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, 1939

Price, 25 cents. **No. 806**

CONTENTS

INVESTIGATIONS IN ORE DRESSING AND METALLURGY, JULY TO DECEMBER, 1939

I

REVIEW OF INVESTIGATIONS

C. S. Parsons

Chief of Division of Metallic Minerals

Reviewing the activities of this Division for the period July to December, 1939, a substantial increase in the number of investigations carried out and formally reported can be recorded over the previous six-month period.

Seventy-five investigations were completed and reported in detail, thirteen of this number being printed in full and appearing in Section II. The remainder are listed by title only in Section III.

Section IV presents a summary of the activities of the chemical, mineralogical, physical testing and heat-treating laboratories, showing the number and origin of samples for chemical analysis, mineragraphic examination, and physical test and heat treatment.

A review of the special investigations in progress and completed is briefly described in Section V.

Summary of Investigations:

Investigations on which reports were prepared	75
Investigations completed, but on which no formal reports were issued	23
Gold-bearing ores and mill products	36
Molybdenite ores	4
Copper-zinc ore	1
Copper ore	1
Tungsten ore	1
Silver-lead ore	1
Silver-copper-cobalt ore	1
Chloridizing plant residue (silver)	1
Mill residue (cobalt)	1
Copper-nickel matte	1
Iron ore containing copper, nickel and cobalt	1
Iron oxide (magnetite) with chromium and nickel	1
Microscopic (special)	9
Steel and alloy products	16
Number of ores investigated	48

Provincially, the above ores originated as follows: Ontario, 27; Quebec, 10; Manitoba, 3; Northwest Territories, 2; New Brunswick, 2; British Columbia, 2; Nova Scotia, 1; Saskatchewan, 1.

The outbreak of the war caused an increased interest in the less abundant minerals of the strategic* class, resulting in numerous enquiries and requests for tests and examinations thereof. The work of the physical testing laboratories increased in this period, and the quickening of the war effort is expected to lead to a further increase in this type of work. The British Aeronautical Inspection Directorate has approved of the laboratories as a check test house and tests are continually being made to confirm the results of commercial test work. These check tests are fairly time-consuming, as they involve the determination of the stress-strain relationships for a material and a plotting of the results obtained. The Department of National Defence metallurgist has freely used the laboratory facilities, and the laboratory staff have assisted him on various problems not recorded in this report. Consultations and advice were an important activity involving various outside industrial concerns, the Department of National Defence, and other Government departments.

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

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

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

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

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

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

Additional Equipment. New equipment added to the new mill comprised the following:

One No. 1 Snyder ore sampler, 27-inch diameter.

One Akins classifier (Submerged Type), 12-inch.

One Dorr tray thickener. Type ATB. 4-foot diameter by 4 feet deep.

One Peacock crusher. Size 12-inch.

One Lorentsen amalgamating machine.

* The word "strategic" is used in the general sense of implying usefulness in the manufacture of munitions. It may be qualified as "sufficient", "insufficient", or "deficient", according to the relative abundance of a mineral in Canada.

II

INVESTIGATIONS THE RESULTS OF WHICH ARE RECORDED IN DETAIL

Ore Dressing and Metallurgical Investigation No. 775

GOLD ORE FROM THE DELNITE MINES LIMITED, TIMMINS, ONTARIO

Shipment. Six bags of gold ore, total weight 370 pounds, were received on May 4, 1939, from Charles S. Stevens, Mill Superintendent, Delnite Mines, Limited, Timmins, Ontario. Previous shipments by this company are covered by reports of the Department.

Location of the Property. The property of the Delnite Mines, Limited, from which the present shipment was received is situated in Deloro Township, Porcupine area, northern Ontario.

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

Gold.................	0·31 oz./ton	Magnesia (as MgO)...	7·07 per cent
Silver.................	0·09 "	Arsenic...............	0·33 "
Iron..................	4·79 per cent	Graphitic carbon......	0·05 "
Sulphur...............	1·03 "	Carbon dioxide.......	11·90 "
Copper................	Trace	Silica (as SiO_2).......	55·45 "
Lime (as CaO).......	6·40 per cent	Alumina (as Al_2O_3)....	10·04 "

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

The *gangue* is composed of dark greenish grey, schistose rock with abundant fine disseminated carbonate and a small quantity of translucent grey quartz. The green colour is possibly due to the presence of chlorite, and the rock may represent a silicified chloritic schist.

Metallic minerals are rather sparse in the sections. Pyrite and arsenopyrite predominate as coarse to fine, irregular grains and subhedral crystals disseminated throughout gangue, alone and intimately associated. Both minerals contain inclusions of gangue and chalcopyrite. Some grains of pyrite are slightly fractured and the fractures are filled with gangue, rarely with chalcopyrite. Chalcopyrite is present as occasional small, irregular grains in gangue, but its total quantity is probably not sufficient to affect cyanidation adversely. Rare tiny inclusions of pyrrhotite occur in pyrite, as do three small grains of galena.

One grain of native gold, 14 microns (−800 Tyler mesh) in size, was observed in dense pyrite.

Investigative Work. The mine operators required the following information on the ore shipment:

1. Whether a flotation tailing low enough to be discarded could be obtained?
2. What primary grind is necessary to get this flotation tailing?
3. What are the extractions possible by cyaniding the flotation concentrate and what are the grinds necessary to obtain these extractions?
4. What reagents, amounts, and place of addition give best results?

The test work gave the following information:

1. A flotation tailing of 0·005 ounce of gold per ton was consistently obtained following the removal of the coarse gold in the ore by means of jigs, traps, or blankets.

2. This flotation tailing of 0·005 ounce gold per ton can be obtained from a fineness of grinding of 73·2 per cent −200 mesh.

3. An extraction of 97·6 per cent of the gold in the combined jig, trap, or blanket concentrates plus the flotation concentrate was obtained by regrinding and cyanidation, this result giving an overall extraction of 96·2 per cent of the gold in the ore. These concentrates were ground to pass 99·5 per cent −325 mesh and agitated 48 hours in cyanide solution.

4. Three pounds of soda ash, 0·05 pound of amyl xanthate, and 0·05 pound of pine oil per ton were used in the grind; and 0·05 pound of amyl xanthate, 0·05 pound of pine oil, and 2·0 pounds of copper sulphate per ton were added to the cells.

Experimental Tests

The test work is divided into two parts. Part I deals with straight flotation of the ore and Part II with concentration of the free gold followed by flotation and cyanidation of the combined concentrates.

Part I

STRAIGHT FLOTATION

Tests Nos. 1 to 9

The ore at −14 mesh was ground in a ball mill to different degrees of fineness. The pulp was transferred to a Denver flotation cell and a flotation concentrate removed. The flotation tailing was assayed for gold, arsenic, and sulphur, and the flotation concentrate was weighed.

The Haultain superpanner was used in conjunction with the microscope to determine whether any free gold or sulphides remained in the pulp after concentration.

The results in detail follow:

8007—2

Feed: gold, 0·31 oz./ton.

Test	Grind, per cent −200 mesh	Flotation, tailing assay			Recovery of gold, per cent	Ratio of concentration	Reagents added, lb./ton	
		Au, oz./ton	Per cent				To ball mill	To flotation cells
			As	S				
1	54·3	0·14	....	0·07	58·5	12·5 : 1	3·0 soda ash............	0·10 amyl xanthate; 0·15 pine oil.
2	76·0	0·055	....	0·06	83·6	13·4 : 1	3·0 soda ash............	0·10 amyl xanthate; 0·15 pine oil.
3	86·0	0·035	....	0·08	89·5	13·3 : 1	3·0 soda ash............	0·10 amyl xanthate; 0·15 pine oil.
4	76·0	0·01	Trace	0·05	96·8	10·6 : 1	2·0 soda ash; 0·07 Aerofloat No. 31.	0·10 amyl xanthate; 0·15 pine oil.
5	76·0	0·08	0·01	0·09	76·9	9·2 : 1	2·0 soda ash; 0·17 Barrett No. 4.	0·10 amyl xanthate; 0·15 pine oil.
6	76·0	0·015	....	0·06	95·6	10·1 : 1	2·0 soda ash; 0·17 cresylic acid.	0·10 amyl xanthate; 0·10 cresylic acid; 0·05 pine oil.
7	76·0	0·04	....	0·03	88·2	11·8 : 1	2·0 soda ash............	0·10 amyl xanthate; 2·0 soda ash; 1·0 copper sulphate; 0·15 pine oil.
8	76·0	0·105	0·02	....	68·9	12·1 : 1	4·0 sodium silicate......	0·10 amyl xanthate; 0·15 pine oil; 1·0 soda ash; 1·5 copper sulphate.
9	76·0	0·09	0·02	....	73·8	10·9 : 1	3·0 sodium silicate......	0·10 amyl xanthate; 0·15 pine oil; 1·0 soda ash; 1·5 copper sulphate.

Results:

On concentration by the Haultain superpanner and microscopic examination, almost all these flotation tailings showed free gold, little or no sulphide, and small quantities of magnetite. The amount of free gold in the different tailings was largest in Tests Nos. 8 and 9, in which sodium silicate was used.

The tests show that removal of the free gold prior to flotation is essential.

DIFFERENTIAL FLOTATION

Test No. 10

Pyrite and arsenopyrite concentrates were obtained and cleaned in a smaller machine. The ore at −14 mesh was ground in a ball mill to pass 80·2 per cent −200 mesh. Two pounds of soda ash per ton of ore was added to the grind. The pulp was transferred to a flotation machine, conditioned with 0·10 pound of amyl xanthate and 0·07 pound of pine oil per ton and a pyrite concentrate was removed and cleaned in a smaller machine. Two pounds of soda ash per ton was then added to the main body of the pulp, bringing the pH up from 8·4 to 9·3, and an arsenopyrite concentrate was removed by the further addition of 1·5 pounds of copper sulphate, 0·05 pound of Reagent No. 301, and 0·07 pound of pine oil per ton. This concentrate was also cleaned in a smaller machine.

A screen test showed the grinding as follows:

Screen Analysis:

Mesh	Weight, per cent
− 48+ 65	0·1
− 65+100	1·8
−100+150	7·8
−150+200	10·1
−200	80·2
	100·0

Results of Flotation:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concen-tration
		Au, oz./ton	As, per cent	Au	As	
Feed	100·00	0·29*	0·33*	100·00	100·00	....
Pyrite concentrate	4·65	4·30	3·56	69·1	50·9	23·2 : 1
Pyrite middling	3·28	2·27	2·08	25·7	21·0	30·5 : 1
Arsenopyrite concentrate	0·65	0·57	10·17	1·3	20·3	154 : 1
Arsenopyrite middling	2·03	0·13	0·82	0·9	5·1	49·3 : 1
Tailing	89·39	0·01	0·01	3·0	2·7	....

*Calculated.

This test concluded the work of straight flotation on the ore. In Part II concentration by means of jigs, traps, or blankets preceded recovery by flotation.

Part II

TRAP CONCENTRATION AND FLOTATION

Test No. 1 (A, B, and C)

The ore at −14 mesh was ground in a ball mill to pass 76 per cent −200 mesh. The pulps were passed through a hydraulic classifier or trap and a trap concentrate was obtained. The trap tailings were conditioned with 3 pounds of soda ash per ton and were floated with the further additions of the following: (lb./ton)

Test No. 1A: 0·08 Barrett No. 4 oil; 1·5 copper sulphate; 0·08 pine oil; 0·10 amyl xanthate.
Test No. 1B: 0·07 Aerofloat No. 31; 1·5 copper sulphate; 0·08 pine oil; 0·10 amyl xanthate.
Test No. 1C: 1·5 copper sulphate; 0·15 pine oil; and 0·10 amyl xanthate.

Results of Trap Concentration:

Feed: gold, 0·31 oz./ton.

Test No.	Grind, per cent −200 mesh	Trap tailing assay, Au, oz./ton	Trap recovery, Au, per cent	Weight trap concentrate, per cent	Ratio of concentration
1A	76·0	0·145	53·2	0·56	179 : 1
1B	76·0	0·14	54·8	0·86	116 : 1
1C	76·0	0·16	48·4	0·96	104 : 1

Results of Flotation of Trap Tailing:

Product	Weight, per cent	Assay			Distribution of gold, per cent	Ratio of concentration
		Au, oz./ton	Per cent			
			As	S		
Test No. 1A						
Feed	100·00	0·145	....	....	100·0	....
Flotation concentrate	9·87	1·33	....	....	90·7	10·1 : 1
Tailing	90·13	0·015	Trace	0·04	9·3	....
Test No. 1B						
Feed	100·00	0·14	....	....	100·0	....
Flotation concentrate	11·29	1·16	....	....	93·6	8·9 : 1
Tailing	88·71	0·01	Trace	0·06	6·4	....
Test No. 1C						
Feed	100·00	0·16	....	....	100·0	....
Flotation concentrate	10·43	1·45	....	....	94·4	9·6 : 1
Tailing	89·57	0·01	Trace	0·06	5·6	....

Summary:

—	Test No. 1A	Test No. 1B	Test No. 1C
Gold recovered in trap concentrate, per cent	53·2	54·8	48·4
Gold recovered in flotation concentrate, per cent	43·4	42·3	48·7
Overall recovery, per cent	96·6	97·1	97·1

A concentration of the flotation tailing on the Haultain superpanner revealed one small piece of free gold in Test No. 1A and no free gold in Tests Nos. 1B and 1C.

CONCENTRATION AND CYANIDATION

Test No. 2

The ore at −14 mesh was crushed in a ball mill to pass 59·1 per cent −200 mesh, and the pulp was passed through a hydraulic classifier or trap as in the previous test. The trap tailing was passed over a corduroy blanket. The blanket tailing was dewatered, sampled, and reground in a ball mill with 3 pounds of soda ash, 0·05 pound of amyl xanthate, and 0·05 pound of pine oil per ton to pass 87·5 per cent −200 mesh. The pulp was transferred to a flotation machine and a flotation concentrate was removed by the additions of 0·05 pound of amyl xanthate, 0·05 pound of pine oil, and 2·0 pounds of copper sulphate per ton. The combined trap, blanket and flotation concentrates were washed and reground in cyanide solution of 2 pounds of sodium cyanide per ton strength to pass 99·5 per cent −325 mesh and were agitated for a 48-hour period.

Screen tests showed the grindings as follows:

Mesh	Weight, per cent	
	Blanket tailing	Flotation tailing
− 35+ 48	0·9	....
− 48+ 65	4·1	....
− 65+100	10·7	0·6
−100+150	14·7	4·2
−150+200	10·5	7·7
−200	59·1	87·5
Totals	100·0	100·0

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Trap and Blanket Concentration:				
Feed	100·00	0·31	100·0	....
Trap and blanket concentrate	3·19	6·38	65·6	31·3 : 1
Blanket tailing	96·81	0·11	34·4	....
Flotation of Reground Blanket Tailing:				
Feed	100·00	0·11	100·0	....
Flotation concentrate	7·91	1·33	95·8	12·6 :1
Final tailing	92·09	0·005	4·2	....

Cyanidation of Combined Concentrates:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton conc.	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
48	4·03	0·095	97·64	1·9	0·6	4·3	17·7

Summary:

	Per cent
Gold recovered in trap and blanket concentrates	65·6
Gold recovered in flotation concentrate	32·9
Gold extracted by cyanidation of combined concentrates	96·2

CONCENTRATION AND CYANIDATION

Tests Nos. 3, 4, and 5

These tests followed the procedure as given in Test No. 2, namely, trap, blanket and flotation concentration followed by cyanidation of the combined concentrates. Variations in the grinding and also aeration of the combined concentrates prior to cyanidation were used as noted.

Test No. 3:

The combined concentrates were ground in a lime pulp to pass 94 per cent −325 mesh and aerated for 16 hours prior to cyanidation. The primary and secondary grinds were similar to those of Test No. 2.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Trap and Blanket Concentration:				
Feed	100·00	0·31	100·0	....
Trap and blanket concentrates	2·88	7·39	68·7	34·7 : 1
Blanket tailing	97·12	0·10	31·3	....
Flotation of Reground Blanket Tailing:				
Feed	100·00	0·10	100·0	....
Flotation concentrate	8·40	1·14	95·4	11·9 : 1
Final tailing	91·60	0·005	4·6	....

Cyanidation of Reground, Aerated, Combined Concentrates:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton conc.	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
48	2·88	0·06	97·9	2·0	0·60	2·30	10·2*

*Ten additional pounds of lime per ton was used during grinding and aeration.

The reducing power of the final cyanide solution was 110 millilitres $\frac{N}{10}$ $KMnO_4$ per litre; KCNS was 0·09 gramme per litre.

Summary:

	Per cent
Gold recovered in trap and blanket concentrates	68·7
Gold recovered in flotation concentrate	29·9
Gold extracted by cyanidation of combined concentrates	96·5

Test No. 4

The blanket tailing was reground to pass 73·2 per cent −200 mesh and the combined concentrate 93·5 per cent −200 mesh. The concentrates were ground in cyanide.

Screen tests showed the grinding as follows:

Mesh	Weight, per cent	
	Flotation tailing	Cyanide tailing of combined concentrates
− 35+ 48	....	0·1
− 48+ 65	0·6	0·1
− 65+100	3·7	0·5
−100+150	10·5	2·1
−150+200	12·0	3·7
−200	73·2	93·5
Totals	100·0	100·0

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Trap and Blanket Concentration:				
Feed	100·00	0·31	100·0	....
Trap and blanket concentrates	2·69	7·72	67·0	37·2 : 1
Blanket tailing	97·31	0·105	33·0	....

Flotation of Reground Blanket Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·105	100·0	....
Flotation concentrate	8·82	1·14	95·6	11·3 : 1
Final tailing	91·18	0·005	4·4	....

Cyanidation of Reground Concentrates:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
48	2·81	0·095	96·6	1·9	0·55	3·10	20·40

An analysis of the final cyanide solutions showed a reducing power of 106 millilitres $\frac{N}{10}$ $KMnO_4$ per litre and 0·15 gramme KCNS per litre.

	Per cent
Gold recovered in trap and blanket concentrates......................	67·0
" " flotation concentrate..............................	31·5
Gold extracted by cyanidation of combined concentrates..............	95·2

Test No. 5

The blanket tailing was ground to pass 67·9 per cent −200 mesh prior to flotation. The combined concentrates were cyanided without regrinding.

Screen tests showed the grind of the flotation tailing and cyanide tailing of combined concentrates as follows:

Screen Analyses:

Mesh	Weight, per cent	
	Flotation tailing	Cyanide tailing of combined concentrates
− 35+ 48	....	3·6
− 48+ 65	0·7	6·6
− 65+100	6·0	7·4
−100+150	12·9	10·6
−150+200	12·5	9·0
−200	67·9	62·8
Totals	100·0	100·0

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Trap and Blanket Concentration:				
Feed	100·00	0·31	100·0	....
Trap and blanket concentrates	2·89	7·37	68·7	34·6 : 1
Blanket tailing	97·11	0·10	31·3	....
Flotation of Reground Blanket Tailing:				
Feed	100·00	0·10	100·0	....
Flotation concentrate	5·59	1·62	90·6	17·9 : 1
Final tailing	94·41	0·01	9·4	....

Cyanidation of Combined Concentrates (Without Regrinding):

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton of solution		Reagents consumed, lb./ton concentrate	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
48	3·75	0·21	94·4	1·8	0·60	2·80	20·2

The final cyanide solution assayed 70 millilitres $\frac{N}{10}$ $KMnO_4$ per litre in reducing power and 0·07 gramme KCNS per litre.

Summary:

	Per cent
Gold recovered in trap and blanket concentrates......................	68·7
" " flotation concentrate..............................	28·4
Gold extracted by cyanidation from combined concentrates...........	91·7

CONCENTRATION AND CYANIDATION

Tests Nos. 6, 7, and 8

A gold jig replaced the trap and blanket of the previous tests. The jig tailings were reground with the same reagents, namely, 3 pounds of soda ash, 0·05 pound of pine oil, and 0·05 pound of amyl xanthate per ton, and were floated with 0·05 pound of pine oil, 0·05 pound of amyl xanthate, and 1·5 pounds of copper sulphate per ton. The combined jig and flotation concentrates were reground and cyanided.

Test No. 6:

The ore at −14 mesh was ground in a ball mill to pass 59·4 per cent −200 mesh. The pulp was passed through a Denver jig and a jig concentrate was taken. The jig tailing was reground to pass 87·5 per cent −200 mesh and was floated. The combined jig and flotation concentrates were reground in cyanide solution of 2 pounds per ton strength to pass 97·3 per cent −200 mesh and were agitated for a 48-hour period.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Jig Concentration:				
Feed..	100·0	0·31	100·0	....
Jig concentrate..............................	2·98	6·50	62·5	33·5 : 1
Tailing.....................................	97·02	0·12	37·5	....
Flotation of Reground Jig Tailing:				
Feed..	100·00	0·12	100·0	....
Flotation concentrate........................	7·42	1·55	96·2	13·5 : 1
Final tailing................................	92·58	0·005	3·8	....

The combined concentrates were reground as follows:

Mesh	Weight, per cent
− 65+100	0·2
−100+150	1·0
−150+200	1·5
−200	97·3
Total	100·0

Cyanidation of Combined Reground Concentrates:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
48	3·15	0·125	96·0	1·3	0·55	2·7	28·4

Summary:

	Per cent
Gold recovered in jig concentrate	62·5
" " flotation concentrate	36·1
Gold extracted by cyanidation from combined concentrates	94·6

Test No. 7:

The grinds in this test were similar to those of Test No. 6. The combined concentrates were reground in a lime pulp and were aerated for 16 hours prior to cyanidation.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Jig Concentration:				
Feed	100·00	0·31	100·0	
Jig concentrate	10·85	2·04	71·3	9·2 : 1
Jig tailing	89·15	0·10	28·7	
Flotation of Reground Jig Tailing:				
Feed	100·00	0·10	100·0	
Flotation concentrate	8·32	1·15	95·4	12·0 : 1
Final tailing	91·68	0·005	4·6	

Cyanidation of Reground and Aerated Combined Concentrates:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
48	1·70	0·055	96·8	1·9	0·55	1·16	8·65*

*Does not include 10 pounds of lime used in aeration.

In the final cyanide solution the reducing power was 75 millilitres $\frac{N}{10}$ $KMnO_4$ per litre and the KCNS was 0·06 gramme per litre.

Summary:

	Per cent
Gold recovered in jig concentrate	71·3
" " flotation concentrate	27·4
Gold extracted by cyanidation from combined concentrates	95·5

Test No. 8:

The grinds in this test were similar to those of Tests Nos. 6 and 7.

In order to determine the ratio of concentration in mill practice the flotation concentrate was cleaned in a smaller machine and the middling product was put back in the primary cell with a fresh batch of ore, the two lots of flotation tailing being sampled separately. The combined jig and flotation concentrates were reground in a lime pulp, aerated for 16 hours, and cyanided for 24- and 48-hour periods.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Jig Concentration:				
Feed	100·00	0·31	100·0	....
Jig concentrate	8·01	2·62	67·6	12·5 : 1
Jig tailing	91·99	0·11	32·4	....
Flotation of Reground Jig Tailing:				
Feed	100·00	0·10*	100·0	....
Flotation concentrate	5·77	1·54	88·9	17·3 : 1
Final middling	1·99	0·295	5·9	....
Tailing (1)	92·44	0·005	5·2	....
Tailing (2)	....	0·005		

*Calculated.

Cyanidation of Reground Aerated Concentrates:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
24	2·21	0·065	97·1	1·8	0·55	0·70	8·4*
48	2·21	0·065	97·1	1·8	0·55	0·85	8·8*

*Does not include lime used during aeration.

In the 24-hour agitation the reducing power was 56 millilitres $\frac{N}{10}$ $KMnO_4$ per litre and the KCNS 0·06 gramme per litre, whereas in the 48-hour agitation the reducing power was 120 millilitres $\frac{N}{10}$ $KMnO_4$ per litre and the KCNS 0·17 gramme per litre.

Summary:

	Per cent
Gold recovered in jig concentrate..................................	67·6
" " flotation concentrate..............................	30·7
Gold extracted from combined concentrates..........................	95·5

Summary of Tests Nos. 2 to 8, Part II:

Test No.	Flotation grind, per cent −200 mesh	Cyanide grind, per cent −200 mesh	Flotation tailing, Au, oz./ton	Overall extraction by cyanidation, per cent	Overall tailing loss, Au, oz./ton	Cyanide consumed, NaCN, lb./ton concentrate	Remarks
2	87·5	99·5*	0·005	96·2	0·012	4·3	....
3	87·5	94·0*	0·005	96·5	0·011	2·3	Aeration
4	73·2	93·5	0·005	95·2	0·015	3·1	....
5	67·9	62·8	0·01	91·7	0·026	2·8	....
6	87·5	97·3	0·005	94·6	0·017	2·7	....
7	87·5	97·3	0·005	95·5	0·014	1·16	Aeration
8	87·5	97·3	0·005	95·5	0·014	0·85	Aeration

*Per cent −325 mesh.

In all these tests the pulp was ground with 3·0 pounds of soda ash, 0·05 pound of amyl xanthate, and 0·05 pound of pine oil per ton, and floated with 0·05 pound of amyl xanthate, 0·05 pound of pine oil, and 1·5 pounds of copper sulphate per ton.

The pH of the pulp was kept between 9·0 and 9·4 in order to ensure the inclusion of the arsenopyrite in the flotation concentrate. The reducing power of the final cyanide solution was low and little fouling of the solution was observable.

The lime titration was kept high, 0·5 pound per ton of solution, as previous work on this ore dictated. The large amounts of carbonate in the ore also require a high lime titration in order to keep the alkalinity and preserve the cyanide content of the solution. Grinding of the concentrate in lime and aeration in a lime pulp gave lower cyanide consumption and improved the gold extraction.

In Test No. 8 a ratio of concentration of 17:1 was obtained in cleaning the primary flotation concentrate.

No increase in the value of the flotation tailing was discernable when the middling product was placed back in the circuit.

SUMMARY AND CONCLUSIONS

The test work shows that straight flotation of the ore is not feasible owing to the content of free gold. When the free gold is removed by jigs or traps and blankets, prior to flotation, a flotation tailing of 0·005 ounce of gold per ton is readily obtainable. In this connection the use of a unit cell in the flotation circuit might be considered.

The combined jig, trap and blanket, and flotation concentrates should be ground in a lime pulp and aerated prior to agitation in cyanide solution. By employing this method, in conjunction with fine grinding of the concentrates, an overall extraction of the gold of over 96 per cent and an overall tailing loss of 0·011 ounce of gold per ton were obtained.

The flotation tailing of 0·005 ounce of gold per ton was secured at a grind of 73·2 per cent −200 mesh and overall extractions of 96 per cent were obtained when the concentrates were reground to pass 94 per cent −325 mesh.

This metallurgical method of extraction of the gold seems to be preferable to the straight cyanidation now employed at the mill and described in previous reports of the Department.

An overall tailing loss of 0·012 ounce of gold per ton should be obtained, or an increase in extraction from the present 92 per cent to 96 per cent of the gold in the ore. By grinding the concentrate in a lime pulp and aerating prior to cyanidation, less fouling of solution should take place. There should be a substantial saving in the amount of floor space required. The amounts of flotation reagents used were small and the milling costs should show no appreciable gain. Whether the capital expenditure involved in a revision of metallurgical practice is economical, is a matter for the company to decide.

Ore Dressing and Metallurgical Investigation No. 776

GOLD ORE FROM THE NORTH ZONE OF THE MacLEOD-COCKSHUTT GOLD MINES, LIMITED, GERALDTON, ONTARIO

Shipment. A shipment consisting of 155 pounds of gold ore was received on May 23, 1939. The ore was said to have been taken from the North Ore Zone of the MacLeod-Cockshutt Gold Mines, Limited. The shipment was submitted by Abbott Rennick, Manager, Little Long Lac Gold Mines, Limited, on behalf of the MacLeod-Cockshutt Gold Mines, Limited.

