

CANADA
DEPARTMENT OF MINES

HON. CHARLES STEWART, MINISTER; CHARLES CAMSELL, DEPUTY MINISTER

MINES BRANCH

JOHN MCLEISH, DIRECTOR

INVESTIGATIONS IN ORE DRESSING AND
METALLURGY

(Testing and Research Laboratories)

1926

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OTTAWA
F. A. AGLAND
PRINTER TO THE KING'S MOST EXCELLENT MAJESTY
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Annual reports on Mines Branch investigations are now issued in four parts, as follows.—

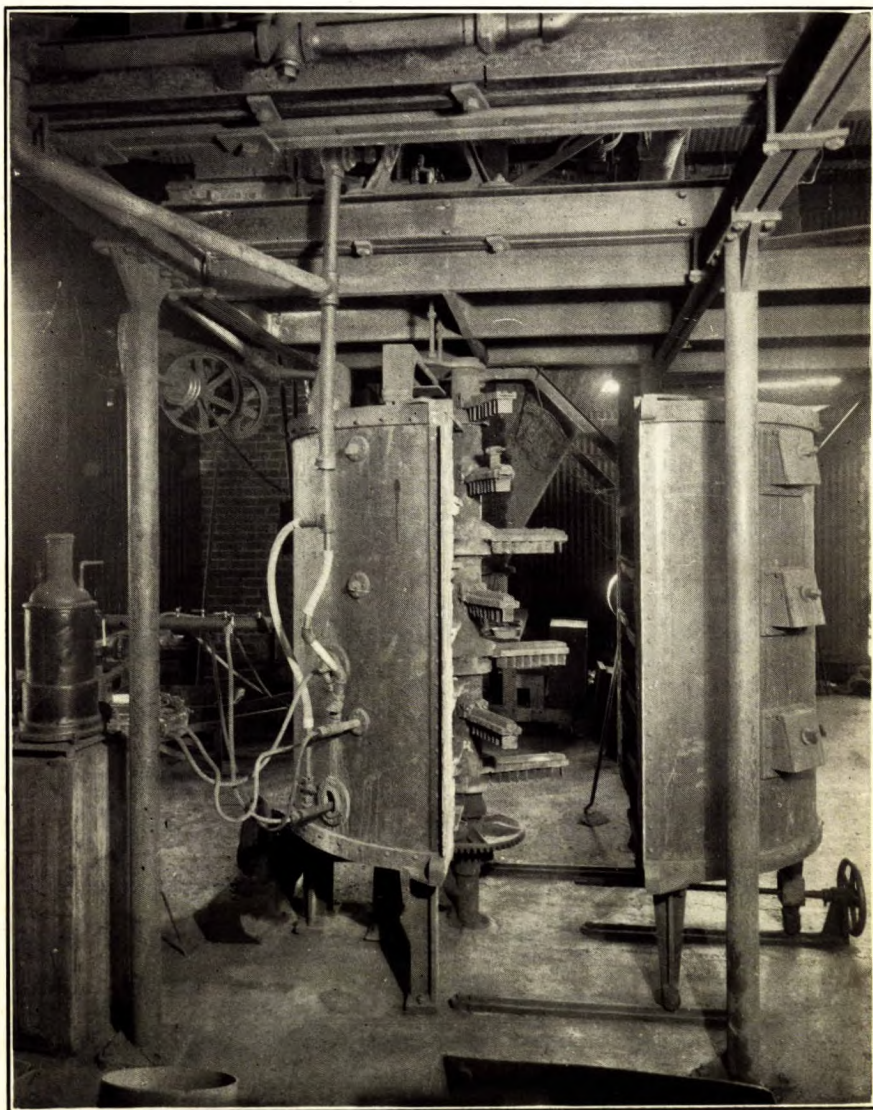
Investigations of Mineral Resources and the Mining Industry.

Investigations in Ore Dressing and Metallurgy (Testing and Research Laboratories).

Investigations of Fuels and Fuel Testing (Testing and Research Laboratories).

Investigations in Ceramics and Road Materials (Testing and Research Laboratories).

Other reports on Special Investigations are issued as completed.



Three-foot, 6-hearth Herreshoff roasting furnace. This furnace is used for the roasting of ores and concentrates for leaching and electrochemical tests.

MINES BRANCH INVESTIGATIONS IN ORE DRESSING AND METALLURGY, 1926

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm
Chief of Division

During the year 1926 the laboratories of the Division, consisting of an ore dressing laboratory for metallic ores, an ore dressing laboratory for non-metallic ores, a pyrometallurgical laboratory, and electrochemical and hydrometallurgical laboratory, and chemical laboratories, were devoted entirely to experimental test and research work on ore treatment. New equipment was installed to meet the requirements of the investigations in progress. The equipment of the non-metallic laboratory was completed. A pamphlet descriptive of the laboratories, their equipment, the nature of the test work performed, and regulations governing the shipment of ore for test purposes, has been revised and brought up to date. Investigatory engineers of the National Research Council were given the use of the laboratories for their investigations on Canadian magnesites.

The investigations have been carried out under the direction of W. B. Timm, chief engineer Division of Ore Dressing and Metallurgy; those in Section II on metallic ores were under the immediate supervision of C. S. Parsons assisted by J. S. Godard, and those on non-metallic ores under R. K. Carnochan; those in Section III under the immediate supervision of R. J. Traill assisted by W. R. McClelland. The work and investigations in the chemical laboratories were under the immediate supervision of H. C. Mabey, chief chemist of the Division, assisted by B. P. Coyne, R. A. Rogers, and L. Lutes. The operation and maintenance of the laboratories was performed by an operating and mechanical staff under the immediate supervision of B. M. Derry, mill foreman.

Section II contains the reports issued to the shippers of ore, giving the results of investigations by C. S. Parsons and J. S. Godard on metallic ores, and by R. K. Carnochan on non-metallics. The investigations deal with the treatment of a wide variety of ores as shown in the list following. The majority of the metallic ores were complex mixtures of various minerals requiring a separation and concentration into marketable products. The gold ores investigated were arsenical, or containing copper and other refractory minerals. The work on non-metallics included investigations on low-grade flake graphite and on the preparation of calcite and bentonite to meet the requirements of the market.

Section III contains the results of the investigatory work in the electro-chemical and hydrometallurgical laboratory, by R. J. Traill and W. R. McClelland, on the treatment of ilmenite ores and high iron-copper concentrates.

Section IV is a summary of the work and investigations in the chemical laboratory by H. C. Mabey, chief chemist of the Division.

Section V is a summary of the work on selective flotation as applied to Canadian ores, by C. S. Parsons.

List of Ores and Metallurgical Products on which Experimental Test and Research Work was Conducted

In the following table is given a list of the ores and products received at the Ore Testing Laboratories for experimental test and research. The tabulated statement includes the report numbers, class of ore or product, source of shipment, name of shipper, and weight of shipment.

Report No.	Page	Ore, or product	Source of shipment	Shipper and address	Weight
241	3	Lead-zinc-gold-silver.	Dunwell mine, Stewat, B.C.	Dunwell Mines Ltd., Victoria, B.C.	200
242	6	Copper-gold...	Horne mine, Que...	Noranda Mines Ltd., Rouyn, Que.	1,618
243	13	Lead-zinc-gold-silver.	J. & L. mine, Revelstoke, B.C.	Porcupine Goldfields Development & Finance Co., Ltd., Montreal.	70
244	16	Gold-arsenic...	Mount Evelyn mine, B.C.	P. J. Job, Vancouver, B.C.....	55
245	28	Lead-zinc.....	Notre-Dame des Anges, Que	British Metal Corporation (Canada) Ltd., Montreal.	40,000
246	31	Calcite.....	Near Perth, Ont....	M. Day Baldwin, Montreal.....	9,551
247	32	Gold-silver.....	Alberni, B.C.....	A. L. Smith, Alberni, B.C.....	70
248	35	Graphite.....	Buckingham, Que...	Crucible Graphite Co., Ltd., Buckingham.	1,000
249	41	Zinc and copper-zinc.	Rouyn, Que.....	Alderson and McKay, Montreal.	140
250	46	Gold-copper....	Gold Hill mine....	Gold Hill Mines, Boston Creek, Ont.	57
251	49	Zinc-copper....	John Bull and Florence claims, Powell River, B.C.	Ladysmith Smelter Syndicate, Victoria, B.C.	78
252	53	Gold-arsenic....	Long Lake, Ont.....	Alderson and McKay, Montreal	200
253	57	Silver-lead-copper-zinc:	La Rose mine.....	Ladysmith Smelter Syndicate, Victoria, B.C.	50
254	60	Lead.....	Perth Road, Ont....	Forbes Galena Mine Ltd., Perth Road, Ont.	78,400
255	66	Gold-copper....	Argonaut mine, Ont.	Argonaut Consolidated Mines Ltd., Argonaut, Ont.	225
256	72	Gold-copper....	Kitchener mine, East-central Manitoba.	Central Manitoba Mines, Winnipeg, Man.	90
257	79	Lead-zinc-gold-silver.	Alice Arm, B.C.....	E. Trethewey, Alice Arm, B.C.	200
258	82	Lead.....	Giant mine, Spillamacheen, B.C.	Pacific Mines, Ltd., Spillamacheen, B.C.	55
259	84	Gold-copper....	English Brook dist., Manitoba.	A. McDonald, Winnipeg.....	70
260	86	Bentonite.....	Rosedale, Alberta..	Rosedale Colliery Co., Rosedale, Alberta.	2,135
261	87	Lead-zinc.....	Whitewater dump, Retallack, B.C.	M. S. Davys, Kaslo, B.C.....	65
262	93	Gold.....	Dome mine, South Porcupine, Ont.	Dome Mines, Ltd., South Porcupine, Ont.	350

II

REPORTS OF INVESTIGATIONS: ORE DRESSING AND METALLURGICAL LABORATORIES

Report No. 241

CONCENTRATION OF THE ORES FROM THE DUNWELL MINES, LTD.,
STEWART, B.C.

C. S. Parsons

Shipment. A shipment of ore was received from the Dunwell Mines, Ltd., Victoria, B.C., December 31, 1925. It consisted of three sacks, gross weight 200 pounds, marked respectively "Sunbeam vein", "Heavy sulphide vein", and "Dunwell vein", from the properties situated on the east side of Bear river, north of Glacier creek, about 5 miles northeast of Stewart, B.C.

Character and Analysis:

Sample	Copper, per cent	Lead, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton	Remarks
Sunbeam Vein.....	0.35	1.10	1.4	0.13	76.09	Small amount of sulphides.
Heavy Sulphide Vein.	0.73	18.47	23.59	0.42	19.42	Larger amount of sulphides.
Dunwell Vein.....	0.17	0.8	1.10	0.12	37.62	Small amount of sulphides and graphite present.

Purpose of Experimental Tests. The shipment was accompanied by a request for mill tests, in order to work out an economical method of treatment, each sample to be run separately, and then a combined sample to be tested to determine if one flow-sheet could be used for treating a combined mill feed composed of the three grades of ore.

EXPERIMENTAL TESTS

Each of the three samples was crushed and assayed separately. A number of flotation and table tests were made on each sample and on the composite sample, the results of which will be discussed briefly.

SUNBEAM VEIN ORE

Heads—Cu 0.35 per cent; Pb 1.10 per cent; Zn 1.40 per cent; Au 0.13 oz./ton; Ag 76.07 oz./ton.

Test No.	Product	Weight, per cent	Analysis				Per cent of values			
			Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
1	Lead concentrate....	13.3	6.2	4.58	0.75	456.05				
	Zinc concentrate....	9.7	1.0	5.52	0.26	66.46				
	Tailing.....	77.0	0.25	0.25	0.05	11.76				
2	Flotation concentrate....	24.3	4.08	5.73	0.50	320.10	90.0	91.6	68.4	92.6
	Table concentrate....	4.1	0.59	0.50	0.70	79.80	2.2	1.3	19.5	3.9
	Tailing.....	71.6	0.12	0.15	0.03	4.07	7.8	7.1	12.1	3.5

Test No. 1

The ore was ground to approximately 65 mesh and selectively floated to produce a lead concentrate and a zinc concentrate. The following reagents were used:

Lead reagents:	Cyanide.....	0.30 lb./ton—15 mins. contact
	Soda ash.....	4.0 "
	Thiocarbamide.....	0.2 "
	Cresylic acid.....	0.2 "
Zinc reagents:	Copper sulphate.....	1.25 "
	Xanthate.....	0.3 "
	Pine oil.....	0.1 "

Test No. 2

The ore was ground to approximately 65 mesh; a bulk concentrate was made of both the lead and zinc, and the flotation tailing tabled to recover the gold associated with the pyrite.

Reagents:

Coal tar.....	0.2 lb./ton added to ball mill
Coal-tar creosote.....	0.75 " " "

HEAVY SULPHIDE VEIN ORE

Heads—Cu 0.73 per cent; Pb 18.47 per cent; Zn 23.54 per cent; Au 0.42 oz./ton; Ag 19.42 oz./ton.

Test No.	Product	Weight, per cent	Analysis				Per cent of values			
			Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
1	Lead concentrate....	28.8	53.4	10.54	0.70	43.3	32.6	13.2	45.4	66.0
	Zinc "	40.8	6.33	46.48	0.34	10.68	13.9	82.5	31.3	23.2
	Tailing.....	30.4	1.16	3.31	0.34	6.78	3.5	4.8	23.3	10.8
4	Lead concentrate....	32.7	52.31	10.97	0.80	46.30	93.0	15.3	63.9	81.3
	Zinc "	41.1	2.31	47.69	0.16	5.74	5.2	83.3	16.1	12.7
	Table "	5.2			0.90	9.80			11.4	2.7
	Tailing.....	21.0	1.25	1.26	0.17	2.93	1.8	1.4	8.6	3.3

Test No. 1

The ore was ground to approximately —65 mesh. The lead and zinc were selectively floated to make a lead concentrate and a zinc concentrate.

Reagents:

Lead reagents:	Soda ash.....	6.0 lb./ton	15 mins. contact
	Thiocarbamilide.....	0.2 "	
	Sodium cyanide.....	0.3 "	
	Cresylic acid.....	0.2 "	
Zinc reagents:	Copper sulphate.....	1.25 "	
	Xanthate.....	0.4 "	
	Pine oil.....	0.1 "	

Test No. 4

The ore was ground to approximately —65 mesh and selectively floated to produce a lead concentrate and a zinc concentrate. The tailings were tailed to recover a portion of the gold remaining. The reagents were the same as used in test No. 1, except that the cyanide was given only a 5-minute contact.

DUNWELL VEIN ORE

Heads—Cu 0.17 per cent; Pb 0.8 per cent; Zn 1.1 per cent; Au 0.12 oz./ton; Ag 37.62 oz./ton. Ore contained graphite.

Product	Weight, per cent	Analysis				Per cent of values			
		Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
Flotation concentrate.....	18.4	4.08	3.83	0.50	179.9	97.8	68.5	73.9	81.0
Tailing.....	80.6	0.02	0.40	0.04	9.56	2.2	31.5	26.1	19.0

The ore was ground to approximately —65 mesh and floated to produce a bulk concentrate.

Reagents:

Coke-oven tar.....	0.2 lb./ton added to ball mill
Coal-tar creosote.....	0.7 " " " "

SUMMARY AND REMARKS ON RESULTS OF TESTS ON INDIVIDUAL SAMPLES

These tests should not be accepted as indicating the maximum grade of concentrate which could be obtained, or the best recoveries of the minerals. They show, however, that the ore from all these veins can be concentrated by flotation and tabling into marketable products, with good recoveries. By tabling the flotation tailing the gold recovery is increased from 11 to 20 per cent with an addition of less than 5 per cent by weight to the concentrate. In order to keep the ratio of concentration as high as possible, the tables should be used on the flotation tailing.

COMPOSITE SAMPLE

A composite sample was made up from the three samples in the following proportion:

Sunbeam Vein.....	5 parts
Heavy Sulphide Vein.....	3 "
Dunwell Vein.....	2 "

The ore was ground to approximately —65 mesh and selectively floated to produce a lead concentrate and a zinc concentrate. The flotation tailing was tailed to recover additional gold by making a pyrite concentrate.

Reagents: Same as used on heavy sulphide vein ore.

Product	Weight, per cent	Analysis				Per cent of values			
		Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
Lead concentrate.....	20.2	28.78	9.66	0.90	233.90	94.2	24.0	72.0	88.2
Zinc ".....	19.3	1.00	31.69	0.14	10.02	3.1	74.8	10.6	4.4
Table ".....	4.0	1.00	1.81	0.64	55.46	0.7	0.9	10.7	3.6
" tailing.....	56.5	0.22	0.05	0.03	3.59	2.0	0.3	6.7	3.8

SUMMARY AND REMARKS

The grades of the lead and zinc concentrates in this test are low, but when it is considered that these concentrates were not cleaned and merely represent a rougher concentrate which would be passed to a cleaner cell in practice, the results obtained are very favourable. The only product it would be necessary to clean is the zinc concentrate, and the character of this concentrate indicates that there would be no difficulty in making a 40 to 45 per cent zinc concentrate.

Report No. 242

THE CONCENTRATION OF THE COPPER-GOLD ORE ("F" ORE-BODY) OF THE NORANDA MINES LIMITED, ROUYN, QUEBEC

C. S. Parsons and J. S. Godard

Shipments. Two shipments of ore were received from Noranda Mines, Ltd., Rouyn, Quebec. Lot No. 1, weight 118 pounds, was received September 9, 1925; and Lot No. 2, weight 1,500 pounds, January 27, 1926.

Characteristics and Analyses. The samples consisted of a heavy sulphide ore containing chalcopyrite, pyrite, pyrrhotite, and gold. The chalcopyrite is disseminated, and fine grinding is necessary to free it. Analyses of the two lots were as follows:—

Lot No. 1	Copper.....	1.55 per cent.	Arsenic.....	0.03 per cent
	Gold.....	0.25 oz./ton	Iron.....	50.39 "
	Silver.....	0.28 "	Sulphur.....	39.86 "
	Lead.....	trace.	Silica.....	2.93 "
	Zinc.....	0.28 per cent.		
Lot No. 2	Copper.....	1.6 per cent.	Gold.....	0.3 oz./ton

Purpose of Experimental Work. The samples were submitted for the purpose of determining whether this type of ore could be concentrated. A concentrate was desired which would contain the gold and copper, the

greater portion of the pyrite and pyrrhotite to be eliminated in a tailing product; also, if possible, to determine with what sulphides the gold is associated.

LOT No. 1

Average head sample for tests: Au 0.25 oz./ton; Ag 0.28 oz./ton; Cu 1.55 per cent; Fe 50.11 per cent.

Flotation Tests

Test No.	Product	Weight, per cent	Assays				Per cent of values			
			Au oz./ton	Ag oz./ton	Cu per cent	Fe per cent	Au	Cu	Ag	Fe
1	Concentrate.....	58.2	0.48	0.47	2.50	46.87	95.5	96.2	92.9	54.3
	Tailing.....	41.8	0.03	0.05	0.14	54.98	4.5	3.8	7.1	45.7
2	Concentrate.....	19.2	0.72	0.66	6.76	44.12	60.8	81.7	51.2	16.9
	Tailing.....	80.8	0.11	0.15	0.36	51.41	39.2	18.3	48.8	83.1
3	Concentrate.....	21.5	0.74	0.70	6.30	44.27	62.8	87.3	51.7	19.0
	Tailing.....	78.5	0.12	0.18	0.25	51.71	37.2	12.7	48.3	81.0
4	Concentrate.....	51.6	0.43	0.45	2.80	46.00	90.2	93.7	80.0	47.3
	Tailing.....	48.4	0.05	0.12	0.20	54.57	9.8	6.3	20.0	52.7
5	Concentrate.....	49.7	0.42	0.44	2.92	45.95	91.3	92.6	78.5	45.7
	Tailing.....	50.3	0.04	0.12	0.23	54.01	8.7	7.4	21.5	54.3
6	Concentrate.....	51.4	0.4	2.74	87.5	89.75
	Tailing.....	48.6	0.06	0.26	12.4	10.25
7	Concentrate.....	38.1	0.54	2.72	78.9	67.0
	Table concentrate.....	4.2	0.48	7.8
	Tailing.....	57.7	0.06	13.3
8	Concentrate.....	46.0	0.42	2.84	81.8	83.0
	Tailing.....	54.0	0.08	0.50	18.2	17.0
9	Concentrate.....	34.5	0.44	3.26	67.8	84.3	Coarse grinding
	Tailing.....	65.5	0.11	0.32	32.2	15.7
10	Concentrate.....	45.4	0.46	3.03	79.2	90.6
	Tailing.....	54.6	0.10	0.26	20.8	9.4
11	Concentrate.....	47.3	0.42	2.64	86.0	82.2
	Tailing.....	52.7	0.06	0.51	14.0	17.8
12	Concentrate.....	36.7	0.53	2.62	78.1	64.0
	Tailing.....	63.3	0.07	0.86	21.9	36.0
13	Concentrate.....	53.0	0.44	2.69	87.5	90.5
	Tailing.....	47.0	0.07	0.32	12.5	9.5
14	Concentrate.....	60.6	0.40	2.18	91.3	91.8
	Tailing.....	39.4	0.06	0.30	8.7	8.2
15	Concentrate.....	17.6	0.85	4.75	51.7	56.0
	Tailing.....	82.4	0.17	0.80	48.3	44.0
16	Concentrate.....	29.3	0.65	3.61	79.2	70.2
	Tailing.....	70.7	0.07	0.64	20.8	29.8
17	Concentrate.....	8.14	1.10	13.98	39.4
	Tailing.....	91.26	0.15	0.38
18	Concentrate.....	36.5	0.52	3.62	73.0	85.6
	Tailing.....	63.5	0.11	0.35	27.0	14.4
19	Concentrate.....	22.5	0.51	3.43	44.0	51.4
	Tailing.....	77.5	0.19	56.0

Reagents:

Test No.	Reagent	Amount lb./ton	Added to	Remarks		
1	Soda ash.....	5.0	Ball mill....	Froth good.		
	Xanthate.....	0.50	Cells.....			
	Pine oil (Century).....	0.12	".....			
2	Lime.....	5.0	Ball mill....	Bubbles small. Froth not good.		
	QED.....	0.10	Cells.....			
	Pine oil No. 5.....	0.08	".....			
	Cresylic acid.....	0.04	".....			
3	Lime.....	5.0	Ball mill....	Bubbles small, improve- ment over test No. 2.		
	Xanthate.....	0.20	Cells.....			
	Pine oil No. 5.....	0.08	".....			
4	Soda ash.....	4.0	Ball mill....	Froth good.		
	Thiocarbamilide.....	0.20	".....			
	Pine oil No. 5.....	0.08	Cells.....			
5	Sodium hydroxide.....	3.00	Ball mill....	Froth good.		
	F.P.L. No. 26.....	0.25	".....			
	Pine oil No. 5.....	0.04	Cells.....			
	Cresylic acid.....	0.04	".....			
6	Soda ash.....	5.0				
	Coke-oven tar.....	1.0				
	Cresylic acid.....	0.2				
	Water-gas tar.....50%	0.2				
	Coal-tar creosote.....50%					
7	Soda ash.....	6.0		Tabled tailing.		
	Water-gas tar.....	0.1				
	Cresylic acid.....	0.16				
	Water-gas tar.....50%	0.16				
	Coal-tar creosote.....50%					
8	Soda ash.....	8.0				
	Coke-oven tar.....	0.1				
	Cresylic acid.....	0.12				
	Water-gas tar.....50%	0.12				
	Coal-tar creosote.....50%					
9	Soda ash.....	8.0				
	Coke-oven tar.....	0.4				
10	Soda ash.....	8.0				
	Coke-oven tar.....	0.2				
11	Soda ash.....	6.0				
	Kiln residue.....	0.08				
	Barrett's No. 4.....	0.4				
12	Soda ash.....	6.0				
	P. T. & T. Co. No. 400.....	0.08				
	Barrett's No. 4.....	0.4				
13	Soda ash.....	6.0				
	Barrett's No. 4.....	0.4				
	Cresylic acid.....	1.0				
14	Soda ash.....	6.0				
	B-14 wood tar.....	0.1				
	Barrett's No. 4.....	0.3				
	Cresylic acid.....	0.5				
15	Soda ash.....	3.5		Cleaned concentrate. Added soda ash at end of test.		
	Barrett's No. 4.....	0.8				
	Water-gas tar.....	0.08				
	F.P.L. No. 31.....	0.12				

Reagents: continued.

Test	Reagent	Amount lb./ton	Added to	Remarks
16	Soda ash.....	8.0		
	Cyanide.....	0.4		
	Barrett's No. 4.....	0.24		
	Water-gas tar.....	0.08		
	F.P.L. No. 31.....	0.12		
17	Lime.....	8.0		Poor froth—added soda ash—no improvement.
	Barrett's No. 4.....	0.24		
	Water-gas tar.....	0.08		
	Cresylic acid.....	0.12		
18	Soda ash.....	6.0		
	Sodium hydroxide (40% solution).....	16.0		
	Barrett's No. 4.....	0.24		
	Water-gas tar.....	0.08		
	Cresylic acid.....	0.08		
19	Soda ash.....	6.0		
	Sodium hydroxide solution.....	16.0		
	Barrett's No. 634.....	0.24		
	Water-gas tar.....	0.08		

Magnetic Separation: A sample, 1,718 grammes, was crushed to -80 mesh and passed through a magnetic separator:

Magnetic.....	762 grammes	45.4 per cent
Non-magnetic.....	918	54.6 "

The non-magnetic portion, 918 grammes, was ground with lime, 3 pounds per ton, and floated using xanthate 0.10, and pine oil No. 5, 0.04 pound per ton. The concentrate was cleaned.

