver recovery by making a high-grade shipping concentrate and cyaniding the tailing was 95 and 97 per cent respectively.

Two selective flotation tests were made producing a lead-silver-gold concentrate and a zinc-iron concentrate, with the object of cyaniding the zinc-iron concentrate. The ratio of concentration was high. The flotation tailing was slightly higher than in the previous tests and the zinc-iron concentrate carried sufficient copper to cause fouling of the cyanide solution. The grade of shipping concentrate carried about 2·46 ounces of gold per ton, 116·26 ounces of silver per ton, and 33·16 per cent of lead. The overall recovery of gold and silver was lower than in the tests in which the bulk tailing is cyanided.

NOTE: In all tests the feed as shown in the tables is calculated from the products.

EXPERIMENTAL INVESTIGATION

A. BULK FLOTATION

Test No. 1

A sample of ore (−14 mesh) was ground in a small ball mill at a pulp dilution of 0·75 to 1, with soda ash, 1 pound per ton, and Aerofloat No. 31, 0·07 pound per ton. The ground pulp was passed over a hydraulic classifier to remove any free gold before going to the flotation cell. No visible gold was seen in the classifier underflow. The classifier overflow was floated using the following reagents:

	Lb./ton
Potassium amyl xanthate	0·1
Soda ash	1·0
Cresylic acid	0·064
Pine oil	0·062

The results are as follows:

Hydraulic Classification:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·18	100·00	
Underflow	0·08	5·64	2·47	1250 : 1
Overflow	99·92	0·178	97·53	

Flotation:

Product	Weight, per cent	Assay						Distribution, per cent				Ratio of concentration
		Oz./ton		Per cent								
		Au	Ag	Cu	Pb	Zn	S	Au	Ag	Cu	Pb	
Feed	100·00	0·178	9·59	0·32	2·27	......	......	100·00	100·0	100·0	100·0	
Bulk concentrate	15·28	1·11	60·0	2·04	14·60	12·04	......	95·24	95·6	97·4	98·1	6·54 : 1
Tailing	84·72	0·01	0·50	0·01	0·05	0·25	0·31	4·76	4·4	2·6	1·9	

Summary of Gold Recovery:

	Per cent
Recovery of gold in classifier underflow	2·47
Recovery of gold in flotation concentrate, 95·24 per cent × 97·53 per cent.	92·89
Overall recovery as concentrates	95·36

Screen Test:

Mesh	Weight, per cent
+65	0·3
− 65+100	3·9
−100+150	12·4
−150+200	15·8
−200	67·6
	100·0

Test No. 2

The ore was given a finer grind. The results are as follows:

Hydraulic Classification:

Product	Weight, per cent	Assay, Au. oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·18	100·00	
Underflow	0·03	16·37	2·69	3333 : 1
Overflow	99·97	0·178	97·31	

Flotation:

Product	Weight, per cent	Assay						Distribution, per cent				Ratio of concentration
		Oz./ton		Per cent								
		Au	Ag	Cu	Pb	Zn	S	Au	Ag	Cu	Pb	
Feed	100·00	0·178	9·41	0·32	2·20	......	......	100·00	100·0	100·0	100·0	
Bulk concentrate	14·89	1·14	60·80	2·00	14·50	12·14	......	95·23	96·2	92·1	98·1	6·72 : 1
Tailing	85·11	0·01	0·42	0·03	0·05	0·23	0·36	4·77	3·8	7·9	1·9	

Summary of Gold Recovery:

	Per cent
Recovery of gold in classifier underflow	2·69
Recovery of gold in flotation concentrate 95·23 per cent × 97·31 per cent.	92·67
Overall recovery	95·36

Screen Test:

Mesh	Weight, per cent
+100	0·7
−100+150	5·2
−150+200	10·5
−200	83·6
	100·0

The results indicate that to obtain a low tailing fine grinding is not necessary.

Hydraulic classification was not used in subsequent tests as no visible free gold was seen in the classifier concentrate.

Test No. 3

The bulk concentrate was cleaned.

Reagents:

Grinding:

	Lb./ton
Soda ash	1·0
Aerofloat No. 31	0·07

Flotation:

Potassium amyl xanthate	0·1
Cresylic acid	0·064
Pine oil	0·062

Grinding: Approximately 65 per cent —200 mesh.

Product	Weight, per cent	Assay						Distribution, per cent			
		Oz./ton		Per cent				Au	Ag	Cu	Pb
		Au	Ag	Cu	Pb	Zn	Fe				
Feed	100·00	0·20	10·15	0·31	2·55	1·99	....	100·00	100·00	100·0	100·0
Cleaner concentrate	14·32	1·18	64·36	2·03	17·06	12·3	24·5	86·14	90·79	93·2	95·6
Middling	1·74	0·06	26·26	0·73	4·96	11·5	....	5·30	4·50	4·1	3·4
Tailing	83·94	0·02	0·57	0·01	0·03	0·28	....	8·56	4·71	2·7	1·0

Ratio of concentration, 6·98 : 1.

Test No. 4

This test was similar to Test No. 3 using 2 pounds of soda ash per ton in grinding. Other reagents were the same.

Product	Weight, per cent	Assay						Distribution, per cent			
		Oz./ton		Per cent				Au	Ag	Cu	Pb
		Au	Ag	Cu	Pb	Zn	Fe				
Feed	100·00	0·20	10·14	0·32	2·54	2·05	....	100·00	100·00	100·00	100·00
Cleaner concentrate	15·34	1·18	62·04	1·98	16·11	12·75	24·9	92·39	93·84	94·8	97·3
Middling	1·82	0·36	18·36	0·45	2·39	3·16	....	3·37	3·29	2·6	1·7
Tailing	82·84	0·01	0·35	0·01	0·03	0·05	....	4·24	2·87	2·6	1·0

Ratio of concentration, 6·52 : 1.

B. SELECTIVE FLOTATION AND CYANIDATION OF TAILING

The object was to make a lead-silver-gold concentrate by depressing as much as possible of the zinc and iron sulphides and cyaniding the flotation tailing. The results are shown in the two following tests.

Test No. 5

A sample of ore was ground with 2 pounds of soda ash per ton, 0·05 pound of sodium cyanide, 0·25 pound of zinc sulphate, and 0·07 pound of Aerofloat No. 31 per ton. Grinding was to a fineness of about 67 per cent

−200 mesh. The following reagents were added in the cell, butyl xanthate 0·05 pound per ton, cresylic acid, 0·064 pound per ton, and pine oil 0·031 pound per ton. A concentrate was floated off and the tailing was reground in water to have 77·8 per cent −200 mesh, it was filtered and cyanided for 24 hours in a solution of strength equivalent to 1 pound of potassium cyanide per ton. The results are as follows:

Flotation:

Product	Weight, per cent	Assay					Distribution, per cent				Ratio of concentration
		Oz./ton		Per cent							
		Au	Ag	Cu	Pb	Zn	Au	Ag	Cu	Pb	
Feed	100·00	0·19	9·64	0·31	2·37	......	100·0	100·0	100·0	100·0	
Ag-Pb concentrate	9·82	1·70	85·44	2·74	22·92	13·66	86·1	87·0	88·2	95·0	10·2 : 1
Tailing	90·18	0·03	1·39	0·04	0·13	0·75	13·9	13·0	11·8	5·0	

Cyanidation of Tailing:

Product	Feed assay, oz./ton		Tailing assay, oz./ton		Extraction, per cent		Consumption, lb./ton tailing		Pulp dilution
	Au	Ag	Au	Ag	Au	Ag	KCN	CaO	
Cyanide tailing	0·03	1·39	0·01	0·34	66·7	75·5	1·35	4·40	1·5 : 1

Summary of Recovery:

	Gold, per cent	Silver, per cent
Recovery in concentrate	86·1	87·0
Recovery by cyanidation of tailing	9·3	9·8
Overall recovery	95·4	96·8

Test No. 6

No cyanide was used in the grind but the zinc sulphate was increased to 2·0 pounds per ton. The tailing was reground to a fineness of 82·8 per cent −200 mesh.

Flotation:

Product	Weight, per cent	Assay					Distribution, per cent				Ratio of concentration
		Oz./ton		Per cent							
		Au	Ag	Cu	Pb	Zn	Au	Ag	Cu	Pb	
Feed	100·0	0·19	9·30	0·33	2·45	......	100·0	100·0	100·0	100·0	
Ag-Pb concentrate	8·8	1·52	82·52	2·78	25·72	6·73	71·0	78·1	74·9	92·2	11·36 : 1
Tailing	91·2	0·06	2·23	0·09	0·21	1·62	29·0	21·9	25·1	7·8	

Cyanidation of Tailing:

Product	Feed assay, oz./ton		Tailing assay, oz./ton		Extraction, per cent		Consumption, lb./ton tailing		Pulp dilution
	Au	Ag	Au	Ag	Au	Ag	KCN	CaO	
Cyanide tailing........	0·06	2·23	0·01	0·83	83·3	62·8	1·36	4·54	1·54 : 1

Summary of Recovery:

	Gold, per cent	Silver, per cent
Recovery in concentrate..................................	71·0	78·1
Recovery by cyanidation of tailing........................	24·2	13·8
Overall recovery.................................	95·2	91·9

Test No. 7

The object was to make a high-grade gold-silver-lead cleaner concentrate and to deslime the tailing to determine the distribution of gold in the slime and sand. The results indicate that the gold is almost equally distributed in the slime and sand of the tailing.

The ore was ground to have about 83 per cent −200 mesh.

Reagents:

Grinding:

	Lb./ton
Soda ash..	2·0
Zinc sulphate..	2·0
Aerofloat No. 31...	0·035

Flotation:

Butyl xanthate...	0·05
Cresylic acid..	0·064
Pine oil...	0·031

Product	Weight, per cent	Assay						Distribution, per cent			
		Oz./ton		Per cent				Au	Ag	Cu	Pb
		Au	Ag	Cu	Pb	Zn	Fe				
Feed...............	100·00	0·19	10·16	0·32	2·54	....	....	100·00	100·00	100·00	100·00
Cleaner concentrate .	7·68	1·82	103·38	3·03	29·66	5·56	24·6	74·40	78·16	72·67	89·71
Middling............	1·15	0·75	36·72	1·00	1·30	....	....	4·58	4·16	3·59	0·59
Slime tailing.........	31·25	0·04	1·72	0·09	0·29	1·70	....	6·65	5·29	8·78	3·57
Sand tailing.........	59·92	0·045	2·10	0·08	0·26	1·90	....	14·37	12·39	14·96	6·13

Ratio of concentration, 13 : 1.

Screen test on sand tailing indicates 77·7 per cent −200 mesh.

Tests Nos. 8 and 9.

The two following tests were run to determine the recovery by making a high-grade cleaner concentrate and cyaniding the tailing. These tests are similar to Test No. 5, but the tailings were not reground.

The ore was ground to have approximately 83 per cent −200 mesh.

Reagents:

Grinding:

	Test No. 8 Lb./ton	Test No. 9 Lb./ton
Soda ash	2·0	....
Zinc sulphate	2·0	....
Sodium sulphite	....	1·0
Aerofloat No. 31	0·035	0·035

Flotation:

	Test No. 8	Test No. 9
Butyl xanthate	0·05	0·05
Cresylic acid	0·064	0·064
Pine oil	0·031	0·031

Flotation:

Test	Product	Weight, per cent	Assay Oz./ton Au	Ag	Assay Per cent Cu	Pb	Zn	Fe	Distribution, per cent Au	Ag	Cu	Pb
8	Feed	100·00	0·21	10·17	0·31	2·86	2·25	....	100·00	100·00	100·00	100·00
	Cleaner concentrate	12·34	1·50	76·0	2·14	21·84	6·85	27·94	88·40	92·17	84·97	94·18
	Middling	0·61	0·41	12·15	0·52	4·46	3·24	....	1·19	0·73	1·03	0·95
	Tailing	87·05	0·025	0·83	0·05	0·16	1·59	....	10·41	7·10	14·00	4·87
9	Feed	100·00	0·17	8·01	0·29	2·38	2·16	....	100·00	100·00	100·00	100·00
	Cleaner concentrate	11·91	1·25	60·80	2·34	18·58	10·32	25·24	87·79	90·36	96·20	93·13
	Middling	0·56	0·565	20·74	0·40	2·59	1·99	7·29	1·89	1·45	0·76	0·61
	Tailing	87·53	0·02	0·75	0·01	0·17	1·05	....	10·32	8·19	3·04	6·26

Ratio of concentrations: Test 8—8·1 : 1.
Test 9—8·4 : 1.

Cyanidation of Flotation Tailings:

Test	Product	Assay, Au, oz./ton Feed	Tailing	Assay, Ag, oz./ton Feed	Tailing	Extraction, per cent Au	Ag	Consumption, lb./ton solids KCN	CaO	Pulp dilution
8	Cyanide tailing	0·025	0·01	0·83	0·26	60·00	68·67	2·60	4·6	1·59 : 1
9	Cyanide tailing	0·02	0·01	0·75	0·15	50·00	80·00	2·09	6·8	1·59 : 1

The analysis of cyanide solution Test No. 9 is as follows:

Reducing power	272 ml. $\frac{N}{10}$ $KMnO_4$/litre
KCNS	0·58 lb./ton soln.
Total alkalinity	0·48 " " as CaO
Copper	0·16 " "
Ferrous iron	Trace

The overall recovery of gold and silver is as follows:

	Per cent		Per cent
Test No. 8—Gold	95·36	Silver	97·55
Test No. 9—Gold	93·90	Silver	98·07

C. SELECTIVE FLOTATION AND CYANIDATION OF IRON-ZINC CONCENTRATE

Two concentrates were made, one a lead-copper-silver-gold concentrate and the other a zinc-iron. The object was to determine the possibility of cyaniding the zinc-iron concentrate.

Test No. 10

Reagents:

Grinding, lb./ton

Lime	1·0
Zinc sulphate	1·0
Sodium cyanide	0·05
Aerofloat No. 31	0·034

Lead concentrate, lb./ton

Butyl xanthate	0·05
Cresylic acid	0·064
Pine oil	0·031

Zinc-iron concentrate, lb./ton

Copper sulphate	1·0
Potassium amyl xanthate	0·10
Soda ash	1·0
Cresylic acid	0·096
Pine oil	0·124

Grinding: 65 per cent —200 mesh

Product	Weight, per cent	Assay						Distribution, per cent				Ratio of concentration
		Oz./ton		Per cent								
		Au	Ag	Cu	Pb	Zn	Fe	Au	Ag	Cu	Pb	
Feed	100·00	0·18	9·81	0·25	2·08	....	....	100·00	100·00	100·0	100·0	
Pb concentrate	5·67	2·30	133·82	3·03	34·28	5·46	....	72·22	77·28	68·7	93·5	17·6 : 1
Zn-Fe concentrate	10·44	0·36	15·74	0·67	1·29	16·40	26·67	20·81	16·74	28·0	6·5	9·6 : 1
Tailing	83·89	0·015	0·70	0·01	0·0	0·10	....	6·97	5·98	3·3	0·0	

Test No. 11

Soda ash was used in grinding in place of lime. The other reagents were the same as in Test No. 5.

Product	Weight, per cent	Assay						Distribution, per cent				Ratio of concentration
		Oz./ton		Per cent								
		Au	Ag	Cu	Pb	Zn	Fe	Au	Ag	Cu	Pb	
Feed	100·00	0·19	8·71	0·31	2·18	....	....	100·0	100·0	100·0	100·0	
Pb concentrate	5·99	2·46	116·26	3·62	33·16	5·77	....	77·2	80·0	70·7	91·0	16·7 : 1
Zn-Fe concentrate	9·64	0·32	12·40	0·67	1·77	17·10	26·36	16·2	13·7	21·1	7·8	10·4 : 1
Tailing	84·37	0·015	0·65	0·03	0·03	0·07	....	6·6	6·3	8·2	1·2	

Test No. 11A

Cyanidation of a sample of the combined zinc-iron concentrates from the above tests. The concentrate was reground and cyanided in a solution of strength equivalent to 3 pounds of potassium cyanide per ton at a pulp dilution of 3 to 1. Lime, 3 pounds per ton, was added as protective alkalinity. The pulp was agitated for 24 hours.

Product	Feed, oz./ton		Tailing, oz./ton		Extraction, per cent		Reagents consumed, lb./ton concentrate	
	Au	Ag	Au	Ag	Au	Ag	KCN	CaO
Cyanide tailing............	0·34	14·07	0·065	3·30	80·9	76·5	15·63	4·10

A screen test of the reground concentrate indicated a fineness of grinding of 97 per cent —325 mesh.

Summary of Recovery (Average of Tests 5 and 6):

	Gold, per cent	Silver, per cent
Recovery in silver-lead concentrate......................	74·70	78·60
Recovery by cyanidation of zinc-iron concentrate........	13·95	11·64
Overall recovery..............................	88·65	90·24

Test No. 12

This was similar to Tests Nos. 5 and 6. In grinding 2·0 pounds of soda ash per ton and 1·5 pounds of zinc sulphate were used. A lead-silver concentrate and a zinc-iron concentrate were made. The latter was reground and cyanided in a solution equivalent to 2 pounds of potassium cyanide per ton for 24 hours at a dilution of 2·68 to 1.

Flotation Results:

Product	Weight, per cent	Assay						Distribution, per cent				Ratio of concentration
		Oz./ton		Per cent								
		Au	Ag	Cu	Pb	Zn	Fe	Au	Ag	Cu	Pb	
Feed...........	100·00	0·15	7·48	0·31	2·39	....	....	100·0	100·0	100·0	100·0	
Pb-Ag concentrate..........	6·72	1·62	84·68	3·26	33·56	5·36	....	73·7	76·0	70·3	94·5	14·88 : 1
Zn-Fe concentrate..........	9·37	0·28	12·30	0·72	1·12	18·12	26·26	17·8	15·4	21·6	4·4	10·67 : 1
Tailing.........	83·91	0·015	0·77	0·03	0·03	0·07	....	8·5	8·6	8·1	1·1	

Cyanidation of Zn-Fe Concentrate:

Product	Feed assay, oz./ton		Tailing assay, oz./ton		Extraction, per cent		Reagents consumed, lb./ton concentrate	
	Au	Ag	Au	Ag	Au	Ag	KCN	CaO
Cyanide tailing............	0·28	12·30	0·15	7·22	46·4	41·3	9·57	13·66

Summary of Recovery:

	Gold, per cent	Silver, per cent
Recovery in silver-lead concentrate......................	73·7	76·0
Recovery by cyanidation zinc-iron concentrate..........	8·3	6·4
Overall recovery..............................	82·0	82·4

Final overall tailing—Au, 0·029 oz./ton

CONCLUSIONS

The investigation shows that two methods of treatment are open for adoption.

The first is the production of a bulk concentrate for shipment to a smelter.

Test No. 4 indicates that 95·7 per cent of the gold, 97 per cent of the silver, and 99 per cent of the lead can be floated in a rougher concentrate. Cleaning this raises the grade to 1·18 ounces of gold, 62·04 ounces of silver per ton, 2 per cent of copper, 16 per cent of lead, and 12·75 per cent of zinc. Continuous operation would show whether the middling product will report with the concentrate or with the tailing. It is expected that the recovery of 95 per cent of the gold should be maintained with a ratio of concentration of 6·5 to 1.

The second method is the production of a higher grade flotation concentrate for shipment to a smelter, followed by cyanidation of the flotation residue.

Test No. 7 shows that a ratio of concentration of 13 to 1 can be obtained and a concentrate containing 1·82 ounces of gold, 103·38 ounces of silver per ton and 29·66 per cent of lead can be produced. Cyanidation of the flotation tailing leaves a residue containing 0·01 ounce of gold per ton. This combination of flotation and cyanidation yields recoveries of from 95 to 97 per cent of the gold in the feed.

It was found impracticable to produce a concentrate for cyanidation. Owing to the copper in the concentrate a high consumption of cyanide was recorded and lower recovery was obtained.

Although this particular shipment of ore does not disclose the presence of coarse free gold, the placing of a jig or unit cell in the grinding circuit would be advisable.

The choice of method will depend largely on the freight rates on the concentrate. Straight flotation shows a ratio of concentration of 6·5 : 1, whereas the second method indicates a ratio of concentration of 13 : 1 with a corresponding increase in the grade of concentrate. To equal the overall recovery of the first method, a cyanide annex would be necessary, the cost of which must be balanced against the freight charges; and the profits from the two methods would have to be compared before deciding on the flow-sheet.

Ore Dressing and Metallurgical Investigation No. 751

GOLD ORE FROM UCHI GOLD MINES, LIMITED, IN THE WOMAN LAKE AREA, KENORA DISTRICT, ONTARIO

Shipment. A shipment of 41 boxes of ore, net weight 2,965 pounds, was received June 10, 1938. The sample was submitted by B. H. Budgeon, Director of Mining, Uchi Gold Mines, Limited, 25 King Street West, Toronto.

Location of Property. The property is in the northwest corner of Earngey Township, near Woman Lake, in the Red Lake division of Kenora District, Ontario.

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

The *gangue* consists of fine-textured glassy quartz and dark grey rock, possessing an erratic schistosity, with abundant disseminated carbonate.

In their decreasing order of abundance the *metallic minerals* are: pyrite, pyrrhotite, ilmenite (?), chalcopyrite, and native gold. The pyrite, which is present as coarse to fine crystals and grains, is disseminated almost solely in the schist. Pyrrhotite occurs in considerable quantity but is not so prevalent as pyrite, with which it is closely associated. Both of these minerals contain numerous inclusions of gangue and of each other. Small irregular grains and rods of a hard grey mineral are locally numerous in the schist and as occasional inclusions in pyrite. This mineral could not be determined definitely in reflected light but it resembles ilmenite. Rare, tiny, irregular grains of chalcopyrite are present in gangue and in pyrite. A dozen grains of native gold are visible in the sections, ranging in size from 25 microns (−560 mesh) down to 2 microns (−2300 mesh). Of this amount, 80·9 per cent occurs in dense pyrite and 19·1 per cent in gangue.

Sampling and Assaying. A feed sample cut from the shipment was assayed and reported as follows:

Gold	0·494	oz./ton
Silver	0·07	"
Iron	5·79	per cent
Sulphur	1·10	"
Pyrrhotite	1·00	"

Experimental Tests

Amalgamation and cyanidation tests were conducted on the ore and it was found that 97 to 98 per cent of the gold was soluble in cyanide solution when the ore was ground 80 per cent through 200 mesh, and that 80 per cent of the gold is free when the ore was ground 60 per cent through 200 mesh. The ore contains some pyrrhotite and it is necessary to add red lead to the grinding circuit to keep the pyrrhotite from fouling the solution.

The tests are described in detail as follows:

BARREL AMALGAMATION

Test No. 1

A sample of the ore was ground 57 per cent through 200 mesh in a ball mill and was amalgamated with mercury for one hour. The amalgamation tailing was assayed for gold.

The object was to determine how much of the gold was free at this grinding.

Results:

Feed sample	0·494	Au, oz./ton
Amalgamation tailing	0·10	"
Extraction	79·8	per cent

CYANIDATION

Tests Nos. 2 to 7

Samples of the ore were ground 57 and 83 per cent through 200 mesh in cyanide and agitated for 24 hours. The solutions were kept at about 1·0 pound of potassium cyanide and 0·20 pound of lime per ton. Dilution ratio was 1·5 : 1 solution to solids. The superpanner showed the presence of coarse gold in each of these tailing samples. The largest piece, measuring about 2·0 × 0·50 millimetre was found in the tailing from Test No. 4.

Summary of Results—Tests Nos 2 to 7:

Feed sample: gold, 0·494 oz./ton.

Test No.	Grind, per cent −200 mesh	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton ore	
				KCN	CaO
2	57·0	0·025	94·94	0·55	3·53
3	57·0	0·025	94·94	0·58	3·56
4	57·4	0·162	67·21	0·55	3·56
5	57·7	0·051	89·68	0·55	3·53
6	82·8	0·03	93·93	0·75	3·65
7	82·8	0·025	94·94	0·75	3·65

Screen Analysis, Cyanide Tailing—Test No. 4:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold	
			Per cent content	Per cent total
+ 65	2·22	3·89	53·31	17·48
− 65+100	7·64	0·205	9·67	3·17
−100+150	16·88	0·10	10·42	3·42
−150+200	15·86	0·145	14·20	4·66
−200	57·40	0·035	12·40	4·06
Cyanide tailing	100·00	0·162	100·00	32·79

Extraction: 67·21 per cent.

Screen Analysis, Cyanide Tailing—Test No. 5:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold	
			Per cent content	Per cent total
+ 65	2·50	0·035	1·72	0·18
− 65+100	7·58	0·11	16·36	1·69
−100+150	16·89	0·08	26·52	2·74
−150+200	15·34	0·09	27·10	2·80
−200	57·69	0·025	28·30	2·91
Cyanide tailing	100·00	0·051	100·00	10·32

Extraction: 89·68 per cent.