Purpose of the Investigation. The investigation was made to determine the response of the ore to the following methods of treatment:

(1) Direct cyanidation of the reground bulk flotation concentrate.

(2) Roasting a bulk flotation concentrate and treating the calcine by cyanidation.

(3) Roasting the ore, followed by cyanidation.

Characteristics of the Ore. Six polished sections were prepared and examined under the reflecting microscope.

The *gangue* is an assemblage of fine-textured, dark greenish grey rock material with small patches of translucent grey quartz and white carbonate. In places the rock shows a schistose structure and in one section a distinct banding.

In their approximate order of decreasing abundance the *metallic minerals* are: pyrite, magnetite, arsenopyrite, pyrrhotite, and chalcopyrite.

Pyrite predominates as coarse to fine disseminated grains and small masses containing numerous small inclusions of gangue. Much of it is finely fractured and the fractures are filled with gangue.

Magnetite is very prevalent as tiny, irregular grains in the banded iron formation; it also occurs to a lesser extent in other parts of the ore. Arsenopyrite is present in considerable quantity, largely as medium to small crystals usually intimately associated with pyrite. Pyrrhotite is rather abundantly disseminated as coarse to fine, irregular grains and small masses in gangue and as tiny inclusions in pyrite and arsenopyrite. A very small quantity of chalcopyrite occurs as rare tiny, irregular grains in pyrite and in gangue.

No native gold or gold minerals were observed and nothing was learned about its mode of occurrence.

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

Gold	0·275 oz./ton
Silver	0·07 "
Copper	Trace
Arsenic	0·68 per cent
Iron	24·50 "
Sulphur	11·18 "
Lime	2·36 "
Magnesia	2·56 "
Carbon dioxide	4·16 "
Insoluble	54·10 "

Experimental Tests

The scope of the investigative test work was divided into three sections, namely:

Section I.—Cyanidation of raw ore and reground concentrate,

Section II.—Cyanidation of roasted flotation concentrate, and

Section III.—Roasting and cyanidation of raw ore.

Section I

CYANIDATION OF RAW ORE AND REGROUND CONCENTRATE

CYANIDATION OF RAW ORE

Test No. 1

Samples of the ore were ground 93 per cent and 97 per cent −200 mesh, respectively, in cyanide solution of a strength of 1·0 pound of sodium cyanide per ton, at a dilution of 4 parts solids to 3 parts solution, using lime for protective alkalinity.

The pulp was diluted to 1 part of solids to $1\frac{1}{2}$ parts of solution and was agitated for 24 hours in a solution of 1·0 pound of sodium cyanide per ton.

Results:

Test No.	Grind, per cent −200 mesh	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton ore		Final titration, lb./ton solution	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
1A	93·4	0·275	0·055	80·0	1·13	5·90	1·10	0·06
1B	97·4	0·275	0·050	81·8	1·31	5·95	1·00	0·04

CYANIDATION OF FLOTATION CONCENTRATE

Test No. 2

Flotation. A sample of ore was ground to 88·5 per cent −200 mesh with 4·0 pounds of soda ash per ton at a dilution of 4:3. The pulp was conditioned in a flotation machine with 0·2 pound of potassium amyl xanthate per ton for 7 minutes, after which 0·05 pound of pine oil per ton was added and a concentrate was removed.

Then, 2·0 pounds of soda ash and 1·0 pound of copper sulphate per ton were added and conditioned for 5 minutes, followed by 0·1 pound of amyl xanthate per ton, which was agitated for 5 minutes.

A further concentrate was removed.

The first concentrate appeared to consist largely of pyrite, and the second of arsenopyrite. Both concentrates were cleaned in separate cells. The tailing from each was designated as middling.

Results:

Product	Weight, per cent	Assay		Units		Distribution, per cent		Ratio of concentration
		Au, oz./ton	As, per cent	Au	As	Au	As	
Feed	100·00	0·273	0·74	27·279	74·18	100·00	100·00	....
Concentrate No. 1	17·88	1·24	0·75	22·171	13·41	81·27	18·08	5·6 : 1
Concentrate No. 2	3·95	0·62	13·90	2·449	54·90	8·98	74·02	25·3 : 1
Combined concentrates	21·83	1·13	....	24·620	....	90·25	....	4·6 : 1
Middling No. 1	5·31	0·35	0·98	1·859	5·20	6·81	7·01	18·8 : 1
Middling No. 2	2·86	0·035	0·23	0·100	0·66	0·37	0·89	35 : 1
Tailing	70·00	0·01	Nil	0·700	....	2·57	....	....

Cyanidation. The combined concentrate was reground to 99·7 per cent −325 mesh in water at a dilution of 4:3. Lime was added to the charge at the rate of 10 pounds per ton of dry solids.

The ground concentrate was filtered and repulped in a solution of 3·0 pounds of sodium cyanide per ton at a dilution of 1 part solids to 3 parts of solution and was agitated for 22 hours.

Results:

Agitation, hours	Grind, per cent −325 mesh	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Final titration, lb./ton solution	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
22	99·7	1·13	0·17	84·96	3·00	18·6	2·8	0·45

Overall extraction, in terms of feed, 84·96 x 90·25..........	76·68 per cent
Consumption of cyanide in terms of feed, $\frac{3 \cdot 0}{4 \cdot 6}$	0·65 lb./ton ore
" lime " " $\frac{18 \cdot 6}{4 \cdot 6}$	4·0 "
Total lime (grind + solution)....... $\frac{28 \cdot 6}{4 \cdot 6}$	6·2 "
Reducing power of cyanide solution........................	275 ml. $\frac{N}{10}$ $KMnO_4$/litre
NaCNS..	0·24 grm./litre
Total alkalinity..............................	0·24 grm. of lime/litre

CYANIDATION OF FLOTATION CONCENTRATE

Test No. 3

Flotation. In order to accumulate flotation concentrate for roasting and cyanidation tests, eighteen charges of 2,000 grammes each were floated.

The ore was ground in ball mills, dilution of 4:3 in water, with 4 pounds of soda ash per ton to 80 per cent −200 mesh. The ground pulp was conditioned in a flotation cell and was floated with the same reagents as were used in the preceding test. The two concentrates were not separated during flotation. The whole concentrate was cleaned in a separate cell. The middling product was added to the succeeding charge to the rougher cell.

The feed for each charge was freshly ground. Charges ground 90 per cent −200 mesh gave the same flotation tailing as the 80 per cent −200-mesh grind.

After floating six or seven charges flotation was discontinued and the cleaner concentrate was filtered and a portion of the fresh concentrate was reground and cyanided.

The remaining concentrate, about 7,500 grammes, was dried, mixed, and sampled, and was used for roasting and cyanidation tests. The results of this part of the investigation are shown under Section II, page 25.

Cyanidation of the Raw Concentrate. Portions of fresh concentrate were reground in water with 10 pounds of lime per ton to 99 per cent −325 mesh, filtered, repulped in a solution of 3·0 pounds of sodium cyanide per ton, and agitated for various periods.

Results of Flotation:

Product	Weight, per cent		Assay, Au, oz./ton	Units, weight per cent × assay, Au	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	100·00	0·2795*	27·945	100·00	....
Concentrate (5 charges)	6·68	....	1·12	7·482	....	....
Concentrate (6 charges)	8·00	....	1·14	9·120	....	....
Concentrate (7 charges)	9·65	....	1·08	10·422	....	....
Combined concentrate	....	24·33	1·11*	27·024	96·70	4·11 : 1
Middling (5)	0·67		0·175	0·117	....	....
Middling (6)	0·28	....	0·13	0·036	....	....
Middling (7)	0·41	....	0·06	0·025	....	....
Combined middling	....	1·36	0·131*	0·178	0·64	73·5 : 1
Tailing (5)	20·24	....	0·01	....	....	....
Tailing (6)	25·13	....	0·01	....	....	....
Tailing (7)	28·94	....	0·01	....	....	....
Combined tailing	....	74·31	0·01	0·743	2·66	....

*Calculated values.

It was noted that seventeen flotation tailings assayed 0·01 ounce of gold per ton. One charge, which was not floated for several hours after grinding, had a tailing of 0·035 ounce of gold per ton. Grinding to 90 per cent −200 mesh did not lower the tailing. Increased amounts of xanthate added after skimming the bulk concentrate in several tests gave the same tailing.

The results of flotation indicate that 0·01 ounce of gold per ton is the minimum tailing.

Results of Cyanidation:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent		Reagents consumed				Final titration, lb./ton solution		Reducing power, ml. $\frac{N}{10}$ $KMnO_4$/litre	KCNS, grm./litre	Total alkalinity, grm./litre
						Lb./ton concentrate		Lb./ton ore						
		Feed	Tailing	Concentrate	Ore	NaCN	CaO	NaCN	CaO	NaCN	CaO			
3A	22	1·13	0·17	85·0	76·7	3·0	18·6	0·65	4·1	2·8	0·45	275	0·29	1·08
3B	24	1·14	0·20	82·5	80·0	5·7	22·4	1·37	6·9	2·3	0·25	230	0·15	1·15
3C	48	1·14	0·19	83·3	80·8	6·3	22·8	1·49	7·2	3·0	0·20	280	0·29	1·04
3D	67	1·12	0·195	82·6	70·0	6·3	23·9	1·53	8·2	2·3	0·10	500	0·49	0·80

Tests Nos. 3 B and 3 C were aerated for 4 hours with lime, filtered, and repulped in cyanide solution. An infrasizer test was made to determine the grind of the concentrate. The 24- and 48-hour tailings were combined.

A period of 67 hours of agitation did not reduce the tailing.

It is assumed that the liberated gold goes rapidly into solution and that the minimum tailing can be obtained within 24 hours.

Results of Infrasizer Test on Cyanide Tailings 3 B and 3 C:

Product, microns	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
−10	39·21	0·115	25·79
+10−14	12·00	0·195	13·38
+14−20	13·93	0·205	16·32
+20−28	14·53	0·22	18·27
+28−40	12·13	0·22	15·25
+40−56	7·13	0·23	9·37
+56	1·07	0·265	1·62
Feed	100·00	0·175	100·00

Infrasizer Test on the Combined Flotation Tailing. This was made to discover the distribution of gold and sulphides in the flotation tailing from Test No. 3.

Results:

Product, microns	Weight, per cent	Assay		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S
−10	25·32	0·01	0·81	24·07	49·7
+10−14	8·97	0·005	0·24	4·28	5·2
+14−20	11·47	0·01	0·20	10·94	5·6
+20−28	13·20	0·01	0·29	12·56	9·3
+28−40	14·13	0·01	0·26	13·42	8·9
+40−56	17·38	0·01	0·26	16·56	11·0
+56	9·53	0·02	0·45	18·17	10·3
Feed	100·00	0·011	0·41	100·00	100·0

FLOTATION AND CYANIDATION OF A PYRITE CONCENTRATE

Test No. 4

A flotation test was made as in Test No. 2, using a grind of 66 per cent −200 mesh instead of 88 per cent.

The ground ore was floated, making a pyrite concentrate and an arsenopyrite concentrate. The same reagents were used as in Test No. 2. The tailing was 0·01 ounce of gold per ton.

The pyrite concentrate was reground in water to 99 per cent −325 mesh and cyanided at a dilution of 1:3 in a solution of 3·0 pounds of sodium cyanide per ton.

Owing to there being insufficient ore there was not enough clean arsenopyrite concentrate to roast and cyanide separately.

Results:

Cyanidation of a Pyrite Concentrate:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Final titration, lb./ton solution	
Feed	Tailing		NaCN	CaO	NaCN	CaO
1·02	0·17	83·33	1·78	15·4	3·0	0·35

Pyrite concentrate:

Arsenic	2·29 per cent
Ratio of concentration	4·4 : 1

Reagents consumed, lb./ton ore:

NaCN	0·41
CaO	3·50
Overall recovery—83·33 x 0·79	65·8 per cent of feed

CYANIDATION OF REGROUND AND PRE-AERATED CONCENTRATE

Test No. 5

A sample of the bulk concentrate was ground 60 per cent −10 microns, and was aerated for 12 hours in water with lime from 0·06 to 0·15 pound per ton at the end of aeration. The pulp was filtered and repulped in cyanide solution, 3·0 pounds of sodium cyanide per ton, at a dilution of 1 part solids to 3 parts of solution.

After cyanidation a portion of the cyanide tailing was infrasized and the fractions were assayed.

Aeration and Cyanidation:

Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate		Final titration, lb./ton solution	
Feed	Tailing		NaCN	CaO	NaCN	CaO
Aeration	....	....	....	69·2	....	0·15
1·04	0·17	83·7	3·54	10·3	2·80	0·30

Lime used in grind..................................	9·3 lb./ton concentrate		
" " aeration................................	69·2	"	"
" " cyanidation...........................	10·3	"	"
Total..................................	88·8	"	"

Infrasizer Test on Cyanide Tailing:

Product, microns	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
−10..	60·14	0·14	49·7
+10−14..	14·07	0·19	15·8
+14−20..	13·23	0·21	16·4
+20−28..	8·30	0·24	11·8
+28−40..	3·30	0·255	4·9
+40−56..	0·60	0·24	1·4
+56..	0·36		
Feed..................................	100·00	0·17	100·0

Section II

CYANIDATION OF CALCINED FLOTATION CONCENTRATE

Roasting Practice. The roasting of the ore or concentrate was carried out in a globar electric muffle type furnace provided with an exhaust pipe and control. The temperature is automatically controlled.

The same general scheme found satisfactory in test work on similar ores and products was used, i.e. roasting for 2 hours or more at 300 to 400°c. until evolution of arsenic and/or sulphur fumes had become very faint, then gradually increasing the temperature in 50 or 100-degree steps and finishing at 600 to 800°c. Certain variations were made to determine if improved conditions resulted.

The charges were roasted in fireclay trays $11\frac{3}{4}\times6\frac{5}{8}\times1\frac{5}{8}$ inches inside dimensions. The ore or concentrate was spread out in a layer from $\frac{1}{4}$ to $\frac{1}{3}$ inch deep. During roasting the charge was rabbled every 15 or 20 minutes. The exhaust was operated in such a manner that sufficient air was drawn over the roast that only a minimum of fumes escaped from the front of the furnace.

The temperatures given are those indicated by a thermocouple placed $5\frac{1}{4}$ inches above the floor of the furnace. These temperatures are about 60 degrees less than the temperatures found by cones placed on the floor of the furnace.

Procedure. Seven roasting tests were conducted on the bulk concentrate and twenty-two cyanide tests made on the calcines.

A summary of results obtained is given in Table I.

Analysis of Raw Flotation Concentrate:

Gold	1·04 oz./ton
Silver	0·20 "
Arsenic	3·00 per cent
Iron	42·60 "
Sulphur	44·00 "
Insoluble	5·45 "

TABLE I

Test No.	Roasting hours		Assay, Au, oz./ton		Extraction, per cent	Agitation, hours	Grind, per cent −325	Reagents consumed, lb./ton of calcine		Cyanide Test No.
	Total time	Final temp., °C	Calcine	Tailing				NaCN	CaO	
7	7¼	2 hr. 650	1·475	0·115	92·2	24	93	1·28	9·8	7–1
	....	....	....	0·12	91·9	48	93	1·98	11·6	7–2
	....	....	....	0·12	91·9	48	99·6	2·54	11·6	7–3
	....	....	....	0·12	91·9	70	99·6	2·62	12·6	7–4
8	8	2 hr. 650	1·47	0·125	91·5	45	98·9	1·0	9·8	8–1
	....	....	....	0·125	91·5	45	98·9	8·7	9·5	8–2
	....	....	....	0·13	91·2	45	98·9	1·28	9·8	8–3
	....	....	....	0·135	90·8	45	98·9	1·84	27·3	8–4
9	8¼	2 hr. 650	1·475	0·105	92·9	25	98·9	1·84	9·8	9–1
	....	....	....	0·09	93·9	45	98·9	2·24	9·8	9–4
13	8	1 hr. 750	1·48	0·07	95·3	48	98·9	1·80	12·2	13–1
15	8	2 hr. 650	1·46	0·095	93·6	48	98	2·00	10·5	15–2
	....	....	....	0·095	93·6	48	75	2·18	10·5	15–3
16	12	4½ hr. 500	1·44	0·13	91·0	48	98	4·88	10·5	16–1
17	8½	1 hr. 750	1·48	0·07	95·3	48	98	1·94	8·7	17–1
	....	....	....	0·07	95·3	48	98	2·26	8·7	17–2

Roasting Conditions

Test No. 7. Two charges of 750 grammes each. Temperature held 1 hour at 300°c, 1 hour at 350°c, 1 hour at 400°c, ½ hour at 450°c, and 2 hours at 650°c. The calcines were ground to different degrees of fineness, pretreated, and cyanided by standard method.

Test No. 8. Two charges of 750 grammes each. Temperature conditions same as Test No. 7, except that the temperature was held at 250°c for ½ hour to find if ignition would start at this range. No action was observed. The calcine was ground and pretreated as usual. Portions were then agitated in lime and sodium cyanide of varying strength.

No. 8-1:	1 pound of sodium cyanide per ton of solution.....	Low lime
No. 8-2:	5 " " "	Low lime
No. 8-3:	2 " " "	Sufficient lime to maintain alkalinity
No. 8-4:	2 " " "	Saturated with lime

Test No. 9. Two charges of 750 grammes each. Temperature held 2½ hours at 300°c, ½ hour at 350°c, ½ hour at 400°c, ½ hour at 450°c, and 2 hours at 650°c. The temperature was held at 300°c without the application of external heat until all indication of fuming and burning of sulphides was over. The temperature rose to 360°c. Sulphur dioxide fumes came off freely and flaming was quite noticeable. The charge then grew duller in appearance and no fuming was evident again until the temperature reached 600°c. The calcines were ground, pretreated, and cyanided as usual.

Test No. 13. One charge of 350 grammes. Temperature held 2½ hours at 300°c, ½ hour at 350°c, ½ hour at 400°c, ½ hour at 450°c, and 1 hour at 750°c. Calcines were ground, pretreated, and cyanided by standard method.

Test No. 15. Two charges of 500 grammes each. Temperature conditions similar to Test No. 9.

No. 15-1..........................	Standard treatment
No. 15-3..........................	Bottle-agitated, filtered, cyanided without regrinding

Test No. 16. One charge of 400 grammes. Temperature held 2½ hours at 300°c, 2 hours at 350°c, 2 hours at 400°c, and 4½ hours at 500°c. Calcine treated by standard method.

Test No. 17. Two charges of 400 grammes each. Temperature conditions similar to Test No. 13. Calcine ground, pretreated, and cyanided by standard method.

Average ignition loss, roasts Nos. 7 and 8.......................	29·5 per cent
" " " 9 " 15.......................	29·7 "
" " " 13 " 17.......................	30·4 "
Ignition loss, roast No. 16..................................	28·5 "

Treatment of Calcines. The calcines were ground, made up 1:6, bottle-agitated 20 hours, filtered, washed, the cake divided into portions, repulped 1:2, and agitated; the stated time in solution containing 2 pounds of sodium cyanide and from 0·1 to 0·25 pound of lime per ton. Any variation from this practice is noted under the descriptions of the tests.

Analysis of Calcines:

	Test No. 7	Test No. 9
Total iron	59·56 per cent	60·00 per cent
Ferrous iron	0·20 "	0·20 "
Arsenic	1·43 "	1·53 "
Sulphur	0·44 "	0·35 "
Gold	1·475 oz.	1·475 oz.

Superpanning Cyanide Tailing. The cyanide tailings from Tests Nos. 7, 8, and 9 were combined and superpanned. The superpanner concentrate resulting from a 10:1 concentration showed almost no increase in gold—composite tailing assayed 0·115 ounce of gold, superpanner concentrate 0·13 ounce. After more intensive superpanning a microscopic examination failed to show the presence of free gold.

SUMMARY OF INVESTIGATIVE WORK ON ROASTING AND CYANIDING CALCINES OF THE BULK CONCENTRATES

The maximum extraction of gold obtainable by roasting and cyaniding the calcine of the bulk concentrate is 93 to 95 per cent.

The most favourable roasting conditions found are: Roasting for a period of at least 2½ hours between 300° and 360°c while active evolution of arsenic and sulphur is taking place, then gradually increasing temperature to 750°c and holding that temperature for at least 1 hour; and free access of air during the period of active ignition.

After suitable calcination, grinding 93 to 98 per cent −325 and agitating in solution of lime and sodium cyanide for at least 45 hours, about 4·7 per cent of the total gold in the calcine remains undissolved. No variation in the pretreatment or cyanidation was found to improve on this. Water-washing and cyaniding without regrinding (75 per cent −325) gave the same extraction as grinding the calcine 98 per cent −325 (Test No. 15).

The method of roasting appears to be the most important factor as variations in grinding, reagent strength, or time (above 24 hours), make little or no difference in the extraction.

Section III

RAW ORE, ROASTING AND CYANIDATION

A sample of the ore was ground to −10 mesh previous to roasting and consisted of the following fractions:

Mesh	Per cent	Assay, Au, oz./ton	Units	Distribution, per cent
−10+ 35	49·5	0·235	11·63	42·51
−35+200	30·5	0·355	10·83	39·58
−200	20·0	0·245	4·90	17·91
	100·0		27·36	100·00

Nine roasting tests were conducted and thirty-two cyanide tests made on the calcines.

A summary of results obtained is given in Table II.

TABLE II

Test No.	Roasting hours		Assay, Au, oz./ton		Extraction, per cent	Agitation, hours	Grind, per cent −325	Reagents consumed, lb./ton calcine		Cyanide Test No.
	Total time	Final temp.,°c	Calcine	Tailing				NaCN	CaO	
1	4¾	1hr. 750	0·305	0·03	90·2	24	68	0·33	4·1	1–2
	....	....	....	0·02	93·4	48	68	0·54	5·0	1–3
	....	....	....	0·02	93·4	96	68	0·78	3·3	1–5
2	7¼	1hr. 750	0·30	0·03	90·0	24	70	0·33	3·0	2–1
	....	....	....	0·02	93·3	48	70	0·33	3·3	2–2
3	2½	1hr. 750	0·29	0·03	89·7	48	62	1·13	4·3	3–3
4	6	1hr. 750	0·29	0·025	91·4	48	20	0·84	1·2	4–1
	....	....	....	0·02	93·1	48	52	0·51	1·8	4–2
	....	....	....	0·02	93·1	48	75	0·51	1·8	4–3
	....	....	....	0·02	93·1	48	98	0·51	4·1	4–4
5	5	1hr. 750	0·29	0·02	93·1	48	92	1·0	3·9	5–1
	....	....	....	0·02	93·1	48	94	0·66	3·3	5–2
	....	....	....	0·02	93·1	48	94	0·45	3·3	5–3
	....	....	....	0·02	93·1	48	94	0·66	3·3	5–4
	....	....	....	0·02	93·1	48	94	0·59	3·6	5–5
6	6¼	1hr. 750	0·29	0·02	93·1	48	82	0·20	1·2	6–1
	....	....	....	0·02	93·1	48	82	1·83	1·2	6–2
	....	....	....	0·02	93·1	48	82	0·36	0·95	6–3
	....	....	....	0·02	93·1	48	82	0·30	10·1	6–4
10	6¼	1hr. 850	0·30	0·020	93·3	48	70	0·65	1·8	10–2
11	13	1hr. 850	0·29	0·020	93·1	48	70	0·76	1·8	11–2
14	12	6hr. 500	0·285	0·02	93·0	48	70	0·85	13·0	14–2

Roasting Conditions

Test No. 1. Two charges of 1,000 grammes each. Temperature held 1 hour at 350°c, 1 hour at 450°c, and 1 hour at 750°c. Calcine was ground, pretreated, and cyanided by standard method for different periods of time.

Test No. 2. Two charges of 1,000 grammes each. Temperature held 1½ hours at 350°c, 1½ hours at 450°c, 1½ hours at 550°c, and 1 hour at 750°c. Calcine was ground, pretreated, and cyanided by standard method.

Test No. 3. Two charges of 1,000 grammes each. Temperature brought up to 750°c in 1½ hours and held 1 hour at 750°c. Calcine was ground, pretreated, and cyanided by standard method.

Test No. 4. Two charges of 1,000 grammes each. Temperature held 1½ hours at 350°c, 1½ hours at 450°c, and 1 hour at 750°c. One portion was bottle-agitated without grinding, filtered, and cyanided. Other portions were ground to different degrees of fineness, then pretreated and cyanided by the standard method.

No. 4–1: −10+35..50·0 per cent, −35+200..26·8 per cent, −200..23·2 per cent
No. 4–2: −200 mesh.. 59 per cent
No. 4–3: −200 mesh.. 85 "
No. 4–4: −200 mesh.. 99 "

Test No. 5. Two charges of 1,000 grammes each. Temperature conditions same as in Test No. 1. One portion was ground in solution of lime and sodium cyanide without any pretreatment. Another portion was ground and cyanided without dewatering; other portions were ground and pretreated for different lengths of time before cyaniding.

No. 5–1....................Ground in solution of lime and sodium cyanide.
No. 5–2....................Ground and cyanided without dewatering.
No. 5–3....................Ground, filtered, repulped, and cyanided.
No. 5–4....................Ground, bottle-agitated 6 hours, filtered, repulped, and cyanided.
No. 5-5....................Ground, bottle-agitated 12 hours, filtered, repulped, and cyanided.

Test No. 6. Two charges of 1,000 grammes each. Temperature conditions same as in Test No. 1. Calcine ground and pretreated as usual, then agitated in solutions of lime and sodium cyanide of different strengths.

No. 6–1:	½ pound sodium cyanide/ton solution		........Low lime.
No. 6–2:	4	" "	....Low lime.
No. 6–3:	1	" "	....Sufficient lime to maintain alkalinity.
No. 6–4:	1	" "	....Saturated with lime.

Test No. 10. Two charges of 750 grammes each. Temperature held 1 hour at 350°c, 1 hour at 450°c, and 1 hour at 850°c. Calcine ground, pretreated, and cyanided as usual.

Test No. 11. One charge of 750 grammes. Temperature held 2 hours at 350°c, 2 hours at 400°c, 1 hour at 450°c, 6 hours at 500°c, and 1 hour at 850°c. Calcine treated by standard procedure.

Test No. 14. One charge of 750 grammes. Temperature held at 350°c for 2 hours, 2 hours at 400°c, 1 hour at 450°c, and 6 hours at 500°c. Calcine ground, pretreated, and cyanided by standard method.

Ignition loss of calcines roasted by the method of No. 1 or No. 2—9·5 per cent. In No. 3 the loss was 8·7 per cent, in No. 10, 10·7 per cent, and in No. 14, 6·7 per cent.

Treatment of Calcine. The calcine was ground, made up 1:6 with water, bottle-agitated 20 hours, filtered, and washed. The filter cake was divided into portions, repulped 1:1·5, and agitated the stated time in solution containing 1 pound of sodium cyanide and 0·1 to 0·25 pound of lime per ton. Any variation from this practice is noted under the descriptions of the tests. During the first part of the agitation, lime is consumed rather quickly. A low alkalinity is then reached and is maintained. The pulp settled well.

Analysis of Calcines:

	Test No. 1	Test No. 2
Ferrous iron	2·06 per cent	3·02 per cent
Arsenic	0·43 "	0·48 "
Sulphur	0·93 "	0·80 "
Gold	0·305 oz.	0·30 oz.