Products	Weight, per cent	Assay				Per cent of values		
		Au oz./ton	Ag oz./ton	Cu per cent	Fe per cent	Au	Ag	Cu
Magnetic.....	45.4	0.07	0.13	0.62	57.73	12.0	22.0	19.1
Concentrate.....	7.2	1.66	0.86	12.04	40.28	44.5	23.1	58.9
Middling.....	10.2	0.41	0.46	1.83	44.72	15.4	17.2	12.4
Tailing.....	37.4	0.20	0.27	0.38	46.45	28.1	37.7	9.6
Head from products.....		0.27	0.27	1.47	50.95			

LOT No. 2

Flotation Tests

Test No.	Product	Weight per cent	Assays		Per cent of values	
			Cu per cent	Au oz./ton	Cu	Au
1	Concentrate.....	31.4	4.77	0.76	90.4	85.2
	Tailing.....	69.1	0.11	0.06	9.6	14.8
2	Concentrate.....	29.1	5.16	0.72	94.6	74.8
	Tailing.....	70.9	0.12	0.10	5.4	25.2

LOT No. 2—Continued

Flotation Tests—Continued

Test No.	Product	Weight, per cent	Assays		Percent of values	
			Cu per cent	Au oz./ton	Cu	Au
3	Concentrate.....	26.8	5.42	0.76	92.5	79.6
	Tailing.....	73.2	0.16	0.07	7.5	20.4
4	Concentrate.....	22.9	6.50	0.98	94.1	80.06
	Tailing.....	77.1	0.12	0.07	5.9	19.94
5	Concentrate.....	32.9	4.88	0.64	95.25	79.2
	Tailing.....	67.1	0.12	0.08	4.75	20.8
6	Concentrate.....	29.7	5.04	0.56	95.0	59.6
	Tailing.....	70.3	0.11	0.16	5.0	40.4
7	Concentrate.....	32.2	4.58	0.64	75.2
	Tailing.....	67.8	0.12	0.10
8	Concentrate.....	32.1	4.57	0.60	70.3
	Tailing.....	67.9	0.16	0.12
9	Concentrate.....	30.4	4.69	0.70	67.0
	Tailing.....	69.6	0.19	0.15
10	Concentrate.....	28.9	4.82	0.74	73.2
	Tailing.....	71.1	0.19	0.11
11	Concentrate.....	42.0	3.41	0.53	83.1
	Tailing.....	58.0	0.23	0.07
12	Concentrate.....	40.6	3.56	0.57	94.5	83.1
	Tailing.....	59.4	0.14	0.08	5.5	16.9
13	Concentrate.....	26.8	5.13	0.68	90.0	86.4
	Tailing.....	73.2	0.21	0.04	10.0	13.6
14	Concentrate.....	49.4	3.0	0.40	97.5	76.3
	Tailing.....	50.6	0.07	0.12	2.5	23.7
15	Concentrate.....	21.8	6.12	0.78	87.8	59.4
	Tailing.....	78.2	0.24	0.15	12.2	40.6
16	Concentrate.....	24.6	5.45	0.59	63.1	65.7
	Tailing.....	75.4	1.04	0.10	36.9	34.3
17	Concentrate.....	21.3	5.53	0.67	55.4	72.1
	Tailing.....	78.7	1.21	0.08	44.6	27.9
18	Concentrate.....	23.9	5.76	0.64	64.7	71.5
	Tailing.....	76.1	0.99	0.08	35.3	28.5
19	Concentrate.....	24.3	6.70	0.61	78.23	75.6
	Tailing.....	76.1	0.63	0.14	21.77	24.3
20	Concentrate.....	29.2	5.58	0.56	77.35	74.09
	Tailing.....	70.8	0.67	0.08	22.65	25.91
21	Concentrate.....	32.3	6.38	0.54	94.0	81.0
	Tailing.....	67.7	0.19	0.06	6.0	19.0
23	Concentrate.....	46.5	4.50	0.46	96.5	78.5
	Tailing.....	53.5	0.14	0.11	3.5	21.5

Test No. 1—1,000 grms. at -50 mesh, 15 minutes in ball mill.

Soda ash	3 grms.
Barrett's No. 4	0.2 gm.
Cresylic acid	0.2 "

" 2—1,000 grms. at -50 mesh, 15 minutes in ball mill.

Soda ash	3 grms.
Barrett's water-gas tar	5 drops
Cresylic acid	0.2 gm.

" 3—1,000 grms. at -14 mesh, 45 minutes in ball mill.

Soda ash	1.5 grms.
NaOH	400 " 40 per cent solution.
Barrett's No. 4	7 drops
Water-gas tar	2 "
Xanthate	0.5 gm.

" 4—1,000 grms. at -14 mesh, 45 minutes in ball mill.

Soda ash	3 grms.
Cyanide	0.1 gm.
Barrett's No. 4	7 drops.
Water-gas tar	2 "
Xanthate	0.5 gm. added to cell.

Added 2 c.c. copper sulphate, produced nice copper float.

" 5—1,500 grms. at -14 mesh, 60 minutes in ball mill.

Soda ash	4 grms.
Cyanide	0.15 gm.
Barrett's No. 4	11 drops
Water-gas tar	3 "
Xanthate	0.15 gm. added to cell.
Copper sulphate	3 c.c.

" 6—As in No. 5, except that hardwood tar was used in place of water-gas tar and double charge of balls used.

" 7—1,000 grms at -14 mesh, 45 minutes in ball mill.

Soda ash	2.5 grms.
Water-gas tar	60%
Cresylic acid	20%
Coal-tar creosote	20%

12 drops

" 8—Soda ash 2.5 grms.

Cresylic acid 15 drops

" 9—Soda ash 2.5 grms.

Coal-tar creosote	50%
Coal tar	50%

12 drops

" 10—Soda ash 2.5 grms.

Dupont flotation oil 12 drops.

" 11—Soda ash 2.5 grms.

P.T. & T. No. 400 12 drops.

" 12—1,000 grms. ore, 45 minutes in mill.

Soda ash	3.0 grms.
P.T. & T. No. 400	6 drops.
Xanthate	0.1 gm. added to cells.

" 13—1,000 grms. ore, 45 minutes in mill.

Soda ash.	3.0 grms.
P.T. & T. No. 400	6 drops.
Xanthate	0.1 gm. added to cells.
Cyanide	0.5 gm. added to ball mill.
Copper sulphate	2 c.c.

Test No. 14—1,000 grms. ore, 45 minutes in mill.

Soda ash	3 grms.
Dupont's creosote	6 drops.
Cresylic acid	8 drops

" 15—1,000 grms. ore, 45 minutes in mill.

Soda ash	3 grms.
Dupont's creosote	8 drops.
Cresylic acid	8 drops
Cyanide	0.5 gm.
Xanthate	0.1 gm. added to cells.

" 16—1,000 grms. ore, 60 minutes in mill (—20 mesh)

Barrett's No. 4	5 drops.
Cresylic acid.	5 drops.
Soda ash	3 grms.

" 17—1,000 grms. ore, 60 minutes in mill (—20 mesh)

Soda ash	3 grms.
Dupont's creosote	5 drops
Cresylic acid	5 drops.

" 18—1,000 grms. ore, 60 minutes in mill (—20 mesh)

Water-gas tar	60%	} 10 drops.
Cresylic acid	20%	
Coal-tar creosote	20%	

" 19—1,000 grms. ore, 60 minutes in mill (—20 mesh).

Mixture as in	
No. 18.	10 drops
Soda ash	2 grms.
Cyanide	0.15 gm.

" 20—1,000 grms. ore, 60 minutes in mill (—20 mesh).

Mixture as in	
No. 18	10 drops
Soda ash	3 grms.

" 21—1,000 grms. ore, 60 minutes in mill (—20 mesh).

Mixture as in	
No. 18	10 drops
Soda ash.	3 grms.
Cyanide	0.15 gm.

After running few minutes added 1 c.c. copper sulphate—copper-coloured froth came up.

" 23—1,000 grms. ore, 20 minutes in mill (—20 mesh).

Soda ash	3 grms.
Cyanide	0.2 gm.
Mixture as in	
No. 18	10 drops.
Xanthate	0.1 gm. added to cell.
Copper sulphate	1.5 c.c. added to cell.

CONCLUSIONS

From the results of the magnetic separation test it is evident that the pyrrhotite does not carry the gold. Attention is drawn to test No. 12 on Lot 1, where 78 per cent of the gold was recovered but only 64 per cent of the copper, and to the fact that when lime is used a high copper concentrate can be obtained with an excellent recovery of the copper, but without a corresponding recovery of gold. These results prove that the gold is not entirely associated with the copper.

One very important point was ascertained, namely, that the tests conducted on the ore immediately after it was received gave very good results (see tests Nos. 1-5, Lot 2), but as the ore became older the results fell off. This characteristic was very noticeable.

Special attention is drawn to the use of cyanide with soda ash. Tests Nos. 4, 5, 13, 15, 19, and 23 were run using cyanide. In the last three of these tests the ore was showing decided signs of oxidation. The results in tests Nos. 4, 5, and 13 are very encouraging, and warrant the conducting of a tonnage-scale test using these reagents.

The experience with this ore shows that the samples used for flotation tests must be selected from freshly mined ore and guarded in every possible way against oxidation.

Report No. 243

EXPERIMENTAL TESTS ON ORE FROM THE J & L MINE NEAR REVELSTOKE, B.C.

J. S. Godard

Shipments. A shipment of 70 pounds of ore was received on January 5, 1926, from the J & L mine near Revelstoke, B.C. It was submitted by the Porcupine Goldfields Development and Finance Co., Ltd., Montreal.

Characteristics of the Ore. The ore is complex and consists principally of the sulphides of arsenic, zinc, lead, and iron, with smaller quantities of the sulphides of copper and antimony. The gold is chiefly associated with the arseno-pyrite and pyrite and the silver with the galena. The gangue is siliceous.

Sampling and Analysis. The entire lot was crushed to $\frac{1}{2}$ inch, cut once, reduced to -10 mesh and cut twice, reduced to -48 mesh and cut twice, then reduced to -150 mesh at which size the head sample was taken. Analysis was as follows:

Gold.....	0.52 oz./ton	Copper.....	0.15 per cent
Silver.....	3.50 "	Lead.....	6.0 "
Zinc.....	4.57 per cent	Iron.....	21.8 "
Arsenic.....	11.90 "	Antimony.....	0.22 "
Insoluble.....	16.38 "		

Purpose of Tests. The purpose of these tests was: (1) to ascertain if the gold could be extracted by cyanidation; (2) to separate the sulphides of lead, zinc, and arsenic.

Tests Nos. 1, 2, 3, and 4

Cyanidation. The purpose was (1) to ascertain if the gold and silver values could be extracted by cyanidation; (2) if finer grinding aided the extraction; (3) if water washing aided the extraction or reduced the cyanide consumption; and (4) if the addition of an oxidizing agent, sodium peroxide, made any appreciable difference in the extraction. In test No. 1, 500 grammes of the ore were dry crushed to -65 mesh. In test No. 2, 500 grammes were dry crushed to -150 mesh. In test No. 3,

500 grammes were wet crushed in a ball mill and screened through 150 mesh, the oversize returned for regrinding, until all passed 150 mesh. The pulp was dewatered before cyanidation. In test No. 4, 500 grammes of ore were crushed as in test No. 3, pulp dewatered. All four tests were cyanided in a 1:3 pulp, using 0.10 per cent KCN. Time of agitation was 48 hours. In test No. 4, 0.25 gramme sodium peroxide was added after 7 hours, and again after 31 hours' agitation.

Summary

Test No.	Heads assay, oz./ton		Tailing assay, oz./ton		Extraction, per cent		Reagent consumption lb./ton		
	Au	Ag	Au	Ag	Au	Ag	KCN	CaO	Na ₂ O ₂
1	0.52	3.50	0.49	2.41	5.8	31.2	5.04	11.40	None
2	0.52	3.50	0.49	3.04	5.8	13.2	5.84	10.48	None
3	0.52	3.50	0.50	3.04	3.8	13.2	5.08	6.60	None
4	0.52	3.50	0.50	3.10	3.8	11.4	5.40	5.40	2.00

CONCLUSIONS

1. The extractions obtained were almost negligible.
2. Grinding—Finer grinding made no practical difference in the gold extraction. The best silver extraction was obtained in test No. 1 when coarse crushing was used.
3. Water washing did not aid in the extraction. It reduced the cyanide consumption slightly in the 150-mesh material.
4. Oxidizer—The addition of sodium peroxide did not aid in the extraction.

In the raw state, this ore cannot be successfully treated by the cyanide process.

Test No. 5

Selective Flotation at 150 Mesh and Tabling the Tailings. The purpose of this test was to ascertain (1) if a practical separation of the lead, zinc, and arsenic could be made by selective flotation; (2) with which sulphide the gold and silver were associated; and (3) if tabling the flotation tailing would assist in the recovery.

A sample of 1,000 grammes of —48 mesh ore was ground to 150 mesh in a ball mill and selectively floated in a Ruth machine. The flotation tailing was tabled on a small Wilfley table.

Results:

Product	Weight, per cent	Assays					Per cent of values				
		Au oz./ton	Ag oz./ton	Pb percent	Zn percent	As percent	Au	Ag	Pb	Zn	As
Lead concentrate	16.6	0.46	15.72	20.67	18.36	7.07	14.2	61.1	54.8	25.6	9.9
Zinc "	22.0	0.37	3.20	4.77	27.97	7.63	15.2	16.5	16.7	51.7	14.2
Arsenic "	26.7	0.80	1.62	2.56	6.44	19.47	40.2	10.1	10.9	14.4	43.8
Table "	6.0	0.91	4.19	5.15	2.31	20.21	10.3	5.9	4.9	1.2	10.2
Table tailing.....	13.8	0.34	0.65	2.05	2.52	10.11	8.8	2.1	4.5	2.0	11.8
Slimes.....	14.9	0.40	1.24	3.46	3.32	8.00	11.3	4.3	8.2	4.2	10.1

CONCLUSIONS

(1) *Separation.* A rough separation can be obtained by selective flotation.

(2) *Association.* The gold is principally associated with the arsenopyrite, the silver with the galena.

(3) *Tabling of Flotation Tailing.* Tabling of the flotation tailing was effective in this test, largely on account of the poor results in the flotation of the arsenic. With improvement in the flotation of the arsenic, this step might be omitted.

Selective Flotation Tests

The purpose of these tests was to separate the lead and zinc, and the arsenic.

Test No.	Product	Weight, per cent	Assays					Per cent of values				
			Au oz./ton	Ag oz./ton	Pb per cent	Zn per cent	As per cent	Au	Ag	Pb	Zn	As
6	Lead concentrate....	13.5	0.52	16.72	19.92	18.21	7.00	14.6	61.4	39.2	21.6	7.2
	Zinc "	25.6	0.30	3.84	6.97	31.08	6.88	15.9	26.7	26.0	69.8	13.4
	Arsenic "	47.6	0.68	0.84	3.49	1.66	21.39	67.3	10.8	24.2	6.9	77.7
	Tailing.....	13.3	0.08	0.30	5.48	1.44	1.67	2.2	1.1	10.6	1.7	1.7
7	Lead concentrate....	11.1	0.48	20.80	23.80	20.12	5.58	9.5	62.0	36.8	19.7	4.8
	Zinc "	20.4	0.22	3.18	6.23	35.01	5.08	8.0	17.4	17.7	62.9	8.1
	Arsenic "	53.2	0.84	1.32	4.48	3.32	20.65	80.0	18.8	33.3	15.5	85.8
	Tailing.....	15.3	0.09	0.43	5.73	1.41	1.12	2.5	1.8	12.2	1.9	1.8
8	Lead concentrate....	10.7	0.54	15.86	20.32	20.79	6.30	11.4	47.4	36.0	19.5	5.3
	Zinc "	35.9	0.39	4.17	6.95	24.04	8.39	27.6	41.8	41.3	75.5	23.8
	Arsenic "	37.8	0.77	0.87	1.47	1.28	22.77	57.3	9.1	9.2	4.2	68.2
	Tailing.....	15.6	0.12	0.39	5.23	0.57	2.16	3.7	1.7	13.5	0.8	2.7
10	Lead concentrate....	6.4	0.62	7.38	11.00	14.53	9.67	8.2	13.7	11.7	8.1	4.7
	Zinc "	26.5	0.23	7.19	12.43	33.95	4.64	12.5	55.2	54.9	78.7	9.4
	Arsenic "	52.8	0.71	1.95	2.56	2.54	20.61	76.6	29.8	22.5	11.7	83.5
	Tailing.....	14.5	0.09	0.31	4.56	1.19	2.16	2.7	1.3	10.9	1.5	2.4
11	Lead concentrate....	32.7	0.48	8.17	11.35	18.34	10.62	30.4	76.0	65.5	53.6	28.4
	Zinc "	16.8	0.48	2.10	3.09	19.06	11.00	15.7	10.0	9.2	28.7	15.1
	Arsenic "	36.6	0.72	1.16	1.78	4.93	18.13	51.2	12.0	11.5	16.2	54.3
	Tailing.....	13.9	0.10	0.50	5.63	1.17	1.92	2.7	2.0	13.8	1.5	2.2
12	Lead concentrate....	15.7	0.52	10.40	13.75	18.55	9.98	15.2	46.8	36.7	25.9	11.7
	Zinc "	23.1	0.37	4.55	7.62	27.66	8.20	15.7	30.2	29.9	56.8	14.1
	Arsenic "	46.9	0.76	1.50	2.54	3.74	20.67	66.0	20.2	20.2	15.6	72.3
	Tailing.....	14.3	0.12	0.68	5.45	1.25	1.78	3.1	2.8	13.2	1.7	1.9
13	Bulk concentrate....	30.9	0.70	1.30	1.99	7.04	19.02	40.6	11.3	10.4	18.9	47.0
	Arsenic "	52.1	0.56	5.84	8.39	17.44	11.70	54.9	85.8	74.3	79.2	48.6
	Tailing.....	17.0	0.14	0.60	5.28	1.27	3.24	4.5	2.9	15.3	1.9	4.4
14	Lead concentrate....	23.7	0.54	10.32	14.29	17.30	10.68	23.3	66.3	56.2	35.6	18.5
	Zinc "	35.5	0.52	2.50	4.21	18.13	13.42	33.8	24.1	24.7	55.9	34.7
	Arsenic "	29.3	0.76	1.04	1.99	2.80	21.24	40.7	8.3	9.7	7.1	45.4
	Tailing.....	11.5	0.10	0.43	4.96	1.40	1.59	2.2	1.3	9.4	1.4	1.4

CONCLUSIONS

The flotation of the sulphides was not satisfactory, poor recoveries and poor separations being made in each test. The ore is slightly oxidized and this interfered with the flotation.

Report No. 244

EXPERIMENTAL TESTS ON ARSENICAL GOLD ORE FROM MOUNT EVELYN MINES, HUDSON BAY MOUNTAIN, B.C.

J. S. Godard

Shipments. A shipment of ore, weighing 55 pounds, was received October 8, 1925, from the property of the Mount Evelyn Mines Limited, situated on the Hudson Bay mountain, Omineca mining division, B.C. Mr. P. J. Job of Vancouver was the shipper.

Characteristics of the Ore. The sample submitted was an oxidized arsenical gold ore in siliceous gangue.

Sampling and Analysis. The entire sample was crushed to —14 mesh and cut four times in a Jones sampler. One cut was reduced to —50 mesh and then cut to 1,500 grammes. This was crushed to —150 mesh and the head sample cut. The analysis was as follows:

Gold.....	0.46 oz./ton	Copper.....	0.18 per cent
Silver.....	0.25 "	Lead.....	Nil
Zinc.....	0.30 per cent	Ni+Co.....	0.08 "
Iron.....	21.52 "	Arsenic.....	15.12 "
Antimony.....	Nil	Insoluble.....	35.20 "

Purpose of Experimental Tests. The purpose of these tests was to ascertain if the gold and arsenic could be recovered by concentration and if the gold could be recovered as base bullion, by amalgamation or cyanidation either with or without roasting.

Test No. 1

Tabling Sized Ore. The purpose of this test was to ascertain what recovery of the gold and arsenic could be obtained by tabling the sized ore. A sample of 2,970 grammes was cut from the —14-mesh material and dry screened on 35 and 80 mesh, the various sizes being tabled separately.

Screening:

Mesh	Weight, grms.	Weight, per cent
—14+35.....	1,500	50.6
—35+80.....	679	22.9
—80.....	784	26.5

Tabling:

Size	Product	Weight, per cent	Assay		Per cent of values	
			Au oz./ton	As per cent	Au	As
-14+35.....	Concentrate.....	28.7	0.68	33.41	37.5	63.4
	Middling.....	15.4	0.44	12.19	13.1	12.4
	Tailing.....	55.0	0.46	6.56	49.4	24.2
-35+80.....	Concentrate.....	36.5	0.68	32.45	45.7	76.8
	Middling.....	45.1	0.48	5.95	39.8	17.3
	Tailing.....	18.4	0.43	4.96	14.5	5.9
-80.....	Concentrate.....	23.6	0.72	36.42	28.3	52.3
	Middling.....	8.1	0.77	24.62	10.3	12.1
	Tailing.....	54.2	0.54	8.38	48.8	27.6
	Slimes.....	14.1	0.54	9.36	12.6	8.0
Combined results....	Concentrate.....	29.1	0.69	33.76	36.7	63.3
	Middling.....	20.3	0.49	10.30	18.1	13.4
	Tailing and slimes....	50.6	0.49	7.14	45.2	23.2

Weight of feed..... 2,963.0 grms.
Loss..... 65.9 "

Weight of products..... 2,897.1 grms.
Per cent loss..... 2.2

Ratio of concentration..... 1:3.5

Conclusions. The recoveries of both gold and arsenic were poor in all three sizes, since in tabling, the oxidized part passed into the tailing. This oxidized material contained considerable quantities of both gold and arsenic. To substantiate this the tailings were passed through an Ullrich magnetic separator, and the magnetic and non-magnetic products were assayed for gold and arsenic with the following results:

Size	Product	Weight, per cent	Assays		Per cent of values	
			Au oz./ton	As per cent	Au	As
-14+35	Magnetic.....	36.0	0.65	10.66	53.3	65.0
	Non-magnetic.....	64.0	0.32	3.22	46.7	35.0
-35+80	Magnetic.....	49.0	0.67	9.61	79.1	89.3
	Non-magnetic.....	51.0	0.17	1.11	20.9	10.7
-80	Magnetic.....	41.6	0.72	11.97	57.4	57.4
	Non-magnetic.....	58.4	0.38	6.32	42.6	42.6

Test No. 2

Amalgamation at 48 mesh. The purpose of this test was to ascertain if the ore was amenable to amalgamation and if so, to what extent.

A sample of 1,000 grammes of —14-mesh ore was ground wet to 48 mesh and amalgamated. The tailing was screened and the products assayed:—

Product	Weight, per cent	Assay Au oz./ton	Per cent Au
+48+65.....	8.5	0.35	6.6
+100.....	12.3	0.42	11.5
+150.....	21.3	0.45	21.1
+200.....	16.9	0.47	17.4
—200.....	41.0	0.48	43.4
Head sample.....		0.46	
Tailing average.....		0.45	

Amalgamation is ineffective on this ore.

Test No. 3

Flotation at 150 mesh. The purpose of this test was to ascertain if the gold and arsenic could be concentrated by flotation, at the same time obtaining a tailing so low in gold to permit its being discarded. A sample of 1,000 grammes of —14-mesh ore was ground to 150 mesh in a ball mill, then floated in a Ruth machine.

Product	Weight, per cent	Assays		Per cent of values	
		Au oz./ton	As per cent	Au	As
Concentrate.....	37.7	0.40	26.85	36.6	62.5
Tailing.....	62.3	0.42	9.65	63.4	37.5

Conclusions. The gold and arsenic cannot be successfully concentrated by flotation. The tailing was too high in both metals to permit its being discarded.

Tests Nos. 4 and 5

Cyanidation. The objects of these tests were (1) to ascertain if the gold in the ore could be extracted by cyanidation; (2) if fine grinding and water washing would aid in the extraction, and lower the cyanide consumption. In test No. 4 the ore was dry crushed to —65 mesh before cyanidation. In test No. 5 the ore was ground wet to —150 mesh and dewatered previous to cyanidation. The pulp dilution in both was 1:2, and the KCN strength was 0.10 per cent. Time of agitation was 48 hours.

Test No.	Assay		Extraction, per cent	Reagent consumption, lb./ton	
	Head	Tailing		KCN	CaO
4	0.46	0.25	45.6	8.0	31.6
5	0.46	0.20	66.5	5.4	30.0

Conclusions. Poor extractions were obtained in each test. Fine grinding and water washing aided the extraction and reduced the cyanide consumption.

TESTS ON ROASTED ORE

Roasting. A sample of 7,500 grammes of —14-mesh ore was roasted to drive off the arsenic and sulphur.

Weight before roasting.....	7,500 grammes		
Weight after roasting.....	5,215	"	
Loss.....	2,285	"	= 30.5 per cent
Analysis of roasted ore: Gold.....	0.69 oz./ton	Arsenic.....	0.18 per cent
Silver.....	0.37	Sulphur.....	0.34

Test No. 6

Water Washing. The purpose of the test was to determine the loss in weight due to water washing the roasted ore. A sample of 300 grammes of —14-mesh ore was agitated for 5 hours with 900 c.c. water. The pulp was filtered, dried, and weighed.

Weight before washing.....	300 grammes		
Weight after washing.....	298	"	
Loss.....	2	"	= 0.7 per cent

Test No. 7

Amalgamation at —28 mesh. The purpose of this test was to determine if the roasted ore could be successfully amalgamated. A sample of 1,000 grammes was ground to pass 28 mesh, then amalgamated. Lime equivalent to 10 pounds per ton was added to the pulp. The tailing was screened and the products assayed:—

Mesh	Weight, per cent	Assay oz./ton	Per cent of values
+ 35	2.8	0.18	0.7
+ 48	18.4	0.40	10.4
+ 65	19.4	0.66	18.5
+100	12.5	0.72	12.8
+150	14.5	0.74	15.2
+200	4.0	0.82	4.7
—200	28.0	0.95	37.7

Head sample before amalgamation.....	0.69 oz./ton
Tailing.....	0.70

Conclusions. The roasted ore is not amenable to amalgamation. The gold is largely in the finer sizes.

Tests Nos. 8, 9, 10, and 11

Cyanidation. The purpose of these tests was to determine (1) if the roasted ore could be successfully cyanided; (2) if water washing decreased

the cyanide consumption; and (3) if fine grinding aided extraction. The procedure in these tests is best illustrated by the following tables:—

Test No.	Mesh	KCN per cent	Pulp dilution	Treatment of ore
8.....	—14	0.10	1:3	Dry crushed. Water washed. Wet crushed and water washed.
9.....	—150	0.10	1:3	
10.....	—14	0.10	1:3	
11.....	—150	0.10	1:3	

Time of agitation—48 hours.