Tests Nos. 8 and 9 (48 Hours)

Samples of the ore were ground 57 per cent through 200 mesh in cyanide solution as was done in Tests Nos. 2 to 7. The pulps were agitated 48 hours, other conditions being the same as in the 24-hour series of tests. Superpanner tests on samples of the cyanide tailings did not reveal the presence of any undissolved free gold.

Summary of Tests Nos. 8 and 9:

Feed sample: gold, 0·494 oz./ton.

Test No.	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton ore	
			KCN	CaO
8	0·0175	96·46	0·71	4·27
9	0·0173	96·50	0·74	4·27

Screen Analysis, Cyanide Tailing:
Test No. 8:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold	
			Per cent content	Per cent total
+ 65	3·10	0·02	3·54	0·13
− 65+100	8·64	0·025	12·33	0·44
−100+150	17·92	0·03	30·68	1·08
−150+200	15·56	0·025	22·20	0·78
−200	54·78	0·01	31·25	1·11
Cyanide tailing	100·00	0·0175	100·00	3·54

Extraction: 96·46 per cent.

Test No. 9:

Mesh	Weight, per cent	Assay, Au, oz./ton	Distribution of gold	
			Per cent content	Per cent total
+ 65	2·20	0·04	5·10	0·18
− 65+100	7·56	0·02	8·76	0·31
−100+150	17·36	0·03	30·16	1·06
−150+200	15·88	0·025	22·98	0·80
−200	57·00	0·01	33·00	1·15
Cyanide tailing	100·00	0·0173	100·00	3·50

Extraction: 96·50 per cent.

From the foregoing series of tests it is obvious that fine grinding with at least 48 hours' agitation would be necessary. If the coarse gold be removed in the grinding circuit and amalgamated, it may be possible to reduce the period of agitation by a few hours. The screen analyses on the tailings from Tests Nos. 8 and 9 indicate that grinding should be carried to 80 per cent or more through 200 mesh.

Owing to the presence in the ore of 1·0 per cent of pyrrhotite, cycle tests were conducted to see if it would cause any fouling of solutions and, if so, what could be done to prevent it.

Cycle Tests Nos. 10 to 17

A sample of the ore was ground 82 per cent through 200 mesh in a ball mill with lime and cyanide. The pulp was agitated 48 hours and filtered. The pregnant solution was de-aerated, precipitated with zinc dust, and the barren solution re-aerated before being used to treat another batch of ore. This was continued for five cycles and in the sixth no lime was used in grinding or agitation. In the seventh no lime was used in grinding or agitation, but red lead was added to the grinding circuit in the proportion of 0·10 pound per ton of ore. In the eighth cycle both lime and red lead were added and agitation was done in a steel jar under pressure to give more thorough aeration. The final solution was then examined for reducing power and harmful ingredients.

Superpanner tests on samples of the tailings showed no undissolved free gold in the first tailing, but did show it in all but one of the rest. The tailing from Test No. 17 assayed 0·01 ounce per ton in gold after the free gold had been removed by amalgamation. The results are shown in Table I. This series of tests seems to show that both lime and lead are necessary in the grinding circuit in order to treat the ore satisfactorily. The lime is needed to neutralize the carbonates in the ore and the lead to precipitate soluble sulphides formed by the pyrrhotite.

It is worthy of note that in Tests Nos. 10 to 15 the tailing assays vary inversely as the amount of lime that has been added to the grinding and agitation circuits. Then in Test No. 16, when some lead was introduced, the tailing improved and in Test No. 17, with both lime and lead in the grinding circuit, the tailing assay is back to normal after a small amount of coarse gold is amalgamated out.

The next cycle test, Series 19 to 23, shown in Table II, is further evidence of the need of both lime and lead.

Cyanide consumption was estimated at 0·85 pound of potassium cyanide per ton of ore.

TABLE I

Summary of Cycle Tests Nos. 10 to 18

Test No.	Lime to grinding and agitation circuit, lb./ton ore	Lime to settling, lb./ton ore	Pb to grinding circuit, lb./ton ore	Final solution, lb./ton		Cyanide tailing, Au, oz./ton	Extraction by cyanidation, per cent gold	Cyanide tailing amalgamated, Au, oz./ton	Total extraction, per cent gold
				KCN	CaO				
10.....	5·0	....	Nil	1·0	0·19	0·0125	97·47	....	....
11.....	4·4	....	Nil	1·48	0·22	0·0175	96·46	....	....
12.....	3·8	....	Nil	1·12	0·14	0·05	89·88	0·03	93·93
13.....	2·4	....	Nil	1·0	0·08	0·06	87·85	0·055	88·87
14.....	4·10	....	Nil	1·20	0·13	0·03	93·93	0·025	94·94
15.....	Nil	1·50	Nil	1·08	0·08	0·13	73·68	0·08	83·81
16.....	Nil	2·0	0·10	1·0	0·20	0·06	87·85	0·06	87·85
17*....	2·0	0·50	0·10	0·80	0·08	0·025	94·94	0·01	97·96
18*....	2·0	1·0	0·10	0·64	0·17	0·0325	93·42	0·01	97·96

* Agitation done in steel jar under pressure.

On examination the final solution was found to contain the following:

Reducing power..................................	288 ml. $\frac{N}{10}$ $KMnO_4$/litre
KCNS..	0·26 grm./litre
Copper..	0·11 "
Iron...	0·07 "

TABLE II

Summary of Cycle Tests Nos. 19 to 23

Test No.	Lime to grinding and agitation circuit, lb./ton ore	Lime to settling, lb./ton ore	Pb to grinding circuit, lb./ton ore	Final solution, lb./ton		Cyanide tailing, Au, oz./ton	Extraction by cyanidation, per cent gold	Cyanide tailing, amalgamated, Au, oz./ton	Total extraction, per cent gold
				KCN	CaO				
19.....	Nil	2·0	Nil	0·92	0·18	0·015	96·96	....	....
20.....	Nil	4·0	Nil	0·98	0·33	0·115	76·70	0·015	96·96
21.....	2·0	0·50	0·10	1·10	0·10	0·015	96·96	....	....
22.....	2·4	Nil	0·10	1·02	0·08	0·0175	96·46	0·015	96·96
23.....	3·6	Nil	0·10	2·06	0·14	0·015	96·96	0·01	97·96

On examination the final solution was found to contain the following:

Reducing power..................................	200 ml. $\frac{N}{10}$ $KMnO_4$/litre
KCNS..	0·19 grm./litre
Copper..	0·013 "
Iron...	0·066 "

SETTLING TESTS

Settling tests were conducted on a sample of pulp in cyanide solution as follows:

The ore was ground 80 per cent through 200 mesh in cyanide solution in a ball mill, with lime for protective alkalinity. The pulp was agitated for 24 hours and transferred to a glass tube 2 inches in diameter, where it was allowed to settle. The pulp level was read and recorded at 5-minute intervals for a period of one hour. Tests were conducted using different dilutions, ratios, and strengths of cyanide.

The results are tabulated as follows:

Time	Lb./ton solution KCN, 1·04 CaO, 0·32	Lb./ton solution KCN, 1·60 CaO, 0·28	Lb./ton solution KCN, 1·95 CaO, 0·26	Lb./ton solution KCN, 0·90 CaO, 0·25
	Dilution ratio, 1·5 : 1	Dilution ratio, 1·5 : 1	Dilution ratio, 1·5 : 1	Dilution ratio, 3 : 1
	feet	feet	feet	feet
Start	2·40	2·45	2·50	4·28
5 minutes	2·33	2·39	2·42	3·92
10 "	2·27	2·33	2·37	3·60
15 "	2·22	2·27	2·30	3·32
20 "	2·16	2·21	2·24	3·06
25 "	2·11	2·15	2·18	2·84
30 "	2·05	2·09	2·11	2·61
35 "	2·00	2·03	2·05	2·40
40 "	1·94	1·97	1·97	2·22
45 "	1·88	1·90	1·91	2·07
50 "	1·82	1·84	1·84	1·95
55 "	1·77	1·78	1·78	1·85
One hour	1·70	1·72	1·71	1·77
Total drop in pulp level	0·70	0·73	0·79	2·51

CONCLUSIONS

The work done on the sample indicates that the ore can be treated by cyanidation if the coarse gold is removed by a jig or other means and is amalgamated. Owing to the presence in the ore of about 1·0 per cent of pyrrhotite, red lead should be added with the ore in the grinding circuit to precipitate soluble sulphides and to prevent the formation of thiocyanates. Sufficient lime to satisfy the carbonates in the ore should also be added to the grinding circuit but, owing to the presence of the pyrrhotite, it would be well to avoid any excess of lime in solution over that needed for settling. For this purpose, 0·30 pound of lime per ton will be found ample.

Owing to the presence of the pyrrhotite, it would not be practicable to attempt concentration of sulphides by any gravity method with the idea of giving them a separate "cyanide" treatment, because the pyrrhotite would concentrate with the pyrite and would surely cause trouble in any separate "cyanide" treatment circuit.

The ore needs fine grinding, 80 per cent or more through 200 mesh, and about 48 hours' agitation, which, coupled with the above-mentioned conditions, should produce a satisfactory tailing.

Ore Dressing and Metallurgical Investigation No. 752

GOLD ORE FROM THE UPPER CANADA MINES, LIMITED, KIRKLAND LAKE, ONTARIO

Shipment. A 159-pound shipment of gold ore from the Upper Canada Mines, Limited, Gauthier Township, East Kirkland Lake area, Ontario, was received on June 23, 1938. The material was submitted by C. W. Tully, Manager, Upper Canada Mines, Limited, Kirkland Lake, Ontario.

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

The gangue is fine-textured, siliceous, somewhat porphyritic rock, mottled greenish grey to pink in color. A considerable quantity of carbonate is present as numerous finely disseminated grains; the carbonate gives a moderately strong microchemical test for iron.

Pyrite is the only abundant *metallic mineral.* It is disseminated throughout the rock as medium to fine grains. A very small quantity of chalcopyrite occurs as small grains associated with the pyrite, occasionally within it. Rare grains of a grey undetermined mineral occur in the gangue, often associated with pyrite. Tests show that this mineral cannot be one that contains gold.

Numerous grains of native gold are visible in the sections. The metal occurs (1) in the gangue; (2) associated with pyrite. That associated with the pyrite occurs (a) in sinuous fractures in pyrite, (b) associated with chalcopyrite along sinuous fractures in pyrite, and (c) in apparently dense pyrite. The grain analysis of the gold seen in the polished sections is shown in the following table:

Microscopic Grain Analysis of the Native Gold:

Mesh	Gold associated with pyrite			In gangue, per cent	Totals, per cent
	Along fractures, per cent	Along fractures with chalcopyrite, per cent	In dense pyrite, per cent		
+ 560	....	....	7·1	....	7·1
− 560+ 800	9·2	....	....	4·0	13·2
− 800+1100	5·7	3·2	3·7	5·9	18·5
−1100+1600	4·5	9·0	4·1	4·5	22·1
−1600+2300	2·7	6·1	....	7·7	16·5
−2300	0·8	4·3	0·8	16·7	22·6
	22·9	22·6	15·7		
Total		61·2		38·8	100·0

Conclusion from Microscopic Examination. The gold occurs in both gangue and pyrite. In both cases there is a considerable proportion of very finely divided gold, and none of the gold seen is coarse.

Sampling and Analysis. The sample of ore was crushed to −14 mesh and sampled by standard method. The sample thus obtained assayed as follows:

Gold	0·57 oz./ton
Silver	0·19 "
Tellurium	Nil
Iron	4·94 per cent
Copper	0·04 "
Sulphur	1·97 "

Experimental Tests

Test work was carried out to determine a method of treatment.

An extraction of 82·12 per cent of the gold was obtained by 70-hour cyanidation of the flotation concentrate. This was increased to 96·23 per cent by cyaniding the flotation tailing for 18 hours.

Twenty-four hours' cyanidation of the feed ground to 80·1 per cent and 90·2 per cent −200 mesh gave 94·74 per cent extraction. Treating the cyanide tailing by flotation, and regrinding and cyaniding the flotation concentrate for 24 hours, increased the extraction to 96·5 per cent. Cyanidation of roasted concentrate increased the overall recovery to 97·0 per cent.

An extraction of 95·6 per cent was obtained by 24-hour cyanidation of the ore ground to 92·0 per cent −325 mesh. The pulp ground to this degree of fineness settled very slowly.

Cyaniding the feed ground to 72·0 per cent −200 mesh, desliming the cyanide tailing, and regrinding and cyaniding the sand gave an overall extraction of 95·6 per cent.

Test No. 1

This was to determine the recovery of gold by the following methods of treatment:

(a) Flotation and cyanidation of the flotation concentrate;

(b) Flotation and cyanidation of the concentrate and the deslimed flotation tailing; and

(c) Flotation and cyanidation of the concentrate and the flotation tailing.

A representative sample of −14-mesh material was ground 90·2 per cent − 200 mesh in a ball mill with 0·5 pound of soda ash and 0·10 pound of potassium amyl xanthate. The pulp was transferred to the flotation machine; 0·10 pound of copper sulphate and 0·18 pound of pine oil per ton were added to the cell.

Results:

Product	Weight, per cent	Gold, assay, oz./ton	Gold, distribution, per cent	Ratio of concentration
Feed	100·00	0·57	100·0	
Flotation concentrate	5·42	8·77	83·4	18·5 : 1
Flotation tailing	94·58	0·10	16·6	

The flotation tailing assayed 0·15 per cent of sulphur.

The flotation tailing was deslimed by hydraulic classifier. The products assayed as follows:

	Weight, per cent of feed	Au, oz./ton
Sand	38·33	0·166
Slime	56·25	0·055

CYANIDATION TESTS

The flotation concentrate was ground 99·6 per cent −325 mesh in a pebble mill with 8·5 pounds of lime, and was agitated for 70 hours. The flotation tailing and the deslimed tailing were agitated for 18 hours. Solutions were kept at about 1 pound of sodium cyanide per ton.

Results:

Product	Cyanidation period, hours	Dilution, liquid : solid	Reagents consumed, lb./ton		Gold assay, oz./ton		Extraction, per cent
			NaCN	Lime	Feed	Cyanide tailing	
Reground concentrate	70	3·13 : 1	10·00	22·1	8·771	0·135	98·46
Sand	18	2·54 : 1	0·07	5·3	0·166	0·025	84·94
Flotation tailing	18	2·01 : 1	0·12	6·0	0·10	0·015	85·00

Summary of Results, Test No. 1:

Treatment of ore	Reagents consumed, lb./ton of ore		Gold assay, oz./ton	Overall recovery, per cent
	NaCN	Lime	Final tailing	
Method (a)	0·54	1·2	0·102	82·12
Method (b)	0·57	3·2	0·048	91·61
Method (c)	0·65	6·9	0·0215	96·23

Test No. 2 (A, B, and C)

The ore was treated by cyanidation and a subsequent flotation of the cyanide tailing and cyanidation of the reground flotation concentrate were made.

Representative samples of ore were ground to various degrees of fineness in lime and cyanide solutions. The pulps were agitated for 24 hours at 2 to 1 dilution. The solutions were kept at about 1 pound of sodium cyanide and 0·5 pound of lime per ton of solution.

Cyanidation of Feed:

Test No.	Per cent −200 mesh	Reagents consumed, lb./ton of ore		Gold, oz./ton		Extraction, per cent
		NaCN	Lime	Feed	Cyanide tailing	
2 A	80·1	1·00	7·3	0·57	0·03	94·74
2 B	90·2	1·00	7·2	0·57	0·03	94·74
2 C	80·1	0·78	6·1	0·57	0·03	94·74

The cyanidation tailings were filtered, thoroughly washed, and repulped for flotation. The following reagents and amounts were added to the flotation cell:

Reagent (lb./ton)	Test No. 2A	Test No. 2B	Test No. 2C
Soda ash	3·0	3·4	0·9
Copper sulphate	1·1	0·57	0·45
Potassium amyl xanthate	0·20	0·17	0·11
Pine oil	0·24	0·21	0·09

Results of Flotation Tests:

Test No.	Weight, per cent	Ratio of concentration	Gold, oz./ton		
			Flotation feed	Flotation concentrate	Flotation tailing
2A	9·50	10·5 : 1	0·03	0·15	0·0175
2B	7·04	14·2 : 1	0·03	0·229	0·015
2C	6·63	15·1 : 1	0·03	0·241	0·015

The flotation concentrate of Test No. 2B was roasted. The decrease of weight due to roasting was 22·4 per cent. The calcine of Test No. 2B and the flotation concentrate of Test No. 2C were ground in the pebble mill to 99·5 and 99·8 per cent −200 mesh, respectively. The pulps were cyanided for 24 hours. The solutions were kept at about 1 pound of sodium cyanide and 0·6 pound of lime per ton.

Results of Cyanidation Tests:

Test No.	Product	Reagents consumed, lb./ton of solids		Gold, oz./ton		Extraction, per cent
		NaCN	Lime	Feed	Cyanide tailing	
2B	Calcine	1·62	11·9	0·294	0·05	83·0
2C	Concentrate	4·8	15·5	0·241	0·095	60·6

Summary of Results:

Test No.	Reagents consumed, lb./ton of ore		Gold, oz./ton	Overall recovery, per cent
	NaCN	Lime	Final tailing	
2B	1·11	8·0	0·017	97·0
2C	1·10	7·1	0·020	96·5

To determine the distribution of gold in the flotation tailing, an infrasizing analysis of Test No. 2A was made. The results are as follows:

Infrasizing Analysis:

Microns	Weight, per cent	Gold, oz./ton	Distribution, per cent
+56	13·33	0·055	41·60
−56+40	12·66	0·035	24·58
−40+20	22·53	0·015	19·18
−20+10	17·57	0·005	4·99
−10	33·91	0·005	9·65
Feed	100·00	0·0176	100·00

The above results show that over 65 per cent of the gold in the flotation tailing is in +325-mesh product. This indicates that grinding to a high degree of fineness is necessary.

Test No. 3

Samples of ore were ground in cyanide solution with different amounts of lime. The pulps were transferred to cyanide agitators and agitated at ratio of dilution of 2 : 1. The solutions were kept at about 1 pound of sodium cyanide and about 0·5 pound of lime per ton, except in Test No. 3F.

In Test No. 3F, 0·3 pound of litharge per ton of ore was added to the grind. The solution during agitation was kept above 1 pound of lime.

Results of Cyanidation Tests:

Feed sample: gold, 0·57 oz./ton.

Test No.	Agitation period, hours	Fineness of grind,		Lime to grind, lb./ton	Reagents consumed, lb./ton ore		Cyanidation tailing assay, Au, oz./ton	Extraction, per cent
		Per cent	Mesh		NaCN	Lime		
3A	24	80·1	−200	1·0	1·00	7·3	0·03	94·74
3B	48	80·1	−200	1·0	1·20	7·7	0·03	94·74
3C	24	90·2	−200	2·0	1·00	7·2	0·03	94·74
3D	24	80·1	−200	3·0	0·78	6·1	0·03	94·74
3E	24	92·0	−325	3·0	1·00	6·7	0·025	95·61
3F	24	92·0	−325	5·0	0·94	12·4	0·03	94·74

The cyanide consumptions in Tests Nos. 3A, 3B, and 3C are slightly high, owing to insufficient lime added to the grind.

SETTLING TEST

Test No. 4

A cyanidation pulp, which had been agitated 22 hours, was transferred to a settling tube. The results were as follows:

Settling time, minutes	Height of pulp level, feet
Start	3·230
5	3·180
10	3·130
15	3·085
20	3·045
25	3·010
30	2·975
35	2·940
40	2·905
45	2·875
50	2·840
55	2·810
60	2·780

Drop in pulp level in 1 hour	0·45 feet
Pulp dilution	2 : 1
NaCN	1·0 lb./ton solution
CaO	0·6 "
−325 mesh	92·0 per cent

At the above fineness of grind, the pulp settles very slowly. This would involve the use of a large thickener area in mill practice.

Test No. 5 (A, B, and C)

The following tests were carried out to determine what gold extraction could be obtained by cyaniding the ore ground to less than 80 per cent −200 mesh, by desliming the cyanide tailing, and regrinding and cyaniding the sands.

Samples of ore were ground in the ball mill with lime and cyanide. The pulps were transferred to cyanide agitators and agitated for 24 hours. The 24-hour cyanidation tailings were filtered and thoroughly washed. The cyanide tailings were deslimed by passing over the Wilfley table. The table sands were reground and cyanided using fresh lime and cyanide solutions.

Cyanidation Tests:

Test No.	Product	Agitation, hours	Dilution liquid: solid	−200 mesh, per cent	Reagents consumed, lb./ton solids		Gold, oz./ton	
					NaCN	Lime	Feed	Cyanide tailing
5A	Feed	24	1·50 : 1	80·1	0·70	5·5	0·57	0·03
	Reground sands	16	1·66 : 1	87·5	0·44	5·9	0·046	0·035
5B	Feed	24	1·50 : 1	72·0	0·78	5·2	0·57	0·045
	Reground sands	6	1·52 : 1	91·0	0·54	4·1	0·075	0·035
5C	Feed	24	1·50 : 1	72·0	0·56	5·6	0·57	0·045
	Reground sands	16	1·45 : 1	91·1	0·77	5·9	0·072	0·025

Summary of Results

Test No.	Table slime			Final tailing, Au, oz./ton	Overall recovery, per cent
	Weight, per cent	Assay			
		Au, oz./ton	S, per cent		
5A..........	62·20	0·02	1·07	0·0256	95·51
5B..........	54·96	0·02	1·09	0·0268	95·30
5C..........	57·38	0·025	0·86	0·0250	95·61

Settling tests were carried out on the cyanidation pulps of Tests Nos. 5A and 5B.

Results of Settling Tests:

Settling time, minutes,	Test 5A Feed	Test 5A Feed	Test 5A Feed	Test 5A Reground sand	Test 5B Feed	Test 5B Reground sand
	Height of pulp, feet	Height of pulp, feet	Height of pulp, feet	Height of pulp, feet	Height of pulp, feet	Height of pulp, feet
Start...............	2·515	3·220	3·220	1·060	2·515	0·990
5..................	2·480	3·155	3·145	0·925	2·450	0·920
10..................	2·430	3·090	3·075	0·800	2·400	0·870
15..................	2·385	3·030	3·015	0·685	2·355	0·820
20..................	2·345	2·970	2·955	0·590	2·310	0·765
25..................	2·310	2·915	2·900	0·530	2·270	0·715
30..................	2·280	2·860	2·845	0·490	2·225	0·665
35..................	2·245	2·810	2·790	0·455	2·190	0·615
40..................	2·215	2·760	2·735	Crit. pt.	2·145	0·570
45..................	2·185	2·705	2·680	0·440	2·105	0·535
50..................	2·150	2·655	2·625	0·440	2·065	0·510
55..................	2·120	2·605	2·575	0·430	2·025	0·485
60..................	2·095	2·550	2·520	0·430	1·980	0·460
Dilution, lb./ton solution.........	1·50 : 1	2·00 : 1	2·00 : 1	1·66 : 1	1·50 : 1	1·52 : 1
−200 mesh, per cent.	80·1	80·1	80·1	87·5	72·0	91·0
NaCN, lb./ton solution..........	0·80	0·80	0·80	0·85	0·88	1·04
CaO, lb./ton solution.........	0·50	0·45	0·45	0·60	0·38	0·34
$(NH_4)_2SO_4$, lb./ton solution.........	....	....	0·50	....	....	....
Drop of pulp level, in feet/hour........	0·42	0·67	0·70	To critical point 1·04	0·535	0·53

SUMMARY AND CONCLUSIONS

The results indicate that over 95 per cent of the gold can be extracted from the ore.

Flotation treatment of the ore gave a concentrate containing 83·40 per cent of the gold. Seventy-hour cyanidation of the flotation concentrate gave an extraction of 82·12 per cent of the gold in the ore; the cyanidation tailing assaying 0·135 ounce of gold per ton. The flotation tailing assayed 0·10 ounce of gold. Desliming the flotation tailing and cyaniding the sand for 18 hours increased the overall extraction to 91·61 per cent. The slime assayed 0·055 ounce of gold. Cyaniding the flotation tailing for 18 hours increased the overall extraction to 96·23 per cent. This would give a final tailing of 0·0215 ounce of gold per ton.

A gold extraction of 94·74 per cent was obtained by 24-hour cyanidation of the feed ground to 80·1 per cent and 90·2 per cent −200 mesh. The cyanidation tailing assayed 0·03 ounce of gold. Treating the cyanide tailing by flotation gave a flotation tailing assaying from 0·015 to 0·0175 ounce of gold. Regrinding and cyaniding the flotation concentrate for 24 hours increased the recovery to 96·5 per cent. Twenty-four hours' cyanidation of the roasted concentrate increased the extraction to 97·0 per cent; the calculated value of the final tailing was 0·017 ounce of gold per ton.