An extraction of 91·4 per cent was obtained by cyaniding the calcine without regrinding after roasting. The calcine consisted of:

Mesh	Per cent	Assay, Au, oz./ton	Units	Distribution, per cent
− 10+ 35	50·0	0·21	10·50	37·15
− 35+200	26·8	0·386	10·34	36·59
−200+325	3·2 }	0·32	7·42	26·26
−325	20·0 }			
	100·0		28·36	100·00

The combined cyanide tailings from Tests Nos. 1, 2, and 6 (Au, 0·02) were infrasized, giving the following results:

Mesh	Weight, per cent	Assay, Au, oz./ton	Units	Distribution, per cent
+56	5·48	0·02	1·096	5·48
−56+40	10·83	0·02	2·166	10·83
−40+28	13·48	0·02	2·696	13·48
−28+20	13·82	0·02	2·764	13·82
−20+14	12·59	0·02	2·518	12·59
−14+10	11·20	0·02	2·240	11·20
−10	32·60	0·02	6·520	32·60
	100·00	0·02	20·000	100·00

It appears that the refractory gold is uniformly distributed in the −200-mesh material.

Summary of Investigative Work on Roasting and Cyaniding Calcine of Raw Ore. The maximum extraction of gold obtainable by roasting the ore and cyaniding the calcine is slightly over 93 per cent.

Satisfactory roasting conditions are: a low initial temperature for 2 hours or more to remove the arsenic and sulphur, then gradual increase to 750°c. It was found that a total roasting time of about 5 hours gave a satisfactory calcine. A short, rapid roast was not satisfactory (Test No. 3).

After suitable calcination, the pretreatment and cyanidation present no special difficulties. About 6·7 per cent of the total gold in the calcine remains refractory and no variation in the grinding, pretreatment, or cyanidation was found to give better extraction (Tests Nos. 4, 5, and 6).

SUMMARY AND CONCLUSIONS

A flotation tailing of 0·01 ounce of gold per ton may be expected, with a ratio of concentration of about 4·1:1.

The best results obtained by cyanidation of the calcined flotation concentrate were in those tests in which the roasting temperature was carried to 750°c.

Comparison of Results:

Straight Cyanidation:

—	Recovery, per cent of original feed	Tailing, oz./ton of original feed
1. Straight cyanidation	81·9	0·05
2. Roasting raw ore and cyaniding calcine	93·3	0·018
3. Cyanidation of flotation concentrate	81·9	0·05
4. Roasting flotation concentrate with cyanidation of calcine	92·7	0·022

It is apparent that the choice between the various metallurgical treatments will depend largely on the economics of each.

Ore Dressing and Metallurgical Investigation No. 777

HIGH-GRADE GOLD ORE FROM THE CHAN YELLOWKNIFE GOLD, LIMITED, YELLOWKNIFE AREA, NORTHWEST TERRITORIES

Shipment. A 50-pound sample of gold ore from the Kilpatrick vein, No. 4 Claim, Chan Yellowknife Gold, Limited, was received on June 6, 1939. The shipment was submitted by E. Miles Flynn, President, Chan Yellowknife Gold, Limited, Suite 503, 67 Yonge Street, Toronto, Ontario.

Characteristics of the Ore. Six polished sections were prepared and examined under the reflecting microscope.

The *gangue* is an assemblage of fine-grained, smoky-grey quartz and dark greenish grey to black rock-material. The quartz exhibits narrow local fractures and stains of iron oxides; the latter disclose the superficial character of the sample.

The polished sections are not heavily mineralized with *metallic minerals.* Arsenopyrite preponderates as coarse to fine disseminated grains and small granular masses. Pyrrhotite is relatively abundant as coarse to fine, irregular grains and narrow, discontinuous stringers cutting quartz. A minor quantity of pyrite, usually admixed with arsenopyrite, is present as medium to small, irregular grains. A small amount of chalcopyrite occurs as small, irregular grains associated with the other sulphides. Magnetite and galena are present in very small quantities as tiny, irregular grains disseminated through gangue; the galena is associated with pyrrhotite in many places. "Limonite" is to be seen as rusty brown stains in quartz and more rarely as small, irregular grains in gangue and along cracks in pyrite and arsenopyrite.

Irregular grains of native gold are very prevalent in the polished sections. Most of them occur in gangue, alone, and in contact with arsenopyrite and pyrrhotite; these two sulphides might have exerted a precipitating effect on the gold. Many of those occurring alone appear interstitial between quartz grains, and one grain is along a fracture in quartz. A small percentage occurs in arsenopyrite both along fractures and apparently entirely enclosed by the sulphide.

Grain sizes and modes of occurrence of the native gold are shown in the following table:

Mesh	Gold in gangue, per cent			Gold in arsenopyrite, per cent		Totals, per cent
	Alone	Against arseno-pyrite	Against pyrrho-tite	Entirely enclosed	Along fractures	
+ 150	....	....	4·2	....	....	4·2
− 150+ 200	5·0	5·3	....	....	....	10·3
− 200+ 280	16·6	....	....	....	....	16·6
− 280+ 400	10·4	1·4	2·6	....	....	14·4
− 400+ 560	9·0	3·2	....	....	....	12·2
− 560+ 800	12·9	6·6	....	0·6	....	20·1
− 800+1100	7·5	2·5	....	0·4	0·4	10·8
−1100+1600	6·3	0·7	0·4	....	....	7·4
−1600+2300	2·8	0·5	....	....	....	3·3
−2300	0·7	....	....	....	....	0·7
	71·2	20·2	7·2	1·0	0·4	100·0
	98·6			1·4		

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

Gold	16·38 oz./ton (Average of 5 assays)
Silver	1·03 oz./ton
Copper	0·05 per cent
Lead	0·09 "
Zinc	0·09 "
Iron	2·99 "
Tellurium	0·007 "
Arsenic	1·01 "
Sulphur	1·21 "
Pyrrhotite	0·31 "
Graphite	0·04 "

Experimental Tests

The investigation consisted of recovery of the free gold by jigs and blankets followed by amalgamation of concentrates; flotation of blanket tailing and cyanidation of flotation concentrate; and amalgamation and cyanidation of the ore.

Cyanidation of flotation concentrate gave a recovery of over 98 per cent, but the combined tailing was high.

Cyanidation of the ore after recovery of the free gold by amalgamation gave a much lower tailing loss.

Details of the tests follow:

Test No. 1

A sample of ore, 5,000 grammes in weight, crushed to 14 mesh, was fed to a Denver mineral jig. The overflow was deslimed by decantation and the sand was ground to have 43 per cent −200 mesh.

The sand product was again fed to the jig, the overflow passing over a blanket strake. The fine was also passed over the blankets.

About 33 per cent of the contained gold was removed in the primary jig, and 57 per cent in the secondary jig and blankets.

Results:

Jig and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·0	16·38	100·00	....
Combined concentrate	5·9	251·79	90·69	16·9 : 1
Blanket tailing	94·1	1·62	9·31	....

Amalgamation of Concentrate:

Feed, Au, oz./ton	Tailing, Au, oz./ton	Recovery, per cent
251·79	2·55	98·9

No indication of fouling of mercury was observed.
Overall recovery of gold by amalgamation.................... 89·7 per cent

Flotation of Blanket Tailing. A portion of the blanket tailing was conditioned with soda ash, 2 pounds per ton, Aerofloat No. 25, 0·035 pound per ton, and 0·1 pound of butyl xanthate per ton for 5 minutes, and a flotation concentrate was removed; 0·5 pound of copper sulphate per ton was used to activate the sphalerite. Pine oil was used as frother.

Results:

Product	Weight, per cent	Assay					Distribution of gold, per cent	Ratio of concentration
		Au, oz./ton	Per cent					
			Cu	Pb	Zn	S		
Feed	100·00	1·66	....	....	....	....	100·00	....
Flotation concentrate	3·77	30·90	0·96	2·60	0·61	....	69·95	26·5 : 1
Flotation tailing	96·23	0·52	....	....	....	0·29	30·05	....

A sample of flotation tailing was concentrated on a Haultain superpanner. The amount of sulphide recovered was extremely small. It was principally arsenopyrite. One small grain of native gold was noted.

Another sample of flotation tailing was given an infrasizing test. The sample was first screened into two products, + 200 and − 200 mesh. The 200-mesh product was infrasized.

The results are tabulated as follows and show the distribution of gold in the different grain size groupings:

Mesh	Microns	Weight, per cent	Assay, Au, oz./ton	Units	Distribution of gold, per cent
+200		72·24	0·65	49·9560	87·57
−200		27·76	0·24	6·6624	12·43
Total		100·00	0·53	53·6184	100·00
	Over 56	22·94	0·45	10·32300	42·7
	−56+40	33·52	0·205	6·87160	28·4
	−40+28	15·44	0·17	2·62480	10·9
	−28+20	10·13	0·135	1·36755	5·7
	−20+14	6·58	0·115	0·75670	3·1
	−14+10	4·55	0·15	0·68250	2·8
	−10 (under)	6·84	0·225	1·53900	6·4
	Total	100·00	0·24	24·16515	100·0

Cyanidation of Blanket Tailing. One sample of tailing was repulped in cyanide solution, 1 pound of sodium cyanide per ton, at a dilution of 2 : 1 and was agitated for 24 hours. Another sample was reground in cyanide to have a fineness of 79·6 per cent −200 mesh and two portions were agitated in cyanide for 24 and 48 hours respectively.

Results:

Feed: gold, 1·62 oz./ton

Grind, per cent −200 mesh	Agitation, hours	Tailing, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton tailing	
				NaCN	CaO	NaCN	CaO
43·0	24	0·16	90·1	0·90	0·25	1·80	4·70
79·6	24	0·065	96·0	1·00	0·30	1·30	8·60
79·6	48	0·045	97·2	1·00	0·25	1·30	8·70

Test No. 2

A sample of −14-mesh ore was treated similarly to that in Test No. 1 by passing over a mineral jig, desliming the overflow, regrinding the sand, and repassing the pulp over the jig and a blanket strake.

The blanket tailing was treated by flotation and a sulphide concentrate was recovered.

The jig and blanket concentrates were amalgamated and the residue was added to the flotation concentrate, which was reground in water to have 99·3 per cent −200 mesh. The pulp was washed to remove flotation reagents and was agitated in cyanide solution, 3 pounds of sodium cyanide per ton, at a dilution of 3 : 1 for 24 hours.

Summary of Results:

Concentration (Jig, Blanket, and Flotation):

Product	Weight, per cent	Au, oz./ton	S, per cent	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	16·38	....	100·00	....
Combined concentrates	10·87	148·39	....	98·48	9·2 : 1
Flotation tailing	89·13	0·28	0·29	1·52	....

Amalgamation and Cyanidation:

Gold in combined concentrates	148·39 oz. ton
Final tailing (amalgamation and cyanidation)	0·40 "
Recovery of gold	99·73 per cent
Cyanide consumption	4·10 lb. NaCN/ton of concentrate
Lime consumption	11·75 " "

Screen Test of Flotation Tailing:

Mesh	Weight, per cent
+ 48	0·1
− 48+ 65	0·3
− 65+100	1·2
−100+150	9·3
−150+200	16·0
−200	73·1
Total	100·0

The overall gold recovery was 98·21 per cent and the combined tailing loss, gold, 0·293 ounce per ton.

Test No. 3

The free gold was recovered by amalgamation and all residues were treated by cyanidation.

A sample of −14-mesh ore was run over the jig and blankets as in previous tests and sand and fine were separated. The sand was reground and again passed over the jig and blankets. The primary fine and the secondary blanket tailing were thickened. All the concentrates, jig and blanket, were reground and barrel-amalgamated and the residue was added to the thickened fine and blanket tailing. This combined product was agitated in two lots for 24 and 48 hours, respectively, in solution of cyanide strength, 2 pounds of sodium cyanide per ton, lime, 3 pounds per ton of ore, and a pulp dilution of 2 : 1.

Cyanidation Results:

Test	Agitation, hours	Total assay, Au, oz./ton	Final titration, lb./ton solution		Reagents consumed, lb./ton ore	
			NaCN	CaO	NaCN	CaO
3 A	24	0·155	2·00	0·25	1·60	3·50
3 B	48	0·08	1·96	0·22	1·68	3·56

Initial feed: gold, 16·38 oz./ton.

Overall Gold Recovery:

24 hours' agitation	99·05 per cent
48 " "	99·51 "

Screen Test on Cyanide Tailing:

Mesh	Weight, per cent
+ 48	0·1
− 48+ 65	0·1
− 65+100	1·1
−100+150	8·5
−150+200	15·5
−200	74·7
Total	100·0

CONCLUSIONS

To obtain a minimum of tailing loss cyanidation is necessary. Although a high recovery is shown by flotation, this loss is appreciable owing to the very high-grade character of the ore.

The gold is very fine-grained and the maximum of recovery will be dependent on very fine grinding. From the infrasizer test on flotation tailing it is shown that 12·3 per cent of the gold in the − 200-mesh product is less than 20 microns in size.

The high recovery obtained by amalgamating jig and blanket concentrates is due to the high grade of the sample, and it should not be expected that a similar recovery would be obtained on lower grade material. It is impossible to predict accurately the recoveries by such methods but owing to the generally fine size of the gold particles and the mode of their occurrence and distribution, not over 60 per cent recovery could be expected on ore of 0·5 to 1·0-ounce grade.

The microscopic examination has shown that this gold is for the greater part enclosed as fine individual grains in the quartz and that only a small proportion is found in the sulphides. This association of the gold and its distribution accounts for the high tailing obtained by flotation of the jig and blanket tailing. If this condition of the gold persists throughout the deposit, flotation can not be counted on as a method for obtaining low tailing losses.

Cyanidation, however, in combination with jigs and blankets, gave tailing assays of 0·045 ounce of gold per ton when grinding to 80 per cent −200 mesh. This is the method of treatment that should be used, when considered from a metallurgical point of view, as undoubtedly the highest recovery of gold can be obtained thereby, regardless of the grade of the ore.

Should present conditions not warrant the expense of a complete cyanide mill, it is suggested that the cost of a skeleton cyanide plant be investigated: the skeleton plant to consist of the crushing and grinding equipment necessary in any mill; jigs and blankets and amalgamating equipment; a cyanide plant with tanks and agitators and thickeners but no filter; and a zinc-dust precipitation unit without the Crowe vacuum system. Such skeleton plants have been found to work satisfactorily.

The total recovery obtained by such a mill should be much greater than that obtained by the use of flotation, in spite of a fairly high soluble loss.

The cost of such a plant should approximate closely the cost of flotation equipment.

Ore Dressing and Metallurgical Investigation No. 778

GOLD ORE FROM THE MALARTIC GOLD FIELDS, LIMITED, NORRIE, QUEBEC

Shipment. Nine bags of ore, total weight 480 pounds, were received on July 3, 1939, from the Malartic Gold Fields, Limited, Norrie, Quebec, per R. A. Halet, Manager.

A previous shipment was received in February, 1939, but the report of its investigation has not been printed.

Location of Property. The property of the Malartic Gold Fields, Limited, from which the present shipment was received is situated in Fourniere, Malartic, and Dubuisson Townships, northwestern Quebec, and is about 1 mile from the Canadian National Railway line.

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

Gold	0·30 oz./ton
Silver	0·07 "
Copper	Trace
Arsenic	0·11 per cent
Sulphur	4·19 "
Iron	9·66 "

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

The *gangue* is composed of an assemblage of dark greenish grey rock, translucent white to grey quartz, and abundant, fine, disseminated carbonate.

In the polished sections *metallic minerals* occur almost entirely within the rock material. Pyrite predominates greatly as coarse to fine disseminated grains; many contain inclusions of gangue, and some are slightly fractured and the fractures are filled with gangue.

Arsenopyrite is in comparatively small quantity, as medium to small irregular grains and subhedral crystals admixed with the pyrite. In many places these two sulphides are very intimately associated. Rare, tiny grains of chalcopyrite and pyrrhotite are visible in pyrite and in gangue, but the total amount of both minerals is almost negligible. A considerable quantity of ilmenite (?), with perhaps some magnetite and specularite, is present as fine, irregular grains in gangue. Under crossed nicols many of these grains show alteration to a white, translucent material, probably leucoxene.

Only one grain of native gold was observed in the sections. It is 32 microns in size and occurs in apparently dense pyrite against a small inclusion of gangue.

9007—4½

Investigative Work

This consisted of a straight cyanidation and also of regrinding of the sulphide concentrates obtained by jig, table, and flotation. Flotation of the ore as a primary operation, prior to cyanidation of the resulting flotation concentrate, was also attempted. The results of straight cyanidation gave an extraction of 96·7 per cent of the gold and a cyanide residue of 0·01 ounce of gold per ton at a grind of 83 per cent − 200 mesh in 30 hours' agitation.

STRAIGHT CYANIDATION

Test No. 1 (A to I)

The ore at −14 mesh was ground in cyanide solution of a strength of 1 pound of sodium cyanide per ton to different degrees of fineness as noted. The pulps were agitated for various periods and the cyanide tailing was assayed for gold.

Screen tests showed the grinding as follows:

Mesh	Weight, per cent — Test No. 1A	1B-1C-1D-1E	1F	1G	1H	1I
− 48+ 65	1·4	....	....	....	....	....
− 65+100	6·8	1·0	0·1	0·1	....	....
−100+150	14·1	5·9	2·4	1·7	1·0	0·5
−150+200	12·3	9·4	5·5	4·5	3·9	2·5
−200	65·4	83·7	92·0	93·7	95·1	97·0
	100·0	100·0	100·0	100·0	100·0	100·0

Results of Cyanidation:

Feed: gold, 0·30 oz./ton

Test No.	Agitation, hours	Grind, per cent −200 mesh	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution NaCN	Titration, lb./ton solution CaO	Reagents consumed, lb./ton ore NaCN	Reagents consumed, lb./ton ore CaO
1A	24	65·4	0·025	91·7	1·0	0·15	0·50	6·1
1B	24	83·7	0·015	95·0	1·0	0·15	0·70	6·1
1C	31	83·7	0·01	96·7	1·0	0·10	0·70	6·2
1D	40	83·7	0·01	96·7	0·9	0·10	0·80	6·3
1E	48	83·7	0·01	96·7	1·0	0·25	0·80	6·0
1F	24	92·0	0·015	95·0	1·0	0·25	0·80	6·1
1G	24	93·7	0·01	96·7	1·0	0·25	0·80	6·1
1H	48	95·1	0·01	96·7	0·9	0·20	1·16	6·1
1I	48	97·0	0·005	98·3	0·9	0·20	1·30	6·2

A portion of the cyanide tailing of Test No. 1C was passed through the Haultain infrasizer, with the following results:

The +200-mesh product was 13·6 per cent of the total weight and assayed 0·01 ounce of gold per ton and 3·00 per cent of sulphur.

The −200-mesh product infrasized as follows:

Microns	Weight, per cent	Assay		Assay units		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S	Au	S
+56	11·3	0·035	9·76	0·395	110·3	46·8	27·1
−56+40	23·5	0·005	4·39	0·117	103·2	13·9	25·3
−40+28	16·1	0·005	3·41	0·085	54·9	10·1	13·5
−28+20	12·5	0·005	3·39	0·063	42·3	7·5	10·4
−20+14	9·9	0·005	3·00	0·049	29·7	5·8	7·3
−14+10	7·4	0·005	2·69	0·037	19·9	4·4	4·9
−10	19·3	0·005	2·43	0·097	46·9	11·5	11·5
Totals	100·0	0·008	4·07	0·843	407·2	100·0	100·0

These results seem to show that most of the gold is in the coarser-size products along with the sulphides and suggests finer grinding if economically possible.

CYANIDATION AND DECANTATION

Test No. 2 (A and B)

This test was to discover whether it was feasible to discard a proportion of the slime from a coarse cyanide grind and agitation, and to regrind and agitate the sand product.

The ore was ground in cyanide solution of a strength of 1 pound of sodium cyanide per ton to pass 65 per cent −200 mesh in Test No. 2A and 55 per cent −200 mesh in Test No. 2B. At the conclusion of a 24-hour agitation period the slime was decanted and the sand was reground in cyanide solution to pass 96 per cent −200 mesh and was agitated for 24 hours. The different products were assayed for gold.

Results:

Primary Cyanidation:

Feed: gold, 0·30 oz./ton

Test No.	Agitation, hours	Grind, per cent −200 mesh	Tailing assay, Au, oz./ton	Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
					NaCN	CaO	NaCN	CaO
2A	24	65·0	0·02	93·4	0·96	0·38	0·28	6·2
2B	24	55·0	0·04	86·7	0·92	0·38	0·34	6·2

Decantation of Cyanide Tailings:

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
2A	Sand	68·9	0·027	92·5
2A	Slime	31·1	0·005	7·5
2B	Sand	73·5	0·053	96·7
2B	Slime	26·5	0·005	3·3

Regrinding and Agitation of Sand Products:

Test No.	Agitation, hours	Grind, per cent −200 mesh	Assay, Au, oz./ton		Extraction, Au, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
			Feed	Tailing		NaCN	CaO	NaCN	CaO
2A	24	96·2	0·027	0·01	63·0	1·0	0·22	1·20	7·1
2B	24	95·1	0·053	0·01	81·0	1·0	0·20	1·66	6·6

Summary:

—	Per cent	
	Test No. 2A.	Test No. 2B.
Gold extracted in primary grinding and agitation	93·4	86·7
Gold extracted from sand	3·8	10·4
Overall recovery of gold	97·2	97·1
Gold discarded in slime product	0·5	0·4
Gold lost in cyanide tailing of sand product	2·3	2·5
Weight of pulp discarded in slime	31·1	26·5

CYANIDATION (CYCLE TEST)

Test No. 3

This test was to see if fouling of the cyanide solution took place after use in grinding and agitating different batches of ore.

The ore at −14 mesh was ground in a ball mill in cyanide solution of a strength of 1 pound per ton to pass 94·3 per cent −200 mesh, the fine grind being used in order to accentuate any fouling. The pulp was agitated for a 24-hour period. The solution was filtered off and was used for grinding and agitating a fresh batch of ore. This procedure was followed for five cycles of agitation. The resulting cyanide residue was assayed for gold, as was the cyanide solution from each cycle for reducing power and KCNS.

A screen test showed the grinding as follows:

Mesh	Weight, per cent
−100+150	1·5
−150+200	4·2
−200	94·3
	100·0

Results of Cyanidation:

Cycle No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore		Solution assay	
				NaCN	CaO	NaCN	CaO	Reducing power, ml.N/10 $KMnO_4$ litre	KCNS, grm./litre
1	24	0·012	96·0	0·96	0·22	0·68	7·0	90·0	0·09
2	24	0·010	96·7	0·90	0·20	0·50	6·0	148·0	0·17
3	24	0·012	96·0	1·00	0·24	0·58	5·9	172·0	0·15
4	24	0·010	96·7	0·96	0·20	0·42	5·9	190·0	0·19
5	24	0·012	96·0	1·00	0·20	0·65	6·0	212·0	0·24

The above results show no noticeable fouling in the cyanide solution.

CYANIDATION AND CONCENTRATION

Test No. 4 (A and B)

The cyanide residue was concentrated on a Wilfley table in Test No. 4A and in a flotation machine in Test No. 4B. These concentrates were reground in cyanide solution of a strength of 2 pounds of sodium cyanide per ton and were agitated for 24 hours. The primary grind was 83 per cent −200 mesh and the concentrates were reground to pass 96 per cent −325 mesh.

Results:

Cyanidation in Primary Grind and Agitation:

Feed: gold, 0·30 oz./ton

Test No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
				NaCN	CaO	NaCN	CaO
4A	24	0·015	95·0	0·96	0·24	0·28	6·5
4B	24	0·015	95·0	0·96	0·24	0·28	6·5

Concentration of the Cyanide Residues:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Test No. 4A:				
Feed	100·00	0·015	100·0	....
Table concentrate	6·56	0·08	35·3	15·2 : 1
Table middling	3·66	0·02	4·7	27·3 : 1
Table tailing	89·78	0·01	60·0	....
Test No. 4B:				
Feed	100·00	0·015	100·0	....
Flotation concentrate	11·37	0·09	67·3	8·8 : 1
Flotation middling	3·03	0·02	4·0	33 : 1
Flotation tailing	85·60	0·005	28·7	....

Regrinding and Agitation of the Concentrates:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton conc.	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
4A	24	0·08	0·03	62·5	1·9	0·30	0·7	9·4
4B	24	0·09	0·035	61·1	1·8	0·25	3·2	12·5

Summary:

—	Per cent	
	Test No. 4A	Test No. 4B
Gold extracted by primary cyanidation	95·0	95·0
Gold recovered by table concentration	2·0	....
Gold recovered by flotation concentration	....	3·5
Gold extracted from table concentrate	1·2	....
Gold extracted from flotation concentrate	....	2·1
Overall extraction	96·2	97·1

This shows that an additional 1·2 per cent of the gold was recovered in Test No. 4A and 2·1 per cent in Test No. 4B. In the table concentration of Test No. 4A part of the gold was lost in the slime.

CONCENTRATION AND CYANIDATION

Test No. 5 (A and B)

The sulphides from the primary cyanide grind were concentrated by passing the pulp through a Denver jig in Test No. 5A and over a Wilfley table in Test No. 5B. The resulting concentrates were reground in cyanide solution to pass 99·0 per cent −325 mesh and agitated for 48 and 36 hours. The jig tailing and table tailing were agitated for 48 and 36 hours. The primary cyanide grind was 83 per cent −200 mesh and gave a product assaying 0·10 ounce of gold per ton in Test No. 5A and 0·095 ounce of gold per ton in Test No. 5B, these figures representing extractions of 66·6 per cent and 68·4 per cent of the gold in the ore.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Test No. 5A: Jig Concentration:				
Feed	100·00	0·10	100·0	....
Jig concentrate	5·47	0·67	34·2	18·3 : 1
Jig tailing	94·53	0·07	65·8	....
Test No. 5B: Table Concentration:				
Feed	100·00	0·095	100·00	....
Table concentrate	5·20	1·15	61·9	19·2 : 1
Table middling	3·67	0·135	5·1	27·2 : 1
Table tailing	91·13	0·035	33·0	....

Cyanidation of Concentrates and Tailings:

Test No.	Product	Agitation, hours	Grind, per cent −200 mesh	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton	
				Feed	Tailing		NaCN	CaO
5A	Concentrate	48	99·0	0·63	0·03	95·5	3·80	19·5
5A	Tailing	48	83·0	0·07	0·01	85·7	0·96	6·0
5B	Concentrate	36	99·0	1·15	0·035	97·0	2·60	15·4
5B	Tailing	36	83·0	0·035	0·01	71·4	0·78	5·5

Summary:

—	Per cent	
	Test No. 5A	Test No. 5B
Gold extracted in primary grind	66·6	68·4
Gold recovered in concentrate	11·4	21·2
Gold extracted from concentrate	10·8	20·5
Gold extracted from tailing	18·8	7·4
Overall recovery	96·2	96·3

Test No. 6

The ore was ground to pass 65·9 per cent −200 mesh and concentrated in a Denver jig. The jig tailing was reground with 3 pounds of soda ash, 0·05 pound of amyl xanthate, and 0·05 pound of pine oil per ton to pass 89·0 per cent −200 mesh. The pulp was transferred to a flotation machine and was floated with 0·10 pound of amyl xanthate, 0·07 pound of pine oil, and 1·5 pounds of copper sulphate per ton. The combined jig and flotation concentrates were washed, reground in cyanide solution to pass 99·0 per cent −325 mesh, and agitated for 48 hours. The flotation tailing was infrasized and analysed for gold and sulphur. The different products were assayed for gold.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Jig Concentration:				
Feed	100·00	0·30	100·0	....
Jig concentrate	5·95	1·80	35·7	16·8 : 1
Jig tailing	94·05	0·205	64·3	....
Flotation Concentration of Jig Tailing:				
Feed	100·00	0·205	100·0	....
Flotation concentrate	15·58	1·22	93·0	6·4 : 1
Flotation tailing	84·42	0·017	7·0	....