The tailings from tests Nos. 8 and 10 were screened on 100 mesh and the products assayed:—

Test No.	Mesh	Weight, per cent	Assay	Per cent of values	Remarks
8.....	+100	66.5	0.16	54.1	Average tailing 0.20 oz./ton.
	—100	33.5	0.27	45.9	
10.....	+100	67.6	0.16	57.2	Average tailing 0.19 oz./ton.
	—100	32.4	0.25	42.8	

Results:

Test No.	Assay		Extraction, per cent	Reagents consumed, lb./ton	
	Head	Tailing		KCN	CaO
8.....	0.69	0.20	71.0	1.5	7.7
9.....	0.69	0.19	72.5	2.2	8.0
10.....	0.69	0.19	72.5	1.4	4.0
11.....	0.60	0.17	75.4	2.8	7.8

Conclusion. The highest extraction was 75 per cent. Water washing decreased cyanide consumption in test No. 10 and increased it in test No. 11. The variation either way was small. Fine grinding did not aid in the extraction.

Test No. 12

Chlorination of roasted ore at —14 mesh. To determine if the gold could be chlorinated. A sample of 300 grammes of roasted ore at —14 mesh was chlorinated in a bottle for 16 hours. The pulp density was 1:0.75.

Summary:

Assay		Extraction, per cent	Reagents used, lb./ton	
Head	Tailing		H ₂ SO ₄	Bleaching powder
0.69	0.10	85.6	70	167

The bleaching powder was of poor quality, 22.8 per cent Cl, and necessitated more sulphuric acid than would be required when using higher grade bleaching powder. The extraction of the gold, 85.6 per cent, was the highest yet obtained.

Test No. 13

Cyanidation by percolation on roasted ore at -14 mesh. To determine what extraction of the gold could be obtained by allowing a strong solution of KCN to percolate through the roasted ore. A sample of 300 grammes of ore at -14 mesh was placed on a filter. It was first washed with 0.05 per cent KCN, then two litres of 0.25 per cent KCN were allowed to percolate through the ore each day. Air was drawn through after each percolation and the solution returned. The time was 5 days.

Head.....	0.69 oz./ton
Tailing.....	0.14 "
Extraction.....	80 per cent

Conclusions. The above treatment extracted 80 per cent of the gold. This compares favourably with 75 per cent extraction obtained by agitation in test No. 11.

Test No. 14

Chlorination at 65 mesh. To determine if chlorination at 65 mesh was more effective than at -14 mesh. Procedure as in test No. 12, excepting that ore was ground to -65 mesh.

Summary:

Assay		Extraction, per cent	Reagents used, lb./ton	
Head	Tailing		H ₂ SO ₄	Bleaching powder
0.69	0.10	85	170	82

Conclusion. Chlorination at -14 mesh was equally as effective as at -65 mesh.

Test No. 15

Cyanidation at 200 mesh. To determine what extraction of the gold could be obtained from ore ground to -200 mesh and agitated for 72 hours in an 0.25 per cent KCN solution. A sample of 500 grammes of roasted ore was ground to -200 mesh in a ball mill, dewatered, and cyanided. Air was blown through the pulp for $\frac{1}{2}$ hour each day.

Summary:

Assay		Extraction, per cent	Reagents consumed, lb./ton	
Head	Tailing		KCN	CaO
0.79	0.14	79.6	2.6	1.9

Conclusion. Long treatment of the finely ground ore in strong cyanide solution recovered only 79.6 per cent of the gold. This is a small improvement over test No. 11.

ADDITIONAL CONCENTRATION TESTS ON RAW ORE

Test No. 16

Flotation and Tabling. The purpose was to determine what percentage of the gold and arsenic could be recovered in the concentrates by flotation followed by tabling the flotation tailing. A sample of 1,000 grammes of ore at —14 mesh was ground to 100 mesh in a ball mill and floated in a Ruth machine. The tailing was tabled on a laboratory Wilfley table.

Summary:

Product	Weight, per cent	Assay		Per cent of values	
		Au oz./ton	As per cent	Au	As
Flotation concentrate.....	39.6	0.78	30.27	57.8	73.1
Table concentrate.....	1.8	0.76	14.44	2.6	1.6
Table middling.....	4.5	0.65	10.43	5.4	2.9
Table tailing.....	40.5	0.34	6.11	25.8	15.0
Slimes.....	13.6	0.33	8.90	8.4	7.4

Recoveries: Gold.....60.4 per cent Arsenic.....71.7 per cent

Test No. 17

Sizing, Tabling, Flotation, and Cyanidation. The purpose of this test was to determine what recovery of the gold and arsenic could be obtained by this treatment. A sample of 1,800 grammes of ore at —14 mesh was screened on 35 mesh and the oversize crushed to —35 mesh. The —35 mesh material was screened on 100 mesh. The —35+100 material was tabled, the tailing reground to —100 mesh, united with the —100-mesh material from sizing, and floated in a Ruth machine. The flotation tailing was tabled and a portion of the table tailing cyanided.

Summary:

Products	Weight, per cent	Assay		Per cent of values	
		Au oz./ton	As per cent	Au	As
Table concentrate.....	21.2	0.65	33.68	25.8	41.7
Flotation concentrate.....	21.1	0.92	28.49	36.1	35.1
Table concentrate.....	1.4	0.93	17.31	2.4	1.4
Table middling.....	2.8	0.72	13.55	3.7	2.2
Table tailing.....	44.0	0.33	6.04	27.0	15.6
Slimes.....	9.5	0.29	7.31	5.0	4.0

A sample of 500 grammes of the tailing was dry crushed to —150 mesh and cyanided for 48 hours in 1:2 pulp, using 0.075 per cent KCN.

Assay		Extraction, per cent	Reagents consumed, lb./ton	
Head	Tailing		KCN	CaO
0.33	0.10	69.8	2.3	33.4

Distribution of the gold:

1. Table concentrate, coarse size.....	25.6 per cent
2. Flotation concentrate.....	36.1 "
3. Table concentrate from flotation tailing.....	2.5 "
4. In cyanide solution.....	18.8 "
5. Table middling.....	3.7 "
6. Table slimes.....	5.1 "
7. Cyanide tailing.....	8.2 "

Items 1-4 show recoveries of the gold which amount to 83 per cent.

Conclusions. By the above method of treatment 78.2 per cent of the arsenic and 64.3 per cent of the gold is recovered as concentrates, weighing 43.7 per cent of the total; 18.8 per cent of the total gold was extracted by cyanidation. The cyanide consumption amounted to 2.3 pounds per ton, which indicates that the cyanicides are largely removed by the concentration; 33.4 pounds lime per ton were used to maintain the necessary protective alkalinity.

Test No. 18

(A) *Sizing, Flotation, and Cyanidation.* The purpose of this test was to determine what recovery of the gold and arsenic could be obtained by the above method of treatment. A sample of 1,600 grammes of the ore was crushed to —35 mesh and screened on 100 mesh. The —35+100 product was tailed and the tailing and middling from the table reground to 100 mesh, mixed with the —100 mesh from the screen and all ground to 150 mesh for flotation in a Ruth machine. The flotation tailing was dewatered, sampled, and cyanided for 65 hours in 1:2 pulp, using 0.075 per cent KCN.

Summary:

Product	Weight, per cent	Assay		Per cent of values	
		Au oz./ton	As per cent	Au	As
Table concentrate.....	9.7	0.64	35.55	11.6	21.9
Flotation concentrate.....	28.5	0.87	28.62	45.8	51.7
Tailing.....	61.8	0.37	6.74	42.6	26.4

Flotation Reagents:

P.T. & T. Co. No. 1580.....	0.15 lb./ton to ball mill.
P.T. & T. Co. No. 350.....	0.15 "
Copper sulphate.....	0.06 lb./ton to cell.
Sulphuric acid.....	7.00 "
Xanthate.....	0.30 "
Pine oil No. 5.....	0.06 "

Cyanidation:

Assay		Extraction, per cent	Reagents consumed, lb./ton	
Head	Tailing		KCN	CaO
0.37	0.10	73.0	2.88	34.7

Distribution of Gold:

Table concentrate.....	11.5 per cent
Flotation concentrate.....	46.0 "
Cyanide solution.....	31.0 "
Final tailing.....	11.5 "

(B) *Cyanidation of Combined Concentrates from Test No. 18.* The purpose of this test was to ascertain if the gold in the combined concentrates could be extracted by cyanidation. Some of the table concentrate was ground to 150 mesh and mixed with some of the flotation concentrate and cyanided for 65 hours in 1:3 pulp, using 0.10 per cent KCN.

Summary:

Assay		Extraction, per cent	Reagents consumed, lb./ton	
Head	Tailing		KCN	CaO
0.84	0.47	44.0	19.6	25.3

Conclusions. By combining the table and flotation concentrates a product which weighs 38.2 per cent of the feed and assays 0.84 ounce gold per ton, and 30.38 per cent arsenic, is obtained. This represents 57.5 per cent of the gold and 73.6 per cent of the arsenic. Cyanidation of the flotation tailing increased the gold recovery by 31.0 per cent, making a total gold recovery of 88.5 per cent. The cyanide consumption was 3.88 pounds per ton and the lime 34 pounds per ton. Cyanidation of the combined concentrates was not satisfactory. The extraction of the gold was low, 44.0 per cent, and large amounts of reagents were consumed.

Test No. 19

Cyanidation of Acid Washed Ore. The purpose of this test was to ascertain if acid washing the ore previous to cyanidation would decrease the cyanide consumption or aid in the extraction. A sample of 600 grammes

of ore was ground to -150 mesh and agitated for 2 hours in 2 per cent sulphuric acid solution, filtered, washed, and cyanided for 65 hours in 1:2 pulp, using 0.075 per cent KCN.

Summary:

Assay		Extraction, per cent	Reagents consumed, lb./ton	
Head	Tailing		KCN	CaO
0.52	0.26	50.0	8.8	49.4

Acid washing previous to cyanidation did not decrease the cyanide consumption nor aid in the extraction.

Test No. 20

(A) *Flotation, Tabling, and Cyanidation.* To ascertain what percentage of the gold and arsenic could be obtained by the above method. A sample of 1,200 grammes of ore at -14 mesh was ground to 200 mesh and floated in a Ruth machine. The flotation tailing was dewatered and cyanided.

Results:

Product	Weight, per cent	Assay		Per cent of values	
		Au oz./ton	As per cent	Au	As
Flotation concentrate.....	36.2	0.85	29.21	59.0	69.9
Table concentrate.....	2.8	0.90	14.44	4.8	2.7
Tailing.....	61.0	0.31	6.79	36.2	27.4
Head sample from products.....		0.522	15.2		
Average combined concentrates.....		0.85	28.61		

Flotation Reagents:

Sulphuric acid.....	3.0 lb./ton to ball mill
Sodium sulphide.....	1.0 "
P.T. & T. Co. No. 1580.....	0.25 "
Pine oil No. 350.....	0.18 "
Xanthate.....	0.30 "
Copper sulphate.....	1.00 "
Sulphuric acid.....	7.50 lb./ton to cell
Copper sulphate.....	0.75 "
Pine oil No. 5.....	0.18 "
Xanthate.....	0.40 "

A sample of 542.3 grammes of the table tailing was cyanided for 48 hours in 1:2 pulp, using 0.075 per cent KCN. The cyanide tailing was screened on 200 mesh.

Mesh	Weight, per cent	Assay Au oz./ton	Per cent of values
+200	6.6	0.09	6.6
-200	93.4	0.09	93.4

Details:

Tailing, grms.	Solution, grms.	KCN per cent	Assay		Extraction, per cent	Reagents consumed	
			Head	Tailing		KCN	CaO
542.3	1,066	0.075	0.31	0.09	71.0	3.68	41.9

Conclusion. The recoveries were gold 89.5 per cent, arsenic 72.5 per cent.

The distribution of the gold was:

In concentrates.....	63.8 per cent
In KCN solution.....	25.7 "
In KCN tailing.....	10.5 "

The gold recovered was distributed as follows:

In flotation concentrate.....	65.9 per cent
In table concentrate.....	5.3 "
In KCN solution.....	28.8 "

(B) *Cyanidation of the Roasted Concentrates.* To ascertain what recovery of the gold could be made by cyanidation of the roasted concentrates. A sample of 415.3 grammes of the combined concentrates was roasted to drive off the arsenic and the sulphur, then cyanided:

Weight before roasting.....	415.3 grammes
Weight after roasting.....	245.0 "
Loss.....	170.3 " -41.0%

Assay of roasted product calculated 1.44 ounces per ton.

Details:

Roasted conc., grms.	Solution, grms.	KCN per cent	Assay		Extraction, per cent	Reagents, lb./ton		Time, hours
			Head	Tailing		KCN	CaO	
245.0	1,000	0.10	1.44	0.27	81.3	10.1	45.8	65

Conclusion. By cyanidation, 81 per cent of the gold in the roasted flotation concentrate can be extracted.

Summary (A) and (B). Distribution of gold on cyanidation of roasted concentrates.

In KCN from cyanidation of roasted concentrate.....	51.9 per cent
In KCN from cyanidation of table tailing.....	25.7 "
In tailing from cyanidation of roasted concentrate.....	11.9 "
In tailing from cyanidation of table tailing.....	10.5 "
Recovered from cyanidation of roasted concentrate.....	51.9 "
Recovered from cyanidation of table tailing.....	25.7 " } 77.6

CONCLUSIONS

The following conclusions are drawn from the tests conducted on this ore:

Amalgamation. Negative results were obtained on both crude and roasted ore.

Cyanidation. Crude ore—The cyanidation of the crude ore was not satisfactory. The extractions were low and the consumption of reagents excessive. Water washing of finely ground ore previous to cyanidation reduced the cyanide consumption and aided in the extraction, but results could not be considered satisfactory.

Roasted Ore. By agitation in cyanide solution 75 per cent of the gold was extracted; and 80 per cent by percolation. The reagent consumption was moderate.

Chlorination. This method extracted 85 per cent of the gold. It necessitates dead roasting, and the production of the chlorine would be costly.

Tabling. Tabling of the sized feed was not satisfactory. The oxidized portions which contained considerable fine gold passed into the tailing. The tailing assayed high in arsenic.

Flotation. Flotation alone produced poor results.

Flotation and Tabling. This recovered 60 per cent of the gold and 75 per cent of the arsenic.

Sizing, Tabling, Flotation, and Tabling. By this method of treatment 75 per cent of the gold and 78 per cent of the arsenic were recovered as concentrate.

Sizing, Tabling, Flotation and Cyanidation. By this method 57 per cent of the gold and 74 per cent of the arsenic are recovered as a combined concentrate amounting to 38 per cent of the original feed by weight and assaying 0.81 ounce gold per ton and 30.38 per cent arsenic. The tailing assayed 0.37 ounce gold per ton and 6.74 per cent arsenic. Cyanidation of this tailing recovered an additional 31 per cent of the gold, leaving a final tailing of 0.10 ounce gold per ton. The cyanide consumption was nearly 3 pounds per ton and the lime 35 pounds per ton. The recoveries obtained by this method of treatment amounted to 88.5 per cent of the gold and 74 per cent of the arsenic.

Flotation, Tabling, and Cyanidation. This method recovered 89.5 per cent of the gold and 72.6 per cent of the arsenic. It necessitates the disposal of the arsenic concentrates which contain 63.8 per cent of the gold. Roasting and cyanidation of the concentrate made a recovery of 81.3 per cent of the gold contained in the concentrates, making a total gold recovery of 77.7 per cent.

Report No. 245

CONCENTRATION OF LEAD-ZINC TAILINGS FROM THE DUMP AT THE
NOTRE DAME (TETREAULT) MINE, NOTRE-DAME DES
ANGES, QUEBEC

C. S. Parsons

Shipment. A shipment consisting of a carload of tailings was received at the laboratories, February 8, 1926, from the British Metal Corporation (Canada) Limited, Montreal, Que. The material was taken from the tailing dump at the Notre Dame (Tetreault) mine at Notre-Dame des Angles, Que.

Character of Material. The dump consists of table tailings produced during the operation of the mine by Messrs. Tetreault Bros., and when under lease to the Zinc Co., Ltd. The tailings are comparatively coarse and contain very little slime. They have not been affected to any extent by oxidation.

Purpose of Tests. The sample of the dump was submitted with a request that flotation tests be made to determine whether the zinc and lead, together with the small amounts of silver and gold present, could be economically recovered by this method of treatment. A flotation plant treating the run-of-mine ore is at present in operation at the mine and it was thought that the old tailings might be treated in the same plant, either mixed with the ore or treated in a separate unit which is already available.

Test No. 1

This is a large-scale tonnage test. The tailings were fed to a 4½-foot Hardinge mill charged with 1,800 pounds of steel balls. A selective separation of the lead and zinc was attempted. The pulp from the mill was delivered to an 8-cell Greenawalt mechanical agitation flotation machine. A lead concentrate was taken from one cell and the concentrate from the remaining cells was returned as a middling to the feed end of the machine. The tailing from the lead cells was pumped to a Callow flat-bottom unit consisting of two roughers operated in parallel and two cleaners in series. The tailing from each cleaner was combined and returned to the roughers.

Results:

Amount of material run.....	4,440 pounds
Total time of running.....	4 hours
Average feed per hour.....	1,110 pounds
Total lead concentrate produced.....	116 "
Total zinc concentrate produced.....	500 "
Heads.....	Lead..... 0.67 per cent Zinc..... 9.96 per cent
Lead concentrate (sample	" 15.34 "
whole run).....	Copper... 3.92 " Silver... 31.7 oz/ton
	Gold..... 0.20 oz./ton
Zinc concentrate (sample	Zinc..... 43.16 per cent Lead..... 0.35 per cent
whole run).....	
Zinc concentrate (sample	" 47.13 "
1 to 3 p.m.).....	" 0.35 "
Tailing (sample whole run)	" 0.35 "

*Reagents:**Lead Flotation:*

Soda ash.....	3.5 lb./ton
Acid coal-tar creosote.....	0.35 "
Sodium cyanide.....	0.20 "
Cresylic acid.....	0.15 "

Zinc Flotation:

Copper sulphate.....	1.5 "
Xanthate.....	0.3 "
Pine oil.....	0.15 "

Test No. 2

The same flow-sheet was used as in test No. 1, but particular attention was given to the operation of the lead cells in order to raise the grade of the lead concentrate.

Results:

Amount of material run.....	6,600 pounds
Total time of running.....	5.58 hours
Average feed per hour.....	1,182 pounds
Total lead concentrate produced.....	149 "
Total zinc concentrate produced.....	1,031 "
Heads.....	Lead..... 0.7 per cent Zinc..... 10.06 per cent
Lead concentrate (sample	" 15.34 "
10.50 -11.40 a.m.)....	Copper... 3.92 "
Lead concentrate (sample	Lead..... 22.81 "
11.40 -3.20 p.m.)....	Copper... 4.23 "
Zinc concentrate (sample	Zinc..... 49.09 "
10.50 -11.40 a.m.)....	Lead..... 0.14 "
Tailing (sample 10.50	" 0.40 "
-11.40 a.m.).....	" 0.06 "
Zinc concentrate (sample	" 47.38 "
11.40 -3.30 p.m.)....	" 0.44 "
Tailing (sample 11.40 a.m.	" 0.10 "
-3.30 p.m.).....	" 0.48 "

*Reagents:**Lead Flotation:*

Soda ash.....	2.0 lb./ton
Acid coal-tar creosote.....	0.35 "
Sodium cyanide.....	0.9 "
Caustic soda.....	0.06 "
Cresylic acid.....	0.6 "

Zinc Flotation:

Copper sulphate.....	1.5 "
Xanthate.....	0.20 "
Pine oil.....	0.15 "

Test No. 3

In this test the lead was floated in the Callow unit and the zinc in the Greenawalt machine. The object of the change was to try and raise the grade of the lead concentrate by the double cleaning action available in the Callow unit. It was necessary to use the Dorr classifier in closed circuit with the Hardinge mill in order to obtain an even distribution of

the feed to the two Callow cells. In such a short test considerable fine lead is trapped and held in the classifier circuit and there is not sufficient time for the circuit to build up, hence the weight of the lead concentrate will be less than in the preceding tests.

Results:

Amount of material run.....	6,640 pounds
Total time of running.....	5½ hours
Average feed per hour.....	1,208 pounds
Weight of lead concentrate.....	137 "
Weight of zinc concentrate.....	744 "

	Lead per cent	Zinc per cent
Heads assay.....	0.77	10.15
Lead concentrate (sample 11 a.m.—2 p.m.).....	15.03	7.67
Lead concentrate (sample 2 p.m.—3.30 p.m.).....	9.76	7.09
Zinc concentrate (sample 11 a.m.—2 p.m.).....	0.05	49.62
Tailing (sample 11 a.m.—2 p.m.).....	0.09	0.88
Zinc concentrate (sample 2 p.m.—3.30 p.m.).....	0.15	46.83
Tailing (sample 2 p.m.—3.30 p.m.).....	0.05	0.14

Reagents:

Lead Flotation:

	lb./ton
Soda ash.....	2.8
Acid coal-tar creosote.....	0.45
Sodium cyanide.....	0.10
Cresylic acid.....	0.15

Zinc Flotation:

Copper sulphate.....	1.5
Xanthate.....	0.25
Pine oil.....	0.04

SUMMARY AND CONCLUSIONS

The zinc can readily be recovered from the dump tailings by flotation. There was some difficulty at times in maintaining a froth which could be brought over the sides of the cells, but as a rule this condition could be adjusted by increasing the quantity of pine oil.

It is possible to obtain a lead concentrate containing over 30 ounces silver and 3 to 4 per cent copper, but low in lead, the best grade made was 22.8 per cent. A quantity of gangue floated with the lead and prevented a higher grade concentrate being obtained. It is possible that on a larger tonnage basis the concentrate can be held back in the cleaner cells and a better cleaning action obtained, thereby producing a higher grade concentrate.

Report No. 246

THE FINE GRINDING OF CALCITE CONTAINING A SMALL AMOUNT
OF GRAPHITE, FROM NEAR PERTH, ONT.

R. K. Carnochan

Shipment. Four shipments of calcite were sent in by M. Day Baldwin, of the Continental Industrial Minerals, Ltd., 719 Pine Avenue W., Montreal, from a property near Perth, Ont. The dates upon which these shipments were received, the number of bags in each, and the shipping weights, were as follows:

August 5, 1925, (35 bags).....	7,080 pounds
October 6, 1925.....	1 pound
October 12, 1925, (2 bags).....	420 pounds
October 23, 1925, (18 bags).....	2,050 pounds

Purpose of Tests. Fine grinding tests were desired on the calcite to determine if products could be prepared from it suitable for use in making paint, linoleum, rubber goods, etc.

Characteristics of the Calcite. The calcite was pure white and crystalline, and contained a small amount of graphite. The graphite was in small flakes.

Sampling and Analysis. A head sample was taken of the lot received on August 5. This sample upon analysis gave the following:

CaO.....	50.55 per cent	Fe.....	0.16 per cent
MgO.....	1.68 "	CaF ₂	0.95 "
SiO ₂	2.80 "	Al ₂ O ₃	trace
C.....	0.35 "	Mn.....	nil

EXPERIMENTAL TESTS

Grinding with Raymond Pulverizer and Gayco Separator. Twenty-three bags of the shipment received August 5 were ground in a No. 0000 Raymond pulverizer. The ground product from the Raymond was separated in a 30-inch Gayco air separator; the fines being finished product and the coarse being fed back as produced to the Raymond. This method of grinding gave a product 99.9 per cent —200 mesh. This product was slightly grey in colour and when mixed with linseed oil it became a dirty greenish grey. The product is therefore unsuitable for use in paints or linoleum.

Twelve bags of the lot received on August 5 were ground in a 4½-foot by 16-inch Hardinge ball mill to see what capacity this machine would have. It was intended to feed the mill product to the Gayco air separator and return the oversize of the separator to the mill. However, the mill discharge was so off colour that this test was not completed.

Removal of Graphite by Screening and Grinding Calcite with Pebble Mill and Gayco. A lot of 1,755 pounds of the shipment received on October 23 was crushed by passing it twice through a small jaw crusher and four times through a small set of rolls, a 35-mesh screen being used to remove the fines after each crushing.

-35 mesh, 1st pass.....	175 pounds
2nd "	408 "
3rd "	469 "
4th "	199 "
5th "	120 "
6th "	99 "
+35 mesh, 6th "	287 "

The +35 mesh contained considerable graphite and -35 of the 1st and 2nd passes contained graphite and were off colour. The -35 mesh of the 3rd, 4th, 5th, and 6th passes contained some graphite, but were otherwise white in colour. The -35 of the 3rd, 4th, 5th, and 6th passes were mixed and ground fine by means of a 4- by 6-foot pebble mill and 30-inch Gayco air separator, the method used being to charge-grind in the mill, and then separate in the Gayco, the oversize being charge-ground again and separated, this being continued until the oversize from the separator was a very small amount. The Gayco fines were 99.9 per cent -200 mesh and were very white in colour; when mixed with linseed oil they gave a light grey colour.

Removal of Graphite by Flotation and Grinding Calcite with Pebble Mill and Gayco. The graphite was removed by oil flotation from a sample weighing 840 pounds. This sample consisted of the -35 mesh, 1st and 2nd passes, and the +35 mesh, 6th pass of the previous test. After removal of the graphite, the calcite was dried and ground fine by means of the 4 by 6-foot pebble mill and treated in a 30-inch Gayco air separator. The method of grinding was the same as in the previous test.

The Gayco fines were 99.8 per cent -200 mesh and were very white in colour; when mixed with linseed oil they gave a light fawn colour.

CONCLUSIONS

The best way to treat this calcite would be to remove the graphite by oil flotation and then dry the cleaned calcite and grind in a pebble mill in closed circuit with an air separator. This would give a large percentage of the crude as very white, fine material.

If a finer product than that obtained by dry grinding is desired, the calcite, after the removal of the graphite, could be ground wet in a pebble mill in closed circuit with a Dorr bowl classifier. The ground pulp could be thickened, filtered, and the filter cake dried and reground.

Report No. 247

THE CONCENTRATION OF A COMPLEX GOLD AND SILVER ORE FROM ALBERNI, B.C.

C. S. Parsons

Shipment. A shipment of ore, weighing 70 pounds, was received from A. L. Smith, Alberni, B.C., on January 26, 1926.

Character and Analysis. The sample received was an oxidized ore containing gold, silver, copper, lead and zinc.

Analysis:

Copper.....	0.92 per cent
Zinc.....	6.54 "
Silver.....	3.52 oz./ton
Lead.....	1.00 per cent
Gold.....	0.70 oz./ton

The sample contained both iron and arsenical pyrites. The sulphides of iron had, in part, undergone oxidation and had been leached, leaving the ore in a porous and weathered condition. A part of the copper sulphide had been altered to carbonates. The oxidized condition of the sample indicated that the gold, on account of its insolubility had been concentrated in this weathered material.