Infrasizing analyses of the flotation tailing showed that over 65 per cent of the gold is in +325-mesh product. This indicates that grinding to a high degree of fineness would be necessary to liberate this gold.

Forty-eight hours' cyanidation of the feed ground to 80·1 per cent −200 mesh gave an extraction of 94·74 per cent. This indicates that maximum extraction at the above fineness of grind is obtained in less than 24 hours. Twenty-four hours' cyanidation of the pulp ground to 92·0 per cent −325 mesh gave a tailing assaying 0·025 ounce of gold, a recovery of 95·6 per cent. The cyanide consumption was 1·0 pound NaCN per ton of ore.

The settling tests showed that the pulp ground to a high degree of fineness settled very slowly. This indicates the need of a large thickener area in mill practice.

Cyaniding the ore for 24 hours ground to 72·0 per cent −200 mesh, desliming the cyanide tailing, and regrinding and cyaniding the sand for 16 hours gave an overall recovery of 95·6 per cent; the calculated value of the final tailing was 0·025 ounce of gold per ton. For the same gold extraction, this method of treatment would require less thickener area than the straight-cyanidation of the feed ground to 92·0 per cent −325 mesh. The cyanide consumption is slightly over 1·0 pound NaCN per ton of ore.

Ore Dressing and Metallurgical Investigation No. 753

CINNABAR ORE FROM THE YALAKOM QUICKSILVER CLAIM, LILLOOET MINING DIVISION, BRITISH COLUMBIA

Shipment. A 50-pound shipment of cinnabar ore from the Yalakom Quicksilver Claim was received on August 2, 1938. The material was submitted by George L. MacInnes, 40 Williams Building, 413 Granville Street, Vancouver, British Columbia.

Location of the Property. The property is in the Bridge River area, situated 30 miles out of Lillooet. It is on the westerly side of the Yalakom River at the junction of Shulaps Creek.

Characteristics of the Ore. Hand specimens of the sample show a gangue composed of a rather fine-textured grey rock, which is crossed by numerous small veins of white carbonate, probably largely calcite. The whole is deeply stained with iron oxides, evidence of surface oxidation. The ore is only sparingly mineralized, the most prominent metallic mineral being cinnabar, which occurs as patches and veinlets along the calcite stringers, and also as veinlets cutting the grey rock. Pyrite is rarely seen as tiny disseminated grains.

Sampling and Assaying. After crushing, cutting and grinding by the standard method, a sample was obtained which assayed as follows:—

Gold	Trace
Silver	0·43 oz./ton
Mercury	0·44 per cent
Iron	6·78 "
Sulphur	0·13 "
Arsenic	Trace
Copper	Trace
Lead	Trace
CaO	7·93 per cent
Insoluble	56·74 "

Experimental Tests

The test work consisted of concentration of the cinnabar ore by flotation.

Test No. 1

A representative sample was ground 65·8 per cent through 200 mesh in a ball mill. The following reagents and amounts were used:—

Reagents to Ball Mill:

	Lb./ton
Sodium silicate	1·5
Sodium cyanide	0·05

Reagents to Flotation Cell:

	Lb./ton
Concentrate Float—	
Sodium Aerofloat	0·08
Copper sulphate	0·10
Cresylic acid	0·30
Pine oil	0·09
Middling Float—	
Sodium Aerofloat	0·07

Results:

Products	Weight, per cent	Mercury, per cent		Iron, per cent	Lime, per cent	Insoluble, per cent	Ratio of concentration
		Assay	Distribution				
Feed	100·00	0·46	100·00				
Concentrate	2·32	18·76	95·48	6·52	5·40	43·24	43·1 : 1
Middling	2·36	0·47	2·44				
Tailing	95·32	0·01	2·08				

Test No. 2

In this test, a sample of ore was ground 49·0 per cent through 200 mesh.

Reagents to Ball Mill:

	Lb./ton
Sodium silicate	1·5

Reagents to Flotation Cell:

Concentrate Float—	
Sodium Aerofloat	0·10
Copper sulphate	0·10
Cresylic acid	0·30
Pine oil	0·05
Middling Float—	
Sodium Aerofloat	0·10

Results:

Products	Weight, per cent	Mercury, per cent		Iron, per cent	Lime, per cent	Insoluble, per cent	Ratio of concentration
		Assay	Distribution				
Feed	100·00	0·44	100·00				
Concentrate	1·04	40·30	94·76	4·40	3·60	25·70	96 : 1
Middling	1·61	0·84	3·05				
Tailing	97·35	0·01	2·19				

Flotation concentrate assayed 0·105 ounce of silver and 0·015 ounce of gold per ton.

Tests Nos. 3 and 4

During grinding, the pulp slimes very readily. In these tests Reagents Nos. 632 and 637 have been tried. These reagents have been developed by the North American Cyanamid, Limited, for depressing carbonaceous and other slime gangue minerals.

The fineness of grind in these tests was similar to that of Test No. 2.

Reagents to Ball Mill:

	Test No. 3, lb./ton	Test No. 4, lb./ton
Sodium silicate	1·5	1·5

Reagents to Flotation Cell:

	Test No. 3, lb./ton	Test No. 4, lb./ton
Concentrate Float—		
Reagent No. 632	0·5	......
Reagent No. 637	......	0·5
Sodium Aerofloat	0·10	0·10
Copper sulphate	0·10	0·10
Cresylic acid	0·30	0·30
Pine oil	0·05	0·05
Middling Float—		
Sodium Aerofloat	0·10	0·10

Results:

Test No.	Product	Weight, per cent	Mercury, per cent		Iron, per cent	Insoluble, per cent	Ratio of concentration
			Assay	Distribution			
3	Feed	100·00	0·41	100·00			
	Concentrate	1·00	36·80	89·34	5·00	27·28	100 : 1
	Middling	1·13	1·28	3·52			
	Tailing	97·87	0·03	7·14			
4	Feed	100·00	0·40	100·00			
	Concentrate	0·91	38·20	86·70	5·10	26·04	110 : 1
	Middling	1·57	1·53	5·99			
	Tailing	97·52	0·03	7·31			

Reagents Nos. 632 and 637 showed no depressing effect on the slime gangue minerals. The recoveries were not so high as in Test No. 2; the flotation tailings assayed 0·03 per cent of mercury.

SUMMARY AND CONCLUSIONS

The ore is amenable to concentration by flotation. A concentrate was obtained assaying 40·30 per cent of mercury, a recovery of 94·76 per cent. The flotation tailing assayed 0·01 per cent. Some of the mercury in the middling product would be recovered in mill practice. This would raise the recovery by 1 or 2 per cent.

North American Cyanamid Reagents Nos. 632 and 637 showed no depressing effect on the slime gangue. These reagents lowered the recoveries. The flotation tailing assayed 0·03 per cent of mercury.

The sample of ore assayed 0·43 ounce of silver. The concentrate of Test No. 2 assayed 0·105 ounce of silver and 0·015 ounce of gold. These values are too low for economical extraction of the precious metals.

Sulphides of iron float readily, and when present in mercury ores, lower the grade of concentrates. Sodium cyanide depresses the pyrite, but has also a slight depressing effect on the mercury minerals. The sample of ore tested was highly oxidized; the sulphides of iron had been weathered to oxides, and so no difficulty from pyrite was encountered.

Ore Dressing and Metallurgical Investigation No. 754

GOLD ORE FROM THE CHESTERVILLE LARDER LAKE GOLD MINING CO., LTD., LARDER LAKE DISTRICT, ONTARIO

Shipment. A shipment of 18 bags of ore, weighing 1,390 pounds, was received on July 8, 1938. It was stated to have been taken from claim H.F. 404 in McGarry Township, Larder Lake District, Ontario.

The shipment was submitted by L. T. Postle, Manager, Chesterville Larder Lake Gold Mining Co., Ltd., Cheminis, Ontario.

Purpose of the Investigation. The investigation was to determine the recovery of gold by various methods of treatment.

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

The *gangue* is composed largely of a grey, highly silicified chloritic and somewhat feldspathic rock, which contains rather abundant disseminated carbonate. Tests for iron failed to reveal its presence in the carbonate, which appears to be mostly calcite. This rock is cut by veins of smoky quartz.

Pyrite is the only abundant *metallic mineral.* It occurs almost wholly in the silicified rock as coarse to fine disseminated irregular grains. The pyrite contains inclusions of gangue, and rare small inclusions of chalcopyrite and pyrrhotite. Magnetite is present in minor quantity as disseminated grains, most of which show considerable alteration to an undetermined mineral, possibly leucoxene. Chalcopyrite is rare as small irregular grains in the gangue, usually associated with pyrite, and also as the rare tiny grains in pyrite already referred to. Pyrrhotite occurs only as tiny inclusions in pyrite, the total quantity being negligible.

No native gold, or minerals that might be gold tellurides, are visible in the polished sections.

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

Gold	0·113	oz./ton
Silver	0·02	"
Copper	0·02	per cent
Arsenic	0·05	"
Sulphur	1·71	"
Iron	7·03	"
Pyrrhotite	Trace	
Lead	Nil	
Zinc	Nil	
Antimony	Nil	

Experimental Tests

The response of the ore to amalgamation, cyanidation, table concentration, and flotation, both separately and in combination, was determined.

Straight cyanidation at a grind of 76 per cent −200 mesh gave an extraction of 91·1 per cent after 8 hours' agitation.

An extraction of 89·9 per cent was obtained by flotation followed by cyanidation of the concentrate.

Four to eight hours' cyanidation of the ore ground 50 to 60 per cent −200 mesh, followed by desliming the tailing and regrinding and recyanidation of the sand portion, gave 95·5 per cent extraction.

Section I

CYANIDATION

Test No. 1

A sample of ore was ground to 85 per cent −200 mesh in cyanide solution, 1·0 pound of sodium cyanide per ton, dilution 4 parts solid to 3 parts solution.

The ground pulp was agitated for 24 hours at a dilution of 1 part solid to 1·5 parts of solution (1·0 pound of sodium cyanide per ton). Lime was used to give protective alkalinity to the solution.

Results:

Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton of ore		Final titration, lb./ton of solution	
Feed	Tailing		NaCN	CaO	NaCN	CaO
0·113	0·015	86·73	0·53	7·70	0·66	0·2

Test No. 2: Time of Agitation

To determine the time of agitation necessary to obtain a minimum tailing from ore ground 76 per cent −200 mesh, samples were agitated for different periods of time. The same conditions as those of Test No. 1 were maintained.

Results:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton of ore		Final titration, lb./ton of solution	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
3......	0·113	0·015	86·73	0·16	2·92	0·66	0·15
4......	0·113	0·015	86·73	0·16	2·86	0·66	0·18
6......	0·113	0·015	86·73	0·22	4·83	0·63	0·25
8......	0·113	0·01	91·15	0·34	4·90	0·55	0·25
16......	0·113	0·01	91·15	0·34	5·10	0·55	0·20
24......	0·113	0·01	91·15	0·30	5·20	0·70	0·25
30......	0·113	0·01	91·15	0·30	7·00	0·70	0·15
48......	0·113	0·01	91·15	0·40	7·20	0·80	0·25

These results indicate that 8-hour agitation is sufficient to obtain a minimum tailing. When compared with the results of Test No. 1, it is apparent that very fine grinding is not required.

Test No. 3

A sample of ore was ground in cyanide solution to 85 per cent −200 mesh and passed through a hydraulic trap. No free gold was observed in the concentrate.

The concentrate was amalgamated. The residue was returned to the trap tailing, which was filtered and repulped in cyanide solution (1·0 pound of sodium cyanide per ton strength). The pulp was agitated for 24 hours at a dilution of 1 : 1·5.

Results:

Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton of ore		Final titration, lb./ton of solution	
Feed	Tailing		NaCN	CaO	NaCN	CaO
0·113	0·01	91·15	0·67	5·73	0·63	0·20

As a similar tailing was obtained in Test No. 2 without amalgamation and at a coarser grind, it is concluded that the gold in the tailing was not liberated.

Test No. 4: Regrinding and Recyanidation of the Sand Portion of a Cyanide Tailing

Desliming at Different Grinds

Samples of ore were ground for different times and agitated for 4 hours in a 1·0 pound of sodium cyanide per ton solution at a dilution of 1 : 1·5. The tailings were then deslimed and the products assayed.

Results:

Screen Test on Cyanide Tailing:

Mesh	Weight, per cent				
	Test No. 4A	Test No. 4B	Test No. 4C	Test No. 4D	Test No. 4E
+ 48	5·8	0·6	......	......	......
− 48+ 65	13·0	4·7	1·6	......	......
− 65+100	18·0	13·7	9·0	0·8	......
−100+150	15·2	19·3	16·1	6·0	2·0
−150+200	8·0	13·8	16·7	11·8	7·2
−200	40·0	47·9	56·6	81·4	90·8
	100·0	100·0	100·0	100·0	100·0

Extraction by Cyanidation after 4 Hours:

Test No.	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton of ore		Final titration lb./ton of solution	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
4A......	0·113	0·035	69·03	0·5	1·8	0·8	0·15
4B......	0·113	0·025	77·88	0·5	1·8	0·8	0·15
4C......	0·113	0·02	82·30	0·5	1·85	0·8	0·10
4D.....	0·113	0·01	91·15	0·5	1·95	0·8	0·04
4E......	0·113	0·01	91·15	0·6	2·00	0·7	0·02

Desliming the Cyanide Tailing:

Test No.	Grind, per cent −200 mesh	Weight, per cent			Assay, Au, oz./ton			Distribution of gold, per cent			Ratio of concentration, sand
		Feed	Sand	Slime	Feed	Sand	Slime	Feed	Sand	Slime	
4A..........	40	100·0	71·2	28·8	0·035	0·045	0·01	100·0	91·8	8·2	1·4 : 1
4B..........	48	100·0	67·6	32·4	0·025	0·035	0·005	100·0	93·7	6·3	1·5 : 1
4C..........	57	100·0	58·9	41·1	0·02	0·03	0·005	100·0	89·4	10·6	1·7 : 1
4D..........	81	100·0	49·5	50·5	0·01	0·02	0·005	100·0	79·6	20·4	2·0 : 1
4E..........	91	100·0	37·5	62·5	0·01	0·02	0·005	100·0	70·7	29·3	2·7 : 1

In a test similar to Test No. 4 the sand was reground and cyanided for different periods of time. With a 73 per cent −200-mesh grind and 4-hour agitation these sands assayed 0·02 ounce of gold per ton. At an 80 per cent −200 mesh grind and 24-hour agitation the values were reduced to 0·005 ounce of gold per ton.

The results indicate that with an initial grind of 56·6 per cent −200 mesh, 82 per cent extraction is obtained within 4 hours.

Forty-one per cent of the tailing, with an assay of 0·005 ounce of gold per ton, can be discarded.

Eighty-nine per cent of the gold not extracted by the 4-hour agitation is found in the sand portion. By regrinding and cyaniding this sand, which constitutes 59 per cent of the weight of original feed, 83 per cent of the contained gold can be recovered, or 13·2 per cent of the gold in the feed. This gives by calculation an overall extraction of 95·5 per cent of the gold.

It is apparent that the reground sand requires somewhat less than 24 hours' agitation to obtain the minimum tailing.

Test No. 5

This was made to note the extraction obtained by grinding the ore to 70 per cent −200 mesh, concentrating and regrinding the sulphides, and cyaniding for periods of 8 and 24 hours.

A sample of ore was ground in cyanide solution, 1·0 pound of sodium cyanide per ton, and concentrated on a Wilfley table. About 10 per cent of the weight was recovered as a concentrate. This concentrate was reground and returned to the table tailing for further cyanidation.

This pulp was agitated with a 1·0 pound of sodium cyanide per ton solution at a dilution of 1:1·5 for 8 and 24 hours. A screen test of the cyanide tailing showed the final grind to be 85 per cent −200 mesh.

Results:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents, consumed lb./ton ore		Final titration, lb./ton solution	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
Grind......	0·25	0·113	0·08	29·2	0·3	2·89	0·4	0·15
5A.........	8·0	0·113	0·01	91·15	1·2	6·1	0·5	0·16
5B.........	24·0	0·113	0·01	91·15	1·2	6·1	0·5	0·10

It is apparent that no increase in extraction is obtained by regrinding the sulphides. The gold in the tailing is either fine gold or pyrite containing gold that escapes table concentration or is enclosed in the gangue.

Test No. 6

This was to determine the extraction that could be obtained from the sulphides and from the gangue.

A sample of ore was ground in water to 39 per cent −200 mesh and concentrated on a Wilfley table.

The table tailing was filtered and a portion was repulped in cyanide solution, 1·0 pound of sodium cyanide per ton, dilution 1 : 1·5, and agitated for 24 hours. A second portion was reground in cyanide solution to 80 per cent −200 mesh and cyanided, as above, for 24 hours.

The table concentrate was reground in cyanide solution to 92 per cent −325 mesh and split in two parts, one of which was agitated for 24 hours and the other for 48 hours. The solution contained 3·0 pounds of sodium cyanide per ton and the dilution was 1 part solid to 3 parts of solution.

Results:

Table Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, of gold, per cent	Ratio of concentration
Feed..	100·0	0·113	100·0	
Table concentrate..............................	10·0	0·77	68·1	10 : 1
Table tailing..................................	90·0	0·04	31·9	

Cyanidation of Table Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton tailing		Final titration, lb./ton of solution	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
6A.........	24	0·04	0·01	75·0	0·30	3·7	0·6	0·30
6B (regrind)	24	0·04	0·005	87·5	0·35	3·7	0·6	0·22

Cyanidation of Table Concentrate:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton concentrate		Final titration, lb./ton of solution	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
6C.........	24	0·77	0·03	96·10	2·3	23·5	3·0	0·9
6D.........	48	0·77	0·025	96·75	3·0	24·2	3·1	0·6

Summary. By table concentration, 68·1 per cent of the gold is found in the concentrate and 31·9 per cent is in the tailing. By cyanidation, 96·1 per cent of the gold in the concentrate, or 65·89 per cent of the total, was extracted. By regrinding and cyanidation, 87·5 per cent of the gold in the table tailing was extracted, or 27·9 per cent of the total.

This method shows an overall extraction of 93·8 per cent of the gold.

The calculated tailing contains 0·007 ounce of gold per ton.

When the table tailing is cyanided without regrinding, the total extraction drops to 89·8 per cent of the gold and the tailing assays 0·012 ounce of gold per ton.

Test No. 7

A cyanide tailing was concentrated by flotation, the concentrate being reground in cyanide solution and agitated for 48 hours. The flotation tailing was divided into sand and slime and the gold contents of these products were determined.

A sample of ore was ground to 85 per cent −200 mesh in cyanide solution and agitated for 17 hours at a dilution of 1 : 1·5 in a 1·0 pound of sodium cyanide per ton solution.

The cyanide tailing was filtered, washed, and repulped in a flotation cell. The pulp was conditioned with 2·0 pounds of soda ash, 1·0 pound of copper sulphate, and 0·2 pound amyl xanthate per ton. Then 0·1 pound of pine oil per ton was added and a concentrate was removed.

The flotation tailing was passed over a Wilfley table on which the sand and slime were separated; sampled, and assayed.

The flotation concentrate was reground in cyanide solution, 3·0 pounds of sodium cyanide per ton, to 95 per cent −325 mesh and was agitated for 48 hours at a dilution of 1 part solid to 3 parts of solution (3·0 pounds of sodium cyanide per ton).

Results:

Cyanidation of the Ore:

Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton ore		Final titration, lb./ton solution	
Feed	Tailing		NaCN	CaO	NaCN	CaO
0·113	0·012	89·38	0·60	4·85	0·8	0·1

Flotation of Cyanide Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	0·012	100·0	
Flotation concentrate	12·2	0·08	78·7	8·2 : 1
Flotation tailing	87·8	0·003	21·3	

Flotation Tailing Deslimed:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, of gold, per cent	Ratio of concentration
Feed	100·0	0·003	100·0	
Sand	53·1	0·005	85·0	
Slime	46·9	0·001	15·0	2·13 : 1

Flotation Concentrate Cyanided:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton concentrate		Final titration, lb./ton solution	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
48	0·08	0·035	56·25	8·87	41·4	2·6	0·33

Summary. An extraction of 89·4 per cent of the gold was made by cyanidation, and 78·7 per cent of that in the tailing was concentrated by flotation, leaving a flotation tailing containing 0·003 ounce of gold per ton.

By cyanidation, 56·25 per cent of the gold in the flotation concentrate was extracted or 4·7 per cent of the gold in the feed. This method gave an overall extraction of 94·1 per cent, with a cyanide consumption of 1·7 pounds of sodium cyanide per ton of feed.

It is apparent that much of the gold in the cyanide tailing is in the sulphide portion, as flotation gives a tailing of 0·003 ounce of gold per ton.

Desliming the flotation tailing showed that the sand portion contains most of the gold.

Section II

CONCENTRATION

Test No. 8

A sample of ore was ground in water with 0·5 pound of soda ash per ton to give a product 64 per cent −200 mesh.

The pulp was conditioned in a flotation cell with 1·0 pound of copper sulphate and 0·1 pound of potassium amyl xanthate per ton. Then 0·1 pound of pine oil per ton was added and a concentrate was removed. Several additions consisting of 0·05 pound of amyl xanthate per ton were added to the pulp until the froth appeared to be free from sulphides. A total of 0·3 pound amyl xanthate per ton was required.

The rougher concentrate was cleaned, yielding concentrate and middling products.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
Feed	100·00	0·113	100·00	
Flotation concentrate	9·85	1·03	89·96	10·15 : 1
Flotation middling	1·26	0·20	2·17	79·4 : 1
Flotation tailing	88·89	0·01	7·87	

Analysis of the flotation concentrate shows:

Silver	0·15 oz./ton
Arsenic	0·49 per cent
Iron	27·34 "
Sulphur	23·44 "
Graphite	0·22 "

An additional test was made in which the flotation tailing contained 0·013 ounce of gold per ton. A flotation concentrate was obtained containing 2·02 ounces of gold per ton, with a ratio of concentration of 56 : 1, and a middling containing 0·48 ounce of gold per ton, with a ratio of concentration of 13 : 1.

The flotation tailing after sampling was split on a Wilfley table into sand and slime, which assayed 0·015 and 0·01 ounce of gold per ton respectively.

Several other tests showed about the same tailing.

Test No. 9

A flotation concentrate was obtained by the methods of Test No. 8.

The concentrate was reground to 99 per cent −325 mesh in a 3·0 pound of sodium cyanide per ton solution, and was agitated for 48 hours at a dilution of 1 to 3 in a solution of the same strength as the grind.

Results:

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	0·113	100·0	
Flotation concentrate	13·5	0·77	92·3	7·4 : 1
Flotation tailing	86·5	0·01	7·7	

Cyanidation of Flotation Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton concentrate		Final titration, lb./ton solution	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
48........	0·77	0·02	97·4	7·3	26·5	2·8	0·4

Summary. A recovery of 92·3 per cent of the gold in the feed was made in the flotation concentrate, 97·4 per cent of the gold in the concentrate was extracted by cyanidation, giving an overall recovery of 89·9 per cent. The combined tailing was calculated and was found to contain 0·011 ounce of gold per ton; 0·98 pound of sodium cyanide and 3·6 pounds of lime per ton of feed were used.

Test No. 10

This was made to note the distribution of gold in the flotation tailing.

The tailing from Test No. 9, after sampling, was split into sand and slime on a Wilfley table, with 57 and 43 per cent weight respectively. Each product assayed 0·01 ounce of gold per ton.

A portion of the sand was reground to 80 per cent −200 mesh and cyanided for 20 hours in a 1·0 pound of sodium cyanide per ton solution at a dilution of 1 : 1·5. The cyanide tailing assayed a trace of gold.

This test shows that the slime portion of a flotation tailing contains 0·01 ounce of gold per ton, whereas the same material from a 4-hour agitation in cyanide solution contains 0·005 ounce of gold per ton.

It also confirms the conclusion that the sand portion on regrinding can be reduced to at least 0·005 ounce of gold per ton by cyanidation.

Test No. 11

A sample of ore was ground in water to 70 per cent −200 mesh and concentrated by a Denver Laboratory Mineral Jig. The jig tailing was deslimed and the sand was repassed twice.

The concentrate was reground to 99 per cent −325 mesh and barrel-amalgamated.

This test was run to obtain a maximum amount of jig concentrate.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Jig feed......................................	100·0	0·113	100·0	
Concentrate..................................	14·0	0·48	59·8	7·14 : 1
Sand tailing..................................	34·0	0·08	24·1	
Slime tailing.................................	52·0	0·035	16·1	

Assay of amalgamation tailing: 0·07 Au, oz./ton.