Cyanidation of Combined Concentrates:

Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
42	1·40	0·03	97·9	1·8	0·25	5·2	18·4

Infrasizing Test on Flotation Tailing:

The +200-mesh product assayed 0·025 ounce of gold per ton and 0·09 per cent of sulphur and formed 14·3 per cent of the total weight.

The −200-mesh material infrasized as follows:

Microns	Weight, per cent	Assay		Assay units		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S	Au	S
+56	4·46	0·073	0·32	0·325	3·25	18·8	10·4
−56+40	20·55	0·02	0·12	0·411	2·47	23·8	7·9
−40+28	17·78	0·015	0·11	0·267	1·96	15·5	6·3
−28+20	15·77	0·01	0·12	0·158	1·89	9·2	6·1
−20+14	11·66	0·01	0·19	0·117	2·21	6·8	7·1
−14+10	9·04	0·015	0·14	0·135	1·26	7·8	4·1
−10	20·74	0·015	0·87	0·311	18·04	18·1	58·1
	100·00	0·017	0·31	1·724	31·08	100·0	100·0

Summary:

	Per cent
Gold recovered in jig concentrate	35·7
Gold recovered in flotation concentrate	59·8
Gold extracted from combined concentrates	93·5

It is apparent, from the infrasizing test on the flotation tailing, that there is no close relationship in the distribution of the gold and sulphur in the different-size material.

CONCENTRATION

Test No. 7

The ore at −14 mesh was ground in a ball mill with 3 pounds of soda ash, 0·05 pound of amyl xanthate, and 0·05 pound of pine oil per ton to pass 88·8 per cent −200 mesh. The pulp was transferred to a flotation machine and was floated with 0·05 pound of amyl xanthate, 0·05 pound of pine oil, and 1·5 pounds of copper sulphate per ton. The resulting flotation concentrate was cleaned on a smaller machine. A portion of the flotation tailing was concentrated on a Haultain superpanner.

Results:

Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·30	100·0	....
Flotation concentrate	10·17	2·58	87·5	9·8 : 1
Flotation middling	4·41	0·565	8·3	22·7 : 1
Flotation tailing	85·42	0·015	4·2	....

The pH of the pulp was 8·6.

Panning of Flotation Tailing:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concentration
		Au, oz./ton	S, per cent	Au	S	
Feed	100·00	0·016*	0·19*	100·0	100·0	....
Concentrate	1·7	0·055	0·46	5·8	4·0	58 : 1
Sand	69·1	0·015	0·18	66·0	64·2	....
Slime	29·2	0·015	0·21	28·2	31·8	....

*Calculated.

The panner concentrate consisted of magnetite and ilmenite and assayed 46·1 per cent of iron and 18·7 per cent of titanium oxide. Under microscopic examination no free gold was visible.

Test No. 8

The ore at −14 mesh was ground to pass 65 per cent −200 mesh and was passed through a hydraulic classifier or trap. The trap tailing was reground and floated, using the same quantities of reagents as in Test No. 7. The flotation tailing was passed over a corduroy blanket.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Trap Concentration:				
Feed	100·00	0·30	100·0	....
Trap concentrate	1·46	3·68	17·9	68·5 : 1
Trap tailing	98·54	0·25	82·1	....
Flotation Concentration of Trap Tailing:				
Feed	100·00	0·25	100·0	....
Flotation concentrate	17·00	1·37	93·4	5·9 : 1
Flotation tailing	83·00	0·02	6·6	....
Blanket Concentration of Flotation Tailing:				
Feed	100·00	0·02	100·0	....
Blanket concentrate	1·32	0·39*	26·0	75·7 : 1
Blanket tailing	98·68	0·015	74·0	....

*Calculated.

The trap and blanket concentrates were examined under the microscope and free gold was not discernible. The blanket concentrate consisted largely of magnetite and ilmenite and assayed 52 per cent iron.

FLOTATION, CONCENTRATION, AND CYANIDATION

Test No. 9 (A, B, C, and D)

The ore at −14 mesh was ground in a ball mill to pass 88·0 per cent −200 mesh. Different reagents were added to the grind as noted. The pulps were transferred to a flotation machine and flotation concentrates were obtained. The flotation tailings were washed, filtered, sampled, and agitated in cyanide solution of a strength of 1 pound of sodium cyanide for 24 hours.

Flotation Reagents: The following reagents were added:

(Quantities given are pounds per ton)

Test No. 9A	Test No. 9B	Test No. 9C	Test No. 9D
To the grind:			
Soda ash........ 3·0	Soda ash......... 3·0	Soda ash......... 3·0	Soda ash......... 3·0
Cresylic acid.... 0·26	Barrett No. 4 oil. 0·17	Aerofloat No. 25. 0·14	Aerofloat No. 31. 0·07
Amyl xanthate.. 0·10	Amyl xanthate... 0·10	Amyl xanthate... 0·05	Amyl xanthate.. 0·05
To the cells:			
Pine oil......... 0·15	Pine oil.......... 0·15	Pine oil.......... 0·05	Pine oil.......... 0·15
Amyl xanthate.. 0·10	Amyl xanthate... 0·10	Amyl xanthate... 0·06	Amyl xanthate... 0·10

Results:

Flotation:

Test No.	Weight of concentrate, per cent	Ratio of concentration	Tailing assay		Recovery of gold, per cent
			Au, oz./ton	S, per cent	
9A	15·0	6·7 : 1	0·02	0·14	93·3
9B	16·4	6·1 : 1	0·02	0·16	93·3
9C	20·0	5 : 1	0·02	0·15	93·3
9D	22·3	4·5 : 1	0·015	0·13	95·0

Cyanidation of Flotation Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
9A	24	0·02	0·005	75·0	0·9	0·20	0·2	1·6
9B	24	0·02	0·005	75·0	0·9	0·20	0·2	1·6
9C	24	0·02	0·005	75·0	0·9	0·20	0·2	1·6
9D	24	0·015	0·005	66·7	0·9	0·20	0·2	1·6

The results seem to show that some of the gold is in the gangue and is not amenable to flotation.

SETTLING TEST

Test No. 10

To see if the pulp settled at a normal rate of speed after grinding and agitation, the ore at −14 mesh was ground in a ball mill, with 1 pound of sodium cyanide per ton of solution and 6·0 pounds of lime per ton of ore, to pass 83 per cent −200 mesh. The pulp was transferred to a tall settling tube of 2 inches diameter and readings were made every 5 minutes in decimals of feet. This procedure was followed for a one-hour period.

Results of Settling Test:

Ratio of liquid to solid	1·5 : 1
Lime added, lb./ton solid	6·0
Sodium cyanide added, lb./ton solution	1·0
Alkalinity of solution at end of test, lime lb./ton	0·42
Overflow solution	Clear
Rate of settling, in feet per hour	0·45

The rate of settling is normal.

SUMMARY AND CONCLUSIONS

Straight cyanidation of the ore was successful in producing a cyanide residue of 0·01 ounce of gold per ton in 30 hours' agitation at a grind of 83 per cent −200 mesh. After grinding to 97 per cent −200 mesh and 48 hours' agitation a residue of 0·005 ounce of gold per ton was obtained.

To avoid fine grinding the whole tonnage, a coarse initial grind may be adopted followed by a shortened period of agitation. By desliming the tailing from this operation and regrinding the sand, as shown in Test No. 2, an extraction of 97 per cent may be obtained.

A cycle cyanidation test showed that no noticeable fouling of the cyanide solutions is to be expected.

Concentration, regrinding and agitation of the sulphides in the ore, either in the mill circuit or as a scavenging operation on the cyanide residue, raised the overall extraction of the gold to a slight extent.

Flotation concentration, as a primary operation, was not successful in producing a flotation tailing lower than 0·015 ounce of gold per ton. This was possibly due to a small amount of the gold being in the gangue and not amenable to concentration by flotation. Owing to the large amount of sulphides in the ore the ratio of concentration was necessarily low.

The infrasizing test on the cyanide residue showed a large proportion of the remaining gold in the coarser-size material and emphasized the necessity for fine grinding. On the flotation tailing the infrasizer showed no close relationship between the gold and the sulphur and suggests that some of the gold was in the gangue.

Straight cyanidation is the method indicated for the treatment of this ore. If after the mill is in operation a further extraction of the gold is thought economical, a scavenging operation, by flotation of the cyanide residue, could be considered.

Ore Dressing and Metallurgical Investigation No. 779

ARSENICAL GOLD ORE FROM THE BARNATO MINERAL CLAIM, NEAR WESTBRIDGE, BRITISH COLUMBIA

Shipment. One bag of ore, weighing 93 pounds, was received on July 5, 1939, from John H. Redden, Caulfield, West Vancouver, British Columbia. The sample shipment was from the Barnato mineral claim, 25 miles north of Westbridge, British Columbia, on the Kettle River.

A previous shipment received by the Department in December 1937, was reported on in February 1938, but not published.

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

Gold	0·87 oz./ton
Silver	0·27 "
Copper	0·18 per cent
Zinc	2·56 "
Iron	24·86 "
Arsenic	11·70 "
Sulphur	19·43 "
Lead	Nil
Antimony	Nil
Tellurium	Trace
Pyrrhotite	3·17 per cent

Characteristics of the Ore:

Six polished sections were prepared and examined microscopically.

The *gangue* is composed of dark greenish grey siliceous rock and grey to white quartz. In general it probably represents a highly, but unevenly, silicified rock.

Metallic minerals and gangue are visible in about equal amounts in the sections. The former are represented mainly by an admixture of arsenopyrite and pyrite. Both sulphides occur largely as coarse-textured masses intimately associated. The arsenopyrite is much fractured and veined with both pyrite and quartz. Massive pyrite contains crystals of arsenopyrite. Large quantities of sphalerite and chalcopyrite are present as small, irregular grains in gangue and in the arsenopyrite-pyrite masses. Rare, tiny grains of pyrrhotite are visible in both pyrite and arsenopyrite.

No native gold or gold minerals were observed in the sections and nothing was learned as to its mode of occurrence.

Experimental Tests

The test work corsisted of concentration, amalgamation, and cyanidation.

The best results were obtained by flotation concentration of the ore in a lime pulp followed by aeration and cyanidation of the flotation tailing.

Amalgamation was not very successful owing to the fact that much of the gold is finely divided and is in intimate association with the sulphides.

The shipment was rather heavily oxidized and on this account the metallurgical picture, as portrayed in this report, is subject to some reservations, particularly as regards the amounts of reagents used in the various tests.

CONCENTRATION, AMALGAMATION, AND CYANIDATION

Test No. 1

The ore at −14 mesh was ground in a ball mill to pass 64·3 per cent −200 mesh. The pulp was passed through a hydraulic classifier or trap, and the trap tailing was passed over a corduroy blanket. The combined trap and blanket concentrates were amalgamated with mercury for one hour in a jar mill. The amalgam residue was added to the blanket tailing and this product was dewatered, reground in cyanide solution of 1 pound per ton strength, and agitated for 24- and 48-hour periods.

Screen tests showed the grinding as follows:

Mesh	Weight, per cent	
	Primary grind	Cyanide grind
− 48+ 65	1·4	....
− 65+100	5·8	0·1
−100+150	14·5	2·3
−150+200	14·0	5·7
−200	64·3	91·9
	100·0	100·0

Results:

Trap and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·87	100·0	....
Combined concentrates	5·85	4·41	29·6	17·1 : 1
Blanket tailing	94·15	0·65	70·4	....

Owing to the large amount of sulphides, blanket concentration was not successful.

An examination of the trap concentrate under the microscope showed an extremely small quantity of very finely divided free gold.

The combined trap and blanket concentrates were amalgamated with mercury and the amalgam residue added to the blanket tailing. This product assayed 0·67 ounce gold per ton.

Cyanidation of Blanket Tailing and Amalgam Residue:

Feed: 0·67 Au oz./ton.

Agitation, hours	Tailing Assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton of solution		Reagents consumed, lb./ton of ore		Solution assay	
			NaCN	CaO	NaCN	CaO	Reducing power, ml. $\frac{N}{10}$ $KMnO_4$/litre	KCNS, grm./litre
24	0·075	88·8	0·96	0·06	2·17	10·95	400	0·29
48	0·07	89·5	1·04	0·12	2·96	14·60	490	0·62

The cyanide solutions showed a certain amount of fouling at the end of the agitation period, as exemplified by the rather high reducing power and KCNS.

Summary of Test No. 1:

	Per cent
Gold recovered in trap and blanket concentrates......................	29·6
Gold recovered by amalgamation from trap and blanket concentrates..	23·0
Gold extracted by cyanidation in 24 hours' agitation..................	68·4
Gold extracted by cyanidation in 48 hours' agitation..................	68·9
Overall recovery (amalgamation + 48 hours' cyanide extraction).......	91·9

CONCENTRATION AND AMALGAMATION

Test No. 2

The ore was ground similarly to Test No. 1, and the pulp was passed through a small Denver jig. The jig tailing was filtered, sampled, and reground with 8 pounds of soda ash, 0·05 pound of amyl xanthate, and 0·05 pound of pine oil per ton, and was floated by the further additions of 0·10 pound of amyl xanthate, 0·05 pound of pine oil, and 1·5 pounds of copper sulphate per ton. The jig concentrate and the flotation concentrate were washed and amalgamated separately with mercury in a jar mill. The different products were assayed for gold.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Jig Concentration:				
Feed..................................	100·00	0·87	100·0	....
Jig concentrate........................	13·20	1·95	29·7	7·6 : 1
Tailing................................	86·80	0·63	70·3	....
Flotation of Jig Tailing:				
Feed..................................	100·00	0·63	100·0	....
Flotation concentrate..................	34·37	1·51	83·2	2·9 : 1
Flotation middling.....................	11·21	0·63	11·2	8·9 : 1
Tailing................................	54·06	0·065	5·6	....

The jig and flotation concentrates assayed 20·9 and 16·4 per cent arsenic.

Amalgamation of Concentrates:

Product	Assay, Au, oz./ton		Recovery, per cent
	Feed	Tailing	
Jig concentrate	1·95	0·775	60·3
Flotation concentrate	1·51	1·12	25·8

A portion of the flotation tailing was concentrated on the Haultain superpanner, with the following results:

Product	Weight, per cent	Assay		Distribution, per cent		Ratio of concen-tration
		Au, oz./ton	As, per cent	Au	As	
Feed	100·0	0·06*	2·91*	100·0	100·0	....
Panner concentrate	1·7	0·72	29·78	20·2	17·4	58 : 1
Panner sand	40·3	0·055	1·24	36·6	17·1	....
Panner slime	58·3	0·045	3·29	43·2	65·5	....

*Calculated.

The panner concentrate showed no free gold under the microscope and consisted largely of arsenopyrite.

Summary of Test No. 2:	Per cent
Gold recovered in jig concentrate	29·7
Gold recovered by amalgamation from jig concentrate	17·9
Gold recovered by flotation	66·4
Gold recovered by amalgamation from flotation concentrate	17·1
Gold recovered in combined concentrates	96·1
Gold recovered by amalgamation from combined concentrates	35·0

The above results show that, owing to the high sulphide content of the ore, bulk flotation produces a comparatively low-grade concentrate owing to the low ratio of concentration necessary.

CYANIDATION

Test No. 3 (A, B, C, and D)

In Tests Nos. 3A and 3B the ore at −14 mesh was ground in cyanide solution of 1 pound per ton strength to pass 70·7 per cent and 91·0 per cent −200 mesh. The pulps were agitated for 24 hours. In Tests Nos. 3C and 3D the ore at −14 mesh was ground in lime pulps to similar degrees of fineness, was aerated for 16 hours in a Wallace agitator, and was agitated for 24 hours in cyanide solution of a strength of 1 pound of sodium cyanide. The cyanide tailings were subjected to screen tests, with the following results:

Mesh	Weight, per cent	
	Tests Nos. 3A and 3C	Tests Nos. 3B and 3D
− 48+ 65	0·2	....
− 65+100	2·8	....
−100+150	10·8	2·3
−150+200	15·5	6·7
−200	70·7	91·0
	100·0	100·0

Results of Cyanidation:

Feed: gold, 0·87 oz./ton

Test No.	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton of solution		Reagents consumed, lb./ton of ore		Solution assays	
				NaCN	CaO	NaCN	CaO	Reducing power, ml. $KMnO_4$/litre	KCNS, grm./litre
3A	24	0·19	78·2	0·8	0·20	2·40	15·6	406	0·24
3B	24	0·12	86·2	0·7	0·20	2·80	16·0	458	0·34
3C	24	0·215	75·3	0·8	0·15	2·00	5·7*	244	0·19
3D	24	0·33	62·1	0·7	0·10	2·20	6·8*	284	0·19

*Ten additional pounds of lime per ton was used in the aeration.

It is apparent that grinding in cyanide solution gives a better extraction of the gold than grinding in a lime pulp prior to aeration and cyanidation. On the other hand, cyanide consumption decreases and the foulness of the solution is corrected when grinding is performed in a lime pulp.

FLOTATION AND CYANIDATION

Test No. 4 (A, B, C, and D)

The ore at −14 mesh was ground in a ball mill to pass 89·7 per cent −200 mesh. The pulp was transferred to a flotation machine and a concentrate was obtained by the addition of different flotation reagents as noted. This concentrate was cleaned in a smaller machine and the flotation tailing was agitated in cyanide solution.

Reagents Added to the Grind:

Test No.	Lime, lb./ton
4A	13·0
4B	14·0
4C	13·0
4D	20·0

Reagents Added to the Cells:

Test No.	Butyl xanthate, lb./ton	Pine oil, lb./ton
4A	0·05	0·05
4B	0·05	0·07
4C	0·04	0·05
4D	0·08	0·07

In Tests Nos. 4A and 4B the flotation tailings were agitated in cyanide solutions of a strength of 1 pound of sodium cyanide per ton.

In Test No. 4C the flotation tailing was aerated in a lime pulp prior to agitation in cyanide solution.

In Test No. 4D the flotation tailing was aerated in a lime pulp and 0·25 pound of lead nitrate per ton added, prior to cyanidation.

Results of Flotation:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Test No. 4A:				
Feed	100·0	0·815*	100·0	....
Flotation concentrate	8·2	6·92	69·6	12·2 : 1
Flotation tailing	91·8	0·27	30·4	....
Test No. 4B:				
Feed	100·00	0·83*	100·0	....
Flotation concentrate	7·80	6·82	64·0	12·8 : 1
Flotation middling	8·25	1·44	14·3	12·1 : 1
Flotation tailing	83·95	0·215	21·7	....
Test No. 4C:				
Feed	100·0	0·84*	100·0	....
Flotation concentrate	2·3	20·10	54·9	43·4 : 1
Flotation middling	2·8	4·42	14·7	35·7 : 1
Flotation tailing	94·9	0·27	30·4	....
Test No. 4D:				
Feed	100·00	0·82*	100·0	....
Flotation concentrate	6·32	9·20	70·8	15·8 : 1
Flotation middling	2·55	1·36	4·2	39·2 : 1
Flotation tailing	91·13	0·225	25·0	....

*Calculated.

Results of Cyanidation of Flotation Tailing:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
4A	24	0·27	0·07	74·1	1·00	0·22	2·00	4·0
4B	24	0·215	0·07	67·5	0·96	0·20	2·65	4·0
4C	24	0·27	0·07	74·1	0·88	0·10	2·02	5·9*
4C	48	0·27	0·075	72·2	1·00	0·04	2·58	8·3*
4D	24	0·225	0·08	64·5	0·96	0·10	0·78	5·6†
4D	48	0·225	0·07	68·9	0·92	0·08	1·60	7·1†

*Aeration. †Aeration and lead nitrate.

A further analysis of the different flotation concentrates resulted as follows:

Test No.	Assay			Recovery, per cent		
	Au, oz./ton	Cu, per cent	As, per cent	Au	Cu	As
4A	6·92	1·45	15·05	69·6	71·7	10·7
4B	6·82	1·49	17·18	64·0	64·5	11·4
4C	20·10	4·24	10·82	54·9	54·2	2·2
4D	9·20	1·92	10·13	70·8	75·0	5·1

The flotation concentrate of Test No. 4D assayed 25·2 per cent of sulphur and 3·7 per cent of insoluble. It is apparent that the gold is in close association with the chalcopyrite and not with the arsenopyrite.

Summary of Test No. 4:

—	Per cent			
	Test No. 4A	Test No. 4B	Test No. 4C	Test No. 4D
Gold recovered in flotation concentrate and middling	69·6	78·3	69·6	75·0
Gold extracted from flotation tailing	22·5	14·6	22·5	16·1
Overall recovery of gold, 24-hour agitation	92·1	92·9	92·1	91·1

SUMMARY AND CONCLUSIONS

Bulk flotation of the ore in a soda-ash pulp is not feasible owing to the large amount of sulphides. By this method, preceded by jig concentration, 96·1 per cent of the gold was recovered, but the resulting concentrates were low grade, assaying 1·95 and 1·51 ounces of gold per ton.

Straight cyanidation of the ore gave an extraction of 86·2 per cent of the gold at a grind of 91·0 per cent −200 mesh. The cyanide solution showed fouling. On grinding in a lime pulp and aerating prior to agitation the extraction was lowered to 75·3 per cent although the cyanide consumption was lessened and the fouling of solution was partly corrected.

Amalgamation of jig and flotation concentrates resulted in a recovery of 35·0 per cent of the gold in the amalgam.

Flotation of the ore in a lime pulp, followed by cyanidation of the flotation tailing, gave an overall recovery of 92·0 per cent of the gold, of which some 70 per cent was recovered in a shipping concentrate assaying 9·2 ounces of gold per ton. The ratio of concentration was 15·8:1.

This last metallurgical procedure appears to be the most suitable. A high-grade flotation concentrate can be obtained as a shipping product and gold remaining in the flotation tailing may be either extracted by cyanidation or retained for future treatment.

Ore Dressing and Metallurgical Investigation No. 780

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

Shipment. A second shipment of ore, composed of twelve separate samples having a combined weight of 1,854 pounds, was received on July 25, 1939, from Athona Mines (1937), Limited, Goldfields, Saskatchewan. The shipment was made by Norman W. Byrne, Resident Manager.

A previous shipment was received on January 31, 1939, and the test work is covered under Investigation No. 771.

Characteristics of the Ore. The ore is similar to that of the first shipment.

Sampling and Assaying. Each sample was crushed, sampled, and assayed separately. For testing a composite bulk sample was prepared to give the required mill feed.

Sample No.	Weight in pounds	Gold, oz./ton
1	160	0·085
2	219	0·0575
3	242	0·36
4	117	0·04
5	116	0·03
6	109	0·08
7	152	0·08
8	172	0·03
9	182	0·135
10	161	0·1025
11	114	0·114
12	110	0·118

The composite bulk sample was composed of the remains of the individual samples with the exception of No. 3, of which only 50 pounds were used.

The composite sample, 1,478 pounds, had a calculated gold content of 0·09 ounce per ton. An assay for tungsten gave a negative result.

Purpose of Investigation. The investigation was for determining the possibility of concentrating the gold in the ore by flotation and its subsequent recovery by cyanidation.

Previous small-scale tests had resulted in a flotation tailing carrying gold, 0·005 ounce per ton. The present investigation was to see if the same results could be obtained from a continuous run in the pilot mill unit.

Results of Investigation. A tailing of 0·005 ounce of gold per ton was obtained in the mill-run tests, a recovery of 94 per cent at a ratio of concentration of 36:1.

A high-grade concentrate was obtained on blankets. This is derived from the free coarse gold in the ore.

Cyanidation of the flotation concentrate gave a satisfactory recovery. On reground concentrate (97 per cent −325 mesh), a tailing of 0·025 ounce of gold per ton was obtained after 48 hours' agitation. The cyanide consumption was 4 pounds of sodium cyanide per ton of concentrate.

Experimental Tests

PRELIMINARY SMALL-SCALE TESTS

Duplicate flotation tests were carried out on Sample No. 2. A charge of ore was ground with 0·5 pound of soda ash per ton and 0·07 pound of Aerofloat No. 25 per ton to have 58·2 per cent −200 mesh. The pulp was transferred to a flotation cell and was conditioned for 3 minutes with 0·5 pound of copper sulphate and 0·1 pound of potassium amyl xanthate per ton and was floated for 5 minutes using 0·062 pound of pine oil per ton as a frother.

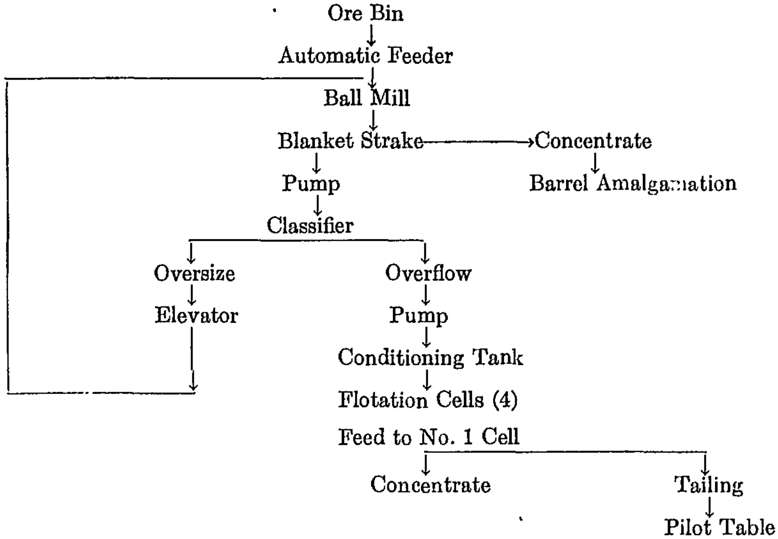
Results:

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
1	Feed (calculated)	100·00	0·042	100·0	....
	Concentrate	1·35	2·74	88·2	74 : 1
	Tailing	98·65	0·005	11·8	....
2	Feed (calculated)	100·00	0·042	100·0	....
	Concentrate	1·56	2·36	88·2	64 : 1
	Tailing	98·44	0·005	11·8	....

Panning of the tailing showed a few grains of sulphide mineral, but no gold.

MILL RUNS

The flow-sheet for the mill runs was as follows:



The pulp was fed to No. 1 Cell and the froth from Cells Nos. 2, 3, and 4 was returned to No. 1.

Blankets were used to collect free gold in place of a jig, because in a grinding circuit of this size the use of blankets provided a closer control of density than could be obtained with the jig in the circuit. In normal mill operation a jig or trap would be quite satisfactory and probably preferable to blankets.

Test Mill Run No. 1

The ore was fed at the rate of 180 pounds per hour. The time of the run was $4\frac{1}{3}$ hours.

The reagents fed were as follows:

Soda ash	0·5 lb./ton	To ball mill
Aerofloat No. 25	0·035 "	To ball mill
Copper sulphate	0·5 "	To No. 2 Cell
Potassium amyl xanthate	0·1 "	To conditioner tank
Pine oil	0·031 "	To No. 2 Cell

Samples were taken every ten minutes during the run.

The assay results are as follows:

Product	Weight, pounds	Assay				
		Oz./ton		Per cent		
		Au	Ag	Zn	Fe	S
Feed	830	0·07	0·07	....	....	....
Ball mill discharge	....	0·06	....	....	....	....
Blanket concentrate	2·0	13·799	....	....	....	....
Blanket tailing	....	0·05	....	....	....	....
Classifier overflow	....	0·03	....	....	....	....
Flotation concentrate	....	0·95	1·25	0·36	6·97	....
Flotation tailing	....	0·005	....	....	....	0·02

The average density of the classifier overflow was 25·6 per cent of solids.

The grinding as indicated by a screen test of the classifier overflow was as follows:

Mesh	Weight, per cent
+ 65	0·4
− 65+100	2·5
−100+150	9·9
−150+200	8·7
−200	78·5
	100·0

The indicated recovery and ratio of concentration calculated from the assays are as follows:

Recovery	93·3 per cent
Ratio of concentration	14·5 : 1

The concentrate was dirty, caused by a heavy non-brittle froth. This was probably due to the action of the Aerofloat. In the subsequent test Aerofloat was not used and a decided improvement was noted.