Purpose of Experimental Tests. The following experimental work was undertaken in order to find a method of concentrating the ore.

EXPERIMENTAL TESTS

Test No. 1

This test was run with the idea of making a selective flotation separation between the copper, lead, and zinc, the copper and lead being floated in one concentrate and the zinc in another. The ore was ground with the following reagents to pass approximately 48 mesh:

Soda ash.....	10.0 lb./ton
Thiocarbamilide.....	0.2 "
Sodium cyanide.....	0.2 "

The flotation of a lead-copper concentrate was not successful, the zinc tending to float in large amounts. The soluble sulphates in the ore, owing to its oxidized condition, destroyed the cyanide and prevented its having a deadening effect on the flotation of the zinc sulphides.

Test No. 2

This test was run to produce a bulk concentrate of the copper, lead, and zinc. The ore was ground to approximately 48 mesh, and the following reagents used for flotation:

Soda ash.....	12.0 lb./ton
Water-gas tar.....	0.08 "

These were ground with the ore, and the following added directly to the flotation cell:

Xanthate.....	0.3 lb./ton
Copper sulphate.....	1.0 "
Sufficient pine oil to froth.	

The flotation concentrate was cleaned and the middling and tailing combined and tabled.

Test No. 3

This test was also run to produce a bulk concentrate, but under different conditions to test No. 2. The object in this test was to determine

the effect of using lime in place of soda ash on the silver and gold values. The ore was ground to approximately 48 mesh, and the following reagents used:

Lime (ground with ore).....	10.0 lb./ton
Xanthate (added to cell).....	0.3 "
Copper sulphate.....	1.0 "
Sufficient pine oil to froth.	

The flotation concentrate was cleaned and the middling and tailing combined and tabled.

Results:

Test No.	Product	Weight, per cent	Assays					Per cent of values				
			Cu per cent	Pb per cent	Zn per cent	Au oz/ton	Ag oz/ton	Cu	Pb	Zn	Au	Ag
1	Concentrate.....	18.5	4.24	5.15	33.54	3.32	18.40	85.0	98.5	94.7	73.7	83.3
	Tailing.....	81.5	0.16	0.02	0.42	0.27	0.71	15.0	1.5	5.3	26.3	14.7
2	Flotation concentrate.....	14.4	3.99	5.95	39.11	1.64	16.84	65.6	88.0	86.9	35.0	65.7
	Table concentrate...	5.1	0.70	0.75	1.40	5.16	5.50	4.1	3.8	1.1	38.7	7.6
	Table tailing.....	80.5	0.33	0.10	0.96	0.22	1.22	30.3	8.2	12.0	26.3	26.7
3	Flotation concentrate.....	13.7	5.14	7.12	41.90	4.64	23.36	73.8	96.3	80.4	33.9	77.5
	Table concentrate...	6.2	0.56	0.50	3.48	3.79	4.01	3.6	3.0	3.0	44.8	6.0
	Table tailing.....	80.1	0.27	0.06	1.48	0.14	0.35	22.6	0.7	16.6	21.3	16.5

SUMMARY AND CONCLUSIONS

The oxidized condition of the sample makes it practically impossible to effect a selective separation by flotation between the copper, lead, and zinc. If there is a large tonnage of this type of ore to be treated and a greater profit can be obtained by marketing a copper-lead concentrate and a zinc concentrate separately, it is possible that the bulk concentrate can be re-treated and a selective separation effected between these minerals by flotation in a separate unit. Such a separation remains to be worked out and it will be necessary to conduct some experimental work on a larger scale than used for these tests.

In cleaning the first flotation concentrate to raise the zinc content to a marketable grade, considerable gold is dropped. By the use of tables the greater part of this gold can be recovered in a concentrate of small bulk and high grade. This is shown clearly in tests Nos. 2 and 3.

The recoveries of gold and silver are not high, but, considering the oxidized condition of the ore, they can be considered as fairly satisfactory.

Report No. 248

THE RE-CONCENTRATION OF LOW-GRADE GRAPHITE CONCENTRATES
FROM BUCKINGHAM, QUEBEC

R. K. Carnochan

Shipments. The Crucible Graphite Company, Limited, Buckingham, Que., sent in two shipments. The first was received March 19, 1926, and consisted of ten bags, shipping weight, 1,000 pounds; five of these bags were marked 'PX' and five 'P12'. The second shipment was received May 6, 1926, and consisted of twenty bags marked 'PX', shipping weight, 2,000 pounds.

Purpose of Tests. The purpose of the test work was to determine an economical method of raising the grade of the concentrates so that they could be marketed more profitably.

Characteristics of the Low-grade Concentrates. The concentrates were produced in the mill of the Crucible Graphite Company. PX is the fines obtained when the dried flotation concentrate is screened on 150 mesh, and P12 the fines produced in bringing up the grade of the No. 2 flake by means of burr mill and screen. Both samples appeared to carry a lot of gangue and were very gritty when smoothed under a spatula. A grab sample of PX was 85.6 per cent —200 mesh, and, of P12, 86.8 per cent —200 mesh.

Sampling and Analysis. A sample was not taken of either of the full lots, but grab samples and samples of the feed to different tests were taken with the following results:

Lot PX

Grab sample.....	29.02 per cent C
One bag, feed to tests Nos. 1, 2, and 4.....	29.60 " C
1,200 pounds, feed to test No. 5.....	19.83 " C

Lot P12

Grab sample.....	45.86 per cent C
One bag, feed to test No. 5.....	29.52 " C

It will be seen from the above that the graphite content of these concentrates varies from bag to bag.

TESTS WITH GAYCO AIR SEPARATOR

These tests were made to see what effect a Gayco separator would have in bringing up the grade of the concentrates and in removing grit and sand. A 30-inch machine was used in all tests.

Lot P12, Test No. 1. About 100 pounds were put through the Gayco, the speed of the machine being 687 r.p.m.

Fines obtained.....	54 pounds	40.98 per cent C	92.6 per cent —200 mesh
Oversize.....	43 "	55.30 " C	67.8 " —200 mesh

Lot P12, Test No. 4. Another test similar to test No. 1 was made, the speed of the Gayco being 624 r.p.m.; the fines were re-run twice. Feed 90 pounds.

Fines obtained.....	49½ pounds	39.28 per cent	C	96.0 per cent	-200 mesh
Oversize obtained.....	37½ "	37.90 "	C	81.8 "	"
Fines, 1st re-run.....	15½ "	36.30 "	C	93.2 "	"
Oversize, 1st re-run.....	21 "	35.00 "	C	81.2 "	"
Fines, 2nd re-run.....	6 "	36.00 "	C	85.8 "	"
Oversize, 2nd re-run.....	14½ "	38.30 "	C	61.2 "	"
Clean-up.....	2 "				

Lot PX, Test No. 1. A test using the 30-inch Gayco was made on lot PX, the speed of the separator being 624 r.p.m.; the fines re-run twice.

Feed.....	88½ pounds	29.60 per cent	C	83.6 per cent	-200 mesh
Fines obtained.....	39 "	35.20 "	C	94.6 "	"
Oversize obtained.....	46 "	22.74 "	C	73.2 "	"
Fines, 1st re-run.....	13 "	30.36 "	C	88.8 "	"
Oversize, 1st re-run.....	32 "	22.82 "	C	68.8 "	"
Fines, 2nd re-run.....	6 "	26.48 "	C	89.2 "	"
Oversize, 2nd re-run.....	25 "	19.30 "	C	66.6 "	"
Clean-up.....	1½ "				

SUMMARY OF GAYCO TESTS

The three tests made demonstrate that the Gayco separator raised the grade slightly, and produced a finer product. Under a spatula, it was found that the fines of the first pass contained very little grit; the fines of the re-runs, a fair amount; and the oversize products, considerably more. The appearances of the products are all about the same. The fines are not very black in colour, and hence appear to carry considerable impurities.

CONCLUSIONS FROM GAYCO TESTS

The Gayco, or any similar separator, would be an unsuitable machine to raise the grade of these concentrates.

SMALL-SCALE FLOTATION TESTS

Lot P12, Test No. 3. A sample of 100 grammes of P12 was ground for 15 minutes with steel balls in a small porcelain jar with 0.25 c.c. coal oil and 0.1 c.c. pine oil. The ground charge was then floated in a small Ruth machine, the concentrates being cleaned.

Results

Product	Weight, grms.	Carbon, per cent	Carbon, grms.	Per cent of values
Concentrate.....	42.0	85.64	35.97	95.14
Middling.....	6.0	17.21	1.03	2.72
Tailing.....	51.4	1.57	0.81	2.14
Feed.....	99.4	38.04	37.81	100.00

Lot P12, Test No. 6. A sample of 100 grammes of P12 was ground for one minute with pebbles in a small porcelain jar mill with 0.25 c.c. coal oil and 0.1 c.c. pine oil. The ground charge was floated in a small Ruth machine, the concentrates being cleaned.

Results

Product	Weight, grms.	Carbon, per cent	Carbon, grms.	Per cent of values
Concentrate.....	40.9	80.37	32.87	83.79
Middling.....	10.0	31.95	3.19	8.13
Tailing.....	48.0	6.60	3.17	8.08
Feed.....	98.9	39.67	39.23	100.00

Lot PX, Test No. 2. A sample of 100 grammes of PX was ground for 15 minutes with steel balls in a small porcelain jar with 0.25 c.c. coal oil and 0.1 c.c. pine oil. The ground charge was floated in a small Ruth machine, the concentrates being cleaned.

Results

Product	Weight, grms.	Carbon, per cent	Carbon, grms.	Per cent of values
Concentrate.....	28.9	86.02	24.86	85.26
Middling.....	7.4	49.86	3.69	12.65
Tailing.....	64.4	0.95	0.61	2.09
Feed.....	100.7	28.96	29.16	100.00

A screen test on the concentrate shows it to be 95.42 per cent —200 mesh.

Lot PX, Test No. 4. A sample of 100 grammes of PX was ground for one minute with pebbles in a small porcelain jar mill with 0.25 c.c. coal oil and 0.1 c.c. pine oil. The ground charge was floated in a small Ruth machine, the concentrates being cleaned.

Results

Product	Weight, grms.	Carbon, per cent	Carbon, grms.	Per cent of values
Concentrate.....	32.4	68.18	22.09	80.68
Middling.....	11.2	11.81	1.32	4.32
Tailing.....	54.3	7.32	3.97	14.50
Feed.....	97.9	27.97	27.38	100.00

SUMMARY OF SMALL-SCALE FLOTATION TESTS

These tests show that excellent results can be obtained by flotation, and that the longer the charge is ground before flotation the higher the grade of concentrate and the greater the recovery.

MEDIUM-SCALE FLOTATION TESTS

Lot P12, Test No. 5. This test was made on a small, continuous unit consisting of feeder, ball mill, rougher and cleaner cells. This unit was manufactured by the General Engineering Company, and as supplied to us had a feeder, ball mill, classifier, and rougher cell. For this test work the classifier was not used, and an extra cell for cleaning was added. Both cells are of the Callow type. This test was divided into two parts; the feed in the first part being 16 pounds per hour, and in the second part $7\frac{1}{2}$ pounds per hour. Small amounts of coal oil and pine oil were fed during both parts of the test, the coal oil to the ball mill and the pine oil to the rougher cell.

Results of First Part

Product	Weight, lb.	Carbon, per cent	Carbon, lb.	Per cent of values
Concentrate.....	4.5	59.70	2.69	86.77
Tailing.....	4.5	9.05	0.41	13.23
Feed.....	9.0	34.44	3.10	100.00

Results of Second Part

Concentrate.....	45.0	47.28	21.28	94.91
Tailing.....	31.5	3.61	1.14	5.09
Feed.....	76.5	29.31	22.42	100.00

Lot PX, Test No. 3. This test was conducted in a similar manner to the second part of test No. 5 on P12. The rate of feed was 16 pounds per hour.

Results

Product	Weight, lb.	Carbon, per cent	Carbon, lb.	Per cent of values
Concentrate.....	37.5	56.80	21.30	97.35
Tailing.....	43.0	1.36	0.58	2.65
Feed.....	80.5	27.18	21.88	100.00

SUMMARY OF MEDIUM-SCALE TESTS

These two medium-scale tests give lower grade concentrates than any of the small-scale tests. This was due to the small size of the mill; the feed passed through the mill without being ground at all and, judging by the appearance of the mill discharge, without being thoroughly mixed. For this reason these tests are not to be considered as giving accurate results. However, they do show that without grinding and with poor mixing a fair concentrate and recovery can be made by flotation on both P12 and PX. These medium-scale tests indicate that the pneumatic type of flotation machine can be used for this material as well as the mechanical type.

LARGE-SCALE FLOTATION TESTS

Lot PX, Test No. 5. In this test, 150 pounds per hour of PX were fed to a 4½-foot by 16-inch Hardinge mill, with 450 pounds of water per hour and a small amount of coal oil. This gives a ratio of three of water to one of solids. The mill carried a ball load of 4,500 pounds. The ground mill discharge was pumped to an 8-cell Greenawalt flotation unit, each cell being 12 inches by 12 inches. Extra water was added at the pump at the rate of 1,085 pounds per hour, giving a dilution of 10.25:1 in the first flotation cell. A little pine oil was added to the feed of the first cell. The froth from the last six cells was returned to the feed of the first cell. The concentrate was re-run twice through the Greenawalt to clean it. A little pine oil was used during each run to give a froth.

Results

Product	Weight, lb.	Carbon, per cent	Carbon, lb.	Per cent of values
Concentrate.....	208.0	81.14	168.77	88.77
1st tailing.....	752.5	0.85	6.40	3.37
2nd tailing.....	29.0	20.43	5.92	3.11
3rd tailing.....	20.0	45.20	9.04	4.75
Feed.....	1,009.5	18.82	190.13	100.00

The following table shows the recovery and grade of concentrate obtained in the above test after each pass through the Greenawalt:

Product	Carbon, per cent	Per cent of values
1st concentrate obtained.....	71.49	96.63
2nd " ".....	77.99	93.52
3rd " ".....	81.14	88.77

Screen tests show the third concentrate to be 85.6 per cent —200 mesh, and the first tailing 99.60 per cent —200 mesh.

Lot PX, Test No. 6. A lot of 400 pounds of PX was ground for two hours in a 4½-foot by 16-inch Hardinge mill with 285 pounds of water and 4,000 pounds of balls. The ground charge was flushed from the mill with 960 pounds water per hour to a pump where 1,125 pounds of water per hour were added. The pump elevated the diluted mill discharge to the Greenawalt machine. The pulp was in the Greenawalt, a small amount of pine oil being added to the feed. The froth from the last six cells was returned to the feed of the first cell. The running time was three hours. This gives a calculated dilution of 15.5:1 in the feed to the first cell. The concentrate obtained was re-run twice through the same flotation machine to clean it, a little pine oil being used during each run.

Results

Product	Weight, lb.	Carbon, per cent	Carbon, lb.	Per cent of values
Concentrate.....	59.0	80.80	47.67	77.40
1st tailing.....	309.5	2.51	7.77	12.64
2nd tailing.....	17.5	9.95	1.74	2.88
3rd tailing.....	14.0	30.42	4.26	6.93
Feed.....	400.0	15.36	61.44	100.00

The following table shows the recovery and grade of concentrate after each pass through the Greenawalt:

Product	Carbon, per cent	Per cent of values
1st concentrate obtained.....	59.30	87.36
2nd " ".....	71.14	84.53
3rd " ".....	80.80	77.60

Screen test shows the third concentrate to be 93.1 per cent —200 mesh, and the first tailing to be 98.9 per cent —200 mesh.

SUMMARY OF LARGE-SCALE TESTS

Both these tests show excellent results and prove that lot PX can be treated on a large scale by flotation. Continuous grinding gave the best results, but this may have been due to the fact that no coal oil was used in the charge grinding. In theory, charge grinding on this kind of material should give equal or better results than continuous grinding, other conditions being the same. The more the ground graphite is floated, the higher the grade of the resulting concentrate and the lower the recovery.

RECOMMENDATIONS

It is advised that the Crucible Graphite Company install a fine wet screen, 80, 100, or 150 mesh, to treat their flotation concentrates, the plus going to the drier and then to the refinery, and the minus to a thickener and then to a continuous ball mill. The discharge of the mill would be floated, the tailing going to the dump, and the concentrate to a drier. The P12 from the refinery would be fed to the ball mill and thus ground and floated with the fines from the screen.

No recommendation is made as to the fineness of grinding, or the amount of floating to be done, as this would depend on what grade of concentrate could be readily sold.

Report No. 249

CONCENTRATION OF THE ZINC AND COPPER-ZINC ORES OF THE ALDERSON AND MACKAY MINE, ROUYN, QUEBEC

J. S. Godard

Shipment. A shipment of 140 pounds, consisting of two types of ore, was received April 8, 1926. The ores were from the Alderson and MacKay mine, Rouyn, Que., and were submitted by Messrs. Alderson and MacKay, New Birks Building, Montreal.

Characteristics of the Ores. Type No. 1 is known as disseminated zinc ore and consists of zinc blende in a siliceous gangue. Type No. 2 is designated as heavy sulphide ore and consists of chalcopyrite, zinc blende, pyrrhotite and pyrite in a siliceous gangue. A small quantity of galena is present, showing only traces of lead in the analysis. Small amounts of gold and silver are also present, mainly associated with the chalcopyrite.

Purpose of Experimental Tests. The purpose of the tests was to concentrate the zinc blende in the disseminated zinc ore, and to separate and concentrate the chalcopyrite and zinc blende in the heavy sulphide ore.

DISSEMINATED ZINC ORE

The entire lot of 45 pounds was crushed to $\frac{3}{4}$ inch and cut once; crushed to 14 mesh and cut three times; crushed to 50 mesh and cut twice, then reduced to -150 mesh before cutting the head samples. The analysis was:

Lead.....	Nil
Copper.....	"
Gold.....	"
Silver.....	0.04 oz./ton
Zinc.....	13.00 per cent
Iron.....	3.23 "
Insol.....	71.95 "

Test No. 1

Flotation at 100 Mesh

Product	Weight, per cent	Assay, Zn per cent	Per cent of values Zn	Reagents	Amount, lb./ton	Added to
Zinc concentrate	25.8	46.13	92.5	Soda ash.....	4.0	Ball mill.
Middling.....	8.9	5.47	3.8	Copper sulphate..	1.0	Flotation cell
Tailing.....	65.3	0.70	3.6	Xanthate.....	0.4	" "
				Pine oil No. 5.....	0.08	" "

Test No. 2

Flotation at 150 Mesh

Product	Weight, per cent	Assay, Zn per cent	Per cent of values Zn	Reagents	Amount, lb./ton	Added to
Zinc concentrate	23.2	48.92	89.8	Soda ash.....	5.0	Ball mill
Middling.....	11.1	8.90	7.8	Barrett's No. 634.	0.4	"
Tailing.....	65.7	0.46	2.4	TT.....	0.2	"
				Copper sulphate...	2.0	Flotation cell
				Pine oil.....	0.12	"
				Xanthate.....	0.05	"

No additional reagents were added in either test to clean the zinc concentrate.

HEAVY SULPHIDE ORE

The entire lot of 95 pounds was crushed to $\frac{1}{2}$ inch and cut twice; crushed to -50 mesh and cut twice; crushed to -150 mesh and head sample cut out. The analysis was:

Lead.....	trace
Copper.....	7.36 per cent
Gold.....	0.03 oz./ton
Silver.....	2.45 "
Zinc.....	6.53 per cent
Iron.....	35.48 "
Insol.....	18.6 "

A number of selective flotation tests were made on this ore, the results of which are tabulated below:—

Test No.	Product	Weight, per cent	Assays				Per cent of values			
			Cu per cent	Zn per cent	Au oz./ton	Ag oz./ton	Cu	Zn	Au	Ag
1	Cu concentrate.....	25.6	26.16	3.01	0.08	8.12	93.3	11.3	74.5	83.5
	Zn "	23.2	1.47	25.64	0.03	1.01	4.7	87.2	25.4	9.4
	Tailing.....	51.2	0.28	0.21	tr	0.35	2.0	1.5	7.1
2	Cu concentrate.....	30.0	23.08	4.58	0.08	7.60	95.6	20.9	80.0	89.0
	Zn "	20.0	1.01	25.17	0.03	0.75	2.8	76.5	19.9	5.9
	Tailing.....	50.0	0.23	0.34	tr	0.26	1.6	2.6	5.1
3	Cu concentrate.....	25.1	26.04	3.63	0.08	7.94	91.4	13.7	50.0	83.0
	Zn "	15.3	2.89	34.88	0.02	1.34	6.2	80.6	7.5	8.5
	Table "	5.5	0.13	0.31	0.11	0.35	0.1	0.3	15.0	0.8
	Tailing.....	54.1	0.30	0.67	0.02	0.34	2.3	5.4	27.5	7.7
4	Cu concentrate.....	28.0	23.44	3.43	0.08	7.86	91.8	14.4	72.7	85.2
	Zn "	12.0	2.89	35.19	0.02	1.20	4.9	63.2	7.8	5.6
	Zn middling.....	19.8	0.73	6.95	0.01	0.61	2.0	20.6	6.5	4.7
	Tailing.....	40.2	0.24	0.29	0.01	0.29	1.3	1.8	13.0	4.5
5	Cu concentrate.....	25.6	24.52	3.17	0.08	7.88	88.0	12.1	71.4	80.6
	Zn "	24.3	2.63	22.27	0.02	1.14	8.9	80.5	17.1	11.1
	Zn middling.....	17.0	0.63	2.08	tr	0.55	1.5	5.3	3.8
	Tailing.....	33.1	0.34	0.42	0.01	0.34	1.6	2.1	11.5	4.5

Test No.	Product	Weight, per cent	Assays				Per cent of values			
			Cu per cent	Zn per cent	Au oz./ton	Ag oz./ton	Cu	Zn	Au	Ag
6	Cu concentrate.....	27.2	25.68	3.06	0.08	8.06	97.0	13.0	66.1	86.8
	Zn ".....	9.2	1.02	48.71	0.02	1.09	1.3	70.0	5.4	3.9
	Zn middling.....	6.0	0.66	7.92	0.06	0.94	0.6	7.4	10.9	2.2
	Tailing.....	57.6	0.14	1.06	0.01	0.31	1.1	9.6	17.6	7.1
7	Cu concentrate.....	30.2	22.62	4.00	0.08	7.12	95.5	18.5	75.3	88.3
	Zn ".....	8.5	1.60	46.03	0.02	1.12	1.9	60.1	5.3	3.9
	Zn middling.....	13.6	0.81	7.93	0.01	0.52	1.5	16.5	4.4	2.9
	Tailing.....	47.7	0.17	0.67	0.01	0.25	1.1	4.9	15.0	4.9
8	Cu concentrate.....	28.0	23.74	3.68	0.08	7.04	93.0	16.1	75.4	76.0
	Zn ".....	9.8	1.22	47.38	0.01	0.81	1.7	72.9	3.4	3.1
	Zn middling.....	9.7	1.07	3.68	0.01	0.71	1.4	5.6	3.4	2.3
	Tailing.....	52.2	0.53	0.65	0.01	0.92	3.9	5.4	17.8	18.6
9	Cu concentrate.....	27.5	22.88	8.03	0.08	6.46	88.1	34.7	79.2	82.3
	Zn ".....	9.3	2.87	38.01	0.01	1.11	3.7	55.6	3.2	4.8
	Zn middling.....	14.5	1.61	2.34	tr	0.61	3.3	5.3	4.1
	Tailing.....	48.7	0.72	0.57	0.01	0.39	4.9	4.4	17.6	8.8
10	Cu concentrate.....	29.0	21.73	4.05	0.08	6.06	86.7	17.9	82.3	80.0
	Zn ".....	12.9	4.39	38.11	0.02	1.56	7.8	75.0	9.2	9.2
	Zn middling.....	12.2	1.49	2.49	0.02	0.67	2.5	4.6	8.5	3.7
	Tailing.....	45.9	0.47	0.35	tr	0.34	3.0	2.5	7.1
11	Cu concentrate.....	20.3	21.22	4.26	0.08	7.68	86.6	19.3	79.0	84.4
	Zn ".....	11.4	3.56	40.79	0.01	1.37	5.5	71.9	3.8	5.9
	Zn middling.....	7.9	2.33	3.68	tr	1.02	2.6	4.5	3.0
	Tailing.....	51.4	0.74	0.54	0.01	0.35	5.3	4.3	17.2	6.7
12	Cu concentrate.....	28.2	24.38	3.37	95.6	14.8
	Zn ".....	10.7	1.19	48.41	1.8	78.3
	Zn middling.....	11.1	0.90	3.32	1.4	5.7
	Tailing.....	50.0	0.17	0.16	1.2	1.2
13	Cu concentrate.....	32.1	21.80	3.32	97.0	16.2
	Zn ".....	16.2	0.55	31.62	1.2	77.7
	Zn middling.....	18.6	0.39	1.45	1.0	4.1
	Tailing.....	33.1	0.18	0.39	0.8	2.0

Reagents Used:

Test No.	Reagents	Amount lb./ton	Added at	Remarks
1	Soda ash.....	5.0	Ball mill.....	Ore ground to 100 mesh.
	Thiocarbamide.....	0.20	".....	
	Sodium cyanide.....	0.40	".....	
	Copper sulphate.....	2.00	Cell.....	
	Xanthate.....	0.40	".....	
	Pine oil No. 5.....	0.10	".....	
2	Cu Lime.....	1.5	Ball mill.....	Ore ground to 200 mesh. Lime added full time contact. Soda ash one-third time contact.
	Soda ash.....	2.0	".....	
	Copper sulphate.....	0.20	Cell.....	
	Xanthate.....	0.20	".....	
	Pine oil No. 5.....	0.08	".....	
	Zn Sodium hydroxide.....	0.60	".....	
	Copper sulphate.....	2.00	".....	
	Xanthate.....	0.40	".....	
	Pine oil No. 5.....	0.08	".....	