A recovery of 85·4 per cent of the gold in the concentrate was made by amalgamation, or 51·1 per cent of the gold in the feed.

Test No. 12

This was made at a coarser grind than the preceding test followed by a regrind of the sand portion of the tailing after the first pass.

A sample of ore was ground to about 60 per cent −200 mesh and passed through the jig. The tailing was deslimed and the sand was reground and passed over the jig twice, desliming after the second pass.

The jig was operated to give a high ratio of concentration.

The concentrate obtained was reground to about 99 per cent −325 mesh and was barrel-amalgamated.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Jig feed	100·0	0·113	100·0	
Concentrate	2·0	3·18	55·7	50 : 1
Sand tailing	39·9	0·06	21·2	
Slime tailing	58·1	0·045	23·1	

Assay of amalgamation tailing: 0·65 Au ,oz./ton.

A recovery of 79·6 per cent of the gold in the concentrate was made by amalgamation, or 44·3 per cent of the gold in the feed.

Test No. 13

This was to determine the settling rate of a cyanide tailing at different grinds and dilutions.

Samples of ore were ground in cyanide solution and agitated for 4 hours in a 1·0 pound of sodium cyanide per ton solution, using lime for protective alkalinity.

The pulp was transferred to a cylinder and the pulp level was noted at 5-minute intervals for one hour. The ore was ground 60 and 65 per cent −200 mesh.

Results:

Grind, 60 per cent −200 mesh			Grind, 65 per cent −200 mesh		Grind, 65 per cent −200 mesh	
Dilution, 1 : 1·23			Dilution, 1 : 1·63		Dilution, 1 : 2	
Time, minutes	Pulp level	Drop, feet	Pulp level	Drop, feet	Pulp level	Drop, feet
0	4·25	0·00	2·665	0·00	3·195	0·00
5	4·19	0·06	2·575	0·09	3·065	0·13
10	4·13	0·06	2·500	0·075	2·945	0·12
15	4·08	0·05	2·430	0·070	2·815	0·13
20	4·03	0·05	2·350	0·080	2·705	0·11
25	3·98	0·05	2·280	0·070	2·585	0·12
30	3·935	0·045	2·205	0·075	2·450	0·135
35	3·88	0·055	2·145	0·060	2·340	0·11
40	3·835	0·045	2·060	0·085	2·220	0·12
45	3·790	0·045	1·985	0·075	2·105	0·115
50	3·735	0·055	1·910	0·075	2·005	0·10
55	3·690	0·045	1·840	0·070	1·900	0·105
60	3·635	0·055	1·775	0·065	1·830	0·070
Drop per hour, feet		0·615		0·89		1·365

Titration of Solutions:

NaCN	0·9 lb./ton	0·8 lb./ton	0·7 lb./ton
CaO	0·3 "	0·33 "	0·28 "

SUMMARY AND CONCLUSIONS

About 51 per cent of the gold in the ore is recoverable by amalgamating a jig concentrate.

Ninety-one per cent extraction was obtained by straight cyanidation. When the sulphides were removed, reground, and recyanided with the main portion, no increase in extraction was noted. When, however, the concentrate was cyanided in a separate circuit (see Test No. 6) the recovery was raised to 93·8 per cent.

Cyanidation followed by regrinding and recyanidation of a flotation concentrate made from the initial cyanide tailing showed an overall extraction of 94·1 per cent.

Flotation with cyanidation of the concentrate resulted in an extraction of 89·9 per cent.

The results of the investigation show that the slime portion of a cyanide tailing contains 0·005 ounce of gold per ton after a short period of agitation, about 4 hours. The slime from a flotation tailing contains 0·01 ounce per ton, and the slime from a cyanide tailing after removing a flotation concentrate contains 0·001 ounce of gold per ton.

It is apparent that the gold contained in an 0·01 ounce per ton cyanide tailing is associated both with the sulphides and with the sand.

The highest overall extraction was obtained by following the procedure of Test No. 4, in which the tailing after a short period of agitation was classified into sand and slime, the sand being reground and recyanided. Ninety-five per cent extraction of the gold was obtained.

From 32 to 60 per cent of the daily tonnage, depending on the fineness of primary grind, can be discarded after about 4 hours' agitation. The reground sand requires somewhat less than 24 hours' agitation to obtain the minimum tailing.

The flow-sheet recommended for this class of ore is that embodied in Test No. 4.

The ore ground in cyanide solution to give a classifier overflow of between 50 to 60 per cent −200 mesh should be agitated for a period of about 4 hours and then thickened, drawing off pregnant solution. The thickener underflow should be deslimed, the slime filtered, washed, and discarded, and the sand should be reground and cyanided in barren solution for about 24 hours. The solution from this circuit can be used at the feed of the mill. The tailing from this secondary circuit should be thickened and filtered in its own section.

In no single-stage operation was a tailing lower than 0·01 ounce of gold per ton obtained.

Alternative flow-sheets yielding lower recoveries than the above are: cyanidation followed by flotation of the cyanide tailing; flotation and cyanidation of the concentrate; and cyanidation with intermediate table concentration and recyanidation.

All the methods outlined require regrind mills and their related classifiers and thickeners.

The results obtained in this investigation apply to ore of a nature similar to that represented by the sample submitted.

Ore Dressing and Metallurgical Investigation No. 755

COBALT-SILVER-NICKEL ORE FROM THE COBALT PRODUCTS, LIMITED, COBALT, ONTARIO

Shipment. A 56-pound shipment of cobalt-silver-nickel ore from the dumps of the Foster Cobalt Mining Company, Cobalt, Ontario, was received on September 6, 1938. The material was submitted by J. E. McDonough, Cobalt Products, Limited, Cobalt, Ontario.

Characteristics of the Ore. The sample of silver-bearing cobalt-nickel ore was said to have been taken from a dump that had been exposed to the atmosphere for some time, and might, therefore, be expected to have suffered alteration with the development of erythrite (cobalt bloom), the hydrous cobalt arsenate, which in fact is fairly abundant in the hand specimens examined.

Masses of hard white metallic minerals occur in the *gangue*, which consists of grey silicates with patches of pink carbonate. Tests on the hard white *metallic mineral* assemblage show it to contain abundant cobalt, nickel, iron and arsenic, and some sulphur and antimony. It is probable that cobaltite, arsenopyrite, safflorite-rammelsbergite, and possibly löllingite and skutterudite, are represented. Irregular patches and grains of galena occur both in gangue and the cobalt-nickel mineral masses, and occasional small grains of tetrahedrite are to be seen in the same relationships. Chalcopyrite is not common, but occasional small masses occur in the gangue. No native silver or silver minerals were identified. It is possible that some of the silver may be contained in the galena and tetrahedrite.

Erythrite may not be susceptible to concentration, thus high cobalt in the tailing may be expected. The grade of cobalt concentrate may be low, owing to the presence of (1) arsenic and iron, which are constituents of some cobalt minerals, and (2) other metallic minerals, such as arsenopyrite, tetrahedrite, and galena.

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

Gold	0·01 oz./ton
Silver	7·67 "
Cobalt	2·33 per cent
Nickel	0·11 "
Copper	0·33 "
Iron	7·52 "
Arsenic	7·91 "
Sulphur	0·90 "

Experimental Tests

Investigations were conducted on the ore to effect a concentration of silver and cobalt. The test work consisted of concentration by flotation and tabling.

The results of the tests were not encouraging. The grades of concentrates were low owing to the presence of sulphide minerals, such as arsenopyrite, and tetrahedrite, which floated with the cobalt and nickel minerals, and of arsenic and iron, which are constituents of some cobalt minerals.

The tailings were high, probably owing to sliming of the metallic cobalt minerals and the presence of erythrite, a very soft mineral that slimes very readily; the classifier slime (Test No. 3) assayed 1·61 per cent of cobalt-nickel. Erythrite cannot be recovered by tabling as the specific gravity is very near to that of gangue minerals; nor can it be floated with collectors suitable for sulphide minerals.

An attempt was made to float erythrite with fatty acids. The concentrate obtained assayed less than 5 per cent of cobalt-nickel, the recovery was less than 10 per cent, and the tailing was not reduced appreciably.

Rougher concentrates, assaying as high as 10 per cent cobalt-nickel, were obtained by flotation. Cleaning the rougher concentrate raised the grade to 12·24 per cent (Test No. 2-B). By tabling, a concentrate assaying 14·01 per cent of cobalt-nickel was obtained. Recoveries of 54·0 and 48·6 per cent (Tests Nos. 4-A and 4-B) were obtained by flotation, using 12 pounds of ammonium sulphide and 8·0 pounds per ton of sodium sulphide, respectively: the tailings assayed 1·30 and 1·49 per cent of cobalt-nickel. The cost of ammonium sulphide may prohibit the use of that reagent in practice.

Large amounts of sodium sulphide decrease the recovery of silver. An attempt was made to float the silver prior to recovery of cobalt (Tests Nos. 5, 6, and 7). Silver concentrates assaying 69·98, 81·80, and 72·68 ounces of silver, and 4·14, 4·66, and 8·54 per cent of cobalt, respectively, were obtained. The recoveries were 34·4, 40·8, and 58·4 per cent. These tests indicate that an appreciable amount of the silver occurs interlocked with the arsenide-sulphide minerals.

Test No. 1

A representative sample of −14-mesh material was ground in the ball mill at 57 per cent solids to 78 per cent −200 mesh. The pulp was treated by flotation, followed by tabling of the flotation tailing. The table tailing was thickened and floated for cobalt bloom.

Reagents Added:

	Lb./ton solids
To Grind:	
Soda ash	3·0
Coal-tar creosote No. 634	0·12
Flotagen	0·20
Sodium sulphide	2·0
To Flotation Cell:	
Aerofloat No. 25	0·21
Potassium amyl xanthate	0·10

Flotation of Table Tailing:

To Flotation Cell:	
Soda ash	1·60
Sodium silicate	1·60
Sodium oleate acid	0·80
Pine oil	0·10

Results:

Product	Weight, per cent	Assay		Distribution, per cent	
		Ag, oz./ton	Co-Ni, per cent	Ag	Co-Ni
Feed	100·00	7·67	2·40	100·0	100·0
Flotation concentrate (head)	8·24	54·80	8·81	58·9	30·3
Table concentrate	2·75	13·16	6·94	4·7	8·0
Table middling	1·56	7·77	3·56	1·6	2·3
Flotation concentrate (table tailing)	26·69	} 3·05	1·63	} 34·8	18·1
Flotation tailing	60·76		1·53		41·3

The flotation concentrate (head) and the table concentrate assayed 0·86 and 0·60 per cent of nickel, a recovery of 57·3 and 15·4 per cent, respectively.

Test No. 2 (A and B)

An attempt was made to float cobalt arsenate (cobalt bloom), using oleic acid as a collector. Oleic acid floats carbonates readily; copper sulphate was added to depress the carbonates.

Representative samples of ore were ground in the ball mill to 78 per cent −200 mesh.

Reagents Added:

—	Lb./ton of solids	
	Test No. 2-A	Test No. 2-B
To Grind:		
Sodium silicate	2·0	....
Soda ash	....	3·0
Sodium sulphide	2·0	2·0
Denver Sulphidizer	0·80	0·80
Coal-tar creosote No. 634	0·12	0·12
To Flotation Cell:		
1st concentrate float—		
Aerofloat No. 25	0·21	0·21
Potassium amyl xanthate	0·10	0·10
2nd concentrate float—		
Sodium silicate	....	2·00
Copper sulphate	0·40	0·40
Oleic acid	0·06	0·12
Cresylic acid	0·06	0·06

The first concentrate of Test No. 2-B was refloated; no reagents were added.

Results:

Test No.	Product	Weight, per cent	Assay		Distribution, per cent		CaO, per cent
			Ag, oz./ton	Co-Ni, per cent	Ag	Co-Ni	
2-A	Feed	100·00	7·73	2·72	100·0	100·0	....
	1st concentrate	8·29	52·37	8·31	56·2	25·3	6·91
	2nd concentrate	5·90	14·20	3·93	10·8	8·5	15·69
	Tailing	85·81	2·97	2·10	33·0	66·2	....
2-B	Feed	100·00	....	2·71	....	100·0	....
	Cleaner concentrate (1st concentrate)	6·48	59·90	12·24	....	29·3	....
	Cleaner tailing (1st concentrate)	6·10	14·95	4·73	....	10·7	....
	2nd concentrate	4·03	....	4·59	....	6·8	18·38
	Tailing	83·39	....	1·73	....	53·2	....

The cleaner concentrate, Test No. 2-B, assayed 1·09 per cent of nickel. Oleic acid gave slimy concentrates high in calcite and low in cobalt.

Test No. 3

A sample of ore was ground to 78 per cent −200 mesh, with the following reagents:

	Lb./ton
To Grind:	
Soda ash	3·0
Coal-tar creosote No. 634	0·12
Sodium sulphide	4·00
Denver Sulphidizer	1·60
To Flotation Cell:	
Aerofloat No. 25	0·28
Potassium amyl xanthate	0·20

The flotation tailing was deslimed by hydraulic classifier, and the sand was concentrated by tabling.

Results:

Products	Weight, per cent	Cobalt-nickel, per cent	
		Assay	Distribution
Feed	100·00	2·24	100·0
Flotation concentrate	9·84	10·00	43·9
Table concentrate	2·42	2·96	3·2
Table middling	26·70	0·98	11·7
Table tailing	30·26	1·40	19·0
Classifier slime	30·78	1·61	22·2

The high cobalt content in the slime may be due to cobalt bloom and to the sliming of the metallic minerals.

Test No. 4 (A and B)

Representative samples of ore were deslimed by decantation to reduce sliming due to grinding. The sands were ground with flotation reagents to about 66 per cent −200 mesh. The ground sands and the slimes were combined for flotation.

Reagents Added:

—	Lb./ton ore	
	Test No. 4-A	Test No. 4-B
To Grind:		
Sodium silicate	2·0	2·0
Ammonium sulphide	12·0	
Sodium sulphide	....	8·0
Coal-tar creosote No. 634	0·25	0·62
Aerofloat No. 31	0·28	0·28
To Flotation Cell:		
Potassium amyl xanthate	0·40	0·40
Coal-tar creosote No. 634	0·55	
Pine oil	0·14	

Note.—Pine oil produced a froth of fine bubbles and wet appearance. To attain a dry, firm froth, coal-tar creosote No. 634 had to be added to the flotation cell.

The flotation tailings were concentrated by tabling.

Results:

Test No.	Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
			Ag, oz./ton	Co-Ni, per cent	Ag	Co-Ni	
4-A	Feed	100·00	7·70	2·27	100·0	100·0	
	Flotation concentrate	12·74	32·22	9·64	53·4	54·0	7·85 : 1
	Table concentrate	2·31	20·48	2·03	6·1	2·1	
	Table middling	22·84	2·94	0·84	8·7	8·4	
	Table tailing	62·11	3·94	1·30	31·8	35·5	
4-B	Feed	100·00	7·67	2·30	100 0	100·0	
	Flotation concentrate	11·38	29·30	9·82	43·5	48·6	8·78 : 1
	Table concentrate	2·35	18·08	2·40	5·5	2·4	
	Table middling	25·15	2·72	0·86	8·9	9·4	
	Table tailing	61·12	5·28	1·49	42·1	39·6	

In Test No. 4-B, the silver recovery was decreased owing to the depressing action of sodium sulphide on the silver minerals.

Tests Nos. 5, 6, and 7

Attempts were made to float the silver prior to recovery of cobalt.

After desliming, the sands were ground as in Test No. 4. The reground sands and slimes were combined for flotation.

Test No. 5

Reagents:

	Lb./ton ore
To Grind:	
Soda ash	3·0
Coal-tar creosote No. 634	0·62
Aerofloat No. 31	0·14
To Flotation Cell:	
Aerofloat No. 31	0·10

The flotation tailing was deslimed by hydraulic classification; the sand was tabled; and the classifier slime and the table tailing and middling were combined for flotation.

Reagents to Flotation Cell:

	Lb./ton ore
Conditioning:	
Soda ash	3·0
Sodium sulphide	4·0
Float:	
Copper sulphate	1·0
Potassium amyl xanthate	0·40
Pine oil	0·14

Results:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Ag, oz./ton	Co-Ni, per cent	Ag	Co-Ni	
Feed	100·00	7·71	2·23	100·0	100·0	
Silver concentrate	3·79	69·98	4·14	34·4	7·0	26·4 : 1
Table concentrate	2·90	32·04	12·56	12·0	16·3	11·7 : 1 (Table and Cobalt concentrates)
Cobalt concentrate	5·63	26·30	7·16	19·2	18·1	
Tailing	87·68	3·02	1·49	34·4	58·6	

Test No. 6

Reagents:

	Lb./ton ore
To Grind:	
Sodium hydroxide	2·0
Coal-tar creosote No. 634	0·62
To Flotation Cell:	
Silver float:	
Potassium amyl xanthate	0·20
Conditioner:	
Sodium sulphide	8·0
Cobalt float:	
Copper sulphate	1·0
Potassium amyl xanthate	0·40
Pine oil	0·14

The flotation tailing was tabled.

Results:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Ag, oz./ton	Co-Ni, per cent	Ag	Co-Ni	
Feed	100·00	7·69	2·21	100·0	100·0	
Silver concentrate	3·83	81·80	4·66	40·8	8·0	26·1 : 1
Cobalt concentrate	4·73	11·46	10·18	7·1	21·8	} 14·2 : 1
Table concentrate	2·31	22·78	14·01	6·8	14·6	
Table tailing	89·13	3·91	1·38	45·3	55·6	

Test No. 7

Reagents:

	Lb./ton ore
To Grind:	
Sodium hydroxide	0·20
Coal-tar creosote No. 634	0·31
To Flotation Cell:	
Silver float:	
Sodium hydroxide	3·0
Potassium amyl xanthate	0·20
Pine oil	0·14

The flotation tailing was tabled.

Results:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Ag, oz./ton	Co-Ni, per cent	Ag	Co-Ni	
Feed	100·00	7·69	2·23	100·0	100·0	
Silver concentrate	6·18	72·68	8·54	58·4	23·7	16·2 : 1
Table concentrate	2·37	9·30	13·80	2·9	16·9	42·2 : 1
Table tailing	91·45	3·26	1·45	38·7	59·4	

The silver concentrate was high in cobalt-nickel. This may account for higher silver recovery than in Test No. 6.

CONCLUSIONS

High recoveries of silver and cobalt cannot be expected in mill practice. The ore contains an appreciable amount of erythrite (cobalt bloom), which cannot be recovered by gravity concentration as its specific gravity is near that of gangue minerals; also the mineral is a cobalt arsenate and cannot be floated with collectors used for sulphide minerals. The fatty acid collectors used for floating oxide minerals float the carbonates very readily, thus lowering the grade of the concentrate.

A recovery of 54·0 per cent of cobalt-nickel was obtained with 12·0 pounds per ton of ammonium sulphide, but the cost of this reagent may prohibit its use in mill practice. Eight pounds of sodium sulphide per ton gave a cobalt-nickel recovery of 48·6 per cent; but this amount of reagent depressed the silver appreciably. The cobalt concentrates were not high grade. A cleaner concentrate of 12·24 per cent cobalt-nickel and a table concentrate assaying 14·01 per cent were obtained.

Silver flotation gave low-grade concentrates and low recoveries. An appreciable amount of silver may be contained in the sulphide minerals which are floated with the cobalt minerals. From a silver concentrate assaying 81·80 ounces of silver per ton and 4·66 per cent of cobalt-nickel, a recovery of 40·8 per cent was obtained. A concentrate assaying 8·54 per cent of cobalt-nickel contained 58·4 per cent of silver.

Ore Dressing and Metallurgical Investigation No. 756

SILVER-LEAD-TUNGSTEN ORE FROM THE REGAL SILVER PROPERTY, NEAR REVELSTOKE, BRITISH COLUMBIA

Shipment. A 700-pound sample of silver-lead-tungsten ore was received on September 8, 1938, from the Regal Silver property[1], Albert Canyon, near Revelstoke, B.C. The shipment was submitted by A. S. MacCulloch, 555 Howe Street, Vancouver, B.C.

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

The *gangue* consists of milky-white quartz and a dense, siliceous, slightly schistose, very dark grey rock with small patches of a soft, black material. Microchemical tests indicate that the latter is a slightly ferruginous carbonate, and the rock may be a silicified schist. The dark colour of both materials is probably due to finely divided graphite as this mineral is visible in the hand specimen.

Metallic minerals predominate over gangue in the sections examined. Pyrite is by far the most abundant, largely as coarse irregular grains and small masses in gangue; an extremely small proportion occurs as small disseminated grains. It is somewhat fractured and veined with quartz, and contains numerous inclusions of gangue. A considerable quantity of galena is present as small masses, irregular grains and veinlets in gangue, frequently associated with patches of dark carbonate. In places it is in contact with pyrite and is sometimes included by the latter mineral. A very light-coloured sphalerite is common as small masses in quartz; it also occurs as occasional inclusions in pyrite, and as rims around the edges of grains and masses of this mineral. Rare, small grains of pyrrhotite are visible in pyrite but its total quantity is negligible. The presence of slight stains of iron oxides around the edges of some pyrite masses indicates that oxidation has proceeded to some extent.

Although no tungsten minerals, such as wolframite, were observed under the microscope, light brown scheelite is abundant in some hand specimens. It is present in quartz as heavy, coarse masses up to 2 inches or more in size.

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

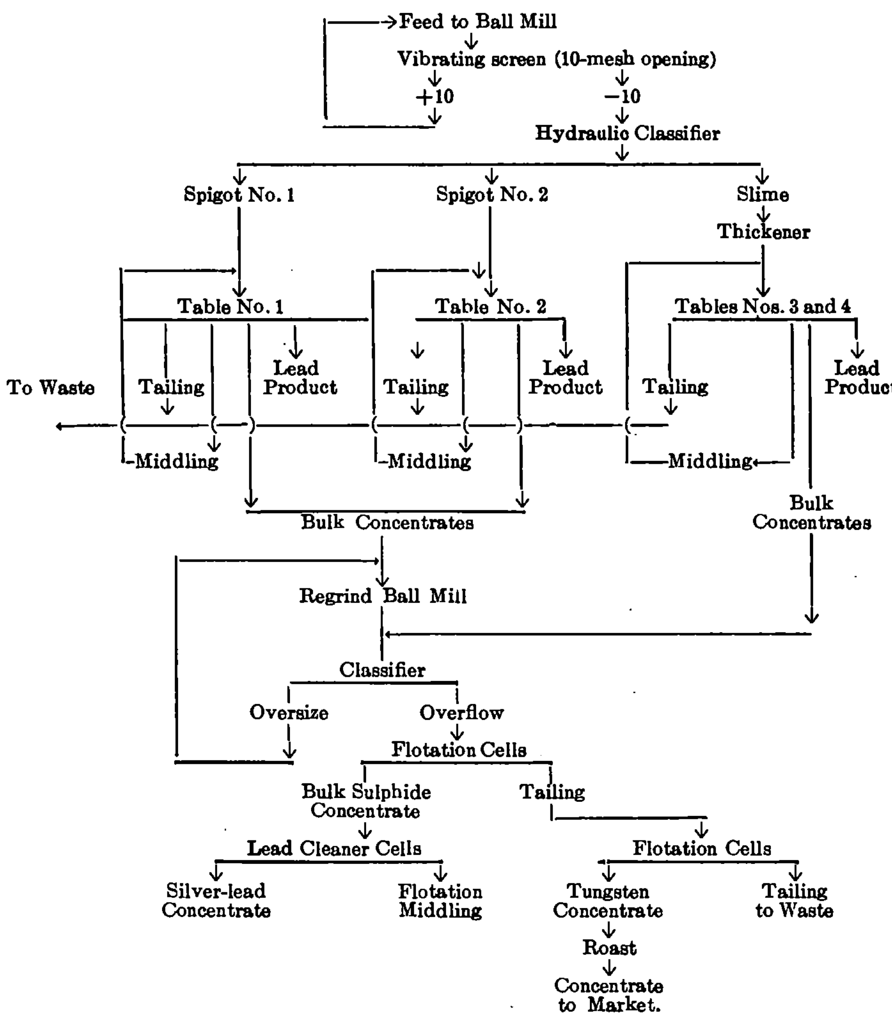
Gold	Trace
Silver	0·955 oz./ton
Lead	0·84 per cent
Zinc	0·36 "
Tungstic oxide	2·42 "
Iron	15·68 "
Sulphur	15·48 "
Graphite	1·42 "

[1]Results of concentration tests on previous sample of ore from this property are given in Invest. Ore Dress. and Met. 1930, Mines Branch, Dept. of Mines. Rept. 724. (Invest. No. 366) pp. 112-115.

Purpose of Investigation. The object was to develop a method of concentrating the silver, lead, and tungsten minerals of the ore with a minimum loss of tungsten due to sliming.