Much coarse gold was observed on the blanket.

The pilot table sand was practically free from sulphide minerals.

Test Mill Run No. 2

The feed rate was the same as in Test Mill Run No. 1, 180 pounds per hour. The duration of the run was 3 hours.

Aerofloat was discontinued, otherwise the reagents were the same as in the first day's run.

The average density of the classifier overflow was 23·5 per cent solids.

The assay results are tabulated below:

Product	Weight, pounds	Assay				
		Oz./ton		Per cent		
		Au	Ag	Zn	Fe	S
Feed	540	0·085	0·09			
Ball mill discharge		0·06				
Blanket concentrate	2·25	23·82				
Blanket tailing		0·05				
Classifier overflow		0·02				
Flotation concentrate		2·92	3·90	1·28	20·30	
Flotation tailing		0·005				0·02

Screen tests indicate the grinding to be as follows:

Classifier Overflow:

Mesh	Weight, per cent
+ 48	0·2
− 48+ 65	0·7
− 65+100	3·9
−100+150	11·0
−150+200	10·0
−200	74·2
	100·0

Flotation Concentrate:

Mesh	Weight, per cent
+ 48	0·1
− 48+ 65	0·3
− 65+100	1·4
−100+150	4·6
−150+200	3·7
−200	89·9
	100·0

The recovery and ratio of concentration calculated from the above assays are as follows:

Recovery	94·3 per cent
Ratio of concentration	36·4 : 1

Observations. According to an approximate calculation less than 2 per cent of the gold in the feed is retained in the mill-classifier circuit.

A jig, trap, or blanket is necessary to remove coarse free gold from the ball mill discharge.

A pulp density around 23 per cent solids appears to be satisfactory.

The grinding in the mill tests was probably a little finer than is necessary. This was due to using a 14-mesh feed. The small-scale tests have indicated that 60 to 70 per cent −200 mesh is sufficient to free the gold and sulphides from the gangue.

BARREL AMALGAMATION OF BLANKET CONCENTRATE

Samples of the two blanket concentrates were barrel-amalgamated separately. The results are as follows:

Mill Run No. 1.	Blanket concentrate........................	Au, 13·799 oz./ton
	Amalgamation tailing.....................	Au, 1·23 "
	Recovery.........................	91·1 per cent
Mill Run No. 2.	Blanket concentrate........................	Au, 23·82 oz./ton
	Amalgamation tailing.....................	Au, 1·485 "
	Recovery.........................	93·8 per cent

These concentrates exhibit no difficulty in barrel amalgamation. There was no indication of mercury fouling.

CYANIDATION OF FLOTATION CONCENTRATE

Cyanidation tests on the flotation concentrates made in the mill runs were carried out to determine the extraction and the action of the concentrate on the solution.

Tests of Raw Concentrate without Regrinding

Samples of the bulk concentrate from Test Mill Run No. 2 were given a preliminary water wash before agitation.

The concentrate was about 90 per cent −200 mesh and assayed gold, 1·54 ounces per ton.

Agitation was carried out for 24- and 48-hour periods in a solution of strength of 2 pounds of sodium cyanide per ton at a pulp dilution of 2 : 1.

Results:

Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
			NaCN	CaO	NaCN	CaO
24	0·09	94·1	1·72	0·18	3·96	7·64
48	0·055	96·2	1·92	0·20	4·36	8·40

Tests of Reground Concentrate

A sample of concentrate was ground in a water pulp to have 97 per cent −325 mesh. The pulp was dewatered and repulped in cyanide solution of strength of 2·0 pounds of sodium cyanide per ton and a pulp dilution of 2 : 1 (solution: solid).

Results:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Final titration, lb./ton of solution		Reagents consumed, lb./ton concentrate	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
24	1·54	0·03	98·0	2·00	0·22	4·00	8·36
48	1·54	0·025	98·3	2·00	0·22	4·00	8·40

The object of washing the concentrate and grinding in water is to remove flotation reagents, that may be harmful to cyanide solution.

An analysis of the solution after 48 hours' agitation indicates no serious fouling of the solution.

Analysis of Solution:

Reducing power	230 ml. $KMnO_4$ /litre
NaCNS	0·23 grm./litre
Iron	0·003 "
Copper	0·14 "

CYCLE TEST OF FLOTATION CONCENTRATE

The test was of a flotation concentrate having a gold content of 1·66 ounces per ton. The concentrate was reground in water to a fineness of 95 per cent −325 mesh. In the initial cycle a sample of the reground concentrate was agitated for 48 hours in a solution of strength of 3 pounds of sodium cyanide per ton. The pulp dilution was 3·2 parts of solution by weight to 1 part of solid.

The pulp was filtered and 37 per cent of the solution was discarded. The remainder of the pregnant solution was clarified, and de-aerated under vacuum with the addition of a small amount (0·01 gramme) of lead acetate. Zinc dust was added to the de-aerated solution to precipitate the gold. The barren solution was filtered and aerated.

The second cycle was made up using a fresh charge of concentrate and the barren solution from the first cycle. Fresh solution was added to make up to the required pulp dilution.

Four cycles were run and the final barren solution was analysed.

The results are as follows:

Results:

Cycle No.	Agitation, hours	Tailing, Au, oz./ton	Extraction of gold, per cent	Additions, lb./ton		Final titration, lb./ton of solution	
				NaCN	CaO	NaCN	CaO
1	48	0·035	97·8	1·34	1·0	3·0	0·25
2	48	0·035	97·8	0·90	1·0	2·9	0·25
3	48	0·04	97·5	0·90	1·0	3·0	0·25
4	48	0·055	96·7	0·69	1·0	2·9	0·35

Barren Solution from 4th Cycle.

Reducing power...	314 ml. $\frac{N}{10}$ $KMnO_4$/litre
NaCNS...	0·34 grm./litre
Iron..	0·01 "
Copper..	0·28 "

There is an indication of lowered extraction on the 3rd and 4th cycles. In mill operation this could probably be overcome by a control of the amount of solution discarded and the addition of suitable reagents at certain points in the clarification-precipitation circuit.

CONCLUSIONS

The ore may be treated satisfactorily by flotation and cyanidation of the concentrate.

The grind reported in the mill runs is finer than is necessary. This was due to the use of a 14-mesh feed. From the results obtained on small-scale tests a grind of between 60 to 70 per cent −200 mesh is sufficient to free the sulphides from the gangue.

The use of jigs, traps, or blankets is necessary at the mill discharge in the mill-classifier circuit to recover coarse free gold, which constitutes an appreciable amount of the gold in the ore.

A pulp density of 23 per cent of solids appeared to give satisfactory results in flotation.

Regrinding of the flotation concentrate prior to cyanide treatment is recommended. The lowest tailing was obtained at a grind of over 90 per cent −325 mesh.

The copper in the concentrate did not report over 0·10 per cent.

No serious fouling of the solution was observed and the moderate consumption of cyanide reported would allow for a reasonable bleeding of solution to maintain the solution within satisfactory limits.

As pointed out in the previous report of investigations on Athona ore, the use of copper sulphate in flotation is important.

The sphalerite contains sufficient gold to make its recovery in the concentrate essential for flotation to be an economic method of treatment.

Ore Dressing and Metallurgical Investigation No. 781

GOLD ORE FROM THE FAYMAR PORCUPINE GOLD MINES, LIMITED, DELORO TOWNSHIP, SOUTH PORCUPINE AREA, ONTARIO

Shipment. Thirty-five bags of gold ore, total weight 2,010 pounds, were received on July 31, 1939, from the Faymar Porcupine Gold Mines, Limited, Timmins, Ontario, D. J. Ludgate, Manager. C. Earl Rodgers, of Toronto, Ontario, is consulting metallurgist. This property is situated in the South Porcupine area, Ontario.

Characteristics of the Ore:

Polished sections from selected samples were made and examined microscopically.

Gangue. The gangue consists of an assemblage of milky-white quartz and greenish grey, schistose rock. The quartz is traversed by narrow, sinuous cracks, and the rock material carries abundant, fine, disseminated carbonate.

Metallic Minerals. Pyrite predominates as coarse-textured masses and coarse to fine irregular grains and subhedral crystals unevenly distributed throughout gangue. It is much fractured and the fractures are filled with gangue and with chalcopyrite. The latter mineral also occurs as coarse to fine irregular grains disseminated in gangue and in pyrite. Its total quantity is comparatively small. Rare tiny inclusions of pyrrhotite are present in pyrite.

Four grains of native gold, ranging in size from 60 microns to 30 microns, are visible in the sections. Three occur along a gangue-filled fracture and one with an inclusion of gangue in pyrite.

Investigative Procedure. The entire lot was crushed and sampled. Analysis showed the shipment to contain:

Gold	0·29 oz./ton
Silver	0·22 "
Iron	3·19 per cent
Sulphur	0·95 "
Copper	0·10 "

The results of the investigation show that the ore is readily amenable to cyanidation. With a grind of 66·5 per cent −200 mesh, 96·6 per cent extraction was obtained within 24 hours, leaving a residue of 0·01 ounce of gold per ton. An extraction of 98·3 per cent is indicated with an agitation time of 48 hours on ore ground 91·8 per cent −200 mesh.

Straight flotation with a grind of 80 per cent −200 mesh gives the same value in the tailing as that obtained by cyanidation, i.e. 0·01 ounce per ton. The concentrate, with a ratio of concentration of 40: 1, contained 11·07 ounces of gold per ton.

Concentration by jigs, blankets, and flotation recovered 98·5 per cent of the gold in the concentrate. A recovery of 95·6 per cent of the gold in the jig and blanket concentrates was made by barrel amalgamation. This represents 53·7 per cent of the gold in the feed.

Experimental Tests

CYANIDATION

Test No. 1

A sample of the ore was ground in cyanide solution, 1 pound of sodium cyanide per ton, together with 5 pounds of lime per ton. The pulp was diluted to 1: 1·5, the solution strength was brought up to 1·0 pound of sodium cyanide per ton. Sufficient lime was added to maintain a protective alkalinity of 0·3 to 0·5 pound of lime per ton. Agitation was carried on for 48 hours.

The tailing was assayed, and a screen analysis and an infrasizing test was made to note the distribution of the gold in the tailing.

Results:

Feed	Au, 0·29 oz./ton
48-hour cyanide tailing	0·01 oz./ton
Extraction	96·6 per cent

Reagent Consumption:

	Lb./ton
NaCN	1·0
CaO	3·8

Titration at End of Agitation:

	Lb./ton
NaCN	0·9
CaO	0·4

Screen Analysis of Tailing:

Mesh	Weight, per cent	Assay	
		Au, oz./ton	S, per cent
+ 65	0·4	0·01	0·24
− 65+100	4·1		
−100+150	17·3	0·015	0·63
−150+200	18·6	0·01	1·16
−200	59·6	0·01	1·10
	100·0		

A portion of the −200-mesh product was infrasized.

Infrasizing Test:

Size, in microns	Weight, per cent	Assay, Au, oz./ton
−200 mesh +56	3·8	0·077
−56+40	25·1	0·005
−40+28	16·3	0·005
−28+20	13·7	0·005
−20+14	11·0	0·005
−14+10	8·7	0·005
−10	21·4	0·005
	100·0	

These results indicate that the gold is amenable to cyanidation; even the +100-mesh sizes have the same assay as the −200-mesh portion. The infrasizer results show that the −200-mesh+56-micron portion contains gold and/or sulphides which have been concentrated.

With very fine grinding, an 0·005-ounce tailing may be anticipated.

Test No. 2

A duplicate test was made to note the distribution of gold in the slime and sand of the tailing of the preceding test. The tailing was separated into sand and slime by decantation.

Results:

Cyanide tailing: Au, 0·01 oz./ton.	Weight, per cent	Assay, Au, oz./ton
Sand	38·8	0·02
Slime	61·2	0·005

Here again it is seen that the fine portions of the tailing have a gold content of 0·005 ounce per ton.

Test No. 3

A series of tests was made to establish fineness of grind and agitation time.

A number of samples were ground to various degrees of fineness and were agitated for different periods of time.

In all cases, the samples were ground in cyanide solution and were agitated at 1 : 1·5 dilution with a solution having 1·0 pound of sodium cyanide per ton.

Results:

Test No.	Grind, per cent −200 mesh	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Final titration, lb./ton		Reagents consumed, lb./ton of ore	
			Feed	Tailing		NaCN	CaO	NaCN	CaO
3A	42·2	24	0·29	0·015	94·8	0·92	0·54	0·56	4·4
3B	56·0	24	0·29	0·015	94·8	0·92	0·58	0·56	4·4
3C	66·5	24	0·29	0·01	96·6	0·84	0·48	0·68	4·5
3D	82·0	32	0·29	0·01	96·6	1·00	0·32	0·52	4·5
3E	82·0	48	0·29	0·01	96·6	1·00	0·36	0·52	4·5
3F	87·3	32	0·29	0·01	96·6	0·96	0·32	0·74	4·5
3G	87·3	48	0·29	0·01	96·6	0·96	0·32	0·58	4·5
3H	91·8	32	0·29	0·01	96·6	0·96	0·30	0·90	4·5
3I	91·8	48	0·29	0·005	98·3	1·00	0·32	1·00	4·5

Settling Tests

Settling tests were made on cyanide pulps at a dilution of 1·5 : 1 and 2 : 1, as shown below:

Time, minutes	Settling rate, feet per hour — Dilution, L : S	
	1·5 : 1	2 : 1
5	0·05	0·10
10	0·10	0·19
15	0·14	0·28
20	0·18	0·37
25	0·22	0·46
30	0·26	0·55
35	0·30	0·63
40	0·34	0·72
45	0·38	0·80
50	0·42	0·88
55	0·46	0·96
60	0·50	1·04

Titration of Solution:

	Lb./ton
1·5 : 1—NaCN	0·9
CaO	0·45
2 : 1—NaCN	0·7
CaO	0·25

STRAIGHT FLOTATION

Test No. 4

A flotation test was made to determine the ratio of concentration and grade of concentrate obtainable.

Grind........ 80 per cent −200 mesh

Reagents:

	Lb./ton
Soda ash	2·0
Potassium amyl xanthate	0·1
Pine oil	0·08

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed (cal.)	100·0	0·287	100·0
Concentrate	2·5	11·07	95·5
Tailing	97·5	0·01	4·5

Ratio of concentration—40 : 1.

The tailing was separated into slime and sand. The slime portion, 62·4 per cent of the weight, assayed 0·01 ounce of gold per ton, whereas the sand contained 0·02 ounce per ton.

Apparently straight flotation can be expected to produce a flotation tailing equal to that obtained by straight cyanidation.

JIG, BLANKET AND FLOTATION CONCENTRATION

Test No. 5

A sample of the ore was ground approximately 85 per cent −200 mesh and was passed through a mineral jig and over corduroy strakes. The strake tailing was floated as in Test No. 4.

The jig and blanket concentrates were combined, reground, and amalgamated.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed (cal.)	100·00	0·30	100·0
Jig and blanket concentrate	2·88	5·85	56·2
Jig and blanket concentrate after amalgamation	2·88	0·255	2·5
Gold from amalgam	....	....	53·7
Flotation concentrate	3·33	3·81	42·3
Flotation tailing	93·79	0·005	1·5

A recovery of 98·5 per cent of the gold was made by jig, blanket and flotation concentration, with a ratio of concentration of 16 : 1.

Amalgamation recovered 95·6 per cent of the gold in the jig and blanket concentrates.

SUMMARY AND CONCLUSIONS

Straight flotation produces a tailing of 0·01 ounce of gold per ton from the grade of ore submitted.

Microscopic examination shows the presence of small quantities of pyrrhotite and chalcopyrite. The analysis of the feed sample shows the presence of 0·10 per cent of copper. These minerals would tend to concentrate, resulting in complications when cyaniding these concentrates. The residues from cyaniding these concentrates would raise the value of the total mill tailing.

Straight cyanidation at a grind of over 66 per cent −200 mesh produces a mill tailing of 0·01 ounce of gold per ton, or 96·6 per cent extraction from the sample furnished.

The absence from the ore of deleterious minerals in any large quantities ensures satisfactory cyanidation.

Over 50 per cent of the gold in the sample can be recovered from concentrates made by jigs and blankets in the circuit.

Straight cyanidation is the method recommended for treatment of this ore. A moderate grind, 75 to 80 per cent −200 mesh, with from 30 to 35 hours' agitation will, doubtless, produce most economic results.

Should the size and quantity of free gold particles in the ore increase and give rise to erratic tailing assays, it would be advisable to install a mineral jig in the ball mill–classifier circuit. The resulting concentrate could be barrel-amalgamated and the residue returned to the classifier.

Ore Dressing and Metallurgical Investigation No. 782

FOUR STAINLESS STEEL TEST SAMPLES FROM BANFF, ALBERTA

Origin of Material and Object of Investigation. On October 6, 1939, four samples of stainless steel sheet were received from Mr. R. A. Gibson, Director, Lands, Parks and Forests Branch, Department of Mines and Resources, Ottawa, Ontario. These samples had been immersed in hot sulphur water at the Upper Hot Springs bath-house in Banff, Alberta, from June 19 to September 29, 1939. An examination of the samples was requested in order to determine the extent of corrosion, and the suitability and probable life of the material.

Descriptions of Material. The following table gives a description of the various samples. The column marked "Composition" contains information given by Mr. Gibson.

Specimen	Composition	Finish	Area of one surface (sq. in.)	Thickness, inches
Staybrite F.S.T......	18 p.c. chromium, 8 p.c. nickel.*	Dull, polished on one side.	12	0·036
Staybrite F.S.L......	18 p.c. chromium, 8 p.c. nickel. Low carbon welding variety.	Descaled..........	18	0·066
Staybrite F.D.P......	18 p.c. chromium, 8 p.c. nickel. Titanium welding quality.	Descaled..........	7·5	0·103
Staybrite F.M.B......	18 p.c. chromium, 8 p.c. nickel. Titanium welding quality.	Descaled..........	15	0·068

*Stated to be 18 per cent chromium, 18 per cent nickel, but Woldman's "Engineering Alloys" gives this as an 18-8 alloy.

The polished surface of Sample F.S.T. showed no signs of corrosion, the stamped identification number being quite sharp. The other samples appeared to be corroded slightly.

Microscopic Examination. All samples were examined under a binocular microscope. It was difficult to estimate the extent of the corrosion on the samples with descaled surfaces as it was not possible to determine whether surface imperfections were caused by the descaling operation or by the action of the hot sulphur waters. It was assumed, however, that the samples were stamped with identification marks after descaling, so an attempt was made to estimate the extent of corrosion by examining these areas. The stamping was not entirely uniform, some sheets being stamped more clearly than others. Making no allowance for this variable, the samples appear to have corroded in the following order: F.M.B., F.S.L., F.D.P., and F.S.T. The first three descaled samples were all perceptibly corroded but there was little difference in the attack on these samples. F.S.T., the polished sample, showed little evidence of corrosion.

CONCLUSIONS

All four steels are of the 18 per cent chromium-8 per cent nickel variety, but three of the specimens are special types that can be welded without danger of weld decay. Weld decay occurs as a result of precipitation of chromium carbide at the weld, the low-chromium areas surrounding these carbides losing their stainless properties. Chromium carbide precipitation is prevented in Samples F.D.P. and F.M.B. by the addition of titanium. In these steels titanium carbide would be precipitated at the weld, with the result that the remaining high-chromium weld material would still have stainless properties. Sample F.S.L. is apparently sufficiently low in carbon to ensure very little precipitation of chromium carbide on welding. Sample F.S.T., according to the information received, is the ordinary 18-8 grade and would be susceptible to weld decay.

Apart from welding, the steels if in the same surface condition should all have about the same resistance to corrosion. This is true of the three samples with descaled surfaces. The better condition of the polished sample was to be expected because the resistance to corrosion of stainless steel is not fully developed unless the surface be polished, as small bits of scale and surface imperfections act as centres of corrosion. The Chemical Foundation recently announced that etching stainless steel in an acid bath containing titanium tetrachloride definitely improves the corrosion resistance of the alloy, as it gives the steel an almost perfect surface.

It is considered that all samples would have been uncorroded if tested in the polished condition. Any of the steels submitted, then, should last indefinitely in Banff hot sulphur waters if used in the polished condition. Material with descaled surfaces should also last almost indefinitely, as there was very little corrosion of the samples in this condition.

If the fittings are to be welded, Steel F.S.T. should not be used and the cheapest of the other three materials should be employed.

If there is no price differential and the material is to be used in the sheet form, the titanium-bearing alloy is probably the best because titanium reduces warping in stainless sheets.

Ore Dressing and Metallurgical Investigation No. 783

GOLD ORE FROM THE CENTRAL CADILLAC MINES, LIMITED,
CADILLAC TOWNSHIP, QUEBEC

Shipment. Four bags of gold ore, comprising four sample lots, were received on September 20, 1939, from the Central Cadillac Mines, Limited, Kewagama, Cadillac Township, Quebec. They were submitted by Mr. G. A. McTeigue, President, 712 Transportation Building, St. James Street West, Montreal, Quebec.

The sample lots were designated as follows:

1. No. 8 vein. Quartz and pyrite, 139 pounds.
2. X-209-N. Arsenopyrite, 118 pounds.
3. X-212-S. Tourmalinized, 110 pounds.
4. No. 11 vein. Pyrite, 154 pounds.

   Total weight of shipment, 521 pounds.

Characteristics of the Ore:

Six polished sections were prepared from each of the four samples for microscopic examination.

No. 8 Vein: Abundant sulphides are disseminated in a gangue consisting of white translucent quartz containing stringers of very fine-textured black silicates and patches of white calcite, which seems to contain only a slight amount of iron.

Pyrite and arsenopyrite are disseminated in irregular grains and poorly formed crystals, and are predominantly coarse, though a small amount of each mineral occurs as very fine grains. Both minerals contain rare, small inclusions of pyrrhotite. Chalcopyrite is very rare, occurring as small grains in both quartz and pyrite.

Three grains of native gold were seen, 90, 20, and 15 microns respectively in size, and all within quartz. No gold is visible in the sulphides.

X-209-N: The gangue of this sample is largely dark silicates, which contain some disseminated carbonate. Arsenopyrite is abundant, pyrite is common; both minerals occur as coarse crystals and masses and contain rare inclusions of pyrrhotite. Chalcopyrite is rarely present in the gangue and is seldom seen in the arsenopyrite and pyrite. No gold is visible.

X-212-S: Except that it contains somewhat less pyrite, this sample is similar to X-209-N. No gold is visible.

No. 11 Vein: The gangue of this sample is grey quartz with dark silicates and a small quantity of carbonate.

Pyrite and arsenopyrite, the former predominating, are disseminated as coarse grains and to some extent form masses of considerable size. Chalcopyrite occurs as small grains in both gangue and pyrite, but the quantity is small. Magnetite is locally abundant as disseminated grains. No gold is visible.

Sampling and Analysis. The four lots were crushed and mixed and a representative feed sample was cut. The assays report as follows:

Gold	0·676 oz./ton
Silver	0·13 "
Arsenic	1·75 per cent
Iron	13·00 "
Copper	0·02 "
Sulphur	5·58 "

Results of the Investigation. Straight cyanidation of the ore showed that a tailing of 0·03 ounce of gold per ton could be obtained with a grind not under 90 per cent −200 mesh.

At a fineness of grind of 80 per cent −200 mesh a barrel-amalgamation test indicated that 55·6 per cent of the gold was free-milling.

By flotation a tailing of 0·05 ounce of gold per ton was obtained, with a ratio of concentration of 6·04 : 1.

Experimental Tests

CYANIDATION

Tests Nos. 1 to 6

A series of straight cyanidation tests on the ore at different degrees of grinding and periods of agitation was carried out.

The ore samples were ground in cyanide and agitated for 24 and 48 hours in solution of 1 pound of sodium cyanide per ton at a pulp dilution of 1·5 parts of solution to 1 part of solids.

Results of Cyanidation Tests:

Test No.	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton ore	
		Feed	Tailing		NaCN	CaO	NaCN	CaO
1	24	0·676	0·045	93·34	1·00	0·16	0·54	4·76
2	48	0·676	0·045	93·34	0·96	0·12	0·92	5·42
3	24	0·676	0·04	94·08	0·92	0·10	0·66	5·05
4	48	0·676	0·03	95·56	0·96	0·10	0·76	5·85
5	24	0·676	0·03	95·56	1·00	0·10	0·86	5·05
6	48	0·676	0·03	95·56	0·96	0·08	1·24	5·88

The grinding is indicated by the following screen tests:

Mesh	Weight, per cent		
	Tests Nos. 1 and 2	Tests Nos. 3 and 4	Tests Nos. 5 and 6
+ 65	0·1	....	....
− 65+100	2·7	0·4	....
−100+150	6·6	3·0	1·1
−150+200	9·5	3·0	1·8
−200	81·1	93·6	97·1
Totals	100·0	100·0	100·0

SETTLING TESTS

For determining the rate of settling of the solids, samples of ore were ground in cyanide and lime to different degrees of fineness and transferred to a glass cylinder, 2 inches inside diameter, and the level of solids was read in feet at five-minute intervals for 1 hour.

The results are tabulated below:

Time, minutes	Increment of Settling, in feet			
	Grind, 81 per cent −200 mesh; pulp dilution, 1·5 : 1	Grind, 81 per cent −200 mesh; pulp dilution, 2 : 1	Grind, 93 per cent −200 mesh; pulp dilution, 1·5 : 1	Grind, 93 per cent −200 mesh; pulp dilution, 2 : 1
0	0	0	0	0
5	0·06	0·12	0·04	0·09
10	0·05	0·11	0·03	0·05
15	0·05	0·11	0·04	0·06
20	0·05	0·11	0·03	0·07
25	0·05	0·11	0·03	0·08
30	0·06	0·11	0·03	0·08
35	0·06	0·12	0·04	0·08
40	0·06	0·12	0·04	0·07
45	0·06	0·11	0·03	0·07
50	0·07	0·11	0·03	0·07
55	0·06	0·10	0·04	0·07
60	0·07	0·09	0·04	0·06

Time, minutes	Increment of Settling, in feet			
	Grind, 81 per cent −200 mesh; pulp dilution, 1·5 : 1	Grind, 81 per cent −200 mesh; pulp dilution, 2 : 1	Grind, 93 per cent −200 mesh; pulp dilution, 1·5 : 1	Grind, 93 per cent −200 mesh; pulp dilution, 2 : 1
Settling, feet per hour	0·70	1·32	0·24	0·85
Overflow	Clear	Clear	Clear	Clear
Titrations: (lb./ton)				
NaCN	0·32	0·24	0·28	0·20
CaO	0·46	0·28	0·42	0·26

AMALGAMATION

Test No. 7

A sample of ore was ground to a fineness of 80·3 per cent −200 mesh and was amalgamated with mercury for one hour in an Abbé grinding jar. The tailing assay was 0·30 ounce of gold per ton, which showed a recovery of 55·6 per cent.

The test indicates that at the grinding shown about 55 per cent of the gold in the ore is free-milling.

FLOTATION

Flotation tests for concentrating the sulphides using different grinds and reagents, were carried out.

Test No. 8

A sample of ore was ground with soda ash, 3 pounds per ton, at a pulp dilution of 0·75 to 1 ($\frac{3}{4}$ part of solution to 1 part of solid), and the pulp was transferred to a laboratory flotation machine. The pulp was conditioned for 5 minutes with reagents and was floated for 12 minutes, using pine oil as a frother.