Test No.	Reagents	Amount lb./ton	Added at	Remarks
3	Cu Soda ash.....	5.00	Ball mill.....	Ore ground to 150 mesh.
	Sodium cyanide.....	0.20	"	
	Thiocarbanilide.....	0.20	"	
	Water-gas tar.....	0.03	"	
	Cresylic acid.....	0.04	"	
	Zn Copper sulphate.....	2.00	Cell.....	
	Xanthate.....	0.20	"	
	Pine oil No. 5.....	0.08	"	
4	Cu Lime.....	2.00	Ball mill.....	Ore ground to 150 mesh. Lime added full time contact. Other reagents added to ball mill half time contact. Denser pulp than in previous tests.
	Soda ash.....	2.00	"	
	Thiocarbanilide.....	0.20	"	
	Sodium cyanide.....	0.50	"	
	Pine oil No. 5.....	0.08	Cell.....	
	Zn Copper sulphate.....	2.00	"	
	Xanthate.....	0.30	"	
	Pine oil No. 5.....	0.08	"	
5	Cu Lime.....	2.0	Ball mill.....	Ore ground to 150 mesh. Lime full time contact. Other reagents added to ball mill half time contact. Pulp density as in test No. 4.
	Soda ash.....	2.0	"	
	Sodium cyanide.....	0.5	"	
	Barrett's No. 4.....	0.4	"	
	Cresylic acid.....	0.05	Cells.....	
	Pine oil No. 5.....	0.04	"	
	Zn Copper sulphate.....	2.0	"	
	Xanthate.....	0.3	"	
6	Cu Soda ash.....	4.00	Ball mill.....	Copper removed at 100 mesh. Tailings dewatered and reground to 200 mesh before floating zinc.
	Sodium cyanide.....	0.3	"	
	Thiocarbanilide.....	0.2	"	
	Cresylic acid.....	0.08	Cells.....	
	Zn Soda ash.....	3.0	Ball mill.....	
	Copper sulphate.....	1.50	Cells.....	
	Xanthate.....	0.3	"	
	Pine oil No. 5.....	0.08	"	
7	Cu Soda ash.....	4.00	Ball mill.....	Copper removed at 100 mesh. Tailing dewatered and reground to 200 mesh before floating zinc.
	Sodium cyanide.....	0.3	"	
	Thiocarbanilide.....	0.2	"	
	Cresylic acid.....	0.08	Cell.....	
	Zn Lime.....	5.00	Ball mill.....	
	Copper sulphate.....	1.5	Cell.....	
	Xanthate.....	0.3	"	
	Pine oil No. 5.....	0.08	"	
8	Cu Soda ash.....	4.0	Ball mill.....	Ore ground to 150 mesh.
	Sodium cyanide.....	0.3	"	
	Thiocarbanilide.....	0.2	"	
	Cresylic acid.....	0.08	"	
	Copper sulphate.....	0.4	Cell.....	
	Zn Copper sulphate.....	1.5	"	
	Xanthate.....	0.3	"	
	Pine oil No. 5.....	0.08	"	
9	Cu Soda ash.....	4.0	Ball mill.....	Ore ground to 150 mesh. Barrett's No. 634 did not mix, xanthate was added.
	Sodium cyanide.....	0.3	"	
	Barrett's No. 4.....	0.25	"	
	Cresylic acid.....	0.08	"	
	Copper sulphate.....	0.4	Cell.....	
	Cresylic acid.....	0.08	"	
	Pine oil No. 5.....	0.08	"	
	Zn Copper sulphate.....	1.5	"	
	Barrett's No. 634.....	0.2	"	
	Xanthate.....	0.35	"	
	Pine oil No. 5.....	0.08	"	

Test No.	Reagents	Amount lb./ton	Added at	Remarks	
10	Cu Soda ash.....	5.0	Ball mill.....	Ore ground to 6% +200 mesh.	
	Sodium cyanide.....	0.2	"		
	Coal tar.....40%}	0.3	"		
	Coal-tar creosote 60%}				
	Cresylic acid.....	0.08	"		
	Pine oil No. 5.....	0.08	Cell.....		
	Zn Copper sulphate.....	1.50	"		
	Xanthate.....	0.3	"		
	Pine oil No. 5.....	0.08	"		
11	Cu Soda ash.....	5.0	Ball mill.....	Ore ground to 2% +200 mesh.	
	Water-gas tar.....	0.3	"		
	Sodium cyanide.....	0.2	"		
	Cresylic acid.....	0.08	"		
	Pine oil No. 5.....	0.08	Cell.....		
	Zn Copper sulphate.....	1.5	"		
	Xanthate.....	0.4	"		
	Pine oil No. 5.....	0.08	"		
	12	Cu Soda ash.....	5.0		Ball mill.....
Sodium cyanide.....		0.35	"		
Thiocarbanilide.....		0.2	"		
Cresylic acid.....		0.08	"		
Pine oil No. 5.....		0.04	Cell.....		
Zn Copper sulphate.....		1.50	"		
Xanthate.....		0.25	"		
Pine oil No. 5.....		0.04	"		
13		Cu Soda ash.....	5.0	Ball mill.....	Ore ground to +200 mesh.
	Sodium cyanide.....	0.35	"		
	Thiocarbanilide.....	0.25	"		
	Cresylic acid.....	0.08	"		
	Pine oil No. 5.....	0.04	Cell.....		
	Zn Copper sulphate.....	1.50	"		
	Xanthate.....	0.3	"		
	Pine oil No. 5.....	0.04	"		

CONCLUSIONS

1. *Disseminated Zinc Ore.* No difficulty was experienced in obtaining a zinc concentrate containing more than 45 per cent zinc with a recovery of over 90 per cent, when the material was ground to —100 mesh.

2. *Heavy Sulphide Ore.* Excellent separation of the copper from the zinc was obtained in these tests on ore ground to —100 mesh. The copper recoveries and the grade of the copper concentrate were both good, 95 per cent of the copper was obtained in the concentrate containing about 24 per cent copper and 3.5 per cent zinc.

The zinc recoveries were good, 70 per cent of the zinc being recovered in three tests, Nos. 6, 8, and 12, in concentrates assaying more than 45 per cent zinc. Grinding to 150 mesh is necessary in order to make a good separation of the zinc from the pyrite and pyrrhotite.

Seventy-five per cent of the gold and eighty per cent of the silver were recovered with the chalcopyrite. The gold in the copper concentrate amounted to 0.08 ounce per ton in each test, and the silver in the same product amounted to about 7 ounces per ton. Three and one-half to five per cent additional gold and about the same quantity of silver were recovered in the zinc concentrate.

In test No. 3 the tailing from the zinc cell was tabled. This operation recovered an additional 15 per cent of the gold values. As only 50 per cent of the gold present in the copper concentrate was recovered in this test, it cannot be assumed that this recovery of gold could be obtained by table concentration where 75 per cent of the gold reports with the copper concentrate. Tabling would no doubt recover sufficient gold in a concentrate of small bulk to pay for the operation and would be very valuable as an indicator of the performance of the zinc cells. Such a product could be mixed with the copper concentrate.

Report No. 250

EXPERIMENTAL TESTS ON A GOLD ORE FROM THE GOLD HILL MINES, BOSTON CREEK, ONT.

J. S. Godard

Shipment. A shipment of ore consisting of five samples was received from the Gold Hill Mines, Boston Creek, Ontario, July 7, 1926. The samples and their weight were as follows:—

Sample:	400-ft level.....	13.5 pounds
" A	500 "	10.0 "
" B	500 "	10.0 "
" A	600 "	12.25 "
" B	600 "	11.75 "

Characteristics of the Ore. It was thought that the gold present in this ore was a gold telluride, but the presence of tellurium was undetected by microscopic examination and by a qualitative test on a 10-gramme sample. Small amounts of lead and copper were present. The gangue was siliceous.

Purpose of Experimental Tests. The purpose of these tests was to determine the occurrence of the gold and an economic metallurgical treatment.

Analysis. The analysis of the samples was as follows:

Sample:	400-ft level.....	2.54 oz./ton gold
" A	500 "	0.11 "
" B	500 "	0.11 "
" A	600 "	0.68 "
" B	600 "	2.26 "

For the purpose of the tests a sample was prepared consisting of the 400-foot level and the B 600-foot level samples. The combined sample assayed 2.30 ounces gold per ton.

CYANIDATION TESTS

Details:

No.	Weight, grms.	Mesh	Density	KCN	Time	Remarks
				per cent	hrs.	
1.....	500	—100	1.2	0.20	65	Ore dry crushed.
2.....	500	—200	1.2	0.20	65	Ore wet crushed and dewatered.
3.....	500	—200	1.3	0.15	65	

The cyanide tailing from test No. 1 was screened and the products assayed:

Mesh	Weight, per cent	Assay Au, oz./ton	Per cent of values
+150	18.9	0.60	46.3
+200	20.1	0.29	23.8
-200	61.0	0.12	29.9
Average tailing....	0.25

Results:

Test No.	Head Au, oz./ton	Tailing Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
1.....	2.30	0.25	89.2	3.32	5.1
2.....	2.30	0.06	97.4	10.84	10.3
3.....	2.30	0.15	93.5	11.12	11.5

AMALGAMATION TESTS

Two amalgamation tests were made on the combined sample. A screen analysis was made on the tailing from each test.

Test No.	Mesh	Weight, per cent	Assay Au, oz./ton	Values per cent	Per cent gold amal- gamated
4.....	+100	2.9	0.49	4.0	84.7
	+150	6.7	0.51	9.7	
	+200	12.0	0.52	17.6	
	-200	78.4	0.31	68.7	
Average tailing.....			0.354		
5.....	+100	1.4	1.64	6.3	84.2
	+150				
	+200	4.4	0.73	8.5	
	-200	94.2	0.33	85.2	
Average tailing.....			0.365		

FLOTATION AND CYANIDATION

Test No. 6—Concentration

Product	Weight, per cent	Au oz./ton	Cu per cent	Per cent of values	
				Au	Cu
Concentrate.....	17.7	9.36	2.42	81.8	96.4
Tailing.....	82.3	0.45	0.02	18.2	3.6

The tailing was ground to —200 mesh and cyanided, with the following results:

Pulp density	KCN per cent	Head, Au oz./ton	Tailing, Au oz./ton	Extraction, per cent	Reagents consumed		Time
					KCN	CaO	
1 : 2.....	0.075	0.45	0.08	82.2	1.3	6.9	72 hrs.

Distribution of gold: In concentrate..... 81.8 per cent
 In cyanide solution..... 14.9 "
 In cyanide tailing..... 3.3 "
 Recovery..... 96.7 "

Test No. 7—Concentration

Product	Weight, per cent	Au oz./ton	Cu per cent	Per cent of values	
				Au	Cu
Concentrate..	20.6	7.76	2.16	78.2	94.9
Tailing.....	79.4	0.56	0.03	21.8	5.1

The tailing was ground to —200 mesh and cyanided, with the following results:

Pulp density	KCN per cent	Head	Tailing	Extraction, per cent	Reagents consumed		Time
					KCN	CaO	
1 : 3	0.05	0.56	0.12	78.6	1.0	6.0	72 hrs.

Distribution of gold: In concentrate..... 78.2 per cent
 In cyanide solution..... 17.2 "
 In cyanide tailing..... 4.6 "
 Recovery..... 95.4 "

SUMMARY

Cyanidation. Good extractions were obtained by cyanidation. These are offset by high cyanide consumption, especially in the tests where the ore is ground to —200 mesh, where over 10 pounds cyanide per ton were used.

Amalgamation. Amalgamation results were satisfactory. Eighty-five per cent of the gold is recovered by this method. This high recovery indicates that the gold is present in a free state and that not more than 15 per cent of it could be as a telluride.

Flotation and Cyanidation. This method of treatment was satisfactory. The recoveries were over 95 per cent and the cyanide consumption was low in both tests. The concentrate was of too low a grade to stand shipment, but this could be remedied by cleaning.

CONCLUSIONS

From the above tests cyanidation of ore would not be practical because of high cyanide consumption and fouling of the solution. Amalgamation is satisfactory and should be inserted in the flow-sheet. Flotation and cyanidation of the flotation tailing is satisfactory. The flotation removed the cyanicides, probably copper, though undetermined. Amalgamation followed by flotation and tabling the flotation tailing; amalgamation followed by flotation; and amalgamation followed by flotation and cyanidation of flotation tailing are other possible flow-sheets for the treatment of this ore.

An insufficient sample made it impossible to attempt tests based on these lines.

Report No. 251

THE CONCENTRATION OF THE ZINC ORE FROM THE JOHN BULL AND FLORENCE CLAIMS, POWELL RIVER SECTION, NANAIMO MINING DIVISION, BRITISH COLUMBIA

C. S. Parsons

Shipment. A shipment of two bags of ore was received July 6, 1926, from the Ladysmith Smelter Syndicate, Victoria, B.C. One bag contained 42 pounds of ore from the John Bull claim, and the other, 36 pounds from the Florence claim. Both claims are situated in the Powell River section, Nanaimo mining division, about 10 miles north of Powell River.

Characteristics of the Ore. The ore from both claims is similar, containing zinc blende of the light resin-coloured variety, a small amount of chalcopyrite, very little pyrite in a metamorphosed limestone gangue rock, and only traces of gold and silver.

Analysis. Samples of the ores gave the following analyses:—

	John Bull	Florence
Zinc.....	23.86 per cent	18.48 per cent
Lead.....	trace	nil
Copper.....	0.07 "	0.73 "
Gold.....	trace	trace
Silver.....	0.26 oz./ton	0.24 oz./ton

Purpose of Experimental Work. Test work on both samples of ore was requested to determine their amenability to concentration by gravity or flotation methods.

EXPERIMENTAL TESTS

Both samples were crushed and sampled. The rejects from the sampling, which were crushed to 14 mesh, were used in the following tests:

Results of Flotation Tests

Ore	Test No.	Product	Weight, per cent	Analysis		Per cent of values	
				Cu per cent	Zn per cent	Cu	Zn
John Bull.....	1	Zinc concentrate.....	44.9	53.8	99.0
		Tailing.....	55.1	0.36	1.0
Florence.....	1	Copper concentrate.....	6.2	10.14	35.7	82.0	12.0
		Zinc concentrate.....	34.8	0.3	46.5	13.5	87.0
		Tailing.....	59.0	0.06	0.36	4.5	1.0
	2	Copper concentrate.....	1.5	24.94	16.9	50.8	1.4
		Copper middling.....	1.3	5.37	31.95	9.4	2.2
		Zinc concentrate.....	35.0	0.60	48.84	28.2	93.7
		Tailing.....	62.2	0.14	0.78	11.6	2.6
	3	Copper concentrate.....	5.2	11.16	23.28	79.8	6.7
		Zinc concentrate.....	33.9	0.35	48.98	16.1	91.6
		Tailing.....	60.9	0.05	0.51	4.1	1.7
	4	Copper concentrate.....	8.3	7.56	17.88	78.9	8.0
		Zinc concentrate.....	34.2	0.39	49.45	16.7	91.0
		Tailing.....	57.5	0.06	0.33	4.4	1.0
	5	Copper concentrate.....	12.4	5.22	39.91	85.5	26.8
		Zinc concentrate.....	32.4	0.22	41.04	9.4	72.2
		Tailing.....	55.2	0.07	0.33	5.1	1.0
	6	Copper concentrate.....	5.3	9.5	20.45	67.0	6.5
		Zinc concentrate.....	31.9	0.68	48.45	28.8	92.0
		Tailing.....	62.8	0.05	0.41	4.2	1.5
	7	Copper concentrate.....	5.5	8.86	15.03	67.2	4.6
		Zinc concentrate.....	37.7	0.57	47.21	29.7	94.3
		Tailing.....	56.8	0.04	0.36	3.1	1.1
	8	Copper concentrate.....	3.2	17.22	20.07	70.7	3.5
		Zinc concentrate.....	31.5	0.56	54.68	22.6	94.3
		Tailing.....	65.3	0.03	0.62	6.7	2.2
	9	Copper concentrate.....	2.8	17.30	13.72	65.9	2.1
		Copper middling.....	2.0	0.72	18.84	2.9	3.0
		Zinc concentrate.....	35.0	0.59	47.51	28.9	93.0
		Tailing.....	58.4	0.03	0.61	2.3	1.9

NOTE.—In test No. 9, the copper concentrates from four lots of 1,000 grammes each were combined and reground to -200 mesh before cleaning.

Reagents Used (Florence Ore)

Test No.	Reagents	Lb./ton	Remarks
1	Soda ash.....	6.0	30 minutes contact in ball mill.
	Cyanide.....	0.2	30 minutes contact in ball mill.
	Thiocarbamilide.....	0.2	30 minutes contact in ball mill.
	Pine oil.....	...	To cells.
	Copper sulphate.....	1.5	To zinc cells.
	Xanthate.....	0.3	To zinc cells.
2	Soda ash.....	6.0	30 minutes contact in ball mill.
	Cyanide.....	0.2	30 minutes contact in ball mill.
	Thiocarbamilide.....	0.2	30 minutes contact in ball mill.
	Pine oil.....	...	To copper cells.
	Copper sulphate.....	1.5	To zinc cells.
	Xanthate.....	0.3	To zinc cells.
3	Pine oil.....	...	To zinc cells.
	Lime.....	3.0	40 minutes contact in ball mill.
	Cyanide.....	0.3	20 minutes contact in ball mill.
	Xanthate.....	0.15	To copper cells (ore ground +150).
	Cresylic acid.....	0.12	To copper cells.
	Copper sulphate.....	2.0	To zinc cells.
4	Xanthate.....	0.6	To zinc cells.
	Pine oil No. 5.....	0.04	To zinc cells.
	Lime.....	4.0	60 minutes contact in ball mill.
	Cyanide.....	0.25	30 minutes contact in ball mill.
	Xanthate.....	0.10	To copper cell (ore ground 200 mesh).
	Pine oil No. 5.....	0.04	To copper cell.
5	Copper sulphate.....	2.0	To zinc cells.
	Xanthate.....	0.5	To zinc cells.
	Pine oil No. 5.....	0.04	To zinc cells.
	Soda ash.....	3.00	60 minutes contact in ball mill.
	Cyanide.....	0.25	30 minutes contact in ball mill.
	Thiocarbamilide.....	0.1	30 minutes contact in ball mill.
6	Cresylic acid.....	0.08	To copper cell (ore ground 200 mesh).
	Copper sulphate.....	2.0	To zinc cells.
	Xanthate.....	0.6	To zinc cells.
	Pine oil No. 5.....	0.04	To zinc cells.
	Lime.....	5.0	60 minutes contact in ball mill.
	Cyanide.....	0.3	30 minutes contact in ball mill.
7	Thiocarbamilide.....	0.1	30 minutes contact in ball mill.
	Pine oil No. 5.....	0.04	To copper cells (ore ground 200 mesh).
	Copper sulphate.....	2.0	To zinc cells.
	Xanthate.....	0.6	To zinc cells.
	Pine oil No. 5.....	0.04	To zinc cells.
	Water-gas tar.....	0.2	60 minutes contact in ball mill.
8	Zinc sulphate.....	1.5	60 minutes contact in ball mill.
	Cyanide.....	0.3	60 minutes contact in ball mill.
	Cresylic acid.....	0.08	60 minutes contact in ball mill.
	Pine oil No. 5.....	0.04	To copper cells (ore ground 200 mesh).
	Copper sulphate.....	2.00	To zinc cells.
	Xanthate.....	0.6	To zinc cells.
9	Pine oil No. 5.....	0.04	To zinc cells.
	Lime.....	4.00	30 minutes contact in ball mill.
	Zinc sulphate.....	1.00	30 minutes contact in ball mill.
	Cyanide.....	0.2	30 minutes contact in ball mill.
	Cresylic acid.....	0.08	30 minutes contact in ball mill.
	Pine oil (Floto).....	0.04	To copper cells.
10	Xanthate.....	0.10	To copper cells.
	Copper sulphate.....	2.00	To zinc cells.
	Xanthate.....	0.60	To zinc cells.
	Pine oil No. 5.....	0.04	To zinc cells.
	Lime.....	5.0	60 minutes contact in ball mill.
	Water-gas tar.....	0.2	60 minutes contact in ball mill.

Reagents Used (Florence Ore)—Concluded

Test No.	Reagents	Lb./ton	Remarks
9	Lime.....	3.00	45 minutes contact in ball mill.
	Zinc sulphate.....	0.80	45 minutes contact in ball mill.
	Cyanide.....	0.20	45 minutes contact in ball mill.
	Cresylic acid.....	0.08	45 minutes contact in ball mill.
	Water-gas tar.....	0.20	45 minutes contact in ball mill.
	Pine oil No. 5.....	0.04	To copper cells (ore ground 150 mesh).
	Copper sulphate.....	2.00	To zinc cells.
	Xanthate.....	0.50	To zinc cells.
	Pine oil No. 5.....	0.04	To zinc cells.
	Lime.....	0.40	20 minutes contact in ball mill.
	Cyanide.....	0.05	20 minutes contact in ball mill.
	Thiocarbamilide.....	0.03	20 minutes contact in ball mill.

Results of Table Tests

John Bull Ore. The ore was crushed and sized as follows for table tests:

Size	Weight		Analysis, Zn per cent	Remarks
	Grms.	Per cent		
- 14+ 20.....	591	11.9	25.07	This size not tabled.
- 20+ 48.....	1,880	37.9	24.64	
- 48+100.....	811	16.3	25.35	
-100.....	1,682	33.9	22.56	
Total.....	4,964	100.0	24.12	

Size	Product	Weight, per cent	Analysis, Zn per cent	Per cent of Zn values
-14+ 20.....	Concentrate.....	50.3	38.05	76.5
	Middling.....	10.4	10.69	4.4
	Tailing.....	39.3	12.22	19.1
-20+ 48.....	Concentrate.....	53.8	34.31	74.9
	Middling.....	24.2	16.78	16.4
	Tailing.....	22.0	9.69	8.7
-48+100.....	Concentrate.....	43.9	39.47	68.3
	Middling.....	22.5	21.10	18.8
	Tailing.....	33.6	9.74	12.9

Florence Ore. The ore was crushed to 14 mesh and sized as follows for table tests:

Size	Weight		Analysis, Zn per cent	Remarks
	Grms.	Per cent		
- 14+ 20.....	617	12.4	17.23	This size not tabled.
- 20+ 48.....	1,786	35.7	16.92	
- 48+100.....	942	18.9	19.40	
-100.....	1,645	33.0	20.29	

Size	Products	Weight, per cent	Analysis, Zn per cent	Per cent of Zn values
-14+ 20.....	Concentrate.....	87.8	18.62	94.8
	Tailing.....	12.2	7.29	5.2
-20+ 48.....	Concentrate.....	46.4	22.15	60.8
	Middling.....	37.2	13.92	30.6
	Tailing.....	16.4	8.92	18.6
-48+100.....	Concentrate.....	47.8	27.07	66.7
	Middling.....	30.6	13.84	28.2
	Tailing.....	12.6	7.81	5.1

SUMMARY AND CONCLUSIONS

John Bull Ore. It is evident from the results of the table test that the ore cannot be concentrated by gravity methods, while on the other hand the test conducted by flotation gives excellent results. The method used was very simple. The ore was ground with cyanide and sodium carbonate to produce an alkaline pulp, the cyanide being added to depress the iron sulphides. Copper sulphate and xanthate were added to the flotation cell together with a frothing oil. The grade of concentrate and the recovery are both good.

Florence Ore. This ore contains copper, but the amount in the sample is less than one per cent, so that the possibility of earning a profit out of the copper content of the ore is not encouraging. A number of selective flotation tests were made in order to determine this possibility. In all the tests, with the exception of Nos. 2 and 9, the content of zinc in the copper concentrate was higher than the copper. Test No. 9 is the only one which is at all encouraging. To obtain this result it was necessary to grind the primary copper concentrate and again subject it to selective flotation. The possible profit, when smelter rates are considered, is so small and such a complicated procedure is required that very careful consideration should be given to the matter before attempting such a separation. The production of a high-grade zinc concentrate with an excellent recovery is a relatively simple matter and the amount of copper which would be present in such a concentrate would not affect its market value.

Report No. 252

EXPERIMENTAL TESTS ON LONG LAKE ARSENICAL-GOLD TAILINGS

J. S. Godard

Shipment. One shipment consisting of twenty-three samples, gross weight, 200 pounds, was received July 17, 1926. Leo H. Timmins, of Alderson and MacKay, was the shipper.

Characteristics of the Tailings. The tailings were those from an old cyanide plant operated from 1912 to 1915 by the Canadian Exploration Company. It is reported that the head sample at the time of opera-

tion was about 0.36 ounce gold per ton, and a tailing of 0.07 to 0.08 ounce per ton was discharged. The ore was crushed in stamps and tube mills to about 150 mesh. The tailings as represented by the composite sample approximated 65 mesh. The sulphide minerals are pyrite and arsenopyrite, with which the gold is associated. The sulphides are partly oxidized and evidently the tailings have become concentrated, as the composite sample assayed 0.20 ounce per ton.

Sampling and Analysis. A composite sample of the twenty-three samples was made by cutting a proportionate cut from each one at —65 mesh. It was sampled in duplicate.

Analysis:

	Sample No. 1	Sample No. 2
Gold.....	0.21 oz./ton	0.20 oz./ton
Silver.....	0.05	0.05
Arsenic.....	1.88 per cent	2.09 per cent

Purpose of Tests. The purpose of these tests was to ascertain if the gold could be profitably extracted from these tailings by regrinding and cyanidation, or by concentration of the sulphides and re-treatment of the concentrates.

CYANIDE TESTS

Tests Nos. 1 and 2—Details:

Test	Dilution	KCN per cent	Time of agitation	Crushing
1.....	1 : 3	0.075	48 hrs.	—65 mesh.
2.....	1 : 3	0.050	48 "	Crushed wet —200 mesh.