Results of Investigations. The results indicate that by employing a coarse primary grind and using a classified feed to the tables, a lead product and a bulk sulphide and tungsten concentrate can be made. The bulk concentrate is reground and a bulk sulphide concentrate obtained by flotation. This is cleaned to obtain a lead-silver concentrate. The flotation tailing is floated and a tungsten concentrate obtained.

A suggested flow-sheet covering these steps is as follows:



Experimental Tests

Test No. 1

A sample, 3,000 grammes, of ore ground to 12 per cent +20 mesh and 20 per cent −100 mesh was fed to a small Wilfley table and a bulk concentrate of sulphides and scheelite was taken off. The table concentrate was reground to a fineness of 69·6 per cent −200 mesh and a bulk sulphide concentrate was obtained by flotation and after conditioning the flotation tailing a tungsten concentrate was floated off.

The feed to the table was not classified, which would account for the high tailing loss. The results are as follows:

Product	Weight, per cent	Assay				Distribution, per cent			Ratio of concentration
		Ag, oz./ton	Per cent						
			Pb	Zn	WO_3	Ag	Pb	WO_3	
Feed*	100·00	0·86	1·00	......	2·76	100·0	100·0	100·0	
Flotation concentrate	26·09	2·84	3·42	0·86	0·05	86·6	89·0	0·5	3·8 : 1
Tungsten concentrate	4·27	0·19	......	......	52·15	0·9	....	80·8	23·4 : 1
Flotation tailing	13·79	0·22	0·03	Nil	1·34	3·5	0·4	6·7	
Table middling	8·32	0·35	0·13	Nil	0·43	3·4	1·1	1·3	
Table tailing	47·53	0·10	0·20	Nil	0·62	5·6	9·5	10·7	

* Feed assays calculated from products.

Reagents used in Sulphide Flotation:

	Lb./ton concentrate
Soda ash	4·0
Aerofloat No. 31	0·046
Copper sulphate	0·65
Potassium amyl xanthate	0·26
Pine oil	0·121
Cresylic acid	0·042

Reagents used for Tungsten Concentrate:

	Lb./ton
Sodium silicate	1·3
Oleic acid	0·42
Pine oil	0·08

Test No. 2

The ore sample was first conditioned in a high-density pulp and a bulk sulphide concentrate was made. The flotation tailing was classified and tabled on a small laboratory Wilfley table. The results were unsatisfactory. Only 29 per cent of the tungsten was obtained in the table concentrate, whereas 70 per cent reported in the table middling and tailing. The silver loss was high.

Test No. 3

In view of the encouraging results obtained in Test No. 1, a larger sample was taken and in order to provide for a sized feed to the table. the ore was screened dry to give five products.

Each product was tabled separately to give a concentrate, middling, and tailing. The concentrates from the three coarsest products were reground before flotation. Those from the two finer products were floated without regrinding.

The ore used was 10,255 grammes.

In flotation a bulk sulphide concentrate was taken off and cleaned to give a lead-silver concentrate. The tailing was conditioned and a tungsten concentrate floated off.

The −14-mesh feed was screened dry to give a sized feed as follows:

Mesh	Weight, per cent
+ 20	11·9
− 20+ 35	38·2
− 35+ 65	25·1
− 65+100	4·4
−100	20·4
	100·0

The figures tabulated below show the results obtained on the three coarser products and the two finer products:

Results:

+65-mesh Products

Product	Weight, per cent	Assay				Distribution, per cent			Ratio of concentration
		Ag, oz./ton	Per cent						
			Pb	Zn	WO_3	Ag	Pb	WO_3	
Feed*	100·00	0·74	0·97	......	2·50	100·0	100·0	100·0	
Tungsten concentrate	4·77	0·06	......	......	46·23	0·4	....	88·1	20·9 : 1
Lead concentrate	6·55	6·94	9·71	1·38	0·47	61·3	65·5	1·2	15·3 : 1
Flotation middling	18·50	0·94	1·07	0·21	Tr.	23·4	20·5	0·0	
Flotation tailing	4·42	0·06	0·18	......	3·61	0·4	0·8	6·4	
Table middling	14·65	0·25	0·18	......	0·74	4·9	2·7	4·3	
Table tailing	51·11	0·14	0·20	......	Tr.	9·6	10·5	0·0	

Sulphur in tungsten concentrate 10·37 per cent.
* Calculated from assays of products.

−65 and −100-mesh Products

Product	Weight, per cent	Assay				Distribution, per cent			Ratio of concentration
		Ag, oz./ton	Per cent						
			Pb	Zn	WO_3	Ag	Pb	WO_3	
Feed*	100·00	1·53	1·43	......	2·95	100·0	100·0	100·0	
Tungsten concentrate	4·23	0·09	......	......	45·26	0·2	....	65·0	23·6 : 1
Lead concentrate	1·97	55·44	65·71	0·26	......	71·6	90·3	....	50·8 : 1
Flotation middling	22·93	0·18	0·20	0·97	......	2·7	3·2	....	
Flotation tailing	13·16	0·06	0·05	......	2·45	0·5	0·5	10·9	
Table slime	57·71	0·66	0·15	......	1·23	25·0	6·0	24·1	

Sulphur in tungsten concentrate 0·80 per cent.
* Calculated from assays of products.

The grade of the lead-silver concentrate as shown in the first table is low. This is due to insufficient lime and depressants being used in the cleaning of the bulk sulphide concentrate.

The combined tables indicate the following overall recoveries.

	Per cent
Tungsten	81·6
Lead	88·5
Silver	80·4

The loss of tungsten as WO_3 in the slime is 6·7 per cent.

The reagents used, in pounds per ton of concentrate, are as follows:

—	Coarse products	Fine products
Grinding:		
Soda ash	2·4	6·0*
Aerofloat No. 25	0·056	0·14*
Bulk Sulphide Flotation:		
Copper sulphate	0·8	2·0
Potassium amyl xanthate	0·16	0·4
Pine oil	0·025	0·02
Cresylic acid	0·051	0·05
Lead Cleaner Concentrate:		
Lime	1·6	12·0
Sodium cyanide	0·08	0·4
Zinc sulphate	1·6	4·0
Butyl xanthate	....	0·2
Tungsten Concentrate:		
Sodium silicate	0·4	0·8
Oleic acid	0·64	1·28
Pine oil	....	0·12

* Added direct to cell.

Desulphurization of Tungsten Concentrate by Roasting:

A portion of tungsten concentrate carrying 46·23 per cent of tungstic oxide and 10·37 per cent of sulphur was roasted in an open dish at a temperature of 550° C.

The calcine was cleaned by re-floating. The final concentrate had the following analysis:

	Per cent
WO_3	55·90
Sulphur	0·32

CONCLUSIONS

A coarse primary grind is clearly essential to keep the loss of tungsten low. The products, flotation middling and table middling, would in plant operation be carried as a circulating load as shown on the flow-sheet.

It is difficult on a small laboratory table to make a clean separation between several minerals of high specific gravity. This difficulty is largely overcome on full-size tables, and so it is possible that better recoveries than those indicated could be made in plant practice.

The high sulphide content of the ore will probably account for sulphur in the tungsten concentrate. This can be reduced by roasting.

Ore Dressing and Metallurgical Investigation No. 757

PLACER MATERIAL FROM RED CEDAR LAKE GOLD MINES, LIMITED
CRILLY, ONTARIO

Shipment. Four bags of placer material, weighing 219 pounds, were received on October 31, 1938, from W. S. Miners, Secretary-Treasurer, Red Cedar Lake Gold Mines, Limited, 133 May Street North, Fort William, Ontario.

Location of the Property. The property of the Red Cedar Lake Gold Mines, Limited is situated adjacent to the Canadian National Railway at Crilly, Ontario, and is 174 miles west of Port Arthur, Ontario.

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

Gold	0·03 oz./ton
Silver	0·04 "
Sulphur	0·19 per cent
Iron	4·37 "

Characteristics of the Shipment. Over 90 per cent of the material screened +½-inch size and consisted of quartz cobbles stained with iron rust and included an extremely small amount of pyrite. This +½-inch material carried over 80 per cent of the gold, the remaining 10 per cent consisting mostly of iron rust and small fragments of quartz. No free gold was seen, the screen analysis showing the gold to follow the pyrite quite consistently.

Investigative Work

Portions of the shipment were screened to different sizes and assayed for gold and sulphur. The fine material was amalgamated with mercury, giving a recovery of less than 1 per cent of the gold.

DRY SCREEN ANALYSIS

Test No. 1

In order to discover the amounts of gold and sulphur contained in the different sizes of material, a screen analysis was made, with the following results:

Product	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S
+ 1″	69·1	0·025	0·16	52·5	46·4
− 1″+ ¾″	11·5	0·035	0·27	12·1	13·0
− ¾″+ ½″	9·6	0·04	0·32	11·5	12·9
− ½″+ ⅜″	1·9	0·05	0·42	2·7	3·4
− ⅜″+ ¼″	1·9	0·06	0·44	3·3	3·5
− ¼″+ ⅛″	1·2	0·075	0·43	2·7	2·2
− ⅛″+10 mesh	1·4	0·075	0·55	3·0	3·2
−10 +14 mesh	0·7	0·065	0·61	1·5	1·8
−14 mesh	2·7	0·13	1·19	10·7	13·5
Totals	100·0	0·033	0·24	100·0	100·0

It can be seen from this test how closely the gold and sulphur follow each other in the different size material.

WET SCREEN ANALYSIS

Test No. 2

The material was screened wet after a preliminary treatment by water washing. The different sizes were assayed for gold.

Results:

Product	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S
+ 1″	77·3	0·02	0·11	56·0	49·1
− 1″+ ¾″	6·1	0·025	0·23	5·4	8·1
− ¾″+ ½″	3·7	0·045	0·28	6·1	6·0
− ½″+ ⅜″	2·8	0·045	0·25	4·7	4·0
− ⅜″+ ¼″	3·0	0·055	0·41	5·8	7·1
− ¼″+ ⅛″	3·3	0·06	0·41	7·2	7·8
− ⅛″+10 mesh	1·7	0·07	0·57	4·3	5·6
−10 +14 mesh	0·6	0·09	0·66	1·8	2·3
−14 mesh	1·5	0·16	1·15	8·7	10·0
Totals	100·0	0·028	0·17	100·0	100·0

WET SCREENING AND AMALGAMATION

Test No. 3

Thirty pounds of material was washed and passed through a ¼-inch screen. The oversize was crushed and assayed. The −¼-inch material was ground in a ball mill sufficiently fine to pass through a 14-mesh screen and amalgamated with mercury for one hour. The different products were assayed for gold.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution, of gold, per cent
+¼″	92·8	0·02	97·7
−¼″	7·2	0·06	2·3

Amalgamation of −¼-inch Mesh Material:

Assay, Au, oz./ton		Recovery per cent
Feed	Tailing	
0·06	0·05	16·7

Summary of Test No. 3:

	Per cent
Gold discarded in +¼-inch material	97·7
Gold remaining in −¼-inch material	2·3
Gold recovered by amalgamation	0·38

FLOTATION

Test No. 4

In order further to elucidate the relation of the gold to the sulphide material, a flotation test was made, with the following results.

After crushing to −14 mesh, the material was ground in a ball mill with 8 pounds of soda ash and 0·17 pound of Barrett No. 4 oil per ton to pass 81·7 per cent −200 mesh. The pulp was transferred to a flotation machine and floated by the addition of 0·10 pound amyl xanthate and 0·07 pound pine oil per ton. The resulting flotation concentrate was transferred to a smaller flotation machine and a cleaner product produced.

Results of Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·032*	100·0	
Flotation concentrate	0·79	2·86	70·8	127 : 1
Flotation middling	1·25	0·44	16·9	80 : 1
Flotation tailing	97·96	0·004	12·3	

* Calculated.

The flotation concentrate assayed 7·98 per cent of sulphur.

CYANIDATION

Test No. 5

In order to determine the effects of cyanidation on this material, part of the shipment at −14 mesh was ground in a ball mill in cyanide solution at a strength of 1 pound of sodium cyanide per ton to pass 62·9 per cent −200 mesh. The pulp was agitated for a 24-hour period. Twelve pounds of lime per ton of material was added during the grinding and agitation.

Results of Cyanidation:

Agitation, hours	Grind, per cent −200 mesh	Assay, Au, oz./ton		Extraction, of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
24	62·9	0·03	Trace	99+	1·0	0·16	1·54	11·70

Although this material cyanides readily, consumption of reagents is excessive, probably owing to the oxidized and weathered condition of the sample.

SUMMARY AND CONCLUSIONS

Owing to the gold following the pyrite, it was found impossible to wash an economical amount of free gold out of this material. This condition is shown in Test No. 1 by the gold and sulphur assays and is also exemplified in Test No. 4 in the flotation concentration. Of the gold in the fine, only 16·7 per cent was found to be amenable to amalgamation.

It is apparent that the material represented by the sample cannot be classed as a placer deposit as such is usually defined. The gold is not free in its present state, but is enclosed in gangue and sulphides, suggesting rather a detrital material. It can be considered as a very low-grade gold ore.

Ore Dressing and Metallurgical Investigation No. 759

COPPER-GOLD ORE FROM THE CHIBOUGAMAU PROPERTY OF THE OBALSKI MINING CORPORATION, MONTREAL, QUEBEC

Shipment. The first shipment, 15 pounds gross weight, consisting of two packages of surface chip samples, was received on July 28, 1938, from W. W. Davis, Amos, Quebec.

Sample 1 consisted of samples in envelopes numbered 1 to 15, and having a net weight of 10·5 pounds.

Sample 2 consisted of samples numbered 16 to 19, and having a net weight of 3 pounds.

A second shipment, consisting of three boxes of bulk samples, received on September 12, 1938, from St. Felicien, Quebec, as detailed in the letter of J. R. Giroux, President and Managing Director, dated September 13, 1938, was made up as follows:

Sample 3. Two boxes of ore samples from vein "A", having a net weight of 161 pounds.

Sample 4. One box of ore samples from vein "C", having a net weight of 81 pounds.

Considerable oxidation was evident in this sample.

The third shipment, consisting of two boxes of bulk samples, was received on September 26, 1938, from St. Felicien, Quebec, as detailed in the letters, dated September 13 and October 1, of J. R. Giroux, President and Managing Director.

Sample 5. Two boxes of ore samples from vein "D", having a net weight of 190 pounds.

In all these samples there was evidence of oxidation, particularly in Sample 4. Sample 5 was the least oxidized. The fact that these samples were oxidized must be borne in mind in interpreting the results of this investigation.

Situation of Property. The property of the Obalski Mining Corporation from which the samples were received is on the northern side of Obalski Township, in the Chibougamau area of northern Quebec.

Sampling and Assaying. After reduction and sampling by accepted standard methods the samples assayed:

Sample No.	Weight, pounds	Au, oz./ton	Ag, oz./ton	Cu, per cent	Fe, per cent	S, per cent	As, per cent
1	10·5	0·930	1·025	1·03	11·12	5·54	Nil
2	3·0	0·447	0·668	1·14	10·60	6·61	0·04
3	161·0	0·403	0·385	2·22	12·55	7·85	0·02
4	81·0	0·760	0·790	0·35	11·62	4·21	0·02
5	190·0	1·176	1·175	0·92	11·56	6·38	Nil

Characteristics of the Ores. These ores may be termed "heavy sulphide, gold-copper" ores. The metallic minerals predominate over the gangue minerals. The three ores are distinctly different. Two ages of pyrite are evident in Samples 3 and 4 but the gold in the latter is much finer. Sample 5 has only one age of pyrite and comparatively coarse gold. The gold occurs for the greater part in the gangue of Sample 5, whereas the pyrite is the major host in Sample 4.

The samples of these ores were badly oxidized, Sample 5 not so extensively as the other two.

The ore of vein "A" (Sample 3) has a preponderance of pyrrhotite and much chalcopyrite, both minerals containing inclusions of other sulphide minerals. Gravity concentration indicates the gold to be particles of appreciable size. (*See* Conclusions).

The ore of vein "C" (Sample 4) has a preponderance of pyrite of two ages. Pyrrhotite is almost absent. The gold is very fine, 62 per cent of it being less than 6 microns in size and 98·7 per cent of it occurs with the sulphides. (*See* Table I).

The ore of vein "D" (Sample 5) has a preponderance of pyrite of only one age. Chalcopyrite is widely disseminated, as is magnetite, whereas pyrrhotite is negligible. The gold occurs very differently, almost 80 per cent being over 20 microns in size, and 90·4 per cent occurring in the gangue. (*See* Table II).

Purpose of Investigation. Mr. W. W. Davis, in his letter of July 25, 1938, desired:

1. To determine a suitable flow-sheet for the various types of ore.
2. To ascertain if the ore or ores can be treated by the cyanide process and what the approximate gold recovery would be.

Owing to the size and highly oxidized nature of the samples first received the scope of the work thereon was limited, and Samples 3, 4, and 5 were sent to supplement.

Summary of Investigations. The flow-sheet that appears most advisable embodies extraction of free gold in jig and strake concentrates, followed by removal of copper by flotation prior to cyanidation of the flotation tailing; recovery of gold in gravity concentrates by fine grinding and cyanidation.

The samples, in order of amenability to this flow-sheet, are:

—	Vein	Extraction, per cent	
		Gold	Copper
Sample 5	D	96·9	90·8
Sample 3	A	94·9	95·6
Sample 4	C	94·5	39·7

MICROSCOPIC INVESTIGATION

Six polished sections of each of Samples 3, 4, and 5 were examined microscopically to determine the general character of the ore.

Sample 3—Vein "A"

The gangue consists largely of impure, milky quartz with a small quantity of very fine, disseminated carbonate, which appears to be dolomite.

Metallic minerals predominate over gangue. In their approximate order of decreasing abundance these minerals are: pyrrhotite, chalcopyrite, pyrite and marcasite, limonite, sphalerite, mineral "X", and covellite.

Pyrrhotite is abundant as fine granular masses containing inclusions of gangue, chalcopyrite, pyrite, marcasite, and mineral "X". Along narrow incipient cracks it has been altered to marcasite, and around the borders of the masses it is frequently replaced by "limonite".

Massive chalcopyrite, which contains inclusions of sphalerite, mineral "X", and pyrite, is common; a small amount is present as small irregular grains in pyrite, pyrrhotite, and gangue.

Pyrite of two ages is recognizable. The earlier occurs as coarse, well crystallized, disseminated grains, and the later as very irregular, rounded, colloform structures which show zones of admixed marcasite. Both are oxidized and show some alteration to "limonite" around the edges and along cracks, and much of the later variety appears to have attacked the pyrrhotite.

A considerable quantity of "limonite" is present as already noted and as rust stains in the gangue.

Occasional small irregular grains of sphalerite are visible largely in association with chalcopyrite, as are rare tiny flakes of covellite.

A minor amount of a soft, white, unidentified mineral "X", occurs as tiny irregular grains and narrow veinlets in pyrrhotite and chalcopyrite. Under crossed nicols this mineral is weakly anisotropic, and etch tests give positive reactions with HNO_3, HCl, and $FeCl_3$.

No native gold is visible in the sections.

Sample 4—Vein "C"

The gangue is essentially the same as in Sample 3 but is more abundant. This sample is very badly oxidized and alteration products are prevalent.

Pyrite is the predominant metallic mineral and occurs in the same two ways as in Sample 3. It shows alteration to both "limonite" and marcasite and contains inclusions of chalcopyrite, pyrrhotite, and gangue.

Chalcopyrite is present in the same modes but in much less quantity than in Sample 3, whereas pyrrhotite is absent except for small irregular inclusions in pyrite.

"Limonite" rims and veins both pyrite and chalcopyrite, and deep stains of iron oxides are common.

Occasional small irregular grains of magnetite are disseminated in gangue, and rare tiny flakes of covellite in chalcopyrite.

The gold visible is very finely divided, as shown in the table. It occurs in the earlier pyrite and in gangue, the former mode of occurrence being much the more common.

Grain Size of Native Gold in Sample 4:

Microns	Gold in early pyrite, per cent					Gold in gangue, per cent	Totals, per cent
	In dense pyrite	Along fractures in pyrite	Associated with inclusions of gangue	Associated with chalcopyrite			
				Along fractures	With inclusions		
−26 +19	12·7						12·7
−19 +13							
−13 + 9			6·3				6·3
− 9 + 6	13·9			5·1			19·0
− 6	42·3	7·0		8·9	2·5	1·3	62·0
	68·9	7·0	6·3	14·0	2·5	1·3	100·0

Sample 5—Vein "D"

This is not so extensively oxidized as Sample 4, nor are its metallic constituents so abundant. Like the other two samples the gangue consists essentially of quartz with irregular patches of a soft, alteration product resembling sericite.

Pyrite occurs only as coarse to fine irregular grains and small masses disseminated throughout gangue but does not show the younger colloidal structures. It contains inclusions of gangue, chalcopyrite, and pyrrhotite, and, in places, is somewhat shattered and healed with quartz.

Chalcopyrite is prevalent as small disseminated grains and irregular masses in gangue, but it is not so abundant as in Sample 3.

A considerable quantity of magnetite occurs as medium to small irregular grains in gangue, and "limonite" has the same modes of occurrence as in the other two samples.

A very minor quantity of pyrrhotite is present as tiny irregular inclusions in pyrite, and rare small grains of sphalerite and covellite are associated with chalcopyrite.

Native gold is visible as small irregular grains in gangue and pyrite, but, unlike Sample 4, most of the grains observed occur in gangue. Its size and modes of occurrence are shown below:

Grain Size of Native Gold in Sample 5:

Microns	Gold in gangue, per cent	Gold in pyrite, per cent	Totals, per cent
−74 +52	51·5	....	51·5
−52 +37	....	....	....
−37 +26	17·9	9·6	27·5
−26 +19	7·2	....	7·2
−19 +13	....	....	....
−13 + 9	6·6	....	6·6
− 9 + 6	1·8	....	1·8
− 6	5·4	....	5·4
	90·4	9·6	100·0

SAMPLING AND ASSAYING INVESTIGATION

The following results, in ounces of gold per ton, illustrate some of the difficulties in assaying Obalski ores and products:

Ore Samples:

Tests Nos. 6, 7, and 8:

Sample 3		Sample 4		Sample 5	
	0·510		0·795		1·225
	0·380		0·735		1·130
	0·335		0·760		1·210
	0·380		0·750		1·140
	0·440				
	0·400				
	0·380				
(Aggregate)	2·825	(Aggregate)	3·040	(Aggregate)	4·705
Average	0·403	Average	0·760	Average	1·176

Flotation Tailings:

Sample 3		Sample 4		Sample 5	
Test No. 18	Test No. 21	Test No. 19	Test No. 22	Test No. 23	Test No. 17
0·160 0·100 0·125 0·110	0·170 0·170 0·135 0·215	0·175 0·165 0·170 0·180	0·28 0·32 0·29 0·30	0·515 0·420 0·470 0·420	0·660 0·700 1·180 0·775 0·890 0·760
(Aggregate) 0·495	0·690	0·690	1·19	1·825	4·965
Average 0·124	0·172	0·172	0·297	0·456	0·709

Cyanidation Tailings:

	Sample 3, Test No. 15	Sample 4, Test No. 16	Sample 5, Test No. 17
	0·180 0·150 0·240 0·225	0·245 0·150 0·150 0·090	0·185 0·285 0·260 0·265
(Aggregate)	0·795	0·635	0·995
Average	0·199	0·159	0·249

Tests Nos. 9, 10, and 11

These tests were made to provide sufficient flotation tailing for use of the assay office to determine assaying difficulties. In Tests Nos. 9 and 10 the flotation concentrates were cyanided and the cyanide solution assayed. In Test No. 11 the flotation concentrates were assayed directly.

Grind:

Sample 5 .... 2,000 grms.
Water .... 1,500 "
Reagents: Sodium ethyl xanthate .... 0·1 "
Lime .... 10·0 "

Time .... 30 minutes.

Conditioning: 5 minutes

Flotation: 7 minutes

Reagents: Pine oil .... 0·075 grm.
pH .... 9·6

Cyanidation:

Test No. 9:

Flotation tailing—dry—5 assay tons.
Water .... 300 grms.
Lime .... 1·0 grm.
NaCN .... 0·2 "
Time .... 24 hours.
Residue—filtered; washed with 155 ml. of water; solids and solution sent to assay office.

Test No. 10:

This duplicated Test No. 9 except the solution which was evaporated at low heat and the dried residue was sent to the assay office.

Assaying. Residues were reweighed in one-assay-ton lots and assayed.

Test No. 11:

Flotation tailing—dry.
Assayed as ordinarily done.