The grinding is indicated by the following screen test:

Mesh	Weight, per cent
+100	1·5
−100+150	6·2
−150+200	7·7
−200	84·6
	100·0

Results:

Product	Weight, per cent	Assay			Distribution of gold, per cent	Ratio of concentration
		Au, oz./ton	Per cent			
			As	S		
Feed	100·0	0·647			100·0	
Concentrate	11·2	5·06	3·15		87·6	8·9 : 1
Tailing	88·8	0·09		0·83	12·4	

Reagents (lb./ton):

—	Soda ash	Reagent 208	Reagent 301	Pine oil	Time, minutes
Grinding	3·0				20
Conditioning		0·1	0·1		5
Flotation				0·15	12

Test No. 9

This was similar to Test No. 8 except that potassium amyl xanthate was used instead of Reagents 208 and 301.

Results:

Product	Weight, per cent	Assay			Distribution of gold, per cent	Ratio of concentration
		Au, oz./ton	Per cent			
			As	S		
Feed	100·0	0·74			100·0	
Concentrate	11·5	5·74	3·65		89·2	8·7 : 1
Tailing	88·5	0·09		0·70	10·8	

Reagents (lb./ton):

—	Soda ash	Potassium amyl xanthate	Pine oil	Time, minutes
Grinding	3·0			20
Conditioning	1·0	0·2		5
Flotation			0·15	15

The potassium amyl xanthate appears to raise the grade of the concentrate, but does not lower the tailing.

Test No. 10

In this test finer grinding was carried out.

Results:

Product	Weight, per cent	Assay			Distribution of gold, per cent	Ratio of concentration
		Au, oz./ton	Per cent			
			As	S		
Feed	100·0	0·746			100·0	
Concentrate	11·9	5·60	2·65		89·4	8·4 : 1
Tailing	88·1	0·09		0·88	10·6	

Reagents (lb./ton):

—	Soda ash	Reagent 208	Reagent 301	Pine oil	Time, minutes
Grinding	4·0				30
Conditioning		0·1	0·1		5
Flotation				0·186	20

Screen Test:

Mesh	Weight, per cent
+100	0·2
−100+150	3·2
−150+200	4·1
−200	92·5
	100·0

Test No. 11

The grinding was the same as for Test No. 10.

Results:

Product	Weight, per cent	Assay			Distribution of gold, per cent	Ratio of concentration
		Au, oz./ton	Per cent			
			As	S		
Feed	100·0	0·73	....	....	100·0	....
Concentrate	12·7	5·23	3·46	....	91·0	7·9 : 1
Tailing	87·3	0·075	....	0·72	9·0	....

Reagents (lb./ton):

—	Soda ash	Potassium amyl xanthate	Pine oil	Time, minutes
Grinding	4·0	....	....	30
Conditioning	....	0·3	....	5
Flotation	....	....	0·217	20

Test No. 12

The reagents used were butyl xanthate (Z-8), copper sulphate, Tarol, and pine oil. The gold content of the tailing was reduced.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·70	100·0	....
Concentrate	16·55	4·00	94·1	6·04 : 1
Tailing	83·45	0·05	5·9	....

Reagents (lb./ton):

—	Soda ash	Copper sulphate	Butyl xanthate	Tarol	Pine oil	Time, minutes
Grinding	4·0	....	....	....	....	20
Conditioning	....	1·0	0·2	....	....	5
Flotation	....	....	....	0·14	0·186	20

Test No. 13

The ore was ground as in Test No. 12 and was floated, using the same reagents, but in slightly larger amounts. The tailing was run over a corduroy blanket.

The final tailing showed a gold content of 0·05 ounce per ton. Thin flakes of free gold were seen in the blanket concentrate, indicating that jigs or traps are necessary prior to flotation.

Test No. 14

In order to determine the extraction of gold by cyanide from the sulphides alone a concentrate was made by flotation. This was reground to have 98 per cent −325 mesh. Two portions were cyanided for 24 and 48 hours respectively in solution of a strength of 3 pounds of sodium cyanide per ton, at a pulp dilution of 3 to 1.

Results of Cyanidation of Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
24	7·04	0·155	97·8	2·8	0·25	4·20	6·25
48	7·04	0·25	96·4	2·9	0·25	5·70	6·25

The results indicate that the gold shown in the above cyanide tailing is locked in the extremely fine grains of sulphide and is not exposed to the action of cyanide.

This is illustrated more clearly in the next test.

Test No. 15

Two lots of ore were ground in cyanide to a fineness of 81 per cent −200 mesh and were agitated for 24 hours as in Test No. 1.

The tailings were filtered, washed, and were conditioned with soda ash in a flotation cell, and the sulphides were floated using copper sulphate, butyl xanthate, Reagent 301, and a mixture of Tarol and pine oil.

The concentrates were reground to a fineness of 98 per cent −325 mesh and were agitated in cyanide for 24 hours.

Results:

Cyanidation:

Test No.	Assay, Au, oz./ton		Extraction of gold, per cent	Final titration, lb./ton solution		Reagents consumed, lb./ton ore	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
A	0·676	0·045	93·34	0·84	0·12	0·74	4·02
B	0·676	0·045	93·34	0·86	0·12	0·71	4·02

Flotation of Cyanide Tailing:

Test No.	Product	Weight, per cent	Assay, Au, oz./ton	Distribution, per cent	Ratio of concentration
A.	Feed	100·0	0·045	100·0	
	Concentrate	16·4	0·172	62·8	6·1 : 1
	Tailing	83·6	0·02	37·2	
B.	Feed	100·0	0·045	100·0	
	Concentrate	14·0	0·198	61·8	7·1 : 1
	Tailing	86·0	0·02	38·2	

Cyanidation of Combined Flotation Concentrates after Regrinding:

(Combined assay: gold, 0·183 oz./ton)

Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Final titration, lb./ton		Reagents consumed, lb./ton concentrate		Pulp dilution
			NaCN	CaO	NaCN	CaO	
24	0·105	42·6	1·9	0·15	5·30	6·55	3 : 1

Summary:

	Per cent
Gold recovery by primary cyanidation	93·34
Gold recovery by cyanidation of flotation concentrate	1·78
Overall recovery	95·12

The composite tailing is, by calculation, Au, 0·033 oz./ton.

These results indicate that the gold represented by the 0·105-ounce tailing is refractory to the action of cyanide, owing to its association with the sulphides. It is probably submicroscopic in size and locked in minute particles of sulphide.

Comparing the results of Test No. 4, in which by straight cyanidation of the ore at a grind of 94 per cent −200 mesh a tailing of 0·03 ounce of gold per ton resulted, with the composite tailing, gold 0·033 ounce per ton, in the above test, it would appear that these assays represent the minimum tailing obtainable by straight cyanidation.

Test No. 16

For determining the association of the refractory gold in the ore, a sample of ore was ground to a fineness of 81 per cent −200 mesh and was agitated for 24 hours in cyanide solution. The tailing assayed 0·04 ounce of gold per ton.

The tailing product was panned on the Haultain superpanner in order to concentrate the respective sulphides, arsenopyrite and pyrite. The analyses of the panner products disclosed an unsatisfactory separation of the respective sulphides, but sufficient information was obtained to throw some light on the association of the gold in the cyanide tailing.

Three products were made on the panner, and one by flotation:

1. Sulphide concentrate
2. Middling product
3. Flotation concentrate of panner overflow
4. Flotation tailing of panner overflow

The last two were slime products.

The assay results of the respective products are tabulated below:

Product	Weight, per cent	Assay				Distribution, per cent			
		Au, oz./ton	Per cent						
			As	Fe	S	Au	As	Fe	S
Feed (cal.)....	100·0	0·037	1·69	13·27	4·73	100·0	100·0	100·0	100·0
1.............	10·1	0·22	11·65	44·41	24·02	59·7	69·5	33·8	51·3
2.............	4·5	0·03	1·08	12·22	4·92	3·6	2·9	4·1	4·7
3.............	2·4	0·05	4·95	12·62	6·34	3·2	7·0	2·3	3·2
4.............	83·0	0·015	0·42	9·56	2·32	33·5	20·6	59·8	40·8

By rough calculation the percentages of arsenopyrite and pyrite in the respective products are as follows:

Product	Arseno-pyrite	Pyrite	Total sulphides
1..........	25·33	35·76	61·09
2..........	2·35	8·38	10·73
3..........	10·77	7·94	18·71
4..........	0·91	4·02	4·93

If the gold in the cyanide tailing is distributed between the two sulphides, it will follow that the gold content of each product will be proportional to the amount of total sulphides. By calculation the gold contents of Products 2, 3, and 4 are 0·03, 0·05, and 0·015 ounce per ton respectively, which check those obtained by assay. The unattacked gold in the cyanide tailing is, therefore, associated with the sulphides.

As the relative proportions of arsenopyrite and pyrite vary in each product and the gold is only proportional to the combined sulphides, the gold must be distributed in both the arsenopyrite and pyrite.

Six polished sections were prepared from each of the superpanner products, Nos. 1, 2, and 3. The results of the microscopic examination are as follows:

Product No 1:

The minerals present are:

Pyrite: abundant
Arsenopyrite: abundant.
Magnetite: common.
Chalcopyrite: rare.
Gangue: rare.

No gold is visible.

Product No. 2:

The minerals present are:

Gangue: abundant.
Pyrite: common, and finer than in No. 1.
Arsenopyrite: common, and finer than in No. 1.
Magnetite: small amount.
Chalcopyrite: rare.

No gold is visible.

Product No. 3:

The sulphides are very fine-grained. Gangue preponderates and pyrite and arsenopyrite occur in very small quantity. Magnetite and chalcopyrite are rare. No gold is visible.

CONCLUSIONS

By straight cyanidation of the ore a minimum tailing of 0·03 ounce of gold per ton is obtainable.

This residual gold is closely associated in the sulphides, arsenopyrite and pyrite; its size being probably submicroscopic, for none is visible in the sections of the panner products examined under the microscope.

The use of jigs or traps in the grinding-classifying circuit is recommended for the recovery of coarse free gold.

Fine grinding, at least 93 per cent −200 mesh, is necessary to obtain the minimum tailing by cyanidation.

Ore Dressing and Metallurgical Investigation No. 784

COPPER-GOLD ORE FROM THE OBALSKI MINING CORPORATION, CHIBOUGAMAU AREA, QUEBEC

Shipment. A shipment comprising three lots of copper-gold ore was received on October 16, 1939, from the Obalski Mining Corporation, Chibougamau area, Quebec.

The lots were designated as follows:

Vein "A"	97 pounds
Vein "C"	57 "
Vein "D"	99 "

The shipment was made by J. R. Giroux, President and Managing Director, the Obalski Mining Corporation, Canada Cement Building, Montreal, Quebec.

Location of the 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.

Characteristics of the Ore. The present samples were taken from the same veins as those of three previous shipments received in 1938 and reported on in Investigation No. 759, Bureau of Mines Report No. 797.

Sampling and Assaying. Each lot was crushed and sampled separately. The assay samples were all ground to pass a 200-mesh screen in order to ensure accurate assaying of the gold content. The gold assays reported below are the average of five assays. These checked satisfactorily.

—	Vein "A"	Vein "C"	Vein "D"
Gold, oz./ton	0·346	0·106	0·61*
Silver, oz./ton	0·42	0·24	0·47
Copper, per cent	2·45	0·35	0·53
Iron, per cent	14·42	16·30	13·92
Zinc, per cent	0·04	0·04	0·05
Arsenic, per cent	Nil	Nil	Nil
Sulphur, per cent	9·20	10·72	7·16

*Average of four assays.

For the purpose of the investigation a composite feed was made up of the following proportions:

40 per cent Vein "A" sample
15 " " "C" "
45 " " "D" "

This had the following analysis:

Gold	0·436 oz./ton.	(Average of four assays)
Silver	0·40 "	
Copper	1·30 per cent	

Purpose of the Investigation. The previous investigation covered flotation and cyanidation of the ore. The present investigation is designed to recover the free gold in jigs with subsequent amalgamation and to determine the most economic grade of copper concentrate that can be made by flotation for shipment to a smelter. Different ratios of concentration were obtained, in some cases by varying the time of floating the primary concentrate and in others by varying the reagent combination.

Experimental Tests

The procedure of the tests consisted in feeding the ground ore pulp to a Denver mineral jig. The jig concentrate was amalgamated with mercury and the amalgam assayed. The jig overflow, combined with the amalgamation tailing, was conditioned in a Denver Sub-A laboratory flotation machine and a copper concentrate was taken off at various ratios of concentration. A feed sample assay was calculated for each test from the gold recovered in the amalgam and the assays of the flotation products.

Test No. 1

Flotation:

Reagents Added:

Lime	10·0 lb./ton
Potassium ethyl xanthate	0·10 "
Pine oil	0·062 "
Conditioning	5 minutes
Flotation	7 "

Rougher concentrate cleaned in small cell.

Results:

Product	Weight, per cent	Assay		Distribution		Ratio of concentration
		Au, oz./ton	Cu, per cent	Gold, per cent	Copper, per cent	
Feed to flotation (cal.)	100·0	0·262	1·27	100·0	100·0	....
Cleaner concentrate	5·1	3·66	21·82	71·2	87·9	19·6 : 1
Middling	1·5	1·30	3·84	7·4	3·9	....
Tailing	93·4	0·06	0·11	21·4	8·2	....

Gold recovered in amalgam	0·131 oz./ton feed
Gold in flotation feed	0·262 "
Feed sample assay (cal.)	0·393 "

Summary of Total Gold Distribution:

	Per cent
Recovered by amalgamation	33·33
Recovered in cleaner concentrate	47·47
Gold in middling product	4·93
Tailing loss	14·27
	100·00

Screen Test of Flotation Tailing:

Mesh	Weight, per cent
+100	0·2
−100+150	2·6
−150+200	4·1
−200	93·1
	100·0

Test No. 2

The grind was the same as in Test No. 1. Two concentrates were taken off during flotation, the first for 2 minutes and the second for 6 minutes of flotation. The reagents were the same as in Test No. 1.

Flotation Results:

Product	Weight, per cent	Assay		Distribution		Ratio of concentration
		Au, oz./ton	Cu, per cent	Gold, per cent	Copper, per cent	
Feed to flotation (cal.)	100·0	0·267	1·28	100·00	100·00	....
1st concentrate	4·3	2·82	16·08	45·34	53·96	23·26 : 1
2nd concentrate	6·0	1·54	8·04	34·54	37·64	....
Tailing	89·7	0·06	0·12	20·12	8·40	....

Gold recovered in amalgam	0·136 oz./ton feed
Gold in flotation feed	0·267 "
Feed sample assay (cal.)	0·403 "

Summary of Total Gold Distribution:

	Per cent
Recovered by amalgamation	33·75
Recovered in 1st concentrate	30·04
Recovered in 2nd concentrate	22·88
Tailing loss	13·33
	100·00

Test No. 3

The flotation periods were four minutes for the first concentrate and four minutes for the second. The reagents and grind were the same as in the previous tests.

Flotation Results:

Product	Weight, per cent	Assay		Distribution		Ratio of concentration
		Au, oz./ton	Cu, per cent	Gold, per cent	Copper, per cent	
Feed to flotation (cal.)	100·0	0·31	1·26	100·00	100·00	....
1st concentrate	6·3	3·10	17·32	52·13	86·51	15·87 : 1
2nd concentrate	3·8	1·24	2·82	14·99	8·50	....
Flotation tailing	89·9	0·08	0·07	22·88	4·99	....

Gold recovered in amalgam	0·131	oz./ton feed
Gold in flotation feed	0·310	"
Feed sample assay (cal.)	0·441	"

Summary of Total Gold Distribution:

	Per cent
Recovered by amalgamation	29·71
Recovered in 1st concentrate	43·67
Recovered in 2nd concentrate	10·54
Tailing loss	16·08
	100·00

The grinding in Tests Nos. 1, 2, and 3 is considered to be unnecessarily fine, and in the following tests the ore was given a coarser grind. This resulted in a higher gold recovery by amalgamation and a lower final tailing in flotation.

Tests Nos. 4, 5, 6, and 7

Tests Nos. 4, 5, 6, and 7 were carried out in a manner similar to those already recorded.

The ore was ground to a fineness of approximately 82 per cent −200 mesh and was passed over a mineral jig. The concentrate was amalgamated and the tailing added to the jig overflow. The combined products were conditioned with lime, 10 pounds per ton, and potassium ethyl xanthate, 0·10 pound per ton, for 5 minutes. Two concentrates were taken off during flotation times of 2 and 6 minutes, 4 and 4 minutes, and 6 and 2 minutes, respectively. In the final test a single rougher concentrate was made with 8 minutes' flotation.

Test No. 4:

Flotation Results:

Product	Weight, per cent	Assay		Distribution		Ratio of concentration
		Au, oz./ton	Cu, per cent	Gold, per cent	Copper, per cent	
Feed to flotation (cal.)	100·0	0·247	1·27	100·00	100·00	....
1st concentrate	4·8	2·95	17·72	57·28	66·88	20·8 : 1
2nd concentrate	5·0	1·30	6·44	26·30	25·32	....
Tailing	90·2	0·045	0·11	16·42	7·80	....

Gold recovered in amalgam	0·141	oz./ton feed
Gold in flotation feed	0·247	"
Feed sample assay (cal.)	0·388	"

Summary of Total Gold Distribution:

	Per cent
Recovered by amalgamation	36·34
Recovered in 1st concentrate	36·46
Recovered in 2nd concentrate	16·74
Tailing loss	10·46
	100·00

Test No. 5:

Flotation Results:

Product	Weight, per cent	Assay		Distribution		Ratio of concentration
		Au, oz./ton	Cu, per cent	Gold, per cent	Copper, per cent	
Feed to flotation (cal.)	100·0	0·251	1·30	100·00	100·00	....
1st concentrate	7·6	2·36	13·54	71·40	79·43	13·1 : 1
2nd concentrate	2·7	1·00	4·22	10·75	8·80	....
Tailing	89·7	0·05	0·17	17·85	11·77	....

Gold recovered in amalgam	0·170 oz./ton feed
Gold in flotation feed	0·251 "
Feed sample assay (cal.)	0·421 "

Summary of Total Gold Distribution:

	Per cent
Recovered by amalgamation	40·38
Recovered in 1st concentrate	42·57
Recovered in 2nd concentrate	6·41
Tailing loss	10·64
	100·00

Test No. 6:

Flotation Results:

Product	Weight, per cent	Assay		Distribution		Ratio of concentration
		Au, oz./ton	Cu, per cent	Gold, per cent	Copper, per cent	
Feed to flotation (cal.)	100·0	0·253	1·27	100·00	100·00	....
1st concentrate	6·7	2·66	15·04	70·50	79·46	14·9 : 1
2nd concentrate	2·6	0·95	3·04	9·77	6·23	....
Tailing	90·7	0·055	0·20	19·73	14·31	....

Gold recovered in amalgam	0·177 oz./ton feed
Gold in flotation feed	0·253 "
Feed sample assay (cal.)	0·430 "

Summary of Total Gold Distribution:

	Per cent
Recovered by amalgamation	41·16
Recovered in 1st concentrate	41·48
Recovered in 2nd concentrate	5·75
Tailing loss	11·61
	100·00

Test No. 7:

A rougher concentrate was made, instead of the two concentrates as in the previous tests.

Flotation Results:

Product	Weight, per cent	Assay		Distribution		Ratio of concentration
		Au, oz./ton	Cu, per cent	Gold, per cent	Copper, per cent	
Feed to flotation (cal.)..........	100·0	0·242	1·23	100·00	100·00	....
Rougher concentrate............	10·4	1·94	11·04	83·34	93·44	9·62 : 1
Tailing.........................	89·6	0·045	0·09	16·66	6·56	....

Gold recovered in amalgam................................	0·195	oz./ton feed
Gold in flotation feed.......................................	0·242	"
Feed sample assay (cal.)....................................	0·437	"

Summary of Total Gold Distribution:

	Per cent
Recovered by amalgamation..	44·62
Recovered in rougher concentrate....................................	46·15
Tailing loss..	9·23
	100·00

Test No. 8

The reagent combination was changed from that of the preceding series. The ore was ground 82 per cent through 200 mesh and was jigged. The jig concentrate was reground and amalgamated with mercury in a mortar. The gold in the amalgam was determined and the amalgamation tailing was reunited with the jig tailing for flotation.

The combined tailing was conditioned for 5 minutes with 10 pounds of lime per ton and 0·10 pound of potassium ethyl xanthate per ton. A concentrate was floated using cresylic acid, 0·32 pound per ton, as frother. The concentrate was not cleaned.

Summary of Results:

Product	Weight, per cent	Assay		Distribution		Ratio of concentration
		Au, oz./ton	Cu, per cent	Gold, per cent	Copper, per cent	
Bulk concentrate...............	4·0	3·70	20·38	60·66	63·44	25 : 1
Flotation tailing................	96·0	0·10	0·49	39·44	36·56	....
Feed to flotation (cal.)..........	100·0	0·244	1·29	100·00	100·00	

Gold recovered in amalgam	0·164	oz./ton feed
Gold in feed to flotation	0·244	"
Feed sample assay (cal.)	0·408	"

Summary of Total Gold Distribution:

	Per cent
Recovered by amalgamation	40·20
Recovered in bulk concentrate	36·27
Tailing loss	23·53
	100·00

Test No. 9

The quantity of lime was reduced to 5·0 pounds per ton of ore and the grinding to 70 per cent through 200 mesh. The ore was fed up to a jig and the concentrate was amalgamated. The gold in the amalgam was determined and the combined tailing was floated with the following reagents:

Lime	5·0	lb./ton
Potassium ethyl xanthate	0·10	"
Cresylic acid	0·32	"

The rougher concentrate was cleaned in another cell.

Results:

Product	Weight, per cent	Assay		Distribution		Ratio of concentration
		Au, oz./ton	Cu, per cent	Gold, per cent	Copper, per cent	
Flotation concentrate	11·48	2·06	10·94	81·18	96·20	8·7 : 1
Cleaner tailing	3·22	0·51	1·01	5·64	2·49	....
Flotation tailing	85·30	0·045	0·02	13·18	1·31	....
Feed to flotation (cal.)	100·00	0·291	1·31	100·00	100·00	....

Gold recovered in amalgam	0·149	oz./ton feed
Gold in feed to flotation	0·291	"
Feed sample assay (cal.)	0·440	"

Summary of Total Gold Distribution:

	Per cent
Recovered by amalgamation	33·86
Recovered in flotation concentrate	53·69
Recovered in cleaner tailing	3·73
Tailing loss	8·72
	100·00

Test No. 10

The quantity of lime was still further reduced to four pounds per ton of ore and the grinding was maintained at 70 per cent through 200 mesh.

The ore was concentrated in a jig as before and the concentrate was amalgamated. The jig overflow and the amalgamation tailing were floated with the following reagents:

Lime	4·0 lb./ton
Potassium ethyl xanthate	0·05 "
Cresylic acid	0·16 "

The rougher concentrate was cleaned in another cell with a small amount of cresylic acid.

Results:

Product	Weight, per cent	Assay		Distribution		Ratio of concentration
		Au, oz./ton	Cu, per cent	Gold, per cent	Copper, per cent	
Flotation concentrate	12·25	1·82	10·08	84·52	93·95	8·16 : 1
Cleaner tailing	2·74	0·56	1·04	5·82	2·17	....
Flotation tailing	85·01	0·03	0·06	9·66	3·88	....
Feed to flotation (cal.)	100·00	0·264	1·31	100·00	100·00	....

Gold recovered in amalgam	0·200 oz./ton feed
Gold in feed to flotation	0·264 "
Feed sample assay (cal.)	0·464 "

Summary of Total Gold Distribution:

	Per cent
Recovered by amalgamation	43·10
Recovered in flotation concentrate	48·09
Recovered in cleaner tailing	3·31
Tailing loss	5·50
	100·00

Test No. 11

The grind was maintained at 70 per cent through 200 mesh and the total lime used was 12·0 pounds per ton. The ore was concentrated in a jig as before and the concentrate was amalgamated. The combined tailing was floated with the following reagents:

Lime	12·00 lb./ton
Potassium ethyl xanthate	0·05 "
Cresylic acid	0·32 "

The rougher concentrate was cleaned with a small amount of cresylic acid.

Results:

Product	Weight, per cent	Assay		Distribution		Ratio of concentration
		Au, oz./ton	Cu, per cent	Gold, per cent	Copper, per cent	
Flotation concentrate	4·41	3·72	26·44	67·89	90·34	22·68 : 1
Cleaner tailing	2·43	0·51	1·30	5·13	2·45	....
Flotation tailing	93·16	0·07	0·10	26·98	7·21	....
Feed to flotation (cal.)	100·00	0·242	1·29	100·00	100·00	....

Gold recovered in amalgam	0·195 oz./ton feed	
Gold in feed to flotation	0·242	"
Feed sample assay (cal.)	0·437	"

Summary of Total Gold Distribution:

	Per cent
Recovered by amalgamation	44·62
Recovered in flotation concentrate	37·60
Recovered in cleaner tailing	2·84
Tailing loss	14·94
	100·00

In the following table, Tests Nos. 4 to 11 are arranged in the descending order of their ratios of concentration, giving comparative recoveries in concentrate and by amalgamation.

Test No.	Ratio of concentration	Assay of concentrate		Recovery in concentrate, per cent total		Recovered by amalgamation, per cent total gold	Per cent total gold recovered*
		Au, oz./ton	Cu, per cent	Au	Cu		
8	25 : 1	3·70	20·38	36·27	63·44	40·20	76·47**
11	22·68 : 1	3·72	26·44	37·60	90·34	44·62	82·22
4	20·8 : 1	2·95	17·72	36·46	66·88	36·34	72·80
6	14·9 : 1	2·66	15·04	41·48	79·43	41·16	82·64
5	13·1 : 1	2·36	13·54	42·57	79·50	40·38	82·95
7	9·6 : 1	1·94	11·04	46·15	93·44	44·62	90·77**
9	8·7 : 1	2·06	10·94	53·69	96·20	33·86	87·55
10	8·16 : 1	1·82	10·08	48·09	93·95	43·10	91·19

*Gold recoveries reported in the last column include gold recovered by amalgamation and gold recovered in primary or cleaned concentrate. Copper recoveries are those in primary or cleaned concentrates only.

**There are no middling products in Tests Nos. 7 and 8.

Analyses of the concentrates from Tests Nos. 4 to 9 are as follows:

Concentrate	Au, oz./ton	Cu, per cent	SiO_2, per cent	Fe_2O_3 per cent	Al_2O_3, per cent	CaO, per cent	MgO, per cent	S, per cent
8	3·70	20·38	9·69	43·76	1·57	0·32	1·18	32·81
4	2·95	17·72	6·79	48·40	2·47	0·05	1·06	37·50
6	2·66	15·04	7·98	47·07	1·39	0·24	1·10	34·91
5	2·36	13·54	9·70	50·80	1·82	0·06	1·27	36·60
7	1·94	11·04	6·60	51·53	1·17	0·14	0·95	38·17
9	2·06	10·94	2·63	57·25	0·52	0·06	0·37	43·83

CONCLUSIONS

It will be noted that, generally, the grade of the concentrates varies directly with the ratio of concentration; recovery is necessarily sacrificed to some degree in order to obtain the higher ratios. The most economical cut-off point will have to be decided on the basis of shipping charges, smelter schedules, and recoveries.

Ore Dressing and Metallurgical Investigation No. 785

TUNGSTEN ORE FROM THE LOWER LEVELS OF THE McKENZIE RED LAKE GOLD MINE, McKENZIE ISLAND, ONTARIO

Shipment. Four samples of ore, total weight 100 pounds, were received on November 6, 1939. The samples were submitted by J. L. Ramsell, Resident Manager, McKenzie Red Lake Gold Mines, Limited, McKenzie Island, Ontario.

Characteristics of the Ore. No microscopic examination was made as all was finely crushed when received. A shipment of gold ore from this property was examined and described in Investigation No. 578, published in 1934. The present shipment differs from the former in that it carries an appreciable quantity of scheelite.