Results:

Test	Head	Tails	Extraction, per cent	Reagents consumed	
				KCN	CaO
1.....	0.20	0.098	51	3.76	37.40
2.....	0.20	0.09	55	2.85	25.90

Tails from Test No. 1 were screened and assayed:

Mesh	Weight, per cent	Assay oz./ton	Per cent of values
+100.....	15.7	0.10	15.9
+150.....	27.8	0.09	25.5
+200.....	17.0	0.06	10.4
—200.....	39.5	0.12	48.2
Average tailing.....	0.093

Tests Nos. 3, 4, 5, and 6—Details:

Test	Dilution	KCN, per cent	Time of agitation	Crushing
3.....	1 : 2	0.05	48 hrs.	-65 mesh dry.
4.....	1 : 2	0.10	48 "	"
5.....	1 : 2	0.05	48 "	{-65 mesh water washed and filtered.
6.....	1 : 2	0.10	48 "	

Results (2nd. Composite Head Sample):

Test	Head	Tails	Extraction, per cent	Reagents consumed	
				KCN	CaO
3.....	0.25	0.093	62.8	1.5	33.0
4.....	0.25	0.055	78.0	2.7	33.0
5.....	0.25	0.100	60.0	1.25	18.2
6.....	0.25	0.076	69.6	2.2	18.4

Screen test on Cyanide Tailings:

Test	Mesh	Weight, per cent	Assay	Per cent of values	Remarks
3.....	+100	20.2	0.07	15.2	Average tailing 0.093 oz./ton.
	+150	15.5	0.08	13.4	
	+200	22.5	0.07	17.1	
	-200	41.8	0.12	54.3	
4.....	+100	21.4	0.04	15.4	Average tailing 0.055 oz./ton.
	+150	14.0	0.04	10.1	
	+200	20.3	0.05	18.4	
	-200	44.3	0.07	56.1	
5.....	+100	15.9	0.08	12.6	Average tailing 0.10 oz./ton.
	+150	13.5	0.08	10.7	
	+200	36.6	0.08	29.2	
	-200	34.0	0.14	47.5	
6.....	+100	21.2	0.09	23.9	Average tailing 0.076 oz./ton.
	+150	17.8	0.06	14.1	
	+200	17.7	0.07	16.4	
	-200	43.3	0.08	45.6	

Tests Nos. 7, 8, 9 and 10

- No. 7—Tailings, 750 grms. at -65 mesh, ground 3 hours in a pebble mill using 1 : 1 pulp, 0.025 per cent KCN solution and lime 25 lb./ton. Cyanided 48 hours 1 : 2 pulp using 0.05 per cent KCN.
- No. 8—Tailings, 600 grms. at -65 mesh, ground 3 hours in a pebble mill using 1 : 1.5 pulp, 0.025 per cent KCN solution and lime 25 lb./ton. Cyanided 48 hours, 1 : 3 pulp, 0.05 per cent KCN.
- No. 9—Tailings, 600 grms. at -65 mesh, ground 3 hours in pebble mill, 1 : 1.15 pulp with lime 10 lb./ton. Cyanided 48 hours 1 : 3 pulp, 0.075 per cent KCN.
- No. 10—Tailings, 600 grms. at -65 mesh, ground 3 hours in 1 : 1.5 pulp, then dewatered and filtered. Cyanided 48 hours, 1 : 3 pulp, 0.075 per cent KCN.

Results, Head Sample 0.25 oz./ton.

Test	Tailing	Extraction, per cent	Reagents consumed	
			KCN	CaO
7.....	0.04	84.0	2.5	41.5
8.....	0.02	92.0	3.0	48.3
9.....	0.02	92.0	3.4	32.1
10.....	0.03	88.0	2.1	26.5

CONCLUSIONS

Tests Nos. 1-6 inclusive showed poor extractions by cyanidation. The tailings were high in all the sizes. The cyanide consumption was moderately high, the lime consumption very high. Water washing reduced the cyanide consumption and cut the lime consumption by 45 per cent. In tests Nos. 7-10 inclusive, where the tailings were very finely ground, good extractions were obtained. The consumption of cyanide and lime was high. Water washing reduced both of these.

CONCENTRATION TESTS

*Test No. 11**Tabling at -65 Mesh:*

Product	Weight, per cent	Assay		Per cent of values	
		Au oz./ton	As per cent	Au	As
Table concentrate.....	21.0	0.54	4.20	60.8	53.2
" tailing.....	64.2	0.10	0.80	34.4	31.0
Slimes.....	14.8	0.06	1.77	4.8	15.8
Head from products.....		0.19	1.66		

*Test No. 12**Flotation, Tabling Flotation Tailing:*

Product	Weight, per cent	Assay		Per cent of values	
		Au oz./ton	As per cent	Au	As
Flotation concentrate.....	22.5	0.57	6.02	67.7	66.2
Table concentrate.....	7.0	0.48	3.31	17.8	11.3
" tailing.....	49.1	0.03	0.39	7.8	9.3
Slimes.....	21.4	0.06	1.26	6.7	13.2
Head from products.....		0.19	2.05		

Test No. 13

Flotation, Tabling Flotation Tailing:

Product	Weight, per cent	Assay		Per cent of values	
		Au oz./ton	As per cent	Au	As
Flotation concentrate.....	20.0	0.75	6.56	75.5	68.8
Table concentrate.....	3.6	0.54	3.97	9.8	7.5
" tailing.....	45.1	0.03	0.31	6.8	7.3
Slimes.....	31.1	0.05	1.00	7.9	16.4
Head from products.....		0.20	1.91		

Flotation Reagents

Test No. 12.—Coal tar 40 per cent, coal-tar creosote 60 per cent, 0.3 lb./ton, soda ash 7.0 lb./ton, cresylic acid 0.08 lb./ton, added to ball mill and ground 30 minutes. Added to cell, sulphuric acid 5.0 lb./ton, copper sulphate 1.0 lb./ton, xanthate 0.5 lb./ton, pine oil 0.1 lb./ton.

Test No. 13.—No. 1,580, 0.4 lb./ton, sulphuric acid 5 lb./ton, copper sulphate 1.0 lb./ton, xanthate 0.3 lb./ton, ground 40 minutes in ball mill. Added to cell, sulphuric acid 10 lb./ton, xanthate 0.10 lb./ton, pine oil 0.1 lb./ton.

CONCLUSIONS

Tabling. Tabling produced very poor results.

Flotation and Tabling. Flotation followed by tabling the flotation tailing was much superior to straight tabling. The combination method showed a ratio of concentration of 1:4.4. The slimes were higher than the sand tailing in both gold and arsenic. The very fine material is too oxidized to float. Relatively large amounts of reagents are required for the flotation.

SUMMARY

The tests on the composite sample indicate that fine grinding and cyanidation is the better method of recovering the gold from these tailings. Water washing previous to cyanidation seems advisable. The present price of arsenic is too low to warrant attention being paid to the arsenic content.

Report No. 253

CONCENTRATION OF A SILVER-LEAD-COPPER-ZINC ORE FROM
LA ROSE MINE, KITSALT RIVER SECTION, B.C.

J. S. Godard

Shipment. A shipment of 50 pounds of this ore was received July 28, 1926. It was sent by the Ladysmith Smelter Syndicate from the La Rose mine, in the Kitsault River section, B.C.

Characteristics of the Ore. The ore consists of a mixture of the sulphides of lead, zinc, and a minor quantity of arsenic and copper. Gold is present to the extent of 0.16 ounce per ton, some of which is free and the balance associated with the above sulphides; 192 ounces of silver per ton is present, mainly associated with the lead and copper. The lead is present as galena, the copper as freibergite, and the zinc as a dark resin blende. The gangue is siliceous.

Sampling and Analysis. The entire shipment was crushed and screened on 20 mesh. The metallics were assayed separately.

Analysis of —20 mesh product:

Gold.....	0.14 oz./ton
Silver.....	180.66 "
Lead.....	2.59 per cent
Zinc.....	3.41 "
Copper.....	0.43 "
Arsenic.....	0.75 "

Assay of metallics +20 mesh:

Gold.....	0.02 oz./ton
Silver.....	12.12 "

Assay of head sample:

Gold.....	0.16 oz./ton
Silver.....	192.78 "

Purpose of Tests. The purpose of these tests was to ascertain the best method of concentrating this ore.

EXPERIMENTAL TESTS

Selective flotation of the lead and copper sulphides from the zinc blende. The flotation tailings were tabled.

Results:

Test No.	Product	Weight per cent	Assays					Per cent of values				
			Pb per cent	Cu per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Cu	Zn	Au	Ag
1	Lead concentrate....	10.4	23.61	3.32	10.39	0.70	1,574.6	96.2	83.7	33.6	73.5	91.8
	Zinc "	8.8	0.98	0.39	24.27	0.15	87.65	3.4	8.3	66.4	13.3	4.3
	Table "	1.5	0.15	0.10	tr	0.30	159.6	0.1	0.2	4.1	1.3
	" middling.....	10.5	0.07	0.07	tr	0.02	18.83	0.3	1.7	2.0	1.1
	" tailing.....	47.1	tr	0.03	tr	0.01	3.65	3.4	5.1	1.0
	" slimes.....	21.7	tr	0.05	tr	0.01	3.63	2.7	2.0	0.5
2	Lead concentrate....	11.0	23.56	3.20	12.75	0.60	1,972.15	96.8	83.8	40.7	60.0	92.8
	Zinc "	8.5	0.67	0.39	23.65	0.15	86.90	2.1	7.9	58.3	11.8	3.2
	Table "	8.8	0.22	0.05	0.05	0.28	77.75	0.7	1.0	0.1	22.7	2.9
	" tailing.....	56.3	nil	0.03	tr	0.01	3.73	4.0	4.5	0.9
	" slimes.....	15.4	0.05	0.09	0.20	0.01	4.21	0.3	3.3	0.9	1.0	0.2
3	Lead concentrate....	9.6	26.44	3.57	15.72	0.925	1,690.33	97.1	84.9	46.0	61.5	90.9
	Zinc "	2.0	1.19	0.59	24.37	0.21	94.06	0.9	3.0	14.8	2.9	1.1
	" middling.....	3.6	0.42	0.34	2.25	0.195	53.21	0.6	3.0	2.5	4.8	1.1
	Table concentrate...	5.1	0.45	0.14	12.13	0.70	173.97	0.9	1.7	18.8	24.2	5.0
	" tailing.....	63.6	tr	0.03	0.64	0.015	4.19	4.7	12.4	6.6	1.5
	" slimes.....	16.1	0.09	0.07	1.11	tr	5.41	0.5	2.7	5.5	0.5
4	Lead concentrate....	9.0	26.94	3.59	13.72	0.75	1,786.75	96.9	83.3	37.6	60.9	93.0
	" middling.....	5.5	0.60	0.18	2.83	0.05	56.40	1.3	2.6	4.7	2.5	1.8
	Zinc concentrate....	4.2	0.71	0.51	39.83	0.09	55.90	1.2	5.4	51.0	3.4	1.4
	" middling.....	4.5	0.16	0.11	1.91	0.08	20.47	0.3	1.3	2.6	3.2	0.5
	Table concentrate...	3.3	0.20	0.07	0.40	0.70	100.55	0.3	0.5	0.4	20.9	1.9
	Tailing.....	55.5	nil	0.02	0.05	0.015	2.94	2.8	0.9	7.5	1.0
	Slimes.....	18.0	nil	0.09	0.51	0.010	3.39	4.1	2.8	1.6	0.4

Reagents Used:

Test No.	Reagent	Lb./ton	Remarks
1	Lead: Soda ash.....	4.0	Added to ball mill and ground 1 hour.
	Thiocarbanilide.....	0.15	" " " "
	Sodium cyanide.....	0.60	" " " "
	Zinc: Copper sulphate.....	2.00	Added to cells.
	Xanthate.....	0.40	Ore ground to 200 mesh.
2	Lead: Soda ash.....	6.0	Added to ball mill and ground 1 hour.
	Thiocarbanilide.....	0.20	" " " "
	Sodium cyanide.....	0.80	" " " "
	Cresylic acid.....	0.08	Added to cell.
	Zinc: Copper sulphate.....	2.00	"
	Xanthate.....	0.40	"
3	Pine oil No. 5.....	0.08	Ore ground to 200 mesh.
	Lead: Soda ash.....	3.0	Added to ball mill and ground 35 minutes.
	Coal tar, 60 per cent.....	0.40	" " " "
	Coal-tar creosote, 40 per cent.....		
	Pine oil No. 5 to froth.....		
4	Zinc: Copper sulphate.....	3.0	Added to cell.
	Xanthate.....	0.5	"
	Pine oil No. 5 to froth.....		
	Lead: Soda ash.....	4.0	Added to ball mill and ground 1 hour.
	Sodium cyanide.....	0.5	" " " "
	Thiocarbanilide.....	0.15	" " " "
	Zinc: Copper sulphate.....	2.0	Added to cell.
	Xanthate.....	0.4	"
	Pine oil No. 5 to froth.....		

Concentrates were cleaned without additional reagents.

CONCLUSIONS

Selective flotation followed by tabling the flotation tailing is a possibility on this ore. In test No. 1, 96 per cent of the lead, 84 per cent of the copper, 73 per cent of the gold, and 92 per cent of the silver reported in the lead-copper concentrate; 66 per cent of the zinc is recovered as a zinc concentrate. Selective flotation is preferable to bulk flotation where high zinc in the concentrate would interfere with the smelting of the concentrate. The 66 per cent of zinc recovered by selective flotation would pay for the operation and would mean lower smelting charges on the lead-copper concentrate.

Report No. 254

THE CONCENTRATION OF A LEAD ORE FROM THE FORBES GALENA MINE, PERTH ROAD, ONTARIO

C. S. Parsons

Shipment. A carload shipment, 78,400 pounds, of lead ore was received August 26, 1926, from the Forbes Galena Mines, Ltd., Perth Road, Ont. The property is better known as the Frontenac lead mine.

Characteristics and Analysis. The ore contains galena in a calcite gangue associated with pyrite and a small amount of zinc blende. The pyrite is present in sufficient quantity to interfere slightly with the concentration of the ore. The analysis of the carload was:

Lead 2.03 per cent

Purpose of Tests. The following mill test was conducted to determine a suitable flow-sheet for the concentration of the ore, which would utilize, with as few additions as possible, the equipment in the mill already on the property.

Experimental Tests. The ore was crushed by jaw crusher and rolls set at $\frac{1}{2}$ inch. A sample was cut by a Vezin sampler taking a tenth cut. This sample was reduced by grade crushing and division until a sample for analysis was obtained. The roll discharge passed over a set of three slotted screens, $\frac{1}{4}$, $\frac{1}{8}$, and $\frac{1}{16}$ inch. The $+\frac{1}{4}$ mesh was held until all the shipment had been crushed, and was then passed through the rolls again. The $-\frac{1}{4}+\frac{1}{8}$ obtained from this recrushing was mixed with the $-\frac{1}{4}+\frac{1}{8}$ obtained from the primary crushing. The $-\frac{1}{8}+\frac{1}{16}$ and $-\frac{1}{16}$ from the recrushing was sampled, weighed, and run separately from the corresponding sizes obtained from the primary crushing. This accounts for portions marked No. 1 and No. 2 in the results of the tests. No. 1 is the portion obtained by primary crushing, and No. 2 by recrushing operation.

Table No. 1 gives the weights and analyses of the sized products to be concentrated.

Table No. 2 gives results of jigging the $-\frac{1}{4}+\frac{1}{8}$ material.

Table No. 3 gives the results of jigging the two portions $-\frac{1}{8}+\frac{1}{16}$.

Table No. 4 gives the weights and analyses of the two $-\frac{1}{16}$ products.

Table No. 5 gives the results of tabling No. 1 portion $-\frac{1}{16}$.

Table No. 6 gives the results of classifying and tabling No. 2 portion $-\frac{1}{16}$. (See notes on operation.)

Table No. 7 gives the results of a table test made on the rejects of the head sample all crushed to 20 mesh. This test was made to check the head sample.

Table No. 8 contains a summary of all the concentrates produced during the tests.

Table No. 9 contains a summary of the hutch products from the jig which were not re-treated.

Table No. 10 contains a summary of all middling products which were not re-treated.

Table No. 11 contains a summary of all tailing products.

Table No. 12 is a balance sheet of the tests on the entire shipment.

Table No. 1

Product	Weight		Analysis Pb per cent	Per cent of values	Remarks.
	Lb.	Per cent			
Head.....	79,647	100.0	2.03	Difference between weight of heads and totals due to material used in sampling.
$-\frac{1}{4}+\frac{1}{8}$	13,057	17.0	1.47	11.9	
$-\frac{1}{8}+\frac{1}{16}$	30,063	39.2	2.08	38.6	
$-\frac{1}{16}$ (1).....	14,982	19.5	2.28	21.1	
$-\frac{1}{16}$ (2).....	18,635	24.3	2.47	28.4	
Total.....	76,737	100.0	Calculated head, 2.11 per cent Pb.

Table No. 2
(Jig details, $-\frac{1}{4}+\frac{1}{8}$)

Feed.....	13,057.0	100.0	1.47	Jig bed used to bed next size, hence difference between head and calculated head.
Concentrate.....	147.0	1.13	82.91	74.2	
Hutch.....	36.5	0.27	17.13	3.7	
Middling.....	124.0	0.95	15.89	12.0	
Tailing.....	12,749.5	97.65	0.13	10.1	
Total.....	13,057.0	100.0	

Table No. 3
(Jig details, $-\frac{1}{8}+\frac{1}{16}$)

No. 1 portion.....	16,259.0	54.08	1.95	51.0	
No. 2 portion.....	13,804.0	45.92	2.20	49.0	
Total.....	30,063.0	100.0	
Concentrate.....	626.5	2.08	84.25	84.2	
Hutch.....	88.0	0.29	38.93	5.4	
Middling.....	178.5	0.59	13.61	3.9	
Tailing.....	29,170.0	97.04	0.14	6.5	
Total.....	30,063.0	100.0	

Table No. 4
(Details $-\frac{1}{16}$ Tabled)

No. 1 portion.....	14,982.0	44.57	2.28	42.6	
No. 2 portion.....	18,635.0	55.43	2.47	57.4	
Total.....	33,617.0	100.0	

Table No. 5
(Tabling No. 1 portion of $-\frac{1}{16}$)

Head.....	14,982.0	100.0	2.28	
Concentrate.....	442.5	2.97	78.83	93.2	
Middling.....	40.5	0.26	21.61	2.2	
Tailing.....	14,499.0	96.77	0.12	4.6	
Total.....	14,982.0	100.0	

Table No. 6
(Tabling No. 2 portion of $-\frac{1}{16}$)

Product	Weight		Analysis, Pb per cent	Per cent of values
	Lb.	Per cent		
Head.....	18,635.0	100.0	2.47	100.0
1st spigot concentrate.....	321.0	1.72	80.71	63.1
1st spigot middling.....	136.0	0.73	9.56	3.1
2nd spigot concentrate.....	53.0	0.28	73.37	9.3
2nd spigot middling.....	367.5	1.97	8.50	7.0
Slime concentrate.....	38.5	0.21	78.63	7.5
Slime middling.....	70.0	0.38	6.78	1.2
Tailing.....	17,649.0	94.71	0.19	8.2
Total.....	18,635.0	100.0		

Table No. 7
(Tabling rejects from sampling)

Concentrate.....	26.50	3.6	62.69	90.2
Middling.....	2.75	0.4	8.50	1.4
Tailing.....	712.75	96.0	0.22	8.4
Total.....	742.00	100.0		

Table No. 8

Concentrates—				
Jig $-\frac{1}{4}+\frac{1}{8}$	147.0	8.9	82.91	9.1
$-\frac{1}{8}+\frac{1}{16}$	626.5	37.9	84.25	39.3
Tabling $-\frac{1}{16}$ (1).....	442.5	26.7	78.83	26.0
Tabling 1st spigot.....	321.0	19.4	80.71	19.3
Tabling 2nd spigot.....	53.0	3.2	73.37	2.9
Slimes.....	38.5	2.3	78.63	2.2
Rejects.....	26.5	1.6	62.69	1.2
Total.....	1,655.0	100.0		
Average of all concentrates.....			81.175%	

Table No. 9

Hutch products—				
$-\frac{1}{4}+\frac{1}{8}$	36.5	29.3	17.13	15.4
$-\frac{1}{8}+\frac{1}{16}$	88.0	70.7	38.93	84.6
Total.....	124.5	100.0		
Average of hutch products.....			32.543%	

Table No. 10

Middlings—				
Jig $-\frac{1}{4}+\frac{1}{8}$	124.0	13.5	15.89	19.4
$-\frac{1}{8}+\frac{1}{16}$	178.5	19.4	13.61	23.8
Table $-\frac{1}{16}$ (1).....	40.5	4.4	21.61	8.6
Table 1st spigot.....	136.0	14.8	9.36	12.5
Table 2nd spigot.....	367.5	40.0	8.50	30.8
Slimes.....	70.0	7.6	6.78	4.7
Rejects.....	2.75	0.3	8.50	0.2
Total.....	919.25	100.0		
Average of all middlings.....			11.062%	

Table No. 11

Product	Weight		Analysis, Pb per cent	Per cent of values
	Lb.	Per cent		
Tailings—				
Jig $-\frac{1}{4} + \frac{1}{8}$	12,740.5	17.0	0.13	15.0
$-\frac{3}{8} + \frac{1}{16}$	20,170.0	39.0	0.14	37.1
Table $-\frac{1}{16}$ (1).....	14,499.0	19.4	0.12	15.9
Table $-\frac{1}{16}$ (2).....	17,649.0	23.6	0.19	30.5
Rejects.....	712.75	1.0	0.22	1.5
Total.....	74,780.25	100.0		
Average of all tailings.....			0.147%	

Table No. 12

Concentrates.....	1,655.00	2.14	81.175	84.3
Hutch products.....	124.50	0.16	32.543	2.5
Middlings.....	919.25	1.18	11.082	6.3
Tailings.....	74,780.25	96.52	0.147	6.9
Totals.....	77,479.00	100.0		
Head sample from entire test.....			2.06% Pb.	

NOTES ON OPERATION

James jigs on $-\frac{1}{4} + \frac{1}{8}$ -inch size. The jigs made an excellent separation on this size, producing a very high-grade concentrate and clean tailing. A higher grade concentrate could have been made without affecting the tailing, by increasing the amount of middling. The jigs were operated in series. There was no difficulty in keeping the iron out of the concentrate.

James jigs on $-\frac{1}{8} + \frac{1}{16}$ -inch size. The jigs also made an excellent separation on this size. The middling was made on the bed of the second jig. After the operation of the jig was adjusted, a clean hutch was obtained, but no clean-up was made to separate the dirty hutch product from the clean, hence the grade of only 38.93 per cent for the total hutch product as shown in table No. 3.

Two James automatic jigs were operated as follows:

No. 1 made a concentrate, a hutch, and a tailing. The tailing was fed to No. 2 which made a concentrate, a hutch, and a final tailing. The $-\frac{1}{4} + \frac{1}{8}$ -inch size was jigged under the following conditions:

No. 1 jig—Water 3-ft. head; $\frac{1}{2}$ -inch throw; about 260 r.p.m.; 12-mesh screen on hutch.

No. 2 jig—Water 2-ft. head; $\frac{3}{8}$ -inch throw; about 260 r.p.m.; 12-mesh screen on hutch.

The $-\frac{1}{8} + \frac{1}{16}$ size was jigged under the following conditions:

No. 1 jig—Water 2-ft. head; $\frac{3}{8}$ -inch throw; about 260 r.p.m.; 12-mesh screen on hutch.

No. 2 jig—Water 2-ft. head; $\frac{1}{4}$ -inch throw; about 260 r.p.m.; 12-mesh screen on hutch.

Rates of feed to jigs: 2,380 lb./hr. (See Table No. 2).

1,700 lb./hr. (See Table No. 3).

Tabling the $-\frac{1}{16}$ inch. A large amount of experimental work was done on this size before a flow-sheet was developed, and classifiers and tables adjusted to work properly. Portion No. 1 was the material used in making and trying out different flow-sheets and machine adjustments. The first flow-sheet used was as follows: the $-\frac{1}{16}$ -inch material screened over a 24-mesh Callow belt screen, sending the -24 to Fahrenwald classifier. The $+24$ mesh going to a quarter-size Butchart roughing table. The six spigot discharges of the classifier were grouped into two products, the coarse product was fed to a quarter-size Plat-O table and the finer sizes product to a quarter-size Wilfley table. The slimes were thickened in a cone tank and fed to a standard Wilfley table. The roughing table treating the $+24$ mesh got the bulk of the feed and made an excellent separation. The large Wilfley handling the slimes also made a good separation, but the two quarter-size tables handling the classified feed did not get sufficient material on them to enable a proper cut to be obtained between the lead and iron sulphides. The flow-sheet was abandoned as being of no practical use to meet the local conditions at the mine.

A second flow-sheet was tried using a Richards launder type classifier with two chambers, straight on the $-\frac{1}{16}$ -inch material. It took considerable time to find the proper adjustments, but one was finally obtained which could be maintained without difficulty and which produced excellent results. This launder type classifier is a direct type pulsator and could be converted into a jig by placing a screen on the sorting column. A coarse spigot product from the classifier was tabled on the Butchart roughing table, an excellent separation being obtained. The classifier was adjusted to give a product containing all the coarse lead. The feed to the table under this condition was light, but very high grade and the bulk of the lead in the $-\frac{1}{16}$ -inch size was obtained from this table. The spigot from the second chamber contained the bulk of the feed, the classifier being operated to de-slime only. The slimes were thickened in a cone tank. The spigot product from this second classifier chamber containing the bulk of the sands was low in lead. This was fed to the Plat-O table and at least two-thirds of the $-\frac{1}{16}$ -inch material passed over this table. By loading the table in this way a very good separation was obtained. Unfortunately, a number of adjustments had to be made and it was not until the latter part of the run that a final and satisfactory adjustment was obtained. A sample taken during this period to represent the grade of concentrate which might be expected under operating conditions gave 80.30 per cent lead.

It is well to point out here that the results obtained on all tables could have been improved greatly if more material had been available to allow correct adjustments to be obtained. The various adjustments which had to be made were, size of plug to use in the spigot discharge of classifier, amount of water, etc. The Plat-O table was being used for the first time and difficulty was experienced in obtaining a suitable device for cutting the concentrate from the middling at the edge of the table. Also the table was run for some time, with too long a stroke, causing the coarse sands to

crowd over the concentrate. It was also found necessary to load the table heavily in order to make it work perfectly and, as the tables have a very large capacity on this class of material, at no time was it possible to get a full load on them. Each of the quarter-size tables would have taken a feed of over one ton per hour. The classifier was fed at a rate of $1\frac{1}{2}$ tons per hour during the latter part of the test on portion No. 2. A head of 12 feet of water was used, and the speed of the coarse compartment was 320 pulsations a minute, or 160 r.p.m. A $\frac{3}{4}$ -inch opening in the plug on the coarse and $\frac{1}{2}$ -inch for the fine discharge was used. The coarse compartment had a $2\frac{1}{2}$ -inch diameter sorting column, the fine chamber a 4-inch diameter sorting column. This type of classifier handled the $\frac{1}{16}$ -inch size (slotted screen) feed perfectly. An 8-foot cone was used for thickening the slime overflow. This gave a clear water overflow. Table No. 6 shows the results of the operation on portion No. 2.