Results:

5 assay tons	145·830 grms.
Lime	1·00 "
Cyanide	0·20 "
Test weight	147·030 "
Residue No. 1	146·35 grms.
Loss No. 1	0·68 "
Residue No. 2	146·63 "
Loss No. 2	0·40 "

Assay:

Test No. 9—Filtered:

	Filtrate	Solution
	0·35 oz./ton	
	0·36 "	
	0·37 "	0·98 mg.
	0·42 "	or
	0·47 "	0·196 oz./ton
Solids, average	0·394 "	
Solution, average	0·196 "	
Flotation tailing	0·590 "	

Assay:

Test No. 10—Dried:

	0·50 oz./ton
	0·53 "
	0·57 "
	0·58 "
	0·60 "
Flotation tailing, average	0·556 oz./ton

Assay:

Test No. 11—As received:

	0·51 oz./ton
	0·59 "
	0·63 "
	0·64 "
Flotation tailing, average	0·5925 oz./ton

Concentration Results:

Product	Weight		Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
	Grammes	Per cent			
Feed	2,000	100·0	1·176	100·00	
Concentrate	70	3·5	17·636	52·50	28·57 : 1
Tailing	1,930	96·5	0·579	47·50	

Conclusion from Sampling and Assaying Investigation. A large part of the gold appears to be in the form of particles of an appreciable size. This confirms the results of the microscopic examination of the various samples, particularly that of Sample 5.

Detailed Investigations

Sample 1

STRAIGHT CYANIDATION

Tests Nos. 1, 2, and 3

Grind:

Ore	2,000 grms.
Water	1,500 "
Time	20 minutes
Reagents: Lime	8·0 grms.
NaCN	2·0 "

Agitation:

Dilution	2 : 1
Strengths: NaCN	0·5 lb./ton
CaO	0·5 "
Periods	10 hours
	28 "
	37 "

Results:

Test	Grind, per cent −200 mesh	Agitation		Strength, lb./ton of L.		Consumed, lb./ton of S.		Assay, Au, oz./ton		Extraction, per cent
		Hours	L : S ratio	NaCN	CaO	NaCN	CaO	Feed	Tailing	
1	75·9	10	2 : 1	0·5	0·5	3·70	17·0	0·930	0·465	50·00
2	75·9	28	2 : 1	0·5	0·5	3·90	20·5	0·930	0·390	58·07
3	75·9	37	2 : 1	0·5	0·5	3·90	23·2	0·930	0·285	69·38

Sample 2

Because of its size and in accordance with the above results, Sample 2 was subjected to further investigation.

Sample 1

CONCENTRATION BY FLOTATION AND AMALGAMATION OF CONCENTRATES

Test No. 4

Grind:

Ore	1,878 grms.
Water	1,400 "

Reagents:

Reagents: Aerofloat No. 25	0·05 grm.
301	0·10 "
Time	20 minutes

Conditioning: 5 minutes

Flotation: 15 minutes

Reagents: Pine oil	0·10 grm.

Amalgamation:

Concentrate	174·6 grms.
Regrind: Time	30 minutes
Lime	0·20 grm.
NaOH	0·05 "
Contact—time	60 minutes

Results:

Product	Weight		Assay, Au, oz./ton	Distribution		Ratio of concentration
	Grammes	Per cent		Units	Per cent	
Concentration:						
Feed	1,878·6	100·00	0·930	93·00	100·00	
Concentrate	246·2	13·11	5·040	66·06	71·00	7·63
Tailing	1,632·4	86·89	0·310	26·94	29·00	
Amalgamation:						
Feed	174·6	100·00	5·748	100·00	71·00	
Tailing	174·6	100·00	2·840	49·39	....	....
Amalgam	....	....	....	50·61	35·95	

Screen Test on Flotation Tailing:

+200	27·2 per cent
−200	72·8 "
	100·0 "

Sample 2

CONCENTRATION BY FLOTATION AND AMALGAMATION OF CONCENTRATE

Test No. 5

Grind:

Ore	963 grms.
Water	700 "
Reagents: Aerofloat No. 25	0·025 grm.
301	0·050 "
Time	20 minutes

Conditioning: 5 minutes

Flotation: 15 minutes

Reagents: Pine oil .... 0·05 grm.

Amalgamation:

Regrind: Concentrate .... 94·4 grms.
Time .... 30 minutes

Lime .... 0·10 grm.
NH_4Cl .... 0·03 "
Contact .... 60 minutes

Results:

Product	Weight		Assay, Au, oz./ton	Distribution		Ratio of concentration
	Grammes	Per cent		Units	Per cent	
Concentration:						
Feed	963·0	100·00	0·447	44·70	100·00	
Concentrate	124·0	12·88	3·102	39·91	89·29	7·76
Tailing	839·0	87·12	0·055	4·79	10·71	
Amalgamation:						
Feed	94·4	100·00	4·40	100·00	89·29	
Tailing	94·4	100·00	0·805	18·30	....	....
Amalgam	....	....	....	81·70	72·95	

Screen Test on Flotation Tailing:

+200 .... 30·6 per cent
−200 .... 69·4 "

100·0 "

Samples 3, 4, and 5

CONCENTRATION OF COPPER BY FLOTATION—CYANIDATION OF FLOTATION TAILING

Tests Nos. 12, 13, and 14

Purpose. To determine if it be practicable to concentrate the copper in a flotation concentrate leaving a maximum amount of the gold in the flotation tailing for subsequent cyanidation.

Grind:

Ore .... 2,000 grms.
Water .... 1,500 "
Time .... 15 minutes
Reagents: Lime .... 10·0 grms.
301 .... 0·1 "

Conditioning:

Sample	*3*	*4*	*5*
Time, minutes	5	5	5
Lime, grammes	0·4	0·4	1·0

Flotation:

Time, minutes	7	6	6
Pine oil, grammes	0·13	0·08	0·08
pH	6·6	5·4	7·4

Cyanidation:

Time .... 24 hours
Dilution .... 2 : 1
Reagent strengths (*See* table of results).

Results of Concentration:

Test No.	Sample No.	Product	Weight, per cent	Assay		Percentage, distribution		Ratio of concentration
				Au, oz./ton	Cu, per cent	Au	Cu	
12	3	Feed..........	100·00	0·403	2·22	100·00	100·00	12·0 : 1
		Concentrate...	8·35	2·180	21·64	45·16	81·40	
		Tailing.......	91·65	0·241	0·45	54·84	18·60	
13	4	Feed..........	100·00	0·760	0·35	100·00	100·00	23·4 : 1
		Concentrate...	4·25	3·015	3·83	16·85	46·50	
		Tailing.......	95·75	0·660	0·17	83·15	53·50	
14	5	Feed..........	100·00	1·176	0·92	100·00	100·00	15·4 : 1
		Concentrate...	6·46	13·935	10·62	76·55	74·60	
		Tailing.......	93·54	0·295	0·25	23·45	25·40	

Results of Cyanidation:

Test No.	Sample No.	Grind, per cent −200 mesh	Reagents				Assay, Au, oz./ton		Percentage extraction on	
			Strength, lb./ton of L.		Consumption, lb./ton of S.		Feed	Tailing	Unit	Ore
			NaCN	CaO	NaCN	CaO				
12	3	62·7	0·3	0·3	3·6	19·4	0·241	0·135	44·00	24·12
13	4	72·5	0·2	0·1	2·0	24·3	0·660	0·350	46·97	39·05
14	5	63·0	0·5	0·5	1·8	18·5	0·295	0·220	25·44	5·97

Recapitulation of Extractions: (Gold)

Test No.	Process	Efficiency	Percentage extraction of original feed
12......................................	Flotation...............	45·16	45·16
	Cyanidation............	44·00	24·12
	Overall.................	....	69·28
13......................................	Flotation...............	16·85	16·85
	Cyanidation............	46·97	39·05
	Overall.................	....	55·90
14......................................	Flotation...............	76·55	76·55
	Cyanidation............	25·44	5·97
	Overall.................	....	82·52

Copper Analyses—Cyanidation:

Test No.	Sample	Before	After
		per cent	per cent
12..	3	0·45	0·38
13..	4	0·17	0·11
14..	5	0·25	0·25

Lime Consumption (lb/ton):

Test No.	Sample	Grinding and flotation	Cyanidation	Total
12	3	10·4	19·4	29·8
13	4	10·4	24·3	34·7
14	5	11·0	18·5	29·5

Screen Analyses of Cyanide Tailings:

Mesh	Sample 3		Sample 4		Sample 5	
	Weight per cent	Au, oz./ton	Weight, per cent	Au, oz./ton	Weight, per cent	Au, oz./ton
+200	37·2	0·287	27·5	0·395	37·0	0·407
−200	62·8	0·045	72·5	0·330	63·0	0·110
Total	100·0	0·135	100·0	0·348	100·0	0·220

Conclusions:

The necessity of finer grinding is shown by the screen analysis.

The lime consumption is prohibitively high although flotation alkalinity is not sufficient for effective depression of the gold and the pyrite in the flotation concentration of the chalcopyrite without removal of free gold prior to flotation.

It is difficult to maintain alkalinity during cyanidation.

CONCENTRATION OF COPPER BY FLOTATION—CYANIDATION OF FLOTATION TAILING

Tests Nos. 15, 16, and 17

Purpose: Similar to that of the previous investigation but with finer initial grinding.

Grind:

Ore		2,000 grms.
Water		1,500 "
Time		30 minutes.
Reagents:	Lime	10·0 grms.
	Sodium ethyl xanthate	0·1 grm.

Conditioning: 5 minutes.

Additional lime was added during the conditional period.

Flotation:

Sample No.	3	4	5
Time, minutes	8	8	7
pH	10·6	7·2	9·6
Pine oil, grammes	0·08	0·13	0·13
Sodium ethyl xanthate, grammes	Nil	0·10	Nil

Cyanidation:

Time .... 24 hours
Dilution .... 2 : 1
Reagent strengths: (*See* table of results).

Results of Concentration:

Test No.	Sample No.	Product	Weight, per cent	Assay		Percentage distribution		Ratio of concentration
				Au, oz./ton	Cu, per cent	Au	Cu	
15	3	Feed	100·00	0·403	2·22	100·00	100·00	13·6 : 1
		Concentrate	7·35	2·962	29·05	54·00	96·20	
		Tailing	92·65	0·200	0·09	46·00	3·80	
16	4	Feed	100·00	0·760	0·35	100·00	100·00	14·2 : 1
		Concentrate	7·03	7·413	2·86	68·55	57·50	
		Tailing	92·97	0·258	0·16	31·45	42·50	
17	5	Feed	100·00	1·176	0·92	100·00	100·00	27·4 : 1
		Concentrate	3·65	10·390	19·14	32·22	75·95	
		Tailing	96·35	0·827	0·23	67·78	24·05	

Results of Cyanidation:

Test No.	Sample No.	Grind, per cent −200 mesh	Reagents				Assay, Au, oz./ton		Percentage extraction	
			Strength, lb./ton of L.		Consumption, lb./ton of S.		Feed	Tailing	Unit	Ore
			NaCN	CaO	NaCN	CaO				
15	3	81·4	0·4	0·3	1·7	9·5	0·200	0·199	Nil	Nil
16	4	88·3	0·6	0·3	1·0	9·8	0·258	0·159	38·38	17·65
17	5	90·5	0·5	1·0	1·0	9·3	0·827	0·249	69·92	47·37

Screen Analyses—Cyanidation Tailing:

—	Test No. 15		Test No. 16		Test No. 17	
	Weight, per cent	Au, oz./ton	Weight, per cent	Au, oz./ton	Weight, per cent	Au, oz./ton
+200	18·5	1·30	11·7	1·72	9·5	1·61
−200	81·5	0·040	88·3	0·065	90·5	0·17
Total (cal.)	100·0	0·273	100·0	0·259	100·0	0·307

Lime Consumption, lb./ton:

Test No.	Sample No.	Grinding and flotation	Cyanidation	Total
15	3	12·0	9·5	21·5
16	4	12·0	9·8	21·8
17	5	12·0	9·3	21·3

Recapitulation of Extractions: (Gold)

Test No.	Process	Efficiency	Percentage extraction of original feed
15	Flotation	54·00	54·00
	Cyanidation	Nil	Nil
	Overall		54·00
16	Flotation	68·55	68·55
	Cyanidation	38·38	17·65
	Overall		86·20
17	Flotation	32·22	32·22
	Cyanidation	69·92	47·37
	Overall		79·59

Tests Nos. 18, 19, and 20

Duplicating Tests 15, 16, and 17, except that the flotation tailings are reground before cyanidation.

Grind:
Conditioning: } Duplicating Tests Nos. 15, 16, and 17.
Flotation:

Flotation pH:

Test	pH
18	9·2
19	8·8
20	9·4

Flotation tailings reground, 30 minutes.

Results of Concentration:

Test No.	Sample No.	Product	Weight, per cent	Assay		Percentage distribution		Ratio of concentration
				Au, oz./ton	Cu, per cent	Au	Cu	
18	3	Feed	100·00	0·403	2·22	100·00	100·00	
		Concentrate	8·77	3·307	23·35	72·00	92·19	11·4 : 1
		Tailing	91·23	0·124	0·19	28·00	7·81	
19	4	Feed	100·00	0·760	0·35	100·00	100·00	
		Concentrate	4·87	12·235	4·30	78·48	59·58	20·5 : 1
		Tailing	95·13	0·1725	0·22	21·52	40·42	
20	5	Feed	100·00	1·176	0·92	100·00	100·00	
		Concentrate	4·07	21·36	18·84	73·90	83·35	24·6 : 1
		Tailing	95·93	0·320	0·16	26·10	16·65	

Results of Cyanidation:

Test No.	Sample No.	Reagents				Assay Au, oz./ton		Percentage extraction	
		Strength, lb./ton of L.		Consumption, lb./ton of S.					
		NaCN	CaO	NaCN	CaO	Feed	Tailing	Unit	Ore
18......	3	0·3	0·2	2·1	10·2	0·124	0·1125	9·28	2·60
19......	4	0·4	0·2	1·7	10·3	0·172	0·0585	66·05	14·18
20......	5	0·5	0·6	1·2	9·3	0·320	0·2050	35·95	9·38

Recapitulation of Extractions: (Gold)

Test No.	Process	Efficiency	Percentage extraction of original feed
18....................................	Flotation..............	72·00	72·00
	Cyanidation............	9·28	2·60
	Overall................	....	74·60
19....................................	Flotation..............	78·48	78·48
	Cyanidation............	66·05	14·18
	Overall................	....	92·66
20....................................	Flotation..............	73·90	73·90
	Cyanidation............	35·95	9·38
	Overall................	....	83·28

Screen Analyses—Cyanidation Tailing:

Mesh	Test No. 18		Test No. 19		Test No. 20	
	Weight, per cent	Au, oz./ton	Weight, per cent	Au, oz./ton	Weight, per cent	Au, oz./ton
+200..........................	4·2	2·25	4·0	0·535	4·9	2·35
−200..........................	95·8	0·04	96·0	0·040	95·1	0·095
Total (calculated).......	100·0	0·1328	100·0	0·0598	100·0	0·2055

Lime Consumption (lb./ton)

Test No.	Sample No.	Grinding and flotation	Cyanidation	Total
18..	3	10·0	10·2	20·2
19..	4	10·0	10·3	20·3
20..	5	10·0	9·3	19·3

Conclusions—Tests Nos. 15 to 20: Fine grinding is essential to extraction by cyanidation, whether effected before or after flotation.

Overall extraction is increased with fine grinding.

In practice it will be advisable to effect the grinding prior to flotation. Such a unit would be more compact and possibly free a larger proportion of gold for extraction by gravity processes or for depression into flotation tailing, that would otherwise be carried into the copper concentrate with the chalcopyrite.

Tests Nos. 21, 22, and 23

Duplicating Tests 18, 19, and 20, except that reground flotation tailings are cyanided for 48 hours.

Grind:
Conditioning: } Duplicating Tests 18, 19, and 20
Flotation:

Flotation. Used 0·16 pound of pine oil per ton, giving quicker and finer float.

Test	pH
21	9·6
22	8·8
23	9·2

Flotation tailings reground, 30 minutes

Cyanidation:

Time	48 hours
Dilution	2 : 1

Results of Concentration:

Test No.	Sample No.	Product	Weight, per cent	Assay		Percentage distribution		Ratio of concentration
				Au, oz./ton	Cu, per cent	Au	Cu	
21	3	Feed	100·00	0·403	2·22	100·00	100·00	10·15 : 1
		Concentrate	9·85	2·491	21·53	60·88	95·55	
		Tailing	90·15	0·175	0·11	39·12	4·45	
22	4	Feed	100·00	0·760	0·35	100·00	100·00	27·05 : 1
		Concentrate	3·70	12·995	3·75	63·28	39·65	
		Tailing	96·30	0·290	0·22	36·72	60·35	
23	5	Feed	100·00	1·176	0·92	100·00	100·00	17·99 : 1
		Concentrate	5·56	13·420	15·02	63·40	90·75	
		Tailing	94·44	0·456	0·09	36·60	9·25	

Results of Cyanidation:

Test No.	Sample No.	Reagents: Strength, lb./ton of L. NaCN	Strength, lb./ton of L. CaO	Consumption, lb./ton of S. NaCN	Consumption, lb./ton of S. CaO	Assay, Au, oz./ton Feed	Assay, Au, oz./ton Tailing	Percentage extraction Unit	Percentage extraction Ore
21.....	3	0·3	0·2	2·4	11·0	0·175	0·119	32·00	12·52
22.....	4	0·4	0·2	2·5	11·4	0·290	0·150	48·28	17·72
23.....	5	0·6	0·4	1·6	10·5	0·456	0·275	39·70	14·53

Recapitulation of Extractions: (Gold)

Test No.	Process	Efficiency	Percentage extraction of original feed
21.....................................	Flotation...............	60·88	60·88
	Cyanidation............	32·00	12·52
	Overall................	....	73·40
22.....................................	Flotation...............	63·28	63·28
	Cyanidation............	48·28	17·72
	Overall................	....	81·00
23.....................................	Flotation...............	63·40	63·40
	Cyanidation............	39·70	14·53
	Overall................	....	77·93

Screen Analyses:

Cyanidation Tailing:

Mesh	Test No. 21 Weight, per cent	Test No. 21 Au, oz./ton	Test No. 22 Weight, per cent	Test No. 22 Au, oz./ton	Test No. 23 Weight, per cent	Test No. 23 Au, oz./ton
+200...........................	3·8	2·260	3·2	1·770	4·0	12·690*
−200...........................	96·2	0·035	96·8	0·095	96·0	0·040
Total (calculated).......	100·0	0·119	100·0	0·149	100·0	0·546

Flotation Tailing:

Mesh	Test No. 21 Weight, per cent	Test No. 21 Au, oz./ton	Test No. 21 Cu, per cent	Test No. 22 Weight, per cent	Test No. 22 Au, oz./ton	Test No. 22 Cu, per cent	Test No. 23 Weight, per cent	Test No. 23 Au, oz./ton	Test No. 23 Cu, per cent
+200.............	13·3	1·360	0·07	13·3	1·390	0·10	12·5	3·180	0·05
−200.............	86·7	0·045	0·08	86·7	0·210	0·22	87·5	0·200	0·11
Total (cal.)....	100·0	0·220	0·078	100·0	0·367	0·204	100·0	0·573	0·103

* Erratic assays.

Lime Consumption (lb./ton):

Test No.	Sample	Grinding and flotation	Cyanidation	Total
21	3	12·0	11·2	23·2
22	4	12·0	11·4	23·4
23	5	12·0	10·5	22·5

Conclusions: With 48-hour instead of 24-hour agitation lime and cyanide consumptions are increased without any important increase in extraction of the gold.

GRAVITY CONCENTRATION—FLOTATION OF CYANICIDES—CYANIDATION OF FLOTATION TAILING

Tests Nos. 24 to 29

Purpose: This was to investigate the possibility of extracting the free gold in the ore by gravity concentration such as provided by jigs and strakes prior to the removal of the cyanicides, such as chalcopyrite, and of cyaniding the flotation tailing.

Grind:

Ore	2,000 grms.
Water	1,500 "
Grind	30 minutes.
Reagents	None

Concentration: Using Denver laboratory jig, pneumatically fed, followed by a single strake 3 inches wide by 16 inches long, at a slope of $1\frac{3}{4}$ inches in 12 inches.

Dilution of feed 1:1.

Feed rate:

Samples 3 and 4	15 minutes for 2,000 grammes
Sample 5	30 minutes for 2,000 "

Conditioning:

	Sample 3		Sample 4		Sample 5	
Test	24	25	26	27	28	29
Time, minutes	5	5	5	5	5	5
Lime, grammes	5	5	5	5	5	5
Sodium ethyl xanthate, grammes	0·1	0·15	0·1	0·2	0·2	0·1

Flotation:

Test	Sample 3		Sample 4		Sample 5	
	24	25	26	27	28	29
Pine oil, grammes	0·05	0·05	0·05	0·08	0·05	0·08
Time, minutes	5	5	5	7	5	5
pH	8·7	8·6	8·8	8·9	9·2	9·5

Cyanidation:

Dilution.. 2 : 1
Time.. 24 hours

Results of Gravity Concentration: (concentrates combined)

Test	Sample No.	Product	Weight per cent	Assay		Distribution, per cent		Ratio of concentration
				Au, oz./ton	Cu, per cent	Au	Cu	
24 and 25	3	Feed	100·00	0·403	2·22	100·00	100·00	....
		Jig concentrate	1·11	23·100	....	63·60	....	90·1 : 1
		Strake concentrate	5·09	0·738	....	9·33	....	19·6 : 1
		Tailing	94·91	0·115	2·22	27·07	....	....
26 and 27	4	Feed	100·00	0·760	0·35	100·00	100·00	....
		Jig concentrate	1·77	15·310	....	35·65	....	56·5 : 1
		Strake concentrate	6·63	1·728	....	15·04	....	15·1 : 1
		Tailing	91·60	0·300	0·33	49·31	....	....
28 and 29	5	Feed	100·00	1·176	0·92	100·00	100·00	....
		Jig concentrate	5·00	14·460	....	61·50	....	20·0 : 1
		Strake concentrate	5·84	2·950	....	14·64	....	17·1 : 1
		Tailing	89·16	0·315	0·81	23·86	....	....

Results of Flotation Concentration:

Test No.	Sample No.	Product	Weight, per cent	Assay		Distribution, per cent	
				Au, oz./ton	Cu, per cent	Au	Cu
24	3	Feed	100·00	0·115	2·22	100·00	100·00
		Concentrate	9·22	0·903	21·22	72·50	88·10
		Tailing	90·78	0·035	0·29	27·50	11·90
25	3	Feed	100·00	0·115	2·22	100·00	100·00
		Concentrate	7·96	0·924	23·38	64·00	83·90
		Tailing	92·04	0·045	0·39	36·00	16·10
26	4	Feed	100·00	0·30	0·33	100·00	100·00
		Concentrate	6·03	3·34	3·45	67·13	63·00
		Tailing	93·97	0·105	0·13	32·87	37·00
27	4	Feed	100·00	0·30	0·33	100·00	100·00
		Concentrate	6·49	3·11	3·93	67·93	77·35
		Tailing	93·51	0·105	0·08	32·07	22·65
28	5	Feed	100·00	0·280	0·77	100·00	100·00
		Concentrate	8·92	2·117	7·50	67·50	87·00
		Tailing	91·08	0·100	0·11	32·50	13·00
29	5	Feed	100·00	0·350	0·84	100·00	100·00
		Concentrate	10·46	2·362	6·75	70·58	84·00
		Tailing	89·54	0·115	0·15	29·42	16·00

Flotation Characteristics:

Test No. 24.—Deep, dry froth, with moderate-size bubbles.
Test No. 25.—Much poorer froth and larger bubbles.
Test No. 26.—A fair froth with rather sparse but small wet bubbles.
Test No. 27.—A fine copious froth. Lime addition made seventeen hours prior to flotation. Extra drop of pine oil greatly increased amount of froth.
Test No. 28.—Splendid froth; appeared to be much less chalcopyrite present.
Test No. 29.—Splendid froth; extra pine oil added in error resulted in a fine but wet froth.