Sampling and Assaying. The four samples received were assayed individually for tungsten and reported as follows:

Sample	Tungstic oxide, per cent
1	6·62
2	1·10
3	3·21
4	0·21

All four samples were then mixed and a composite feed sample was assayed for gold and tungsten. It was reported as follows:

Gold	0·03 oz./ton
Tungstic oxide	2·48 per cent

Experimental Tests

Samples of the ore were concentrated by gravity methods to determine the recovery and grade of concentrate. A recovery of 84 per cent of the tungsten was made on a small table in a product averaging about 50 per cent of tungstic oxide. The concentrate carried 5 per cent of sulphur in the form of pyrite, which could not be separated on the small tables without considerable sacrifice of recovery. On a large table it is possible the pyrite could be taken out and, as it carries gold, could be sent to the cyanide plant for further treatment.

The tests are described in detail as follows:

TABLE CONCENTRATION

Test No. 1

A sample of the ore was dry-crushed through 20 mesh and screen-sized as follows:

Mesh	Weight, per cent	Calculated assay, WO_3, per cent	Distribution of WO_3, per cent total
− 20+ 35	30·82	2·38	29·58
− 35+ 48	15·01	2·37	14·35
− 48+ 65	13·97	2·33	13·13
− 65+100	12·00	2·67	12·92
−100	28·20	2·64	30·02
Feed sample	100·00	2·48	100·00

The screened fractions were concentrated on a small table and the products were assayed for tungstic oxide (WO_3).

Summary of Results:

Mesh size of fraction	Table product	Weight, per cent	Assay, WO_3, per cent	Distribution of WO_3	
				Per cent content	Per cent total
− 20 + 35	Concentrate	4 96	47·67	99·20	29·34
	Tailing	95·04	0·02	0·80	0·24
	Fraction (cal.)	100·00	2·38	100·00	29·58
− 35 + 48	Concentrate	4·19	54·77	96·91	13·91
	Middling	7·92	0·59	1·97	0·28
	Tailing	87·89	0·03	1·12	0·16
	Fraction (cal.)	100·00	2·37	100·00	14·35
− 48 + 65	Concentrate	4·84	46·94	97·32	12·78
	Middling	4·32	0·19	0·35	0·05
	Tailing	90·84	0·06	2·33	0·30
	Fraction (cal.)	100·00	2·33	100·00	13·13
− 65 +100	Concentrate	4·31	55·16	88·90	11·48
	Middling	4·09	2·11	3·23	0·42
	Tailing	91·60	0·23	7·87	1·02
	Fraction (cal.)	100·00	2·67	100·00	12·92
−100	Concentrate	3·15	46·89	56·03	16·81
	Middling	11·97	2·45	11·13	3·34
	Tailing	84·88	1·02	32·84	9·87
	Fraction (cal.)	100·00	2·64	100·00	30·02

Recovered in concentrates........................... 84·32 per cent total WO_3.
Recovered in middlings............................. 4·09 " "
Tailing loss.. 11·59 " "

The table concentrates produced on the small table were low in grade owing to the presence of 10 to 11 per cent of pyrite, with which much gangue was associated. It was not possible to separate the pyrite from the scheelite without too great a sacrifice of recovery, but this might be done on large, commercial-size tables, or, alternately, the pyrite might be floated away from the table concentrates.

The shipment received was too small for a test run on a large table.

TABLE CONCENTRATION

Test No. 2

The ore was crushed dry to pass through a 10-mesh screen and was sized on 20-, 35-, 48-, 65-, and 100-mesh screens.

The object of the test was to see if improvement could be made in the grade of concentrate by hand-panning the final table-product to remove surplus gangue and pyrite. The coarsest fraction was treated in a jig and the remaining five were tabled. The jig produced a concentrate assaying 69 per cent of tungstic oxide but it contained 4·32 per cent of sulphur in the form of pyrite. Hand-panning made no improvement.

The table concentrates after hand-panning averaged slightly more than 60 per cent of tungstic oxide but they still carried about 5 per cent of sulphur in the form of pyrite, rendering them unsalable.

A pyrite concentrate floated from a sample of the ore assayed 0·86 ounce per ton in gold and contained about 60 per cent of the gold in the feed sample. Such a product could be sent to the cyanide plant for further treatment.

An attempt to float the scheelite from the pyrite tailing was unsuccessful, owing to presence of carbonates and other gangue material which floated readily, producing a very low-grade product, nor did the scheelite float efficiently, and consequently recovery was low.

CONCLUSIONS

The tests indicate that the scheelite would be free and amenable to gravity concentration if the ore be crushed through 10 mesh. Care should be taken in crushing to produce as little fine as possible, because scheelite slimes readily and is likely to be lost in any form of gravity concentration.

The crushing could be done with jaw crusher and rolls, the finished product being screened out after each pass to prevent the formation of excessive fine. In the summary of results it will be observed that the tailing assay begins to rise in the −65 +100-mesh product and goes up sharply in the −100-mesh product.

In preparing the ore for concentration hydraulic classification is recommended as preferable to screening, as it gives a more desirable product for the table separation. At least three spigot products should be produced and the slime overflow should be thickened in a thickener or cone before being fed to the slime tables.

In regard to the removal of pyrite from the table concentrate, this can be done in a number of ways and should present no real difficulty. Flotation of the table concentrate would undoubtedly be the first step in this operation and if the sulphur were not lowered to the 0·5 per cent required by the usual specifications, a slight roasting would suffice to do this.

For ready sale in Canada, the following specifications are required:

Tungstic oxide	70·00 per cent minimum
Sulphur	0·50 per cent maximum
Phosphorus	0·05 per cent maximum
Manganese	1·00 per cent maximum
Copper	0·05 per cent maximum
Arsenic, antimony, tin	0·10 per cent maximum each and total of 0·25 per cent maximum
Molybdenum, lead, bismuth	0·10 per cent maximum each

Ore Dressing and Metallurgical Investigation No. 786

GOLD ORE FROM THE THOMPSON-LUNDMARK MINE, YELLOWKNIFE, NORTHWEST TERRITORIES

Shipment. One bag of gold ore, weighing 62 pounds, was received December 1, 1939, from E. V. Neelands, Consulting Engineer for the Thompson-Lundmark Gold Mines, Limited, 2810, 25 King Street West, Toronto, Ontario. The shipment was said to come from the "Fraser Vein" on the property.

On November 8, 1938, a shipment had been received from the "Kim Vein" and is covered by Investigation No. 761 (1938).

Location of the Property. The property of the Thompson-Lundmark Mines, Limited, from which the present shipment was received, is situated on Thompson Lake, Yellowknife Mining Division, Northwest Territories.

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

Gold	0·80 oz./ton
Silver	0·21 "
Iron	2·80 per cent
Sulphur	0·41 "
Pyrrhotite	0·14 "
Copper	0·02 "
Arsenic	Trace

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

The *gangue* consists largely of fine-textured, impure grey quartz with some glassy quartz. Some finely disseminated carbonate is present. In some places indistinct schistosity indicates that the rock is probably a highly silicified schist.

The *metallic minerals* are very sparsely scattered through the siliceous gangue. In their order of abundance they are: pyrite, arsenopyrite, pyrrhotite, galena, sphalerite, chalcopyrite, and native gold. Of these, only pyrite is present in quantities that might be considered larger than traces. Only a single small grain of gold (about 400 mesh in size) is visible in the polished sections, but coarse gold was readily discernible in the jig concentrate.

Experimental Tests

The test work undertaken was as follows:

As the extraction in the proposed mill might at the start be by amalgamation only, it was required to ascertain what extraction could be expected with a jig between the ball mill and the classifier, with the classifer overflow passing over amalgamating plates followed by corduroys (with provision to place the corduroys ahead of the plates if this gave improved results) and with the tailing going to a shaking table so that a concentrate could be recovered and stored for future treatment. It was desired also to ascertain the extraction from the amalgam residue and table concentrate by cyanidation.

A number of tests were also made on the ore by straight cyanidation.

By following the suggested flow-sheet over 89 per cent of the gold was recovered bÿ amalgamation. Of the remaining 11 per cent, about 6 per cent was recovered in the amalgam residue and table concentrate, and over 90 per cent of the gold in these products was extracted by cyanidation.

PLATE AMALGAMATION AND TABLE CONCENTRATION

Test No. 1

The ore at -14 mesh was ground in a ball mill to pass 61·2 per cent -200 mesh. The pulp was passed over an amalgamation plate and the plate tailing was concentrated on a Wilfley table. The resulting table concentrate was agitated in cyanide solution of 3 pounds per ton for 40 hours. This concentrate was not reground prior to agitation.

A screen test showed the grinding on the plate feed as follows:

Mesh	Weight, per cent
− 48 + 65	1·6
− 65 +100	8·3
−100 +150	13·2
−150 +200	15·7
−200	61·2
	100·0

Results:

Plate Amalgamation:

Assay, Au, oz./ton		Recovery of gold, per cent
Feed	Tailing	
0·80	0·12	85·0

Table Concentration of Plate Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·12	100·0	....
Table concentrate	3·83	1·67	53·3	26·1 : 1
Table middling	4·67	0·22	8·6	21·4 : 1
Table tailing	91·50	0·05	38·1	....

The table tailing assayed 0·18 per cent of sulphur.

Cyanidation of Table Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
40	1·67	0·12	92·8	2·9	1·1	5·9	11·2

Summary of Test No. 1:

	Per cent
Gold recovered by plate amalgamation	85·0
" " table concentration	9·3
" extracted from table concentrate	8·6
Overall recovery by amalgamation and cyanidation	93·6

CONCENTRATION, AMALGAMATION, AND CYANIDATION

Test No. 2

The ore at −14 mesh was ground in a ball mill to pass 79·6 per cent −200 mesh. The pulp was passed through a Denver jig and the jig tailing was passed over a corduroy blanket. The blanket tailing was run over an amalgamation plate and the plate tailing was concentrated on a Wilfley table. The combined jig and blanket concentrates were reground and amalgamated with mercury in a jar mill; the resulting amalgam residue was combined with the table concentrate and agitated in cyanide solution of 3 pounds of sodium cyanide per ton for 40 hours.

A screen test showed the grinding of the jig feed as follows:

Mesh	Weight, per cent
− 65+100	1·0
−100+150	6·2
−150+200	13·2
−200	79·6
	100·0

Results:

Jig and Blanket Concentration:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·80	100·0	....
Jig and blanket concentrate	1·55	43·35	84·0	64·5 : 1
Blanket tailing	98·45	0·13	16·0	....

The jig concentrate was 1·1 per cent by weight of the feed and the blanket concentrate was 0·45 per cent.

Plate Amalgamation of Blanket Tailing:

Assay, Au, oz./ton		Recovery of gold, per cent
Feed	Tailing	
0·13	0·075	42·3

Table Concentrate of Table Tailing:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed	100·00	0·075	100·0	....
Table concentrate	2·78	0·71	26·4	36 : 1
Table middling	2·07	0·37	10·2	48·3 : 1
Table tailing	95·15	0·05	63·4	....

The table tailing assayed 0·20 per cent of sulphur.

Barrel Amalgamation of Jig and Blanket Concentrates:

Assay, Au, oz./ton		Recovery of gold, per cent
Feed	Tailing	
43·35	0·585	98·6

Cyanidation of Amalgam Residue and Table Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
40	0·66	0·12	81·8	2·9	0·65	6·3	13·0

Summary of Test No. 2:

	Per cent
Gold recovered as jig and blanket concentrates	84·0
Gold recovered as table concentrate	3·4
" " by plate amalgamation	6·8
" " " barrel amalgamation	82·8
" " " cyanidation of table concentrate and amalgam residue	3·7
Overall recovery, by amalgamation and cyanidation	93·3

Test No. 3

This was similar to Test No. 2, except that the corduroy and amalgam plate were interchanged, the plate receiving the jig overflow and the blanket feed being the plate tailing.

The jig concentrate was 0·55 per cent by weight of the feed. The amalgam plate tailing assayed 0·075 ounce of gold per ton.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Blanket Concentration of Plate Tailing:				
Feed	100·00	0·075	100·0	....
Blanket concentrate	0·68	1·53	13·9	147 : 1
Blanket tailing	99·32	0·065	86·1	....
Wilfley Table Concentration of Blanket Tailing:				
Feed	100·00	0·065	100·0	....
Table concentrate	2·00	0·60	18·5	50 : 1
Table middling	3·10	0·18	8·6	32·2 : 1
Table tailing	94·90	0·05	72·9	....

The table tailing assayed 0·19 per cent of sulphur.

The jig concentrate and blanket concentrate were reground and amalgamated, the amalgam residue assaying 1·57 ounces of gold per ton.

Cyanidation of Amalgam Residue and Table Concentrate:

Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton concentrate	
	Feed	Tailing		NaCN	CaO	NaCN	CaO
40	1·01	0·12	88·1	2·9	1·1	5·9	11·2

Summary of Test No. 3:

	Per cent
Gold recovered by barrel and plate amalgamation	89·5
Gold recovered by table concentration	2·2
Gold recovered by cyanidation of amalgam residue and table concentrate	3·5
Overall recovery, by amalgamation and cyanidation	93·0

STRAIGHT CYANIDATION

Test No. 4 (A, B, C, and D)

Portions of the ore at −14 mesh were ground in a ball mill in cyanide solution of 1 pound of sodium cyanide per ton to pass 56·7 per cent −200 mesh in Tests Nos. 4A and 4B, and 77·8 per cent −200 mesh in Tests Nos. 4C and 4D. The pulps were agitated for 24- and 48-hour periods.

Results of Cyanidation:

Feed: gold, 0·80 oz./ton

Test	Agitation, hours	Grind, per cent −200 mesh	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore	
					NaCN	CaO	NaCN	CaO
4A.........	24	56·7	0·015	98·1	1·00	0·34	0·40	2·8
4B.........	48	56·7	0·01	98·7	0·96	0·32	0·48	2·9
4C.........	24	77·8	0·01	98·7	1·00	0·22	0·40	3·1
4D.........	48	77·8	0·005	99·4	0·85	0·22	0·70	3·1

SUMMARY AND CONCLUSIONS

The test work showed that 89·5 per cent of the gold could be extracted by concentration and amalgamation.

Of the other 10·5 per cent, about 1·2 per cent remained in the amalgam residue and 3·8 per cent was recovered in a table concentrate assaying 0·70 ounce of gold per ton. When this concentrate was combined with the amalgam residue and agitated in cyanide solution, a further 3·5 per cent of the gold was extracted. The remaining 6 per cent was lost in the table tailing, which assayed 0·05 ounce of gold per ton and 0·2 per cent of sulphur.

These results were obtained at a grind of 79 per cent −200 mesh.

Using a coarser grind, 61 per cent −200 mesh, 85 per cent of the gold was caught on an amalgamation plate and some 9 per cent was concentrated on the Wilfley table, the table tailing showing a 6 per cent loss of gold and assaying 0·05 ounce of gold per ton and 0·18 per cent of sulphur.

Straight cyanidation of the ore gave a cyanide residue of 0·01 ounce of gold per ton in 48 hours at a grind of 56·7 per cent −200 mesh.

The ore presents no serious metallurgical problem and is easily amenable to either amalgamation or cyanidation.

From the work on the present shipment from the "Fraser Vein", there does not appear to be any apparent difference in the advisable metallurgical treatment from that for the former shipment from the "Kim Vein" on the same property.

Ore Dressing and Metallurgical Investigation No. 787

GOLD ORE FROM THE LEITCH GOLD MINES, LIMITED, STURGEON RIVER AREA, NORTHERN ONTARIO

Shipment. Four bags of gold ore, total weight 155 pounds, were received on October 14, 1939, from W. H. Segsworth, Consulting Engineer, Leitch Gold Mines, Limited, 67 Yonge Street, Toronto, Ontario.

Previous shipments had been received on December 24, 1935, and July 22, 1936, and are covered by Investigations Nos. 667 and 693.

Location of the Property. The property of the Leitch Gold Mines, Limited, from which the present shipment was received is situated in the Sturgeon River area, Thunder Bay District, about 5 miles from Beardmore, Ontario, on the Canadian National Railway.

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

Gold	0·79 oz./ton
Silver	0·06 "
Iron	2·86 per cent
Sulphur	0·80 "
Arsenic	0·21 "
Copper	0·01 "

The shipment, which came from the ball mill feeder belt, also contained about 2 pounds of added lime per ton.

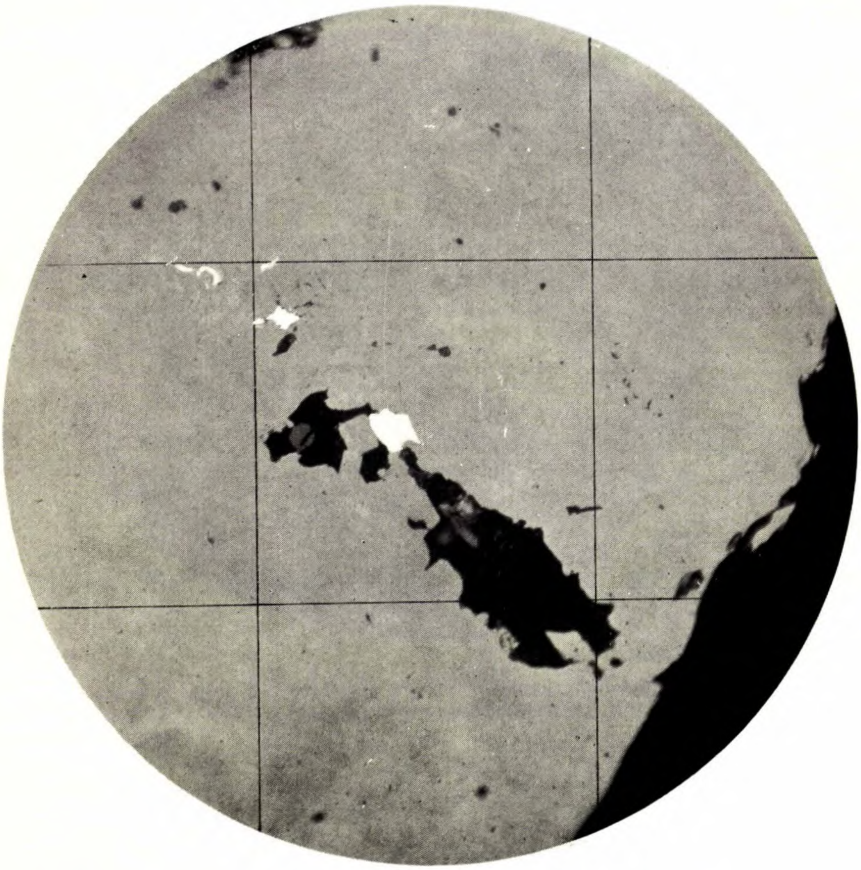
Characteristics of the Ore. Six polished sections were prepared and examined under the reflecting microscope.

The *gangue* is an assemblage of dark greenish-grey, somewhat fissile rock, fine-textured white quartz, and a small amount of fine disseminated carbonate which appears to be dolomitic.

Metallic minerals are not abundant and occur almost entirely in the rock material. Pyrite and arsenopyrite preponderate, largely as medium to extremely fine subhedral crystals and irregular grains disseminated through rock. Practically negligible quantities of chalcopyrite and of two unknown grey minerals are visible as rare, tiny grains in pyrite and in gangue. Etch reactions failed to identify either of the unknown minerals, and the particles are too small to obtain material for microchemical or spectrographic tests.

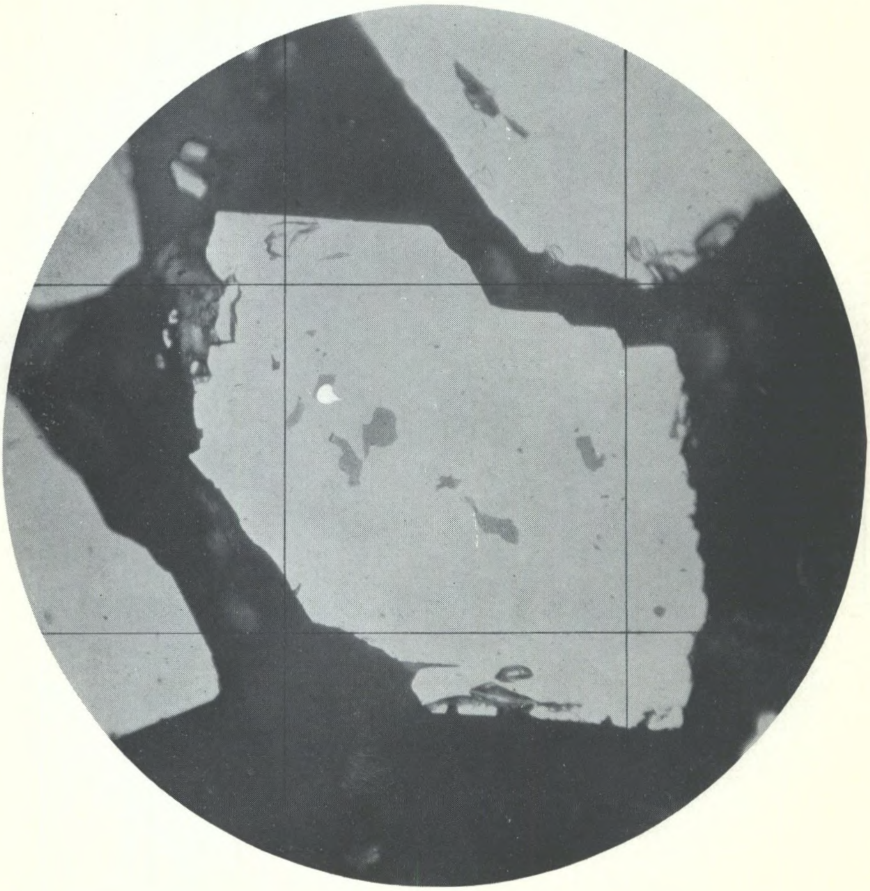
Ten tiny grains of native gold, ranging from 12 microns (−1100 mesh) down to 1 micron (−2300 mesh) in size, were observed. All occur in pyrite; some are alone in the dense mineral, some are associated with inclusions of gangue, and one is associated with a small grain of one of the unknown minerals. *(See* Plates I and II.)

PLATE I.



Small grains of native gold (white) in pyrite (light grey); some associated with inclusions of gangue (dark grey to black). Magnification, approximately 500. Oil immersion. A 200-mesh grid is superimposed.

PLATE II.



Tiny grain of native gold (white) associated with a bluish-grey unknown mineral (grey) in dense pyrite (light grey). Gangue, dark grey to black. Magnification, approximately 500. Oil immersion. A 200-mesh grid is superimposed.

Experimental Tests

The present shipment was submitted in order to determine:

(1) Whether the character of the ore mined has changed since the previous shipment in 1936;

(2) What cyanide residue could be obtained in 36 hours' agitation instead of 48 hours';

(3) Whether as good results could be obtained from agitation in a pulp denser than the 2 : 1 dilution being used.

It was also desired to ascertain the effect of increasing the strength of the cyanide solution used in the grinding and agitation periods and to discover whether concentration and regrinding of the sulphides would have a beneficial effect on the final cyanide residue.

As regards (1), there is a noticeable increase in the amounts of sulphides since the previous shipment in 1935, which assayed 0·85 ounce of gold per ton, 0·03 per cent of arsenic, and 0·13 per cent of sulphur. The microscopic examination of the polished sections showed that a portion of the gold was in dense pyrite in a very fine stage of subdivision.

In regard to (2) and (3), the following tests were carried out:

CYANIDATION

Test No. 1 (A to R)

Portions of the ore at −14 mesh were ground in a ball mill in cyanide solution of 1 pound of sodium cyanide per ton to varying fineness of grind. The pulps were agitated at the dilution ratio and for the length of time as set forth.

Results of Cyanidation:

Feed: gold, 0·79 oz./ton.

Test No.	Agitation, hours	Dilution ratio	Grind, per cent −200 mesh	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Titration, lb./ton solution		Reagents consumed, lb./ton ore		Reducing power of final solution ml. $\frac{N}{10}$ $KMnO_4$
						NaCN	CaO	NaCN	CaO	
1A........	24	2 : 1	56·4	0·04	94·9	1·0	0·30	0·30	2·4	36
1B........	36	2 : 1	56·4	0·035	95·6	1·0	0·20	0·30	2·5	38
1C........	48	2 : 1	56·4	0·035	95·6	1·0	0·20	0·30	2·5	40
1D*.......	48	2 : 1	57·8	0·035	95·6	1·96	0·20	0·80	2·5	....
1E*.......	48	1 : 1	57·8	0·04	94·9	2·0	0·25	0·80	2·5	....
1F........	36	2 : 1	61·0	0·035	95·6	1·0	0·20	0·4	2·3	....
1G........	36	1·5 : 1	61·0	0·035	95·6	1·0	0·20	0·4	2·3	....
1H........	36	1 : 1	61·0	0·04	94·9	0·96	0·20	0·45	2·5	....
1I........	36	2 : 1	63·6	0·04	94·9	1·0	0·25	0·35	2·5	....
1J........	24	1 : 1	68·7	0·065	91·8	1·0	0·3	0·4	2·4	64
1K........	36	1 : 1	68·7	0·04	94·9	1·0	0·3	0·4	2·4	70
1L........	48	1 : 1	68·7	0·035	95·6	1·0	0·3	0·4	2·4	80
1N........	36	1 : 1	71·7	0·04	94·9	0·96	0·25	0·45	2·5	....
1N........	24	2 : 1	74·8	0·035	95·6	1·0	0·2	0·4	2·3	....
1O........	24	1·5 : 1	74·8	0·035	95·6	0·9	0·2	0·4	2·3	....
1P........	24	1 : 1	74·8	0·05	93·7	1·0	0·2	0·4	2·4	....
1Q........	48	2 : 1	87·0	0·03	96·2	1·0	0·4	0·6	2·8	....
1R........	48	1·5 : 1	87·0	0·03	96·2	1·0	0·4	0·6	2·8	....

* Two pounds of sodium cyanide per ton.

The reducing powers of the final solutions were normal and showed no fouling.

Portions of the cyanide residue of Tests Nos. 1F and 1M were concentrated on the Haultain superpanner, with the following results:

Test No. 1F:

Product	Weight, per cent	Assay		Distribution per cent	
		Au, oz./ton	S, per cent	Au	S
Feed	100·00	0·036*	0·80	100·0	100·0
Concentrate	0·82	1·37	42·20*	31·1	43·3
Sand	74·57	0·025	0·47	51·7	43·8
Slime	24·61	0·025	0·42	17·2	12·9

Test No. 1M:

Feed	100·00	0·038*	0·80	100·0	100·0
Concentrate	0·86	1·325	31·56*	30·0	44·9
Sand	67·42	0·03	0·47	53·3	39·6
Slime	31·72	0·02	0·39	16·7	15·5

* Calculated.

On examination under the microscope, the concentrates were seen to consist of pyrite, arsenopyrite, and magnetite. No free gold was visible.

A portion of the cyanide residue from Test No. 1G was screened into +200- and −200-mesh sizes. The +200 mesh, which was 38·2 per cent by weight of the tailing, assayed 0·04 ounce of gold per ton and 0·58 per cent of sulphur. The −200-mesh product, consisting of the remaining 61·8 per cent, was passed through the Haultain infrasizer with the following results:

Haultain Infrasizer Test:

Microns	Weight, per cent	Assay,		Units		Distribution, per cent	
		Au, oz./ton	S, per cent	Au	S	Au	S
Above 56	12·7	0·085	2·17	0·108	27·56	35·9	30·7
56 to 40	15·2	0·025	0·94	0·038	14·29	12·6	15·9
40 to 28	14·6	0·025	0·90	0·037	13·14	12·3	14·6
28 to 20	10·9	0·023	0·86	0·025	9·37	8·3	10·5
20 to 14	8·9	0·020	0·81	0·018	7·21	6·0	8·0
14 to 10	7·6	0·020	0·69	0·015	5·24	5·0	5·8
Below 10	30·1	0·020	0·43	0·060	12·94	19·9	14·5
Totals	100·0	0·030	0·90	0·301	89·75	100·0	100·0

The gold follows the sulphides fairly consistently through the different size products. Even the finest-size particles at −10 microns contain 0·02 ounce of gold per ton and were not amenable to cyanidation.