Table No. 5 is given so that a balance may be struck and enable a calculation to be made to check the original head sample. No attention should be given this table in regard to indicating separations, as the products obtained are a mixture of a number of different flow-sheets.

RECOMMENDED FLOW-SHEET

The flow-sheet recommended is as follows: crushing to $\frac{1}{4}$ -inch, sizing on $\frac{1}{8}$ - and on $\frac{1}{16}$ -inch screens. This gives two jig sizes. In order to obtain capacity on the jigs and still make a separation between the iron and the lead, this screen ratio of 2 is believed to be necessary. The $\frac{1}{16}$ -inch material can be fed to two hydraulic classifiers of simple construction. This gives three table sizes, coarse, medium, and slime overflow, the slime overflow to be thickened in a cone tank. The classifiers should be adjusted to give the proportions as described above. The middling from the jigs and tables can be reground and returned with the original feed.

This flow-sheet can be adapted to the present mill without any radical changes being necessary, and no additional machinery will be required except the two simple hydraulic classifiers which can be constructed on the job.

The tables should be loaded as heavily as possible and two spigot products seem all that are necessary in the way of classification.

Report No. 255**EXPERIMENTAL TESTS ON ORE FROM THE ARGONAUT MINE,
LARDER LAKE, ONT.****J. S. Godard**

Shipment. Two shipments of ore were received from the Argonaut Consolidated Mines Limited, Larder Lake, Ont. Shipment No. 1, received August 31, 1926, consisted of 70 pounds of sample No. 1 and 80 pounds of sample No. 2. Shipment No. 2 received September 23, 1926, weighed 75 pounds and was marked "1018."

Characteristics of the Ore. The samples received were of a gold-copper ore. The gold was present mostly as free gold, and the copper as a massive chalcopyrite. The gangue resembles syenite in composition. Sample 1018 contained a small amount of chalmersite, a magnetic chalcopyrite of about the following composition— Cu_2S , Fe_4S_5 . Considerable magnetite and some graphite were also present.

Analysis:

	Copper
Sample No. 1.....	0.80 per cent
" 2.....	1.35 "
" 1018.....	3.02 "

Purpose of Experimental Tests. The flow-sheet of the Argonaut mill is as follows: grinding in a ball mill, followed by a classifier in closed circuit with a tube mill. The classifier discharge passes over a set of amalgamation plates before being floated in a K & K flotation machine. The tube mill discharge passes over another set of amalgamation plates before discharging into the classifier. Ore from different levels in the mine is at present being milled.

The results from floating the ore represented by Sample No. 1 were unsatisfactory. The recoveries were poor and the froth watery, and greyish in colour. Ore as represented by sample No. 2 behaved normally. The purpose of these tests was to ascertain, if possible, the reason for the difference in behaviour of the two samples and to apply corrective measures.

*Experimental Tests**Sample No. 1:*

Test No.	Product	Weight, per cent	Analysis, Cu per cent	Per cent of values	Remarks
1	Concentrate.....	6.0	12.24	94.0	Ore ground -100 mesh. Froth fair.
	Tailing.....	94.0	0.05	6.0	
2	Concentrate.....	4.4	16.52	92.7	Ore ground 18% +200 mesh. Bluish grey froth.
	Tailing.....	95.6	0.06	7.3	
3	Concentrate.....	7.2	10.56	93.0	Bluish grey froth.
	Tailing +200.....	17.8	0.07	1.5	
	-200.....	75.0	0.06	5.5	
4	Concentrate.....	4.5	16.84	96.3	Froth bright colour.
	Tailing +200.....	15.9	0.03	0.6	
	-200.....	79.6	0.03	3.1	
5	Concentrate.....	4.3	16.52	88.2	Ore ground 15% +200 mesh. Bluish grey froth. Table concen- trate high in magnetite.
	Table concentrate....	6.5	0.23	1.8	
	" middling.....	2.7	0.09	0.2	
	" tailing.....	61.5	0.08	6.1	
	Slimes.....	25.0	0.12	3.7	
7	Concentrate.....	6.6	10.46	90.8	Froth bluish grey
	Tailing +200.....	8.4	0.12	1.3	
	-200.....	85.0	0.07	7.9	
8	Concentrate.....	5.0	14.66	96.3	Froth superior to test No. 7. Colour bright.
	Tailing +200.....	7.4	0.03	0.3	
	-200.....	87.6	0.03	3.4	
9	Concentrate.....	4.7	19.08	93.1	Head sample high, 0.96% Cu. Froth good, colour bright.
	Tailing +200.....	9.6	0.07	0.7	
	-200.....	85.7	0.07	6.2	
10	Concentrate.....	4.8	14.08	94.7	Froth good, colour bright. Pulp alkaline.
	Tailing +200.....	7.0	0.04	0.4	
	-200.....	88.2	0.04	4.9	
11	Concentrate.....	6.1	12.32	95.2	Froth good, colour bright. Pulp alkaline.
	Tailing +200.....	7.5	0.04	0.4	
	-200.....	86.4	0.04	4.4	
12	Concentrate.....	7.7	10.50	95.8	Conditions good.
	Tailing +200.....	6.1	0.04	0.2	
	-200.....	86.2	0.04	4.0	
13	Concentrate.....	9.5	7.46	90.4	Froth bluish grey.
	Tailing +200.....	6.3	0.15	1.1	
	-200.....	84.2	0.08	8.5	

6. Test for soluble salts:

Iron, ferrous.....	0.0025 grm./litre
Lime.....	0.0113
Solution alkaline—	
Magnesia.....	0.0086 "
Sulphur trioxide.....	0.0103 "

Reagents Used:

Test No.	Reagent	Amount, lb./ton	Added to
1	Soda ash.....	3.0	Ball mill.
	Coal-tar.....40%	0.35	"
	Coal-tar creosote.....60%	0.04	Cell.
	Pine oil Floto brand		
2	Lime.....	4.0	Ball mill.
	Xanthate.....	0.2	Cell.
	Pine oil No. 5.....	0.04	"
3	Lime.....	3.0	Ball mill.
	Xanthate.....	0.2	Cell.
	Pine oil No. 5.....	0.04	"
	Copper sulphate.....	0.5	"
4	Soda ash.....	3.00	Ball mill.
	Xanthate.....	0.2	Cell.
	Pine oil No. 5.....	0.04	"
5	Lime.....	3.00	Ball mill.
	Coal-tar creosote.....	0.30	"
	Cresylic acid.....	0.08	"
	Pine oil No. 5.....	0.04	Cell.
7	Lime.....	3.0	Ball mill.
	Barrett's No. 4.....	0.34	"
	Pine oil No. 5.....	0.04	Cell.
8	Soda ash.....	3.00	Ball mill.
	Barrett's No. 4.....	0.34	"
	Pine oil No. 5.....	0.04	Cell.
9	Lime.....	2.00	Ball mill.
	Coal-tar creosote.....	0.25	"
	Cresylic acid.....	0.08	"
	Pine oil No. 5.....	0.04	Cell.
10	Soda ash.....	2.00	Ball mill.
	Coal-tar creosote.....	0.25	"
	Cresylic acid.....	0.08	"
	Pine oil No. 5.....	0.04	Cell.
11	Lime.....	1.00	Ball mill.
	Coal-tar creosote.....	0.20	"
	Cresylic acid.....	0.08	"
	Pine oil No. 5.....	0.04	Cell.
12	Soda ash.....	1.00	Ball mill.
	Coal-tar creosote.....	0.20	"
	Cresylic acid.....	0.08	"
	Pine oil No. 5.....	0.04	Cell.
13	Lime.....	6.00	Ball mill.
	Coal-tar creosote.....	0.20	"
	Cresylic acid.....	0.08	"
	Pine oil No. 5.....	0.04	Cell.

Sample No. 2:

Test No.	Product	Weight, per cent	Analysis, Cu per cent	Per cent of values	Remarks
1	Concentrate.....	8.2	16.32	95.4	Ore ground -100 mesh.
	Tailing.....	91.8	0.07	4.0	Froth good.
2	Concentrate.....	6.0	21.30	95.8	Ore ground +150 mesh.
	Tailing.....	94.0	0.06	4.2	Froth good.
3	Concentrate.....	7.6	17.50	95.3	Ore ground 18% +200 mesh.
	Table concentrate....	6.9	0.25	1.2	Froth good. Table concentrate high
	" middling.....	3.3	0.10	0.2	in magnetite.
	" tailing.....	48.9	0.06	2.1	
	" slime.....	33.3	0.05	1.2	
5	Concentrate.....	7.6	17.38	96.5	Conditions good.
	Tailing +200.....	7.6	0.06	0.4	
	-200.....	84.8	0.05	3.1	

4. Soluble salts:

Iron, ferrous.....	0.0015 gm./litre.
Lime.....	0.0137
Solution alkaline—	
Magnesia.....	0.0149 "
Sulphur trioxide.....	0.0103 "

Reagents Used:

Test No.	Reagent	Amount, lb./ton	Added to
1	Soda ash.....	3.0	Ball mill.
	Water-gas tar.....	0.37	"
	Pine oil No. 5.....	0.04	Cell.
2	Lime.....	4.00	Ball mill.
	Xanthate.....	0.20	Cell.
	Pine oil No. 5.....	0.04	"
3	Lime.....	3.00	Ball mill.
	Coal-tar creosote.....	0.28	"
	Cresylic acid.....	0.08	"
	Pine oil No. 5.....	0.04	Cell.
5	Lime.....	1.0	Ball mill.
	Coal-tar creosote.....	0.20	"
	Cresylic acid.....	0.08	"
	Pine oil No. 5.....	0.04	Cell.

Sample 1018

Test No.	Product	Weight, per cent	Analysis Cu per cent	Per cent of values
1	Concentrate.....	26.0	14.68	96.5
	Tailing +200.....	8.8	0.40	0.9
	-200.....	65.2	0.16	2.6
2	Concentrate.....	27.0	14.44	98.2
	Tailing +200.....	2.9	0.21	0.2
	-200.....	70.1	0.09	1.6
3	Concentrate.....	24.9	16.04	97.8
	Tailing +200.....	3.2	0.33	0.3
	-200.....	71.9	0.11	1.9
4	Concentrate.....	30.2	13.30	97.8
	Tailing +200.....	4.4	0.25	0.3
	-200.....	65.4	0.12	1.9
5	Concentrate.....	23.4	16.74	97.8
	Tailing +200.....	3.3	0.25	0.2
	-200.....	73.3	0.11	2.0
6	Concentrate.....	25.1	15.64	98.1
	Tailing +200.....	2.3	0.21	0.1
	-200.....	72.6	0.10	1.8

Reagents Used:

Test	Reagents	Lb./ton	Added to	Remarks
1	Lime.....	1.5	Ball mill...	Froth good, colour good, time of contact 20 minutes.
	Coal-tar creosote.....	0.20	"	
	Cresylic acid.....	0.08	"	
	Pine oil No. 5.....	0.08	Cell.....	
2	Soda ash.....	1.5	Ball mill...	Froth good, colour good, time of contact 20 minutes.
	Coal-tar creosote.....	0.20	"	
	Cresylic acid.....	0.08	"	
	Pine oil No. 5.....	0.08	Cell.....	
3	Lime.....	1.5	Ball mill...	Froth good, colour good, contact 1 hour.
	Thiocarbamilide.....	0.20	"	
	Pine oil No. 5.....	0.08	Cell.....	
4	Soda ash.....	1.5	Ball mill...	Froth good, lively; colour good. Conc. not so good as No. 3.
	Thiocarbamilide.....	0.20	"	
	Pine oil No. 5.....	0.08	Cell.....	
5	Lime.....	1.5	Ball mill...	Lime 60-minute contact. Xanthate 5-minute contact. Froth good, colour good.
	Xanthate.....	0.20	"	
	Pine oil No. 5.....	0.08	Cell.....	
6	Soda ash.....	1.5	Ball mill...	Soda ash 60-minute contact. Xanthate 5-minute contact. Froth good, colour good.
	Xanthate.....	0.20	"	
	Pine oil No. 5.....	0.08	Cell.....	

Composite Samples:

Test	Product	Weight, per cent	Analysis, Cu per cent	Per cent of values	Ore used
A	Concentrate.....	6.8	10.76	89.7	1,000 grms. sample No. 1.
	Tailing +200.....	9.9	0.09	1.1	
	-200.....	83.3	0.09	9.2	
B	Concentrate.....	11.0	8.78	94.0	500 grms. sample No. 1. 500 " " " 2.
	Tailing +200.....	12.6	0.10	1.2	
	-200.....	75.5	0.07	4.8	
C	Concentrate.....	15.9	14.42	97.0	500 grms. sample No. 1. 500 " " " 1018.
	Tailing +200.....	7.3	0.12	0.4	
	-200.....	76.8	0.08	2.6	
D	Concentrate.....	19.4	13.32	97.2	500 grms. sample No. 2. 500 " " " 1018.
	Tailing +200.....	5.2	0.17	0.3	
	-200.....	75.4	0.09	2.5	
E	Concentrate.....	13.5	14.48	97.0	333 grms. sample No. 1. 333 " " " 2. 333 " " " 1018.
	Tailing +200.....	10.1	0.08	0.4	
	-200.....	76.4	0.07	2.6	
F	Concentrate.....	17.5	11.10	97.7	Same as E, adding 8.3 grms. carbonaceous material as found in sample No. 1018.
	Tailing +200.....	5.7	0.12	0.4	
	-200.....	76.8	0.05	1.9	
G	Concentrate.....	7.1	10.08	95.1	1,000 grms. sample No. 1.
	Tailing +200.....	8.6	0.04	0.4	
	-200.....	84.3	0.04	4.5	
H	Concentrate.....	7.5	9.48	94.0	1,000 grms. sample No. 1.
	Tailing +200.....	8.2	0.04	0.4	
	-200.....	84.3	0.05	5.6	

Reagents:

Tests A-F: Lime 1.5 lb./ton added to ball mill, 45-minute contact.

Xanthate 0.14 lb./ton added to ball mill, 5-minute contact.

Pine oil No. 5, 0.08 lb./ton added to cell.

Test G: Same as above except that no lime was added.

Test H: Same as A-F, copper sulphate 0.40 lb./ton added to ball mill with xanthate, 5-minute contact.

The froth in each test was satisfactory. Colour, golden.

CONCLUSIONS

Flotation Tests on Sample No. 1. Generally good results were obtained on ore as represented by this sample. The addition of three or more pounds per ton of lime to the ball mill affected the froth, making it bluish grey in colour and the recovery somewhat lower. Using soda ash to provide the alkalinity gave good recoveries. The soluble salts were insufficient in quantity to cause any trouble.

Flotation Tests on Sample No. 2. Good results were obtained on this sample. The quantity of soluble salts present was negligible.

Flotation Tests on Sample No. 1018. Good results were also obtained on this sample.

Flotation Tests on Mixtures of all Three Samples. Good results were obtained on the various mixtures used to constitute the feeds.

RECOMMENDATIONS

The following suggestions are offered as a means of avoiding the undesirable bluish watery froth, and decreasing the copper in the tailing.

(1) Grinding. That the dilution of the ball and tube mills be decreased to about 60 per cent solids. Too thick a pulp in grinding is known to affect flotation.

(2) That a small quantity of lime, not exceeding 1.5 pounds per ton be added to the ball mill.

(3) That a soluble reagent, such as xanthate, be used in place of the coal-tar creosote and pine oil mixture at present being used. The xanthate should be added in the form of a solution at the end of the amalgamation plates. Coal-tar creosote should be added to the ball mill in order to allow a thorough mixing with the pulp and provide a time contact, but as it would seriously interfere with the amalgamation if added here, the change to xanthate is suggested.

Report No. 256

EXPERIMENTAL TESTS ON GOLD ORE FROM THE KITCHENER MINE,
EAST CENTRAL MANITOBA

J. S. Godard

Shipments. Two shipments of similar ore were received during September, 1926, from the Rice Lake district, Manitoba, one weighing 25 pounds and the other 64 pounds.

Characteristics of the Ore. The samples received contained about 0.75 ounce gold per ton, about 90 per cent of which is free, the remainder being associated with small quantities of the sulphides of iron, copper, and nickel.

Purpose of Tests. The purpose of these tests was to compare concentration and cyanidation as means of recovering the gold.

Sampling and Analysis. The head sample of the combined shipments contained—

Gold.....	0.78 oz./ton
Copper.....	0.40 per cent
Lead.....	nil
Zinc.....	trace
Iron.....	1.67 per cent
Nickel.....	0.009 "
Insoluble.....	92.54 "

Test No. 1. Flotation and Tabling

Results:

Product	Weight, per cent	Assay		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate.....	4.2	7.84	8.36	56.5	92.6
Table concentrate.....	6.5	3.18	0.16	35.4	2.6
Table tailing.....	67.6	0.06	0.02	7.0	3.7
Slimes.....	21.7	0.03	0.02	1.1	1.1
Head from products.....		0.58	0.38		

Recovery in concentrates: Au..... 91.9 per cent
Cu..... 95.2 "

Test No. 2. Flotation and Tabling

Product	Weight, per cent	Assays		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate.....	3.2	11.13	10.71	54.3	89.4
Table concentrate.....	8.8	2.70	0.16	36.2	3.6
Table tailing.....	59.4	0.09	0.03	8.2	4.7
Slimes.....	23.6	0.03	0.03	1.3	2.3
Head from products.....	0.66	0.38

Recovery in concentrates: Au..... 90.5 per cent
Cu..... 93.0 "

Test No. 3. Amalgamation

Screen test on tailings—

Mesh	Weight, per cent	Assay Au oz./ton	Per cent of values
+48.....	0.7	0.14	1.0
+65.....	7.3	0.68	5.5
+100.....	14.9	0.12	17.0
+150.....	16.5	0.12	18.8
+200.....	16.3	0.10	15.5
-200.....	44.3	0.10	42.2
Average tailing.....	0.105

Recovery, 85.7 per cent

Test No. 4. Amalgamation

Screen test on tailings—

Mesh	Weight, per cent	Assay Au oz./ton	Per cent of values
+100.....	1.5	0.12	2.1
+150.....	9.0	0.11	11.6
+200.....	21.1	0.09	22.2
-200.....	68.4	0.08	64.1
Average tailing.....	0.085

Recovery, 88.5 per cent

Test No. 5. Amalgamation and Tabling

Product	Weight, per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Table concentrate.....	8.5	0.68	1.42	53.9	30.0
Table tailing.....	63.8	0.06	0.28	35.8	44.5
Slimes.....	27.7	0.04	0.37	10.3	25.5
Amalgamation tailing.....		0.107	0.40		
Table tailing screened on 150 mesh—					
+150.....	13.7	0.06	0.06	13.7	3.0
-150.....	86.3	0.06	0.31	86.3	97.0
Average tailing.....		0.06	0.28		

Distribution of gold: In amalgam..... 86.3 per cent
 table concentrate..... 7.4 "
 table tailing..... 4.9 "
 slimes..... 1.4 "
 Recovery..... 93.7 "

Test No. 6. Amalgamation and Flotation

Product	Weight, per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Flotation concentrate.....	5.3	1.16	6.62	76.5	92.5
Flotation tailing.....	94.7	0.02	0.03	23.5	7.5
Amalgamation tailing from products.....		0.08	0.38		

Distribution of gold: In amalgam..... 89.7 per cent
 flotation concentrate..... 7.9 "
 flotation tailing..... 2.4 "
 Recovery..... 97.6 "

Test No. 7. Amalgamation, Flotation, and Tabling

Product	Weight, per cent	Assay		Per cent of values	
		Au oz/ton	Cu per cent	Au	Cu
Flotation concentrate.....	5.2	1.12	6.96	71.5	93.9
Table concentrate.....	3.8	0.28	0.09	13.0	0.9
Table tailing.....	54.2	0.023	0.01	15.4	1.4
Slimes.....	36.8	tr.	0.04		3.8
Amal. tailing from products.....		0.081	0.39		
Table tailing screened on 200 mesh—					
+200.....	15.3	0.04	0.01	26.5	15.0
-200.....	84.7	0.02	0.01	73.5	85.0
Average tailing.....		0.023	0.01		

Distribution of gold: In amalgam..... 90.0 per cent
 flotation concentrate..... 7.2 "
 table concentrate..... 1.4 "
 table tailing..... 1.4 "
 Recovery..... 98.6 "

Head sample tests Nos. 5-7: Gold, 0.78 oz./ton. Copper, 0.40 per cent.

Test No. 8. Six-cycle Cyanidation Test

A six-cycle cyanide test was made to obtain some data on the following points:

1. The cyanide and lime consumption.
2. The amount of cyanide soluble copper.
3. The effect of the fouling of solution due to presence of copper on the dissolution of gold.

The procedure was as follows:

For cycle No. 1, six pebble jars were used for grinding the ore. Each jar was charged with 750 grammes of ore at -20 mesh. The grinding was done in 1:1 pulp with lime equivalent to 2 pounds per ton added to mill. The ore for cycle No. 1 was ground in water, that for all other cycles in solution from previous cycle.

For cycle No. 2, five lots of ore were used and for each succeeding cycle one lot less than for the previous one was used. Thus, for cycle No. 6 one lot of ore was used.

On completion of the grinding the pulp was washed into small glass agitators and agitated for 48 hours in 1:2.5 pulp. The cyanide strength was maintained at 0.075 per cent KCN by additions of cyanide twice each day. The protective alkalinity was kept at approximately 0.02 per cent CaO. Any solution withdrawn was replaced by solution from the previous cycle. After 48 hours' agitation the contents of each agitator were combined in a pressure filter and washed. The amount of wash solution used was based on 450 c.c. for every 750 grammes of tailings. The 450 c.c. consisted of 150 c.c. 0.05 per cent KCN, 150 c.c. 0.025 per cent KCN, and 150 c.c. water. The washings were combined with the pregnant solution for the succeeding cycle. Precipitation of the gold was not attempted between cycles. The solution from each cycle was analysed for copper content. On the completion of cycle No. 1 it was realized that the free gold interfered with the results, as it settled out in the agitators and made the results erratic. It was, however, decided to continue the work to obtain information on points 1 and 2. Head sample: gold 0.78 oz./ton; copper 0.40 per cent.

Cycle	Tailing assay Au oz./ton	Extraction, per cent	Reagents used lb./ton		Cu grm./l. in solution	—
			KCN	CaO		
1	0.20	74.46	2.67	3.41	0.1207	
2	0.37	52.6	3.41	2.48	0.2406	Ore 2% +200 mesh.
3	0.39	50.0	2.64	2.52	0.3422	Ore 2% +200 mesh.
4	0.31	60.2	2.77	2.03	0.4180	Ore 0.7% +200 mesh.
5	0.27	65.4	2.90	1.96	0.4810	Ore 1.1% +200 mesh.
6	0.10	87.2	2.78	2.20	0.5120	

The extent of the interference of free gold is shown by the following results taken from cycle No. 4 as being typical.

Cyanide tailing before amalgamation—

Mesh	Weight, per cent	Assay Au oz./ton	Per cent of values
+200.....	0.7	15.71	35.6
—200.....	99.3	0.21	64.4
Average.....		0.31	

Cyanide tailing after amalgamation..... 0.077 oz./ton.
 Gold amalgamation..... 75.2 per cent.

Test No. 9. Cyanidation Tests in Winchesters

A sample of 900 grammes of the ore at —20 mesh was ground in a pebble mill with 1 pound lime per ton. The pulp was filtered and divided into about three equal parts. One part was used for a head sample and each of the others put into a winchester bottle and made up to 1:2.5 dilution with the clear filter solutions. The cyanide strength was 0.075 per cent KCN and strengthened after 24 hours' agitation. The protective alkalinity was approximately 0.02 per cent CaO. Time of agitation 48 hours.

Results:

Part	Mesh	Weight, per cent	Assay Au oz./ton	Per cent of values
(1)	Head sample after grinding screened on 200 mesh.			
	+200.....	6.8	2.22	31.0
	—200.....	93.2	0.35	69.0
	Average.....		0.48	
(2)	Cyanide tailing screened on 200 mesh.			
	+200.....	6.1	0.12	28.0
	—200.....	93.9	0.02	72.0
	Average.....		0.026	
(3)	Cyanide tailing screened on 200 mesh.			
	+200.....	6.7	0.044	13.4
	—200.....	93.3	0.02	86.6
	Average.....		0.022	

Part	Assay		Extraction, per cent	Reagents used, lb./ton	
	Head	Tailing		KCN	CaO
(2).....	0.48	0.026	94.5	1.90	3.5
(3).....	0.48	0.022	95.4	2.27	3.5

Test No. 10—Amalgamation and Cyanidation (Three Cycles)

Cycle No. 1. Three lots of ore, each 900 grammes, were ground to about 5 per cent +200 mesh, then amalgamated one hour with 90 grammes of mercury in a 1:1 pulp. The amalgamation tailings were dewatered and

agitated 48 hours in 0.075 per cent KCN and 1:2 pulp. The washings were not united with the main solution. The tailings from the three agitators were screened on 200 mesh and assayed separately.

Cycle No. 2. Two lots of ore each of 900 grammes at -20 mesh were ground and amalgamated as in cycle No. 1. After dewatering the dilution was made up to 1:2 with solution from cycle No. 1. The agitation period and conditions were the same as in No. 1.

Cycle No. 3. One lot of ore of 900 grammes was used for this cycle, otherwise procedure was the same as for No. 2. The solution from each cycle was analysed for copper and nickel.

Results:

Cycle No. 1—Tailings screened on 200 mesh.

Part	Mesh	Weight, per cent	Assay Au oz./ton	Per cent of values	—
A	+200.....	5.6	0.18	21.2	Average tailing 0.0417 oz./ton.
	-200.....	94.4	0.04	78.8	
	Average.....		0.048		
B	+200.....	2.9	0.58	30.2	
	-200.....	97.1	0.04	69.8	
	Average.....		0.056		
C	+200.....	3.3	0.04	6.3	
	-200.....	96.7	0.02	93.7	
	Average.....		0.021		

Reagent consumption, lb./ton.

Part	KCN	CaO
A.....	2.60	2.84
B.....	2.67	3.10
C.....	2.67	3.00
Average.....	2.65	2.98

Cycle No. 2—Tailings screened on 200 mesh.

Part	Mesh	Weight, per cent	Assay Au oz./ton	Per cent of values	—
A	+200.....	4.8	0.13	24.6	Average tailing 0.018 oz./ton.
	-200.....	95.2	0.02	75.4	
	Average.....		0.025		
B	+200.....	4.8	0.07	17.1	
	-200.....	95.2	0.01	82.9	
	Average.....		0.011		

Reagent consumption, lb./ton.