Results of Cyanidation:

Test No.	Sample No.	Agitation, hours	Reagents				Assay, Au, oz./ton		Extraction, per cent	
			Strength, lb./ton of L.		Consumption, lb./ton of S.					
			NaCN	CaO	NaCN	CaO	Feed	Tailing	Unit	Flotation tailing
24	3	24	0·2	0·1	3·36	7·68	0·035	0·025	28·55	7·85
25	3	48	0·2	0·1	4·33	9·18	0·050	0·025	50·00	18·00
26	4	24	0·4	0·1	2·10	8·33	0·110	0·045	59·20	19·46
27	4	48	0·4	0·1	3·15	9·62	0·105	0·025	76·40	24·50
28	5	24	0·3	0·1	2·68	8·53	0·100	0·040	60·00	20·50
29	5	48	0·3	0·1	3·34	10·00	0·115	0·050	56·50	16·64

Recapitulation of Extractions: (Gold)

Test No.	Process	Efficiency	Percentage extraction of original feed
24	Jigging	63·60	63·60
	Straking	9·33	9·33
	Flotation	72·50	19·62
	Cyanidation	28·55	2·12
	Overall		94·67
25	Jigging	63·60	63·60
	Straking	9·33	9·33
	Flotation	64·00	17·31
	Cyanidation	50·00	4·88
	Overall		95·12
26	Jigging	35·65	35·65
	Straking	15·04	15·04
	Flotation	67·13	33·08
	Cyanidation	59·20	9·01
	Overall		92·78
27	Jigging	35·65	35·65
	Straking	15·04	15·04
	Flotation	67·93	33·50
	Cyanidation	76·40	12·08
	Overall		96·27
28	Jigging	61·50	61·50
	Straking	14·64	14·64
	Flotation	67·50	16·10
	Cyanidation	60·00	4·66
	Overall		96·90
29	Jigging	61·50	61·50
	Straking	14·64	14·64
	Flotation	70·58	16·89
	Cyanidation	56·50	3·94
	Overall		96·97

NOTE.—In the above recapitulation the extractions determined by combining the jig and strake concentrates of each sample have been assumed representative for both tests on the same sample.

CYANIDATION OF JIG AND STRAKE CONCENTRATES

Because the amounts of these concentrates were small it was possible only to run indicative tests by grinding in cyanide solution with pestle in an agate mortar, leaching overnight, and assaying both solution and solids to determine feed value.

Cyanide and lime strengths were kept low, the reading of the solutions before assaying at the end of 16 hours' contact being:

—	NaCN, lb./ton of L.	CaO lb./ton of L.
Jig Concentrates:		
Tests Nos. 24-25	0·1	0·2
" " 26-27	0·15	0·25
" " 28-29	0·50	0·30
Strake Concentrates:		
Tests Nos. 24-25	0·15	0·08
" " 26-27	0·20	Nil
" " 28-29	0·50	0·10

Extractions:

—	Tests Nos. 24-25	Tests Nos. 26-27	Tests Nos. 28-29
Jig concentrates ... per cent	4·75	11·73	4·15
Strake concentrates ... per cent	56·60	99·10	36·18

These extraction figures merely indicate what might be expected. Neither the grind effected in the mortar nor the leaching process employed could be considered satisfactory.

They do, however, show that with finer grinding and proper strengths of reagent and periods of contact a large proportion of the gold could be recovered by cyanidation.

No extraction of copper was indicated in the gravity concentration, suggesting that little trouble might be experienced from copper in the cyanide solution in the cyanidation of these products. A separate regrind and agitation circuit would be preferable to cyanide the gravity concentrates prior to thickening and filtration with the cyanide mill tailing.

Conclusion: At a moderate grind still finer grinding is necessary to effect reasonable extraction of the gold of the gravity concentrates by cyanidation on the property.

JIGGING—STRAKING—CYANIDATION OF CONCENTRATES

Tests Nos. 31, 32, and 33

Purpose. To determine the extraction of the gold from the jig and blanket concentrates by cyanidation.

Procedure:

Grind:

Ore............................... 6,000 grms. of each sample, in 2,000-grm. lots
Time............................... 30 minutes

Jigging:

Time: Ore A............................... 130 minutes
" C............................... 160 "
" D............................... 140 "

Regrind of Concentrates:

Time............................... 40 minutes
Lime: Ore A............................... 6 grms.
" C............................... 3 "
" D............................... 9 "

Cyanidation of Concentrates:

Aeration:

Time............................... 120 minutes

Agitation:

Liquids, Solids............................... 2 : 1

Reagent Strengths:

Lime............................... 0·1 lb./ton
Cyanide............................... 0·1 "
Time............................... 17 hours

Results of Concentration:

Test No.	Sample No.	Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
31	3	Feed	100·00	0·403	100·00	
		Jig concentrate	7·48	3·874	81·15	13·3 : 1
		Strake concentrate	2·93			34·1 : 1
		Tailing	89·59	0·085	18·85	
32	4	Feed	100·00	0·760	100·00	
		Jig concentrate	2·85	9·680	68·90	35·1 : 1
		Strake concentrate	2·56			39·1 : 1
		Tailing	94·59	0·250	31·10	
33	5	Feed	100·00	1·176	100·00	
		Jig concentrate	11·07	6·514	76·97	9·04 : 1
		Strake concentrate	2·82			35·5 : 1
		Tailing	86·11	0·315	23·03	

Results of Cyanidation:

Test No.	Sample No.	Reagents				Assay, Au. oz./ton		Percentage extraction	
		Strength, lb./ton of L.		Consumption, lb./ton of S.					
		NaCN	CaO	NaCN	CaO	Feed	Tailing	Unit	Ore
31	3	0·1	0·1	2·36	28·7	3·87	2·08	46·25	37·53
32	4	0·1	0·1	3·44	36·8	9·68	5·52	43·00	29·62
33	5	0·1	0·1	2·16	18·8	6·51	3·78	41·90	32·25

Screen Analyses—Cyanidation Residue:

Mesh	Test No. 31		Test No. 32		Test No. 33	
	Weight, per cent	Assay, Au, oz./ton	Weight, per cent	Assay, Au, oz./ton	Weight, per cent	Assay, Au, oz./ton
+150	1·7	47·36	....	....	1·8	49·26
−150+200	12·9	2·88	1·2	44·73	7·3	7·19
−200	85·4	1·38	98·8	4·95	90·9	2·64
Total (calculated)	100·0	2·35	100·0	5·43	100·0	3·81

AMALGAMATION AND FLOTATION CONCENTRATE

Test No. 30—Sample 5

Grind:

Sample 5	2,000 grms.
Water	1,500 "
Time	15 minutes
Reagents: Lime	10·0 grms.
301	0·1 "

Conditioning:

Time	5 minutes
Lime	1·0 grm.

Flotation:

Time	6 minutes
Pine oil	0·05 lb./ton

Amalgamation:

Time	60 minutes
Concentrate	Unground

Results:

Test	Product	Weight, per cent	Assay, Au, oz./ton	Percentage, distribution		Ratio of concentration
				Unit	Ore	
30	Feed	100·00	1·176	100·00	100·00	15·71 : 1
	Concentrate	6·36	8·610	53·44	53·44	
	Tailing	93·64	0·585	46·56	46·56	
	Amalgam	....	....	61·00	32·60	
	Amalgamation tailing	6·36	2·48	39·00	20·84	

Copper-gold concentrate floated directly yields 61·0 per cent of its gold by amalgamation. After removal of a large proportion of "free" gold by gravity methods prior to flotation, amalgamation of such a flotation concentrate is expected to show a decreased extraction.

Finer grinding of the flotation concentrate before amalgamation should be investigated.

CONCLUSIONS

Samples 1 and 2. Because of the smallness and the oxidized condition of these samples, the results of the work thereon cannot be considered as representative of the metallurgy of the ore.

Samples 3, 4, and 5. Fine grinding is needed to liberate the gold and the chalcopyrite; it must be ground to pass 200 mesh. Overgrinding should be avoided, however, particularly of ore like Sample 5. (Vein "D". *See* microscopic sizing of gold particles.)

The hydrometallurgical investigations suggest that the gold occurs in vein "A" somewhat similarly to that in vein "D", although the sections made of Sample 3 of vein "A", when examined under the microscope, did not reveal any particular form of occurrence. (*See* Tests Nos. 24 and 25.)

The flow-sheet that seems most applicable to a mill feed of ore from the three veins, A, C, and D, is outlined in Figure 1. The finer points of this treatment have yet to be worked out. The proper treatment for the gravity concentrates alone demands much further work. Whether to effect a much finer overall grind of these concentrates, decrease or increase periods of cyanide contact and strength of reagent, calcine or stock-pile these concentrates before or after cyanidation are points that should be investigated on a pilot-mill scale rather than on that of laboratory tests before the construction of any regrind circuit.

The use of strakes (blanket tables) may be dispensed with upon continued mill operation and with improvements in operating details and operating skill. Until this stage is reached, their use will reduce the amount of free gold in the shipping copper concentrate.

A small amount of cyanide returning to the main grinding-classifying circuit by way of the regrind unit thickener underflow may be beneficial in increasing the copper grade of the shipping concentrate without a soluble loss of gold.

It is recommended that such details as are outlined above be investigated by a mill-scale test before the details of final construction and equipment are settled. Facilities for this work are now provided at the Ore Dressing Laboratories of the Bureau of Mines at Ottawa.

The high consumptions of lime can be reasonably expected to be lessened in practice, for the following reasons:

Lime consumption in laboratory testing is always higher than in practice, owing to adherence of carbonate to glass equipment, similar to the "primary absorption" of a regular mill circuit.

Mill feed ore will probably be less oxidized than the samples under investigation.

Lime used in the flotation circuit will be largely recoverable in the overflow of the flotation tailing dewatering thickener, such an overflow possibly being used as grinding solution. The possible fouling of the flotation circuit due to this return solution would not be serious, as in this flow-sheet the function of flotation is to remove the cyanicide chalcopyrite prior to cyanidation, not to effect gold extraction in the concentrate.

The flow-sheet (Figure 1) is only intended to outline that treatment which is indicated by this investigation as being most suitable.

Several phases of this flow-sheet must be further investigated before being considered in practice:

1. Degree of primary grind;
2. Degree of secondary grind for gravity concentrates;
3. Period of cyanidation contact in regrind unit; and
4. Subsequent treatment most advisable for regrind unit discharge.

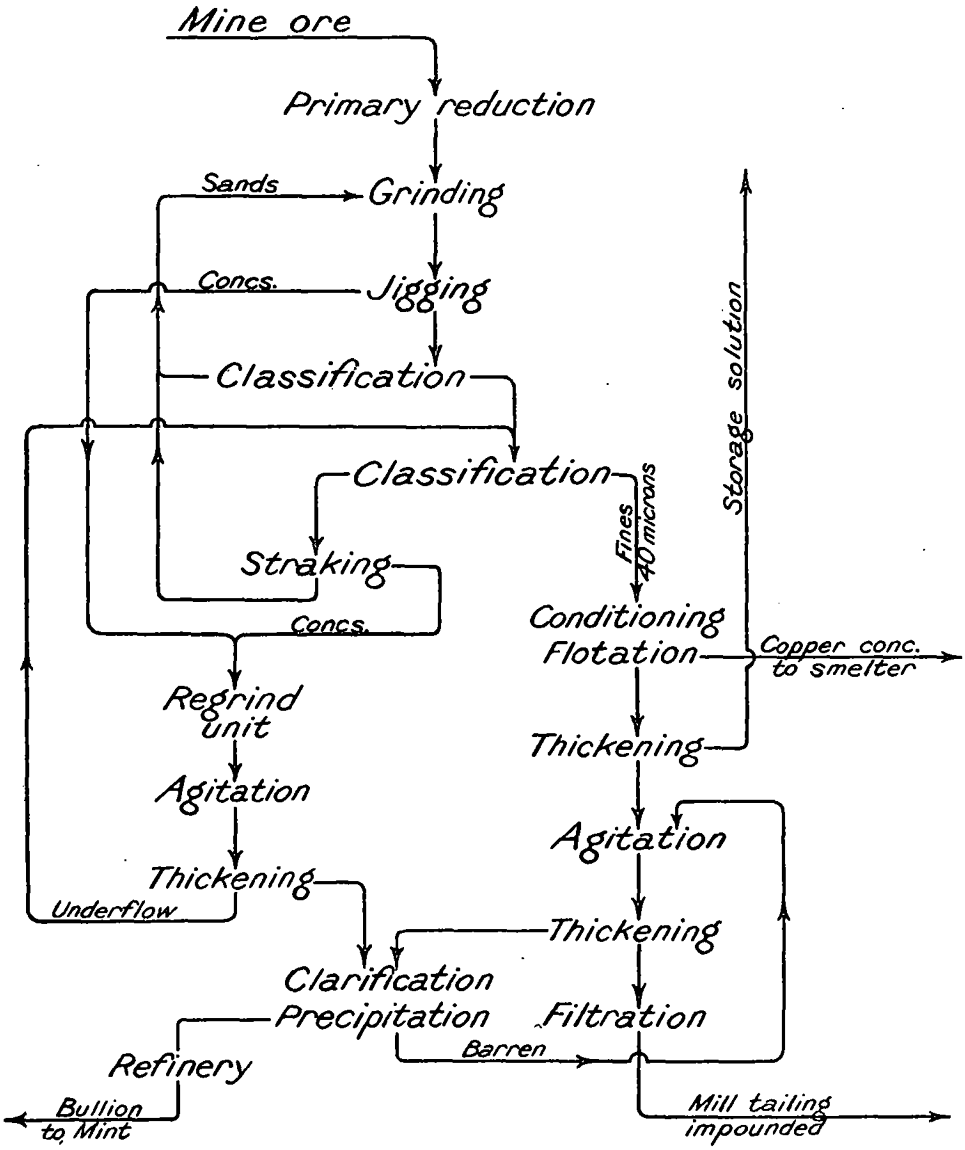


Figure 1. Flow-sheet of suggested treatment of ore from Chibougamau property, Obalski Mining Corporation.

Ore Dressing and Metallurgical Investigation No. 760

FLOTATION CONCENTRATE FROM THE TOMBILL GOLD MINES, LIMITED, EMPIRE, ONTARIO

Shipment. One can of flotation concentrate, one can of flotation water, and one can of barren solution were received on November 3, 1938, from the Tombill Gold Mines, Limited, Empire, Ontario. The samples were submitted by W. S. Hargraft, Mill Superintendent.

Purpose of Investigation. From the beginning of operations in the Tombill mill, difficulties were encountered in the cyanidation of the flotation concentrate. It was suggested that an unknown condition was causing re-precipitation of the gold from solutions at various points in the circuit.

The investigation was carried out to check this suggestion and to provide a method for improved recovery.

Analysis of the Concentrate. The pulp was mixed and a grab sample assayed as follows:

Gold	3·76 oz./ton
Silver	0·64 "

Results of Investigation. The results of the investigation show that a tailing of 0·175 ounce of gold per ton is made by grinding in water against tailings of from 0·35 to 0·73 ounce of gold per ton by grinding directly in cyanide solution. This improved extraction is primarily due to finer grinding and may be influenced somewhat by the removal of soluble salts in a water grind.

The experimental data and a discussion of the results follow in detail.

Experimental Results

Analyses of Solution Submitted

A.—Flotation Water

Reducing power	Nil
NaCNS	Nil
Ferrous iron	Nil
Lime (CaO)	0·08 lb./ton
Magnesia (MgO)	0·024 "
Nickel	Nil
Chromium	Nil
H_2S metals	Nil

B.—Barren Solution

Reducing power	120 c.c. $\frac{N}{10}$ $KMnO_4$/litre
NaCNS	0·10 grm./litre
Ferrous iron	0·11 "
Nickel	0·003 "
Chromium	less than 0·0000005 per cent
Total alkalinity	0·92 lb./ton (as CaO)
pH	10·70

There is no indication of any harmful material in the flotation water. The barren solution shows a relatively high content of ferrous iron, which indicates a reduced condition of the pulp.

CYANIDE TREATMENT OF CONCENTRATE

Tests Nos. 1 to 4

Samples of concentrate were ground in cyanide solution to a fineness of approximately 91 per cent —200 mesh. The ground pulp was diluted to 3 : 1 and was agitated in cyanide solution, strength 2 pounds of sodium cyanide per ton. Lime was added to maintain a protective alkalinity. The agitation periods for the respective samples were 4, 8, 12, and 24 hours.

Results:

Test No.	Agitation, hours	Tailing, Au, oz./ton	Extraction of gold, per cent	Final titration, lb./ton solution		Reagent consumption, lb./ton	
				NaCN	CaO	NaCN	CaO
1	4	0·35	90·7	1·64	0·10	1·68	3·20
2	8	0·47	87·5	1·80	0·08	2·20	3·76
3	12	0·40	89·4	1·92	0·14	2·04	4·58
4	24	0·73	80·6	1·56	0·12	2·22	3·64

The results of these tests would indicate that re-precipitation was taking place, but somewhat similar tests carried out later (*See* Tests Nos. 15-17) do not substantiate such a conclusion.

Tests Nos. 5 and 6

In Test No. 5 the sample of concentrate was first washed and filtered before being ground and cyanided as in Test No. 4.

In Test No. 6 the sample was dried before grinding and agitation. The period of agitation was 24 hours.

Results:

Test No.	Tailing, Au, oz./ton	Extraction of gold, per cent	Final titration, lb./ton solution		Reagent consumption, lb./ton	
			NaCN	CaO	NaCN	CaO
5	0·56	85·1	1·60	0·08	2·40	3·76
6	0·69	81·6	1·88	0·10	2·76	3·20

The tailing from the washed concentrate is slightly lower. (Compare Tests Nos. 4 and 5). Analysis of the solutions from these tests shows slightly less fouling in the solution from the washed concentrate (Test No. 5).

—	Test No. 4	Test No. 5	
		Washing	Cyanide solution
Reducing power, c.c. $\frac{N}{10}$ $KMnO_4$/litre	152	4	128
NaCNS, grm./litre	0·12	....	0·10
Ferrous iron, grm./litre	0·06	Nil	0·05
Total sulphur, grm./litre	....	0·04	....

The water wash of the concentrate shows the presence of soluble sulphur, which may be due to residual flotation reagents.

Tests Nos. 7 and 8

Two portions of the tailing from Test No. 5 (gold, 0·56 ounce per ton) were re-agitated in fresh cyanide, strength 2 pounds of sodium cyanide per ton, for 6 hours and 24 hours respectively. There was no change in 6 hours, but in 24 hours the tailing was lowered to 0·345 ounce per ton.

This indicates that fresh solution after long agitation promotes extraction of exposed gold. By exposed gold is meant that which comes in contact with the solution, but which may have been coated or otherwise inhibited to dissolution in the initial period of agitation.

Tests Nos. 9 to 14

This series was carried out under the following conditions:

a. Concentrate ground in cyanide with 0·5 pound of litharge per ton.
b. Potassium permanganate, 0·2 pound per ton, added to agitation.
c. Concentrate ground in water-lime circuit, filtered, and repulped in cyanide solution.

The grinding for each test was approximately the same, 91 per cent —200 mesh. The strength of solution was 2 pounds of sodium cyanide per ton and the pulp dilution, 3 : 1.

Results:

Test No.	Agitation, hours	Special reagent	Tailing, Au, oz./ton	Extraction of gold, per cent	Final titration, lb./ton solution		Reagent consumption, lb./ton	
					NaCN	CaO	NaCN	CaO
9	6	Litharge	0·54	85·6	1·72	0·20	2·04	5·40
10	24	Litharge	0·51	86·4	1·92	0·22	2·04	6·34
11	6	Potassium permanganate	0·515	86·3	2·04	0·40	0·48	4·80
12	24	Potassium permanganate	0·47	87·5	1·96	0·10	1·22	6·70
13	6	None (water grind)	0·45	88·0	1·60	0·16	3·00	5·52
14	24	None (water grind)	0·41	89·1	1·84	0·10	3·68	7·70

The above tests show no indication of re-precipitation. The consumption of cyanide is influenced by the addition of oxidizing agents. This is most apparent in Tests Nos. 11 and 12, in which potassium permanganate was used. The water grind shows the lowest tailing, although the consumption of cyanide is higher than in the other tests. Analysis of the solutions shows that in the water grind the fouling is lower than in the other tests of the series.

Analysis of Solutions:

Test No.	Agitation, hours	Reducing power, c.c. $\frac{N}{10}KMnO_4$/l.	KCNS, grm./litre	Ferrous Fe grm./litre
9	6	140	0·13	0·033
10	24	176	0·13	0·028
11	6	168	0·13	0·028
12	24	244	0·16	0·033
13	6	76	0·06	0·011
14	24	152	0·13	0·022

The following tests were carried out with finer grinding than in the preceding tests.

Tests Nos. 15, 16, and 17

The concentrate was ground in a cyanide pulp to a fineness of 91 per cent −325 mesh. The periods of agitation were 6 and 24 hours. Lime and cyanide were added at intervals to maintain a lime content of around 0·10 pound per ton and a cyanide strength of 2 pounds per ton.

Results:

Test No.	Agitation, hours	Tailing, Au, oz./ton	Extraction of gold, per cent	Final titration, lb./ton of solution		Reagent consumption, lb./ton	
				NaCN	CaO	NaCN	CaO
15	6	0·35	90·70	1·92	0·22	4·24	8·34
16	24	0·24	93·62	1·88	0·12	3·96	7·64
17	24	0·34	90·90	1·98	0·06	4·66	11·82

The wide variation in lime consumption recorded in these tests is probably due to oxidation of the concentrate, as Tests Nos. 15 and 17 were carried out several weeks later than Test No. 16.

A decided increase was found in both lime and cyanide consumption on concentrate after standing for some time.

Tests Nos. 18 to 21

These were carried out by grinding in water and 2 pounds of lime per ton of concentrate to a fineness of 95 per cent −325 mesh. The pulp was filtered and repulped in cyanide solution at a dilution of 3 : 1. In Tests No. 18 and 19 the cyanide strength was maintained at 2 pounds of sodium cyanide per ton and in Tests Nos. 20 and 21 at 3 pounds sodium cyanide per ton.

Results:

Test No.	Agitation, hours	Tailing, Au, oz./ton	Extraction of gold, per cent	Final titration, lb./ton of solution		Reagent consumption, lb./ton	
				NaCN	CaO	NaCN	CaO
18	6	0·235	93·75	1·92	0·06	2·24	5·82
19	24	0·24	93·62	1·96	0·04	2·92	9·88
20	36	0·175	95·35	2·52	0·16	5·84	17·55
21	48	0·175	95·35	3·18	0·14	8·66	18·58

An analysis of the solutions from Tests Nos. 19 and 21 indicates that finer grinding promotes greater fouling.

Test No.	Agitation, hours	Reducing power, c.c. $\frac{N}{10}KMnO_4$/l.	NaCNS, grm./litre	Ferrous Fe, grm./litre	Nickel
19	24	292	0·20	0·06	—
21	48	480	0·34	0·08	Trace

The results obtained in Tests Nos. 18 to 21 show a decided improvement in extraction and indicate that fine grinding in water is a definite factor in the promotion of making lower tailings.

The high consumption of reagents is largely due to the reason already mentioned and should not occur in freshly made concentrate.

INFRASIZER TEST ON CYANIDE TAILING

Test No. 19

A sample of the cyanide tailing (Au, 0·24 oz./ton) was infrasized in the Haultain infrasizer in order to determine the gold content of the sized grains of sulphide. Grinding: 95 per cent −325 mesh.

Results of Infrasizer Test:

Microns	Weight per cent	Assay		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S
Over 56	4·14	0·53	30·12	9·09	4·15
Between 56 - 40	13·11	0·44	28·90	23·91	12·61
" 40 - 28	17·01	0·29	29·52	20·44	16·72
" 28 - 20	15·75	0·235	31·10	15·34	16·31
" 20 - 14	13·06	0·20	32·50	10·82	14·13
" 15 - 10	10·20	0·155	29·88	6·55	10·15
Under 10	26·73	0·125	29·14	13·85	25·93
Total	100·00	0·24	30·04	100·00	100·00

The results show that fine grinding is essential to expose the gold to the action of cyanide. The assays indicate that grinding to a fineness of under 20 microns would expose a considerable portion of the gold at present held in the pyrite grains. The −10-micron grains are shown to contain 13·85 per cent of the gold in the tailing. This amount of gold may be considered as non-recoverable by cyanidation.

CONCLUSIONS

The maintaining of uniform surface conditions on a single sample of flotation concentrate during an investigation covering a period of weeks is difficult. Oxidation is likely to occur that may result in an increased consumption of reagents. These changes in the concentrate, although slight, might easily account for any variation in similar tests carried out during the period of the investigation. Despite this difficulty, information has been gained that throws considerable light on the problems encountered in the cyanidation of the concentrate.

The investigation indicates definitely that very fine grinding is essential for extraction of the gold by cyanidation. The relationship of gold content to size of sulphide grains is shown in the result of infrasizing the cyanide tailing from Test No. 19. The tests also show that the gold is soluble within 48-hours' agitation time on finely ground concentrate. (*See* Test No. 21.)

With regard to the action of reducing salts and possibly adhering flotation reagents, the evidence, although not conclusive, suggests that they may be a factor in causing poor cyanidation. The analysis of the sample of barren solution submitted shows the presence of considerable ferrous iron. This condition suggests oxygen depletion. Addition of an oxidizing agent, such as potassium permanganate (*See* Tests Nos. 11 and 12), has a marked effect on the amount of cyanide consumed. It is assumed that this salt prevents the formation of ferrous and sulphide compounds. These, in their transitory combinations before becoming stable cyanide salts, tend to set up reversible chemical reactions, which apart from consuming cyanide also tend to retard the dissolution of gold.