JIG CONCENTRATION AND CYANIDATION

Test No. 2 (A and B)

The ore at −14 mesh was ground in cyanide solution of 1 pound of sodium cyanide per ton to pass 70·7 per cent −200 mesh in Test No. 2A and 84·0 per cent in Test No. 2B. The pulps were passed through a Denver gold jig and the resulting hutch products reground in cyanide solution to pass 99 per cent −325 mesh. The reground concentrates were combined with the jig tailings and agitated in cyanide solution for 24- and 48-hour periods.

Results. In Test No. 2A the jig concentrate weighed 5·0 per cent of the weight of the ore.

In Test No. 2B the jig concentrate weighed 3·5 per cent of the weight of the ore.

Cyanidation of Combined Products:

Test	Dilution ratio	Initial grind, per cent −200	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton ore	
						NaCN	CaO
2A......	2 : 1	70·7	24	0·035	95·6	0·80	3·0
2A......	2 : 1	70·7	48	0·032	95·9	1·00	3·2
2B......	2 : 1	84·0	24	0·03	96·2	1·20	3·5
2B......	1·5 : 1	84·0	24	0·03	96·2	1·20	3·6

TABLE CONCENTRATION AND CYANIDATION

Test No. 3 (A and B)

A Wilfley table replaced the jig used in the previous tests. Conditions otherwise were similar, the table concentrate being reground and combined with the table tailing prior to agitation.

Results. In Test No. 3A the table concentrate weighed 6·5 per cent of the weight of the ore.

In Test No. 3B the table concentrate weighed 3·6 per cent of the weight of the ore.

Cyanidation of Combined Products:

Test	Dilution ratio	Initial grind, per cent −200 mesh	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton ore	
						NaCN	CaO
3A......	1·5 : 1	62·9	24	0·04	94·9	0·95	2·9
3A......	2 : 1	62·9	36	0·03	96·2	1·15	3·0
3B......	2 : 1	87·0	48	0·025	96·8	1·30	3·4
3B......	1·5 : 1	87·0	48	0·025	96·8	1·30	3·5

CYANIDATION AND FLOTATION

Test No. 4 (A, B, and C)

A scavenging operation was conducted on the cyanide residue. The ore at −14 mesh was ground in a ball mill in cyanide solution of 1 pound of sodium cyanide per ton to pass 73·0 per cent −200 mesh in Test No. 4A, 79·6 per cent in Test No. 4B, and 82·0 per cent in Test No. 4C. The pulps were agitated for 48 hours at a dilution ratio of 1·5 : 1. The cyanide residues were filtered, washed, sampled, repulped, and transferred to a flotation machine. The pulps were conditioned with 2 pounds of soda ash per ton and floated with 1·0 pound of copper sulphate, 0·10 pound of amyl xanthate, and 0·07 pound of pine oil per ton. The resulting flotation concentrates were washed and reground in cyanide solution of 3 pounds of sodium cyanide per ton to pass 99·0 per cent −200 mesh and were agitated for 48 hours. The different products were assayed for gold.

Results:

Initial Cyanidation:

Feed; gold, 0·79 oz./ton.

Test	Dilution ratio	Initial grind, per cent −200 mesh	Agitation, hours	Tailing assay, Au, oz./ton	Extraction of gold, per cent	Reagents consumed, lb./ton ore	
						NaCN	CaO
4A......	1·5 : 1	73·0	48	0·035	95·6	0·62	2·8
4B......	1·5 : 1	79·6	48	0·03	96·2	0·67	3·0
4C......	1·5 : 1	82·0	48	0·03	96·2	0·75	3·0

Flotation and Cyanide Residues:

Test No. 4A

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
Feed..	100·00	0·035	100·0	....
Flotation concentrate........................	4·18	0·61	72·6	23·9 : 1
Flotation tailing............................	95·82	0·01	27·4	....

Test No. 4B

Feed..	100·00	0·03	100·0	....
Flotation concentrate........................	4·25	0·48	68·0	23·5 : 1
Flotation tailing............................	95·75	0·01	32·0	....

Test No. 4C

Feed..	100·00	0·03	100·0	....
Flotation concentrate........................	5·38	0·38	68·3	18·6 : 1
Flotation tailing............................	94·62	0·01	31·7	....

Regrinding and Agitation of Flotation Concentrates:

Test	Dilution ratio	Grind, per cent −325 mesh	Agitation, hours	Assay, Au, oz./ton		Extraction of gold, per cent	Reagents consumed, lb./ton concentrate	
				Feed	Tailing		NaCN	CaO
4A	3 : 1	99·0	48	0·61	0·41	32·8	2·1	7·9
4B	3 : 1	99·0	48	0·48	0·38	20·8	2·2	8·1
4C	3 : 1	96·1	48	0·38	0·31	18·4	2·4	8·0

The flotation concentrates from the different tests assayed as follows:

—	Au, oz./ton	As, per cent	S, per cent
Test 4A	0·61	5·36	17·44
Test 4B	0·48	5·18	17·09
Test 4C	0·38	4·13	13·22

Summary of Test No. 4:

—	Test No. 4A	Test No. 4B	Test No. 4C
Gold extracted by straight cyanidation, per cent	95·6	96·2	96·2
Gold extracted from flotation concentrate, per cent	1·4	0·8	0·5
Overall recovery, per cent	97·0	97·0	96·7
Overall tailing loss, Au. oz./ton	0·024	0·024	0·026

SUMMARY AND CONCLUSIONS

Straight cyanidation of the ore showed an extraction of 95·6 per cent of the gold and a cyanide residue of 0·035 ounce of gold per ton at a grind of 56·4 per cent −200 mesh in 36 hours' agitation. This extraction was not bettered until the ore was ground to pass 79·6 per cent −200 mesh, when a residue of 0·03 ounce of gold per ton was obtained. Increasing the agitation period from 36 to 48 hours did not result in any appreciable added extraction of the gold.

No beneficial results were apparent when the strength of cyanide solution was raised to 2·0 pounds of sodium cyanide per ton, the residue remaining at 0·035 ounce of gold per ton at a grind of 57·8 per cent −200 mesh.

The results obtained from different ratios of dilution showed that 1·5 : 1 and 2 : 1 gave similar values in the tailing of 0·035 ounce of gold per ton, whereas at 1 : 1 the residue showed an increase to 0·04 ounce per ton.

The superpanner and infrasizer tests of the cyanide residues showed that even the smallest-size sulphide particles carried 0·02 ounce of gold per ton, this result being given by the slime from the superpanner and the −10-micron product of the infrasizer.

Jig concentration in the grinding circuit was of little apparent benefit, an overall tailing loss of 0·03 ounce of gold resulting from this method. Table concentration in the grinding circuit gave a slightly better result, an overall tailing of 0·025 ounce of gold per ton being obtained.

A scavenging operation by means of flotation concentration of the tailing, followed by regrinding and agitation, gave an overall residue of 0·024 ounce of gold per ton. An analysis of the concentrate obtained by this method gave as high as 5·3 per cent of arsenic or 11·6 per cent of arsenopyrite.

It is apparent from the work performed on this shipment that the ore from the lower levels is showing an increasing amount of sulphides and that these sulphides carry about 3 per cent of the gold, or 0·025 ounce of gold per ton, in a very refractory form, which is not susceptible to exceedingly fine grinding and prolonged agitation in cyanide solution. This portion of the gold appears to be locked up somewhat as in some of the ores of the Long Lac district, which require roasting to free the gold.

A cyanide residue of 0·035 ounce of gold per ton can be obtained with a grind of 60 per cent −200 mesh in 36 hours' agitation at a dilution ratio of 1·5 : 1.

III

INVESTIGATIONS THE DETAILS OF WHICH ARE NOT PUBLISHED

Ore or Product	Source of Shipment	Address
Molybdenite	Maniwaki Molybdenite Company	Maniwaki, Que.
Gold	Early Bird Claim	Moresby Island, Que.
Gold	Algold Mines, Limited	Goudreau, Ont.
Molybdenite	Cheabella Mine	Montbeillard, Témiscamingue County, Que.
Gravel	Vermillion River, Sudbury District, Ont.	F. J. Adair, Capreol, Ont.
Chromite	Lake Abitibi, Ont	L. W. Coon, Haileybury, Ont.
Silver-copper-cobalt	Harrison-Hibbert Mines, Limited	Cobalt, Twp. of Bucke, Ont.
Gold	Paulpic Gold Mines, Limited	Tashota Station, Ont.
Gravel	H. H. Smith	Niagara Falls, Ont.
Molybdenite	Molybdenite Corporation of Canada, Limited.	LaCorne Township, Que.
Silver-lead	Sheffield Mine	Nicola, B.C.
Gold	Brunne Copper Lake Telluride Mines, Limited.	Tartan Lake, Man.
Arsenical gold	Madoc Township, Hastings County, Ont.	A. T. Westbrook, Peterborough, Ont.
Gold	Broulan Porcupine Mines, Limited	Pamour, Ont.
Gold-silver	Brunne Copper Lake Telluride Mines, Limited.	Athapapuskow Mining Division, Man.
Gold	Melisek Property	Tyrrell Township, Gowganda Mining Division, Ont.
Calcine	J. Murray Riddell	Duluth, Minnesota, U.S.A.
Gold	Claim No. K.K. 677, Herb Lake, Man.	Alex Cupples, Gurney, Man.
Gold	Lake Rowan Gold Mines, Limited	Red Lake, Ont.
Gold ore and mill water	A. Bethune	Emo, Ont.
Tailing dump residue	Nova Scotia Mine, Cobalt, Ont	J. S. Crosscombe, Haileybury, Ont.
Mill products	Chesterville Larder Lake Gold Mining Company, Limited.	Cheminis, Ont.
Gold ore and mill tailings	Cochenour Willans Gold Mines, Limited.	McKenzie Island, Ont.
Black sand	Saint's Rest or Taylor's Island, near Saint John, N.B.	W. J. Wright, Provincial Geologist, Fredericton, N.B.
Chromium-nickel-iron table concentrate.	Canadian Johns-Manville Company, Limited.	Asbestos, Que.
Molybdenite	Greenlee Mines, Limited	Bompas Township, Ont.
Placer sand	Saw Mill Creek Claims, Fort Steele Mining Division, B.C.	F. C. McAlpine, Calgary, Alta.
Placer sand	H. H. Smith	Niagara Falls, Ont.
Gold	Theresa Gold Mines, Limited	Long Lac, Ont.
Gold	Falcon Lake, Man	W. J. Richards, Kenora, Ont.
Mill residues	Peterson Lake, Cobalt, Ont	Progress Smelting and Refining Co., Toronto, Ont.
Gold	Bayside Malartic Mines, Limited	Taschereau, Que.
Gold	Astoria and Jewel Claims, Bissett, Man.	M. J. Thorarinson, Winnipeg, Man.
Copper	Frontenac County, Ont	W. McG. Brown, Toronto, Ont.
Gold	King Fissure Mine	Brookfield Mines, Queens Co., N.S.
Gold	Claim K-9194, District of Kenora, Ont.	Dr. A. L. MacDonald, Dyment, Ont.
Copper-zinc	Normetal Mining Corporation, Limited.	Dupuy, Que.
Nickel matte	Falconbridge Nickel Mines, Limited	Falconbridge, Ont.
Gold	Golconda Mines, Limited	Devangue, Que.
Gold	International Mining Corporation (Quebec), Limited.	Senneterre, Que.

Examinations were made of:

Utilization of the iron, nickel, and chromium values in tailings. (Canadian Johns-Manville Company, Limited, Asbestos, Que.)

Use of the Chrom-X additions in manufacturing steel in the high-frequency induction furnace. (Chromium Mining and Smelting Corporation, Limited, Sault Ste. Marie, Ont.)

Two connecting rod gudgeon pins. (Department of National Defence.)

A three-foot section of a Bolingbroke centre section spar flange. (R.C.A.F.—Department of National Defence.)

Two austenitic manganese steels. (Lynn, MacLeod Engineering Supplies, Limited, Thetford Mines, Que.)

A stained Alclad sheet. (Ottawa Car Manufacturing Company, Limited.)

A broken airscrew from aircraft CV-CCW. (Department of Transport, Ottawa.)

Four bearing metal materials. (T. Reid, M.P., Newton, B.C.)

A broken Dilts hydrafiner shaft. (Alexander Fleck, Limited, Ottawa.)

A rock drill tappet. (Paymaster Consolidated Mines, Limited, Schumacher, Ont.)

A failed austenitic manganese steel dipper tooth casting. (Joliette Steel, Limited, Joliette, Que.)

The steel of an austenitic manganese steel ball mill liner. (Sorel Steel Foundries, Ltd., Sorel, Que.)

A broken landing gear part from R.C.M.P. Aircraft CF-MPE. (Royal Canadian Mounted Police, Ottawa.)

Three austenitic manganese steels. (Sorel Steel Foundries, Limited, Sorel, Que.)

Four paper mill burrs. (Northern Foundry and Machine Company, Limited, Sault Ste. Marie, Ont.)

Microscopic examinations and identifications were made of:

Bismuth-bearing mineral in sample of cobalt ore from Cobalt, Ont. (Submitted by A. A. Cole, Cobalt, Ont.)

Hyland ore. (Submitted by C. M. Campbell, Vancouver, B.C.)

Auriferous arsenopyrite from Seal Harbour Gold Mines, Goldboro, N.S.

Arsenopyrite from Seal Harbour Gold Mines, Goldboro, N.S.

Ore from Cranberry Lake area, northern Manitoba. (Submitted by S. L. May, Saskatoon, Sask.)

Gold ore from the Goldcrest Claims, east end of Lake Athabaska, Sask.

Metallic mineral in specimen of rock from near Portage du Fort, Que.

Ten special samples of gold ore from Sullivan Consolidated Mines, Limited, Sullivan P.O., Que.

Manganese-bearing samples. (Submitted by J. P. Messervey, Department of Mines, Halifax, N.S.)

IV

ACTIVITIES OF THE CHEMICAL, MINERALOGICAL, PHYSICAL TESTING, AND HEAT-TREATING LABORATORIES, LISTING ANALYSES AND TESTS, INCLUDING MISCELLANEOUS ITEMS, TESTS, AND SAMPLING JOBS

Chemical Laboratories:

During the half-year, July 1 to December 31, 1939, 2,995 samples of ores, minerals, and metal products were analysed by the staff of the chemical laboratories and complete records were issued thereon.

This work included a total of 8,011 chemical and assay determinations, in which 50 different mineral constituents were involved.

The sources of the samples were:

—	No. of Samples	Per cent of Total
Metallic mill	2,353	78·56
Bureau of Geology and Topography	19	0·64
Pyrometallurgical Laboratory	104	3·47
Industrial Minerals Division	286	9·55
Fuel Testing Laboratory	16	0·53
Custom assays	217	7·25
	2,995	100·00
Total determinations	8,011	
Total gold assays	2,805	35·01
Total silver assays	405	5·06

Mineragraphic Laboratory:

The following is a summary of the work:

A. Investigations Completed:	
Gold ores	22
Mill products	3
Miscellaneous samples	5
Special studies	12
Total	42
B. Spectrographic analyses	22
C. Polished sections prepared:	
For Mineragraphic Laboratory	373
For others	59
Total	432
Thin sections prepared	5
Photomicrographs taken	36
Photographs prepared	259

Physical Testing and Heat-Treating Laboratories:

The determination of the stress-strain relationships for eighty aluminium alloy test specimens for the British Aeronautical Inspection Directorate (forty-five from the Inspector at the Aluminum Company of America, twenty-eight from the Inspector at the Ottawa Car Manufacturing Co., Limited, five from the Inspector at Canadian Vickers, Limited, and two from the Inspector at Fairchild Aircraft Co., Limited, Longueuil, Que.).

Calibration of four thermocouples (Ottawa Car Manufacturing Company, Limited).

Determination of impact strengths of six steels. (Dominion Engineering Company, Limited, Lachine, Que.)

Drop tests and hardness test on grinding ball. (Canada Foundries and Forgings, Limited, Welland, Ont.)

Hardness Test on bronze casting. (Mr. Moffatt, Timmins, Ont.)

Hardness tests on two steel castings. (Hull Iron and Steel Foundries, Limited.)

Tensile tests on three cast bronze test bars. (Department of National Defence.)

Heat treatment of six dozen steel aircraft studs. (R.C.A.F., Department of National Defence.)

Aircraft steel spar member, heat-treated. (R.C.A.F., Department of National Defence.)

Determination of melting point of silver solder. (Canadian Car and Foundries Company, Limited.)

Miscellaneous:

Microscopic examination of gold ore from Mallard Lake Gold Mines, Limited, Kirkland Lake, Ont.

Microscopic examination of iron ore submitted by A. S. Watson, Dunham, Que.

Note on the character of iron ore from the head of the Bay of Fundy, N.S.

Photomicrographs of sections of coconut shells for Fuel Research Laboratories.

Microscopic examination and photomicrographs of marl from White Valley Chemical Company, Limited, Toronto, Ont. (Submitted by M. F. Goudge.)

A 500-pound silico-chromium steel ingot made for Atlas Steel Company, Limited.

Three tons of nickel-chromium magnetite concentrate were sintered and shipped to reduction furnace at Shawinigan Falls. (Shawinigan Chemicals, Limited and Johns-Manville Company, Limited.)

Twenty-two nickel-chromium alloy heat-resisting trays were cast. (Royal Canadian Mint.)

Anodized anti-gas respirator valve parts were examined. (Department of National Defence.)

Two hundred pounds of manganese ore were given a reducing roast. (Pan American Alloys, Limited.)

V

RÉSUMÉ OF SPECIAL INVESTIGATIONS AND RESEARCH COMPLETED, IN PROGRESS, OR UNDER CONSIDERATION

The activities of this section, as in previous years, comprised the conducting of special investigations to:

(1) Assist in solving problems presenting difficulties that arise during the investigation of ores submitted for determination of the best method of treatment for the recovery of their valuable contents.

(2) Assist mine operators in solving problems in current milling practice.

(3) Develop new procedures or processes to further the use of the natural resources of the country.

(4) Improve present practice in milling with the purpose of increasing recovery and grade of product.

(5) Assist the metal industries by standard and special tests involving physical, chemical, microscopic, and spectrographic examinations.

(6) Investigate possible production of new alloys.

In connection with the investigations of the first category eight ores from the following sources were studied:

Gold ore: MacLeod-Cockshutt Gold Mines, Limited, Geraldton, Ont.

Gold ore: Paymaster Consolidated Mines, Limited, South Porcupine, Ont.

Gold ore: W. J. Richards, Falcon Lake, Man.

Gold ore: Melisek property, Tyrell Township, Gowganda, Ont.

Gold ore: Cochenour-Willans Gold Mines, Limited, McKenzie Island, Ont.

Gold ore: Tamarac Gold Mines, near Ymir, B.C.

Manganese ore: Cowichan Lake, B.C.

Tungsten ore: McKenzie Red Lake Gold Mines, Limited, McKenzie Island, Ont.

The MacLeod-Cockshutt ore was of the gold-bearing pyrite-arsenopyrite class with refractory gold in the sulphides. Details will be found in Investigation No. 776, pages 18 to 32 of this report.

The Paymaster Consolidated investigation was conducted on a flotation concentrate (pyritic) from the plant cyanide tailing and involved roasting tests and cyanidation of the calcines to determine recovery obtainable. Straight cyanidation at a very fine grind showed an extraction of only 35 per cent of the gold. Two parcels of concentrate assaying 0·27 ounce and 0·20 ounce of gold per ton respectively were received and the results showed that recoveries of 92 per cent and 79 per cent could be obtained by keeping the temperature low (350° C.) for the first hour with minimum air supply, then gradually increasing the temperature with freer air supply to a maximum of 650° C.

The Falcon Lake ore contained 3·5 per cent of pyrrhotite. The pyrrhotite causes excessive fouling of the cyanide solution in cycle tests with decreased extraction in each cycle. Aeration, with addition of lead salts, reduced fouling materially and resulted in more consistent extraction.

The Melisek ore carried most of the gold in the sulphides, and flotation of the sulphide minerals with subsequent roasting of the concentrate under controlled conditions and cyanidation of the calcine was found to be the most satisfactory treatment. Results showed 92 per cent of the gold extractable.

The Cochenour-Willans ore also had a substantial amount of the gold locked up in pyrite in an extremely fine condition. The best method of treatment comprised direct cyanidation of the ore, yielding 83·7 per cent of the gold, with re-treatment of the cyanide tailing by flotation to concentrate the gold-bearing pyrite. By roasting the pyrite concentrate and cyaniding the calcine, an additional 12·6 per cent of the gold was recovered, making a total of 96·3 per cent.

The Tamarac ore was an arsenopyrite assaying 17·6 per cent of arsenopyrite, 20 per cent of pyrite, and 0·52 ounce of gold per ton. Straight cyanidation gave an extraction of 75 per cent, which could not be bettered by any other method of treatment except roasting.

The manganese ore from Cowichan Lake consisted of a mixture of rhodochrosite, rhodonite, and an undetermined oxide of manganese in quartz gangue. Concentration by tabling or flotation was unproductive of a product of marketable grade. Leaching with sulphur dioxide extracted 50 per cent of the manganese, whereas leaching with sulphuric acid followed by sulphur dioxide made 75 per cent of the manganese soluble.

The scheelite from McKenzie Red Lake Gold Mines occurs in the lower levels of the orebody and is associated with heavy pyrite. No particular difficulty was encountered in producing a concentrate containing 60 per cent of tungstic oxide by tabling, but it carried also a substantial amount of pyrite. Flotation removes a large proportion of the pyrite, but roasting probably would be necessary to reduce the sulphur to the amount required by specification.

Under the second category, two projects were investigated, namely,

Mill tailing: Tyranite Gold Mines, Limited, Gowganda, Ont.

Mill tailing: Sand River Gold Mines, Beardmore, Ont.

The loss in the Tyranite mill tailing was 0·035 ounce of gold per ton. In grinding, this ore produces a large amount of fine slime, which hinders settling and possibly, also, extraction. By removal of the slime and regrinding the sand, tailing loss was reduced by half. By flotation, a flotation tailing of 0·0075 ounce was obtained.

The Sand River mill tailing also carried an appreciable amount of gold, but it was found to be locked up in sulphides and gangue in too finely divided a condition to make recovery practicable.

Of the third category there were three investigations.

A request was received to investigate the possible separation of copper and nickel in matte by a process of roasting and leaching with ammonia and ammonium salts. Difficulty in roasting the matte through its hardening into cake even at very low temperatures was solved by continuous rabbling. Extraction with ammonium salts was not satisfactory, inasmuch as both copper and nickel leached readily from low-temperature roasts, and similarly from high-temperature roasts, although with much reduced extraction. Remelting the matte and granulating it by pouring it in a fine stream into water yielded somewhat better results by selective leaching, especially when using sulphuric acid as the leaching medium. The investigation was suspended on account of pressure of other work.

The reduction of tin concentrate was attempted with the idea of producing electrolytic tin, either directly from impure metallized tin using the impure tin as anode, or by leaching the metallized tin and electro-winning from electrolyte. Reduction of over 90 per cent of the tin by using hydrogen or coal gas was readily obtained. Proper facilities were not immediately available for making good anodes from metallized tin and the research was not followed up. Leaching tests were fairly satisfactory, but in electro-deposition the problem of avoiding spongy deposition was not solved when the work had to be suspended.

In the study of the behaviour of associated minerals, such as sulphides, in milling and cyanidation, time permitted experiments with sphalerite only. Very little reaction occurred in grinding in water or lime, but in the presence of cyanide some reaction was noticeable, and zinc was present in the solution, expecially in solutions low in lime. High-lime pulps showed lower solubility of the zinc.

In a short study of the roasting of arsenopyrite concentrate, the most favourable results were obtained by roasting at a low initial temperature just sufficient to start the arsenic fuming and by limiting the air supply. Upon diminishment of arsenic fumes the air can be increased with temperature increase. Some evidence was obtained of a beneficial effect from keeping dioxide circulating with the air in the middle roasting stage ; arsenic was more completely removed and slightly higher recovery of gold was obtained by subsequent cyanide leaching. A more detailed study of this procedure should be made, arrangement being provided for collecting, sampling, and analysing the gases during roasting.

A low-grade bullion submitted was described as being high in zinc and difficult to refine; the bulk of it proved to be lead. A simple fluxing for removal of zinc, copper, etc., followed by cupellation solved the difficulty.

A gold precipitate was submitted as giving serious trouble in refining. Analysis proved it to contain zinc, copper, lead, antimony, and other lesser impurities. Roasting, followed by sulphuric acid leaching for removal of zinc and copper and leaching the lead with hot brine, produced a bullion of 820 fine gold and 170 fine silver with extremely low losses. Roasting and fluxing could also be employed, with re-treatment of the slag, which carried a substantial amount of gold.

In the Spectrographic Laboratory (set-up and equipment described in Report No. 805, 1939, pages 192 to 194) a study was made of the general problem of qualitative analysis. The method finally adopted was that involving specially prepared "Master Plates", by the use of which a rapid survey can be made of the spectrum of an unknown substance, and the results of analysis can be reported within two hours of receipt of the sample. The elements capable of being so detected are the following:

I. Infra-red group:

Lithium	Caesium
Sodium	Calcium
Potassium	Strontium
Rubidium	Barium

II. Ultra-violet group:

Aluminium	Gold	Rhodium
Antimony	Indium	Ruthenium
Arsenic	Iridium	Scandium
Beryllium	Iron	Silicon
Bismuth	Lead	Silver
Boron	Magnesium	Tellurium
Cadmium	Manganese	Thallium
Chromium	Molybdenum	Thorium
Cobalt	Nickel	Tin
Columbium	Osmium	Titanium
Copper	Palladium	Tungsten
Gallium	Phosphorus	Vanadium
Germanium	Platinum	Zinc
		Zirconium

Sixty-nine qualitative analyses were made, comprising precipitates, derived from investigations in the mill, for constituents, gold beads for detection of metals of the platinum group, etc., from the Chemical laboratory; mineral crystals, etc., from the Mineragraphic laboratory; metals and alloys from the Pyrometallurgical laboratory; fuel ashes for vanadium, etc., from the Fuels Research laboratory; and various mineral specimens from the Industrial Minerals laboratory for detection of contained elements.

A photo-electric densitometer is being specially built for the Department by Baird Associates of Cambridge, Mass. Upon delivery, it will be possible to proceed with the development of quantitative methods.

Much auxiliary equipment has been built up, notably rotating sectors, special diaphragms, etc., and improvements were made in the air stand and optical condensing system. The photographic technique has been studied and markedly improved.

Pending the arrival of the densitometer, little work has been done on the spark method, the chief application of which is in the quantitative study of alloys. A program of work has, however, been drawn up for the analysis of duralumin, the first step in which is the preparation of a comprehensive set of standard alloys. Accurate measurements of specific gravities of materials submitted by the Industrial Minerals Division have occasionally been made.

In the Metallurgical Laboratory, the demands from industry and the preparations for handling the expected increase in the work of check testing arising from the production of war materials prevented further progress on the investigations contemplated with the limited staff available. Some of the work of this laboratory is listed in Sections III and IV. Further alterations were made to improve the accuracy of the machine for testing damping capacity.

The proposed studies in the use and application of the X-ray equipment have been suspended.

The vacuum furnace purchased early in the year has been completely equipped and a few trial melts were made to check the operating conditions. A study involving the use of and behaviour of uranium as an alloyed element in steel was being planned.

INDEX