Grinding in water prior to agitation in cyanide has the advantage of removing any possible flotation reagents or soluble salts. Although the results by extraction do not indicate any conclusive advantage by this treatment, the cyanide solutions from water grinding show a decidedly lower thiocyanate and ferrous iron content. (*See* Tests Nos. 13 and 14.)

Ore Dressing and Metallurgical Investigation No. 761

GOLD ORE FROM THE THOMPSON LUNDMARK GOLD MINES, LIMITED, YELLOWKNIFE, NORTHWEST TERRITORIES

Shipment. One bag of sample rejects, weighing 77 pounds, was received on November 9, 1938, from E. V. Neelands, consulting engineer for the Thompson Lundmark Gold Mines, Limited, 2810, 25 King Street West, Toronto, Ontario.

Location of the Property. The property of the Thompson Lundmark Gold Mines, Limited, from which the present shipment was received is situated at Thompson Lake, Yellowknife Mining Division, Northwest Territories.

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

Gold	0·50 oz./ton
Silver	0·12 "
Copper	0·01 per cent
Iron	2·03 "
Sulphur	0·09 "
Arsenic	Nil

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

The *gangue* consists of translucent white quartz and an assemblage of highly siliceous, schistose rock minerals. In some hand specimens it appears to be a silicified tourmaline-mica schist cut by stringers of quartz and locally stained with iron oxides.

Metallic minerals are present in the sections in extremely small quantities. In their approximate order of decreasing abundance they are: pyrite, pyrrhotite, "limonite", sphalerite, chalcopyrite, and native gold. Pyrite is present largely as medium to small irregular grains disseminated in both schist and quartz. A number of grains show alteration to "limonite", some being completely rimmed by this mineral. Small irregular grains of pyrrhotite containing tiny inclusions of gangue are disseminated in the schist, as are rare tiny grains of chalcopyrite. As already noted, "limonite" is visible attacking pyrite, and as rust stains in gangue. One small mass of rather light-coloured sphalerite occurs in quartz.

Two grains of native gold are visible under the microscope. Both occur in quartz. One grain is coarse and visible megascopically, and the other is relatively fine, being 30 microns (−400 mesh) in size. In a test made, coarse gold was panned from the concentrate.

Experimental Tests

The ore was tested with a view to discovering whether treatment by mechanical recovery plus amalgamation was feasible; also whether the ore was amenable to flotation; and finally as to the response of the ore to cyanidation.

Concentration by means of traps, jigs, and blankets, followed by amalgamation of these concentrates, was successful in recovering up to 87 per cent of the gold in the ore.

Owing the the comparatively large amount of rather coarse gold in the ore and the small amount of sulphides, straight flotation was not feasible. Cyanidation proved successful, the gold dissolving readily in cyanide and giving recoveries of 97 to 98 per cent at a grind of under 50 per cent −200 mesh.

The investigative work is divided into two parts. Part I covers concentration and amalgamation, and Part II covers cyanidation.

Part I

BARREL AMALGAMATION

Test No. 1 (A and B)

The ore at −14 mesh was ground in a ball mill to pass 41·8 per cent −200 mesh in Test No. 1A and 68·9 per cent in Test No. 1B. The pulps were amalgamated with mercury in a jar mill, the amalgam was separated, and the amalgamation tailings assayed for gold.

Results of Amalgamation:

Test No.	Grind, per cent −200 mesh	Assay, Au, oz./ton		Recovery of gold, per cent
		Feed	Tailing	
1A	41·8	0·50	0·055	89·0
1B	68·9	0·50	0·035	93·0

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

TRAP AND FLOTATION CONCENTRATION

Test No. 2

The ore at −14 mesh was ground in a ball mill to pass 69·3 per cent −200 mesh. The pulp was passed through a hydraulic classifier or trap and the trap tailing was transferred to a Denver flotation machine. The pulp was conditioned with 2 pounds of soda ash per ton and floated with 0·10 pound of amyl xanthate and 0·05 pound of pine oil per ton.

A screen test showed the grinding as follows:

Mesh	Weight, per cent
− 48 + 65	0·2
− 65 +100	3·5
−100 +150	12·0
−150 +200	15·0
−200	69·3
	100·0

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Trap Concentration:				
Feed	100·00	0·50	100·0	
Trap concentrate	0·61	54·26	66·2	164 : 1
Trap tailing	99·39	0·17	33·8	
Flotation:				
Feed	100·00	0·17	100·0	
Flotation concentrate	1·89	7·06	78·0	53 : 1
Flotation tailing	98·11	0·04	22·0	

The pH of the pulp was 8·9.

The trap concentrate was amalgamated, with the following results:

Assay Feed	Au, oz./ton Tailing	Recovery, per cent
54·26	0·425	99·2

Summary of Test No. 2:

	Per cent
Gold recovered as trap concentrate	66·2
" " flotation concentrate	26·4
Overall recovery of gold	92·6
Gold recovered by amalgamation of trap concentrate	65·7

TRAP, BLANKET, AND FLOTATION CONCENTRATION

Test No. 3

This was conducted similarly to Test No. 2, with the exception that a corduroy blanket was placed between the trap and flotation machine. The blanket was set at a slope of 2·5 inches per foot.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Trap Concentration:				
Feed	100·00	0·50	100·0	
Trap concentrate	0·66	54·68	72·2	152 : 1
Trap tailing	99·34	0·14	27·8	
Blanket Concentration:				
Feed	100·00	0·14	100·0	
Blanket concentrate	0·52	11·62	43·1	192 : 1
Blanket tailing	99·48	0·08	56·9	
Flotation Concentration:				
Feed	100·00	0·095*	100·0	
Flotation concentrate	1·30	6·28	84·6	77 : 1
Flotation tailing	98·70	0·015	15·4	

* Calculated.

The trap and blanket concentrates were combined, reground, and amalgamated. The flotation concentrate was treated similarly.

Amalgamation:

Product	Assay, Au, oz./ton		Recovery of gold, per cent
	Feed	Tailing	
Trap and blanket concentrates	36·63	0·46	98·7
Flotation concentrate	6·28	1·22	80·6

Summary of Results:

	Per cent
Gold recovered as trap concentrate	72·2
" " blanket concentrate	12·0
" " flotation concentrate	13·4
Overall recovery	97·6
Gold recovered by amalgamation of trap and blanket concentrates	83·1
" " " flotation concentrate	10·8

JIG AND FLOTATION CONCENTRATION

Test No. 4

The ore at −14 mesh was ground in a ball mill to pass 53·1 per cent −200 mesh and the pulp passed through a Denver jig. The jig tailing was reground with 2 pounds of soda ash and 0·08 pound of Barrett No. 4 oil per ton to pass 69·3 per cent −200 mesh and was transferred to a flotation machine. A flotation concentrate was obtained by the addition of 0·10 pound of amyl xanthate and 0·05 pound of pine oil per ton. The jig concentrate and the flotation concentrate were reground and amalgamated separately.

Screen tests showed the grinding as follows:

Mesh	Weight, per cent	
	Jig tailing	Flotation tailing
− 48 + 65	1·7	0·2
− 65 +100	9·8	3·5
−100 +150	19·7	12·0
−150 +200	15·7	15·0
−200	53·1	69·3
	100·0	100·0

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Jig Concentration:				
Feed	100·00	0·50	100·0	
Jig concentrate	4·46	9·50	84·7	22·4 : 1
Jig tailing	95·54	0·08	15·3	
Flotation of Jig Tailing:				
Feed	100·00	0·09*	100·0	
Flotation concentrate	1·78	4·33	84·0	56·2 : 1
Flotation tailing	98·22	0·015	16·0	

* Calculated.

Amalgamation of Concentrates:

Product	Assay, Au, oz./ton		Recovery of gold, per cent
	Feed	Tailing	
Jig concentrate	9·50	0·135	98·6
Flotation concentrate	4·33	1·53	64·7

Summary of Results, Test No. 4:

	Per cent
Gold recovered as jig concentrate	84·7
" " flotation concentrate	12·8
Overall recovery	97·5
Gold recovered by amalgamation of jig concentrate	83·5
" " " flotation concentrate	8·3

JIG, BLANKET, AND FLOTATION CONCENTRATION—AMALGAMATION

Test No. 5

This was conducted similarly to Test No. 4, with the exception that a corduroy blanket was placed after the jig and prior to regrinding the pulp for flotation concentration.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·50	100·0	
Jig concentrate	4·32	9·36	80·9	23·1 : 1
Jig tailing	95·68	0·10	19·1	
Blanket Concentration:				
Feed	100·00	0·10	100·0	
Blanket concentrate	0·34	13·30	45·2	294 : 1
Blanket tailing	99·66	0·055	54·8	
Flotation:				
Feed	100·00	0·055	100·0	
Flotation concentrate	1·51	2·66	73·1	66·2 : 1
Flotation tailing	98·49	0·015	26·9	

Amalgamation of Concentrate:

Product	Assay, Au, oz./ton		Recovery of gold, per cent
	Feed	Tailing	
Jig and blanket concentrate	9·62	0·275	97·15
Flotation concentrate	2·66	2·65	....

Summary of Results, Test No. 5:

	Per cent
Gold recovered as jig concentrate	80·9
" " blanket concentrate	8·6
" " flotation concentrate	7·7
Overall recovery	97·2
Gold recovered by amalgamation from jig and blanket concentrates	86·95

FLOTATION CONCENTRATION

Test No. 6

In order to show the effect of flotation, without prior removal of the coarse gold, the ore was ground with 1·5 pounds of soda ash and 0·12 pound of Barrett No. 4 oil per ton to pass 80·3 per cent −200 mesh.

The pulp was transferred to a flotation machine and a flotation concentrate was obtained by the addition of 0·05 pound of pine oil and 0·10 pound of amyl xanthate per ton. This flotation concentrate was cleaned in a smaller machine and the flotation tailing passed over a corduroy blanket.

Results:

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·50	100·0	
Flotation concentrate	2·20	10·82	47·6	44·5 : 1
Flotation middling	1·87	5·91	22·1	53·5 : 1
Flotation tailing	95·93	0·155	30·3	

The pH of the pulp was 8·4.

Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·155	100·0	
Blanket concentrate	0·50	28·02	90·4	200 : 1
Blanket tailing	99·50	0·015	9·6	

A portion of the flotation tailing was panned and showed rather coarse free gold particles.

Summary of Results:

	Per cent
Gold recovered as rougher flotation concentiate	69·7
Gold recovered as blanket concentrate	27·4
Overall recovery	97·1

Portions of the flotation tailing from Tests Nos. 3 and 4 and the blanket tailing from Test No. 6 were concentrated on the Haultain superpanner. The resulting concentrates showed no gold under the microscope and consisted mostly of tourmaline and a little pyrrhotite.

Part II

STRAIGHT CYANIDATION

Test No. 1 (A, B, C, and D)

The ore at −14 mesh was ground in a ball mill in cyanide solution of 1 pound per ton strength to different degrees of fineness. The pulps were agitated for 24- or 48-hour periods. Four pounds of lime per ton of ore was added to the grind in order to maintain protective alkalinity.

Screen tests showed the grinding as follows:

Mesh	Weight, per cent	
	Tests Nos. 1A and 1B	Tests Nos. 1C and 1D
− 35 + 48	0·6	....
− 48 + 65	3·6	0·1
− 65 +100	15·6	2·6
−100 +150	21·0	11·5
−150 +200	15·7	15·5
−200	43·5	70·3
	100·0	100·0

Results of Cyanidation:

Feed: gold, 0·50 oz./ton

Test No.	Agitation, hours	Grind, per cent −200 mesh	Tailing assay, Au, oz./ton	Extraction, per cent	Titration, lb./ton of solution		Reagents consumed, lb./ton of ore	
					NaCN	CaO	NaCN	CaO
1A	24	43·5	0·015	97·0	1·00	0·30	0·30	3·4
1B	48	43·5	0·01	98·0	0·92	0·35	0·35	3·4
1C	24	70·3	0·005	99·0	0·84	0·28	0·40	3·5
1D	48	70·3	0·005	99·0	0·96	0·26	0·45	3·5

The pregnant solutions were assayed for reducing power and KCNS, with the following results:

Test No.	Agitation, hours	Reducing power, $\frac{N}{10}KMnO_4$ c.c./litre	KCNS, grm./litre
1A	24	16	0·016
1B	48	24	0·02
1C	24	28	0·016
1D	48	32	0·03

Test No. 2

CONCENTRATION—AMALGAMATION—CYANIDATION

The ore at −14 mesh was ground in cyanide solution of 1 pound of sodium cyanide per ton to pass 70·3 per cent −200 mesh. The pulp was passed through a Denver jig and the jig concentrate reground and amalgamated. The amalgam residue was added to the jig tailing and this product was agitated in cyanide solution for 24 hours.

After grinding in cyanide solution the jig feed assayed 0·32 ounce of gold per ton.

Results:
Jig Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·32	100·0	29·2 : 1
Jig concentrate	3·43	6·94	74·3	
Jig tailing	96·57	0·085	25·7	

After amalgamation the combined amalgam residue and jig tailing assayed 0·09 ounce of gold per ton.

Cyanidation of Amalgam Residue and Jig Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
2	24	0·09	0·01	88·9	1·00	0·30	0·30	3·4

Summary of Results, Test No. 2:

	Per cent
Gold extracted by cyanide grind	36·0
" recovered by amalgamation of jig concentrate	46·0
" extracted by agitation	16·0
Overall recovery	98·0

Test No. 3

This was similar to Test No. 2 with the exception that traps and blankets were substituted for the jig. After grinding in cyanide solution to pass 70·3 per cent −200 mesh the trap feed assayed 0·35 ounce of gold per ton.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Trap Concentration:				
Feed	100·00	0·35	100·0	278 : 1
Trap concentrate	0·36	29·41	30·3	
Trap tailing	99·64	0·245	69·7	
Blanket Concentration:				
Feed	100·00	0·245	100·0	233 : 1
Blanket concentrate	0·43	44·23	77·6	
Blanket tailing	99·57	0·055	22·4	

The trap and blanket concentrates were reground and amalgamated. The amalgam residue was added to the blanket tailing, this product assaying 0·06 ounce of gold per ton.

Cyanidation of Amalgam Residue and Blanket Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, gold, per cent	Titration, lb./ton solution		Reagents, consumed, lb./ton ore	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
3..........	24	0·06	0·005	91·7	0·92	0·46	0·30	3·1

Summary of Results, Test No. 3:

	Per cent
Gold extracted by cyanide grind	30·0
Gold recovered by amalgamation of combined concentrates	58·0
Gold extracted by agitation	11·0
Overall recovery	99·0

SUMMARY AND CONCLUSIONS

The metallurgy of this ore presents no great problems.

About 85 to 90 per cent of the gold can be concentrated by a simple flow-sheet consisting of jigs at the ball mill discharge followed by blanket concentration of the classifier overflow. The combined concentrates, representing a concentration of 80 : 1, after barrel-amalgamation are reduced from 9·62 ounces to 0·27 ounce of gold per ton. This represents an overall recovery of 86·95 per cent.

Flotation alone is not suitable. The coarse gold tends to pass into the tailing. By passing the flotation tailing over blankets, a total recovery of 97 per cent from the two concentrates is indicated.

Flotation preceded by jigging to remove coarse gold shows a recovery of 97·5 per cent as concentrates. When these are barrel-amalgamated a total recovery of 91·8 per cent of the gold in the mill feed is obtained.

Fine grinding is not essential to obtain the above results; 70 per cent −200 mesh apparently is all that is necessary.

Straight cyanidation with a jig in the ball mill-classifier circuit and a grind of 70 per cent −200 mesh results in an extraction of 98 to 99 per cent within 24 hours. The cyanide and lime consumptions are low, with no evidence of solution fouling.

An intermediate flow-sheet worthy of consideration would be jigs and flotation with amalgamation and cyanidation of the concentrates.

The flow-sheet to adopt can best be determined by a study of the economics. The least costly plant will yield the lowest recovery, whereas a fully equipped cyanide plant will yield the highest extraction.

III

INVESTIGATIONS THE DETAILS OF WHICH ARE NOT PUBLISHED

Ore or Product	Source of Shipment	Address
Gold	Ronda Gold Mines, Limited	Westree, Sudbury District, Ont.
Molybdenite	Molydor Mines, Limited	Loon, Ont.
Mill tailing	Asbestos Corporation	Thetford, Que.
Gold	Amm Gold Mines, Limited	Amos, Que.
Gold	Magnet Consolidated (1936) Mines, Limited.	Little Long Lac, Township of Errington, Ont.
Mill run	Canadian Wood Molybdenite Company.	Quyon, Que.
Molybdenite	Amorada Gold Mines, Limited	Dorothea Township, Beardmore, Ont.
Concentrate	Payore Gold Mines, Limited	Bourlamaque, Que.
Gold	Cochenour Willans Gold Mines, Limited.	McKenzie Island, Ont.
Gold-quartz	Preston East Dome Mines, Limited.	South Porcupine, Ont.
Gold ore and mill products	Arntfield Gold Mines, Limited	Arntfield, Que.
Gold ore and mill products	Sand River Gold Mining Company, Limited.	Beardmore, Ont.
Mill tailing	Orelia Mines, Limited	Rainy River District, Northwestern Ont.
Gold	"Dugan Option" of Tyranite Mines, Limited.	Gowganda, Ont.
Asbestos tailing	Canadian Johns-Manville Company, Limited.	Asbestos, Que.
Gold	Dome Mountain Mine	Smithers, B.C.
Gold	Tyranite Mines, Limited	Gowganda, Ont.
Gold	Hiawatha Gold Mines, Limited	Oba, Ont.
Gold	Magpie Junction	District of Algoma, Sault Ste. Marie Mining Division, Ont.
Silver	Coniagas Mine	Cobalt, Ont.
Gold	"Dugan Option" of Tyranite Mines, Limited (Supplementary).	Gowganda, Ont.
Gold	Slave Lake Mine	Outpost Island, Great Slave Lake, N.W.T.
Gold	Pan-Canadian Gold Mines, Limited.	Heva River, Que.
Cobalt	W. E. MacCready	Cobalt, Ont.
Gold-silver-copper	Grotto Mine	Usk, B.C.
Gold	Townships of Kennebec and Barrie	Frontenac County, Ont.
Gold	Alsac Mines, Limited	Beardmore, Ont.
Mill product	Lapa Cadillac Gold Mines, Limited	Heva River, Que.
Mill product	Tyranite Mines, Limited	Gowganda, Ont.
Gold	Halcrow-Swayze Mines, Limited	Township of Bryce, Ont.
Pitchblende	Eldorado Gold Mines, Limited	Great Bear Lake, N.W.T.
Gold	Rochette Gold Mines Company, Limited.	Launay Township, Northwestern Que.
Concentrate and ore	Hard Rock Gold Mines, Limited (work incomplete).	Geraldton, Ont.
Gold	Chesterville Larder Lake Gold Mining Company (second shipment, work incomplete).	Larder Lake District, Ont.
Cast steel grinding balls	Britannia Mining and Smelting Company, Limited	Britannia Beach, B.C.

An examination of the steel of the gratings of two low-discharge tube mills. (Lake Shore Mines, Limited.)

An examination of an austenitic manganese steel ball mill liner. (Sorel Steel Foundries, Limited.)

A determination of the elastic properties of two austenitic stainless steels. (Atlas Steels, Limited.)

An examination of two defective "bronze" bolts from H.M.C.S. "Gaspe". (Department of National Defence.)

The testing of a wire hoisting cable. (Lamaque Gold Mines, Limited.)

A determination of the elastic properties of three duraluminium test bars. (Department of National Defence.)

An examination of two austenitic manganese steels of special analysis. (Sorel Steel Foundries, Limited.)

An examination of three austenitic manganese steels. (Sorel Steel Foundries, Limited.)

An examination of two austenitic manganese steels. (Sorel Steel Foundries, Limited.)

Identification of worn numbers on bird bands. (National Parks Branch, Department of Mines and Resources.)

Impact tests on steels. (Canada Car and Foundry.)

Hardness tests on steel grinding balls. (Hull Iron and Steel Foundries, Limited.)

The casting of thirty nickel-chromium steel heat-resisting trays. (Royal Mint.)

Microscopic examination of two aeronautical structural steels. (Department of National Defence.)

A determination of the impact strength of a steel. (Dominion Engineering Company, Limited.)

Microscopic study of products from Beattie Gold Mines, Limited, Duparquet, Que.

Microscopic examination of specimens from Quebec Manitou Mines, Val d'Or, Que.

Microscopic examination of magnetic product from Canadian Johns-Manville Corporation, Asbestos, Que.

Examination of rock sample from Cape Breton Island, N.S.

Examination of two specimens from Aldermac Copper Corporation, Arntfield, Que.

Microscopic examination of sample of gold ore from Chan Yellowknife Gold Mines, Limited, Yellowknife, N.W.T.

Grain analysis of pyrite concentrate from Aldermac Copper Corporation, Arntfield, Que.

Microscopic examination of two mill products from Aldermac Copper Corporation, Arntfield, Que.

Microscopic examination of sulphide ore from Hard Rock Gold Mines, Limited, Geraldton, Ont.

Study of mode of occurrence of gold in a table concentrate from Sherritt-Gordon Mines, Limited, Sherridon, Man.

Investigation of mode of occurrence of nickel in tailings from Canadian Johns-Manville Company, Asbestos, Que.

Examination of five rock samples submitted by A. L. Wilson, Ignace, Ont.

Investigation of the mode of occurrence of gold in froth from No. 1 Thickener at Sullivan Consolidated Mines, Limited, Sullivan Post Office, Que.

Microscopic analysis of pyrite concentrate from Aldermac Copper Corporation, Arntfield, Que.

IV

A RÉSUMÉ OF SPECIAL INVESTIGATIONS AND RESEARCH COMPLETED, IN PROGRESS, OR UNDER CONSIDERATION

In the Research Section, problems arising in the treatment of ores received for usual mill-test investigation have occupied most of the time and effort of the staff, and several research projects in progress or under consideration have had to be temporarily suspended.

During the investigations in ore treatment, such troubles as serious fouling of cyanide solution, low extraction due to refractory or submicroscopic gold, poor or erratic precipitation of gold from cyanide solution, and mineral separation by flotation were encountered. These were studied and tests were made to get better results or to determine the maximum of extraction and recovery.

The effect on the extraction and precipitation of gold from ores containing chromium was a problem submitted for investigation as of interest to the operators of several milling plants. The investigation is not yet complete but it has not been established that chromium is the interfering constituent; some evidence is forthcoming, however, that the offending element may be nickel.

Among the ores or mill products received for testing, those from the following called for special attention:

Uchi Gold Mines, Limited, Kenora District, Ont.

Gold Cup Mining Company, Limited, Rossland, B.C.

Sand River Gold Mining Company, Limited, Beardmore, Ont.

Chesterville Larder Lake Gold Mining Company, Larder Lake, Ont. (Incomplete.)

Hard Rock Gold Mines, Limited, Geraldton, Ont. (Incomplete.)

Tyranite Gold Mines, Limited, Gowganda, Ont.

Beattie Gold Mines, Limited, Duparquet, Que.

Tombill Gold Mines, Limited, Empire, Ont.

Obalski Gold Mines, Limited, Chibougamau, Que.

Yalakom Quicksilver Claim, Lillooet Mining Division, B.C. (Cinnabar.)

Manitou Mining Co., Limited, Bridge River, B.C. (Cinnabar.)

Cobalt Products, Limited, Cobalt, Ont.

Regal Silver Property, Revelstoke, B.C.

Canadian Johns-Manville Company, Asbestos, Que. (Asbestos tailing.)

Each ore presented some problem that resulted in the adoption of a mode of treatment differing in some respect from normal practice. As pointed out in the last report the number of complex or refractory ores being submitted for test treatment is increasing.

On account of pressure of other work little progress was made in the study of the part played by accessory minerals such as sulphides in milling and cyanidation, but it will be continued at the first opportunity. Some preliminary data have been collected on the behaviour of pyrite, pyrrhotite, and galena separately.

In the Metallurgical Laboratory also, little time was available for research. The machine for testing damping properties showed defects and is being redesigned, and this has held up work on the correlation of the damping with other physical properties of cast iron and on the damping qualities of drill steel, until the machine is operating satisfactorily.

The investigation of uranium as an alloying element has been continued. A quantity of high-carbon ferro-uranium was made from the black oxide, and some ferro-uranium from sodium uranate. Six uranium steels of different uranium and carbon contents have been made and were rolled into one-inch bars. The properties of the bars in the "as rolled" and heat-treated states were determined. Difficulty arose in making these steels, owing to the rapid oxidation of the uranium. A new furnace being installed will permit melting and pouring in vacuo, which may largely prevent the rapid oxidation.

A modern type of grating spectrograph is being added to the equipment of these laboratories and is expected to be ready early in January 1939.

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