Part	KCN	CaO
A.....	2.40	3.97
B.....	2.23	3.45
Average.....	2.31	3.71

Cycle No. 3—Tailing screened on 200 mesh.

Mesh	Weight, per cent	Assay Au oz./ton	Per cent of values
+200.....	4.8	0.06	13.1
-200.....	95.2	0.02	86.9
Average.....		0.022	

Reagent consumption, lb./ton.

KCN	2.12
CaO.....	3.64

Summary—Head sample: Au 0.78 oz./ton, Cu 0.40 per cent, Ni 0.009 per cent.

Cycle	Tailing, average assay	Extraction, per cent	Reagent consumption (average)		Analysis solution	
			KCN	CaO	Cu, grm./l.	Ni, grm./l.
1.....	0.021	97.3	2.65	2.98	0.11	0.0031
2.....	0.018	97.7	2.31	3.71	0.21	0.0042
3.....	0.022	97.2	2.12	3.64	0.28	0.0064

CONCLUSIONS

Concentration: Good recoveries were made by flotation and tabling. The ratio of concentration is 1:10. Flotation alone is not sufficient to recover the gold. A good recovery of the gold remaining in the flotation tailing is made in table concentrate. The flotation concentrate, uncleaned, contains about 10 per cent copper, which when cleaned would be doubled with an increase in the gold content. This would make a good smelter product. If this method were employed the table concentrate from the flotation could probably be successfully cyanided as its copper content is low, 0.16 per cent.

Amalgamation: Amalgamation gave very good results on this ore, 85 to 90 per cent being recovered by this treatment.

Amalgamation and Concentration: Amalgamation, followed by tabling, recovered 93.7 per cent of gold. The concentrate is low in both gold and copper—only 30 per cent of the copper present is recovered. When amalgamation is followed by flotation, good recoveries in both gold and copper are obtained. The flotation concentrate is higher in gold than that from the table. The grade of this concentrate could be considerably increased by a cleaning operation. When amalgamation is followed by flotation and tabling, 98.6 per cent gold and 94.8 per cent copper are recovered. The table concentrate is very low in grade.

Cyanidation:

1. *Extraction.* The extractions in test No. 8 were very poor, due to presence of free gold. In test No. 9 where bottles replaced agitators the extractions were greatly improved, 94 to 95 per cent being extracted. In test No. 10 where amalgamation preceded cyanidation, 97 per cent gold is recovered. The free gold would not interfere in large-scale operations.

2. *Reagent Consumption.* Two to three pounds of cyanide were used in all tests. The tests conducted in bottles showed a lower consumption than those in agitators. The ore was overground, which increases the consumption. Copper is the main cyanicide, though some nickel enters the solutions as shown in test No. 10. Two to nearly four pounds of lime were consumed to maintain a fair protective alkalinity. Good settling was obtained in the agitators when about three pounds per ton were used.

3. *Fouling of Solution.* Where agitators are used about one-tenth of a gramme per litre of copper appears after the first cycle, which gradually decreases until in the sixth cycle when three one-hundredths gramme per litre is the amount dissolved. On the completion of the sixth cycle 0.512 gramme per litre copper was present in the solution. This would represent a little more than one pound per ton of solution. Fineness of grinding is a big factor in the amount of soluble copper in a cyanide solution. While nickel enters the solution, it is of secondary importance, the copper being the greater cyanicide. From the above tests, amalgamation followed by flotation appears as the most satisfactory method of treating this ore. The operations would not be effected by a possible increase in the copper content with increase in the depth of mining operations. The problem of the disposal of the concentrate is the chief objection to this method.

The ore is apparently amenable to cyanidation, with a fairly high consumption of cyanide, but there is a possibility of the interference of the copper in the extraction and precipitation operations. Should this occur, running to waste of a certain amount of the solution might be sufficient remedy. At any rate it would prevent indefinite accumulation of the copper.

Report No. 257

CONCENTRATION AND SEPARATION OF THE VALUES IN A LEAD-ZINC -GOLD-SILVER ORE FROM BUNKER HILL CLAIMS, ALICE ARM, BRITISH COLUMBIA

C. S. Parsons

Shipment. A shipment of 200 pounds of ore was received October 16, 1926, from Edgar Trethewey, Alice Arm, B.C. The sample was from the Bunker Hill claims, Alice Arm.

Characteristics and Analysis of Sample. The sample received was a heavy sulphide ore containing galena, zinc blende, gold, and silver. The analysis of the sample was as follows:

Lead.....	12.72 per cent
Zinc.....	7.63 "
Gold.....	0.28 oz./ton
Silver.....	17.98 "
Iron.....	28.67 per cent

Purpose of Tests. The purpose of the experimental tests which follow was to determine a method of concentration and separation by which marketable lead and zinc concentrates would be produced. The recovery of the gold and silver values presents the most difficult problem and considerable experimental work was performed in an endeavour to increase the recoveries of these two metals.

EXPERIMENTAL TESTS

Tests Nos. 1 to 5 were a series of flotation tests, and tests Nos. 4a and 5a were made in an attempt to recover part of the gold remaining in the flotation tailings. In the selective flotation tests 1,000-gramme samples were used; these samples were taken from the original large sample which had been crushed to -14 mesh. The general procedure followed in making these tests was to grind the ore in a small ball mill with certain reagents and float first for the lead and then for the zinc after the addition of other reagents.

Reagents

Test No. 1

Soda carbonate.....	6.0 lb./ton	} Added to ball mill. Time contact 20 mins.
Sodium cyanide.....	0.25 "	

NOTE.—When tested before flotation the pulp was found to be acid.

Lead flotation—

Soda carbonate.....	3.0 lb./ton
Phospho-cresylic.....	0.15 "

Zinc flotation—

Soda carbonate.....	1.0	"	} Time contact of 3 mins. before floating.
Sodium sulphate.....	2.0	"	
Copper sulphate.....	2.0	"	
Xanthate.....	0.2	"	
Pine oil to froth			

Test No. 2

Soda carbonate.....	6.0 lb./ton	} Added to ball mill. Time contact 25 minutes.
Soda sulphite.....	2.0 "	

Lead flotation—

Soda carbonate.....	2.0 lb./ton
Phospho-cresylic.....	0.15 "

Zinc flotation—

Soda ash.....	1.0 lb./ton	} Time contact of 3 minutes before flotation.
Copper sulphate.....	2.0 "	
Xanthate.....	0.2 "	

Test No. 3.

A duplicate of No. 2, but with 2 pounds additional soda ash added to ball mill, making 8 pounds.

Test No. 4.

A duplicate of No. 1, but adding 10 pounds per ton of soda ash to ball mill and no additional soda ash to cells.

Test No. 5.

The ore was ground finer in this test.

Soda ash.....	12.0 lb./ton	} Time contact in ball mill 30 minutes.
Sodium cyanide.....	0.4 "	
Lead flotation—		
Phospho-cresylic.....	0.15 lb./ton	
Zinc flotation—		
Copper sulphate.....	1.5 "	} Time contact of 3 minutes before flotation.
Xanthate.....	0.3 "	

Test No. 5a. A cyanide test was made on the tailing from test No. 5. The extremely basic nature of this ore causes high consumption of cyanide; so that the ordinary method of 24 to 48 hours' contact is not practical. It was hoped that the gold and silver remaining in this tailing might be in such a form that would go into solution rapidly and that a 60 to 75 per cent recovery might be obtained in a few hours' contact, which would keep the consumption of cyanide within economic limits. The following test was, therefore, made with this idea in view:

Amount of flotation tailing used.....	500 grammes
Pulp made up to.....	1 : 2
KCN (0.1 per cent).....	0.78 gramme NaCN
Lime (5 lb./ton).....	1.25 " CaO
Time cyaniding.....	4 hours
Cyanide consumption.....	2.16 lb./ton
Lime consumption.....	2.9 "
Assay of flotation tailing (cyanide head).....	0.16 oz./ton
Assay of cyanide tailing (4 hours' contact).....	0.12 "
Extraction by cyaniding.....	23 per cent

Screen Test on Cyanide Tailing:

Mesh	Weight, grms.	Weight, per cent	Assay Au oz./ton	Content	Per cent of values
+200.....	57.5	11.3	0.17	1.58	12.9
-200.....	450.0	88.7	0.12	10.64	87.1

SUMMARY

Five flotation tests were run, one table test on the flotation tailing from test No. 4, and a cyanide test on the tailing from test No. 5. The table test and cyanide test were run in order to determine if part of the gold remaining in the flotation tailing could be recovered. In the first four flotation tests the ore was ground to approximately —65 mesh, 75 per cent passing 200 mesh. In the fifth test the ore was ground finer so that 87 per cent passed 200 mesh. Two different depressant reagents were used for the zinc and iron, namely, sodium cyanide and sodium sulphite.

CONCLUSIONS

The best practice indicated by these tests points to the production of a lead concentrate high in iron sulphides in order to recover as much gold as possible. A grade of 50 per cent lead seems to strike a balance. The lower the grade of lead product, the greater the loss of zinc in that product. The tests indicate that 90 per cent of the lead may be recovered in a concentrate containing 50 per cent lead, and 85 per cent of the zinc in a concentrate containing 45 per cent zinc. The grade of the zinc concentrate can without doubt be raised to 48 or 50 per cent by cleaning, without affecting the recovery. Approximately 50 per cent of the gold and 80 per cent of the silver can be recovered in the lead concentrate. The silver reporting in the zinc concentrate can also be considered recovered. The gold in the zinc concentrate is too low to be paid for by the smelter.

In regard to reagents, there is very little to choose between the use of cyanide and of sodium sulphite. Cyanide seems to be slightly the better reagent. Time contact of these reagents is essential, and a mill will require surge tanks to obtain at least 30 minutes' contact before the lead flotation and 10 minutes between the lead and zinc flotation.

The table and cyanide tests for the recovery of the gold and silver in the flotation tailing were not successful. These values are evidently locked up in the pyrite.

Results of Flotation Tests

Test No.	Product	Weight per cent	Assays					Percent of values			
			Pb percent	Zn percent	Au oz./ton	Ag oz./ton	Fe percent	Pb	Zn	Au	Ag
1	Lead concentrate.....	13.2	72.23	0.20	0.52	81.20	6.84	75.2	0.4	25.7	60.8
	Lead middling.....	10.6	12.27	7.47	0.66	30.3	29.88	10.2	11.0	26.3	18.2
	Zinc concentrate.....	13.8	5.48	44.95	0.02	10.02	12.27	5.7	82.8	1.0	7.5
	Zinc tailing.....	62.9	1.31	0.67	0.20	3.82	34.86	8.9	5.8	47.0	13.5
2	Lead concentrate.....	22.9	45.30	4.35	17.3	86.7	13.9
	Zinc ".....	13.2	6.69	35.84	17.0	7.4	66.0
	Tailing.....	63.9	1.10	2.25	35.02	5.9	20.1
3	Lead concentrate.....	19.2	59.90	3.58	0.84	77.76	11.06	85.3	9.0	47.7	79.4
	Zinc ".....	14.0	3.62	46.70	0.20	7.92	11.66	3.8	85.8	8.4	5.9
	Tailing.....	66.8	2.20	0.60	0.22	4.14	35.85	10.9	5.2	43.8	14.7
4	Lead concentrate.....	20.5	62.0	2.76	0.76	75.32	10.70	92.1	7.6	54.9	82.7
	Zinc ".....	15.9	1.51	41.37	0.28	6.17	16.0	1.7	87.9	9.4	4.4
	Tailing.....	63.6	1.35	0.52	0.16	3.80	36.02	6.2	4.5	35.7	12.9
4a	Tabling flotation tailing—
	Table concentrate.....	20.1	0.22	2.48	30.2	15.1
	" middling.....	34.4	0.10	2.48	23.4	25.8
	" tailing.....	23.8	0.14	3.46	22.7	17.9
5	" slimes.....	21.7	0.16	6.26	23.7	41.2
	Lead concentrate.....	23.2	49.29	4.64	0.52	65.4	91.7	14.6	49.0	83.2
	Zinc ".....	14.4	2.21	41.53	0.18	8.58	2.6	81.0	10.5	6.8
	Tailing.....	62.4	1.13	0.52	0.16	2.94	5.7	4.4	40.5	10.0

Report No. 258

CONCENTRATION TESTS ON ORE FROM THE GIANT MINE, SPILLAMACHEEN, BRITISH COLUMBIA

C. S. Parsons

Shipment. A shipment of 55 pounds of ore was received October 19, 1926, from the Pacific Mines, Spillamacheen, B.C. The sample was from the Giant claims.

Characteristics and Analysis. The sample received consisted of galena associated with a barite gangue. Analysis:

Lead.....	10.1 per cent
Zinc.....	trace
Gold.....	nil
Silver.....	1.36 oz./ton
Barite.....	47.10 per cent
Insoluble (other than barite).....	36.10 "

Purpose of Tests. This ore contains two marketable minerals, namely, lead and barite (barium sulphate). The tests which follow were made for the purpose of determining economic methods for concentrating these two minerals into marketable products.

Experimental Tests. Tests Nos. 1, 2, and 4 are flotation tests made for the recovery of the lead. The procedure followed was to grind the ore in a small ball mill and float in a laboratory flotation machine. Two table tests were also made for the recovery of the barite in the flotation tailings of tests Nos. 2 and 4. An elaborate gravity test was also made on the crude ore to determine whether a lead and barite concentrate could be obtained on tables. Ore at -14 mesh was classified in an hydraulic classifier into three products, each of which was tabled. The separations obtained were very poor, practically no separation of the barite resulted. The lead concentrates were of poor grade and the recovery was low. The assay results on this test are not given.

Results of Tests

Test No.	Product	Weight, per cent	Assays		Per cent of values	
			Pb per cent	Ag oz./ton	Pb	Ag
1	Lead concentrate.....	18.0	54.24	6.60	99.4	90.6
	Tailing.....	82.0	0.07	0.15	0.6	9.4
2	Lead concentrate.....	21.5	42.75	94.7	
	Tailing.....	78.5	
4	Lead concentrate.....	7.5	75.85	4.10	58.8	
	Lead middling.....	9.9	31.78	4.52	37.7	
	Tailing.....	82.6	0.95	8.5	

Table Tests on Flotation Tailings:—

2a	Barite concentrate.....	19.7	93.4	% BaSO ₄	Recovery: 14.8 tons per 100 tons ore.
	Middling.....	30.0	31.8	"	
	Tailing.....	50.3	45.4	"	
4a	Barite concentrate.....	24.6	91.5	"	Recovery: 19.5 tons per 100 tons ore.
	Tailing.....	75.4	33.28	"	

SUMMARY AND CONCLUSIONS

The lead in the ore can readily be concentrated by flotation. In the above tests the ore was crushed to 65 mesh.

The recovery of the barite is more difficult. A clean white product can, however, be tabled from the flotation tailing which runs high in barium sulphate. In practice on large tables, considerably more barite should be recovered. These tests show that between 15 and 20 tons of barite, 90 per cent BaSO₄, can be obtained from 100 tons of ore. In practice this could probably be increased to 25 tons.

Report No. 259

THE CONCENTRATION OF A GOLD-COPPER ORE FROM ENGLISH
BROOK DISTRICT, EAST CENTRAL MANITOBA

J. S. Godard

Shipment. A shipment of 70 pounds of gold-copper ore was received at the laboratories from A. McDonald, 905-908 McArthur Building, Winnipeg, November 19, 1926. The ore was from the English Brook district, east central Manitoba.

Description of Ore, and Assay. Gold, free and associated with small quantities of chalcopyrite, pyrite, and arsenopyrite. Gangue, siliceous. Assay of the head sample:

Gold.....	0.14 oz./ton
Silver.....	0.34 "
Copper.....	0.49 per cent.

Amalgamation Tests

(A)—Amalgamation at 65 mesh.

Screen Analysis of Amalgamation Tailing—

Mesh	Weight, per cent	Assay Au oz./ton	—	Per cent of values
+65.....	1.8	0.065	2.2
+100.....	12.7	0.06	14.0
+150.....	23.3	0.04	17.1
+200.....	11.5	0.05	10.7
—200.....	50.7	0.06	56.0
Assay of tailing.....		0.054	
Assay of head sample.....		0.14	
Per cent gold amalgamated.....			61.5	

(B)—Amalgamation at 20 per cent +200 mesh.

Screen Analysis of Amalgamation Tailing—

Mesh	Weight, per cent	Assay Au oz./ton	—	Per cent of values
+200.....	21.1	0.06	21.1
—200.....	78.9	0.06	78.9
Assay of tailing.....		0.06	
Assay of head sample.....		0.14	
Per cent gold amalgamated.....			57.2	

Concentration Tests (flotation and tabling):

Test	Product	Weight, per cent	Assays		Per cent of values	
			Au oz./ton	Cu per cent	Au	Cu
A	Flotation concentrate.....	3.9	2.80	10.54	68.9	88.1
	Table concentrate.....	4.4	0.79	0.14	22.0	1.3
	Table tailing.....	66.3	0.018	0.04	7.5	5.7
	Slimes.....	25.4	0.01	0.09	1.6	4.9
	Head from products.....	0.16	0.47
{(Flotation concentrate contained 2.58 per cent As)						
{(Table " " 0.54 " " As)						
Table tailing screened on 150 mesh—						
	+150.....	20.3	0.01	0.04	11.3	20.3
	—150.....	79.7	0.02	0.04	88.7	79.7

Test	Product	Weight, per cent	Assays		Per cent of values	
			Au oz./ton	Cu per cent	Au	Cu
B	Flotation concentrate.....	4.2	3.00	10.24	83.2	93.5
	Table concentrate.....	8.4	0.20	0.05	11.1	0.9
	Table tailing.....	55.2	0.01	0.026	3.6	2.8
	Slimes.....	32.2	0.01	0.04	2.1	2.8
	Head from products.....	0.152	0.46
Table tailing screened on 200 mesh—						
	+200.....	25.9	0.01	0.03	25.9	34.5
	—200.....	74.1	0.01	0.02	74.1	65.5

Amalgamation and Concentration Tests:
Results of Concentration:

Product	Weight, per cent	Assays		Per cent of values	
		Au oz./ton	Cu per cent	Au	Cu
Flotation concentrate.....	3.4	0.78	10.98	48.6	85.5
Table concentrate.....	2.1	0.44	0.29	16.8	1.4
Table tailing.....	62.7	0.02	0.03	22.9	4.3
Slimes.....	31.8	0.02	0.12	11.7	8.8
Head sample (amalgamation tailing) from products.....	0.055	0.44

Distribution of Values:

	Gold, per cent	Copper, per cent
In amalgam.....	60.9
In flotation concentrate.....	18.7	85.5
In table concentrate.....	6.8	1.4
In table tailing.....	8.9
In slimes.....	4.7

SUMMARY AND CONCLUSIONS

The ore of this shipment can hardly be considered of economic grade. The tests indicate two methods of treatment, namely: concentration by flotation and tabling; and amalgamation followed by flotation and tabling. Given an ore of economic value possessing the same characteristics, the second method would probably be adopted, as an appreciable recovery of gold values could be made at the mine by amalgamation and cyaniding the table concentrates, leaving only a flotation concentrate carrying the copper values to be shipped to a smelter. No straight cyanide tests were made, as cyanidation of a gold ore containing 0.5 per cent copper has not been a commercial success to date.

Report No. 260

GRINDING TEST ON ROSEDALE BENTONITE

R. K. Carnochan

Preliminary Crushing. A lot of 2,135 pounds of bentonite from the Rosedale Colliery Company, Rosedale, Alberta, was crushed in a No. 103 Austin gyratory crusher. The feed was in lumps, the largest of which were about 6 inches in their greatest dimension. The crusher was set at 1-inch opening and gave a $-1\frac{1}{2}$ -inch product. The crushing time was 2 minutes, 50 seconds. This is a capacity of 23 tons per hour. The discharge of the gyratory was crushed to $-\frac{1}{4}$ inch in a No. 00 Sturtevant hammer mill fitted with a $\frac{1}{2}$ -inch grate. The crushing time was 54 minutes. This is a capacity of 1.2 tons per hour.

Fine Grinding. The hammer mill discharge was fed to a No. 0000 Raymond pulverizer. The Raymond product was separated in a 30-inch Gayco air separator, the oversize from the separator being fed back to the Raymond as produced. The Gayco fines were finished product and tested 99.99 per cent -200 and 99.85 per cent -325 mesh. In colour they were a very light fawn. The capacity of the Raymond and Gayco on this material is about 50 pounds per hour, with about 20 pounds of Gayco oversize being returned to the Raymond per hour.

Observations. It was noticed that when all the fresh feed had been fed to the Raymond and nothing going into it but the Gayco oversize, the Gayco fines became grey in colour. In cleaning out the Raymond a large quantity of coarse sand was found in the beater chamber. To determine just what was taking place, samples of the different products were taken and analysed for non-colloidal material. This non-colloidal determination is made by diluting very highly and filtering, the material not passing the filter being non-colloidal. The following results were obtained:

	Per cent non-colloidal
Feed to Raymond.....	12.03
Gayco fines (sample whole run).....	9.24
Gayco fines (sample near end of run).....	13.7
Coarse sand from Raymond.....	92.6

A study of the above leads to the conclusion that the grit builds up to a certain extent in the circuit and then starts to grind up and go into the Gayco fines.

When no fresh feed is going to the Raymond the amount of grit in the Gayco fines is greatly increased and as this grit wears the iron off the beaters and beater chamber much more than the bentonite, the Gayco fines become darker.

It might be possible to separate the grit out of the Gayco oversize either by an air separator or screen before feeding it back to the Raymond, and this would give a Gayco fine product with much less non-colloidal matter.

Report No. 261

EXPERIMENTAL TESTS ON WHITEWATER LEAD-ZINC TAILINGS FROM RETALLACK, BRITISH COLUMBIA

C. S. Parsons

Shipment. A shipment of 65 pounds of lead-zinc tailings was received December 9, 1926, from M. S. Davys, Kaslo, B.C. This sample was from the Whitewater tailing dumps at Retallack, B.C.

Characteristics and Analysis. The sample received was a tailing from the early operations of a gravity concentration mill on the Whitewater mine ore. The Whitewater tailing dump is of considerable tonnage, lying in the bed of Kaslo creek, extending for about one-half mile below the mine. The chief values are zinc and silver, but galena up to one per cent is present, and also small values in gold. Part of the galena is oxidized, making its recovery by flotation difficult.

Analysis of the Sample:

Lead.....	1.46 per cent
Zinc.....	6.65 "
Silver.....	7.31 oz./ton

Purpose of Tests. The sample was submitted for the purpose of having flotation tests made to obtain information as to the relative merits of selective flotation as compared to bulk flotation followed by tabling to separate the lead and zinc in the bulk concentrate.

EXPERIMENTAL TESTS

The various tests will be described separately as each one was made to obtain certain information to compare the different methods of treatment.

Test No. 1

Selective Flotation. The crude tailings were crushed dry to 14 mesh. A 1000-gramme sample was taken and ground in a small ball mill for 15 minutes.

Reagents:

Soda carbonate.....	10.0 lb./ton	added to ball mill
Cyanide.....	0.25 "	15 minutes' contact
Zinc sulphate.....	2.0 "	" " "
Phospho cresylic acid.....	0.15 "	for lead flotation
Copper sulphate.....	1.5 "	" zinc "
Xanthate.....	0.3 "	" " "

Results:

Product	Weight, per cent	Assays				Per cent of values			
		Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
Lead concentrate..	3.5	17.30	24.11	127.5	40.5	12.2	59.6
Zinc concentrate...	15.4	1.56	36.01	0.01	12.5	16.1	79.4	25.7
Tailing.....	81.1	0.80	0.72	tr.	1.36	43.4	8.4	14.7

Tests Nos. 2 and 3

Two tests were made by floating a bulk concentrate containing both the lead and zinc and endeavouring to make a selective separation on the bulk concentrate. The two tests were unsuccessful and the results too poor to report.

Test No. 4

This is a selective flotation test using cyanide alone and adding it to the ball mill. The zinc concentrate was cleaned.

Reagents:

Soda carbonate.....	10.0 lb./ton	added to ball mill
Cyanide.....	0.25	" " "
Phospho cresylic acid.....	0.15	" " for lead flotation
Copper sulphate.....	1.5	" " } 3 minutes' contact before
Xanthate.....	0.3	" " } zinc flotation.

Results:

Product	Weight, per cent	Assays				Per cent of values			
		Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
Lead concentrate...	3.9	14.78	14.42	0.14	112.38	40.3	8.0	60.9
Zinc concentrate...	9.0	1.30	57.25	0.02	12.78	8.2	73.0	15.9
Zinc middling.....	4.6	3.0	20.52	tr.	17.50	10.4	13.3	11.2
Tailing.....	82.5	0.72	0.49	tr.	1.06	41.1	5.7	12.0

Test No. 5

The purpose of this test was to determine if a flotation concentrate containing the bulk of the lead but only a small proportion of the zinc, could be taken off the cells, after which the zinc would be floated in the usual manner by the addition of copper sulphate. In this operation the copper sulphate used to promote the flotation of zinc would not be added until after the first concentrate had been floated. The advantage of such an arrangement would be that a smaller quantity of material would have to be tabled for the lead and zinc separation—it being necessary to table the first float only. The second float would, after cleaning, be a final zinc concentrate. In this test the zinc concentrate was not cleaned. In practice this concentrate should be cleaned and the middling from the cleaner cell be sent back to the head of the second flotation.

Reagents Used:

Soda carbonate.....	10.0 lb./ton added to ball mill
Acid creosote (D. T. & C. Co.).....	0.35 " " " "
Floated first concentrate with these reagents and then added:	
Copper sulphate.....	1.5 lb./ton, 3 minutes' contact
Xanthate.....	0.4 " added to cell

Results:

Product	Weight, per cent	Assays				Per cent of values			
		Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
Lead concentrate..	9.7	7.04	23.70	0.02	43.18	45.9	32.4	64.0
Zinc concentrate...	11.6	2.06	37.76	tr.	14.64	16.1	62.1	23.5
Tailing.....	78.7	0.72	0.50	tr.	1.16	38.0	5.5	12.5

Remarks: It is difficult to see from the results of this test how it would pay to table the second concentrate which only contains 2.06 per cent lead. The extra loss of silver and zinc in slimes resulting from tabling would also be considerable.

Test No. 6

On this test a bulk flotation of the lead and zinc was made, the concentrate being cleaned. The object was to obtain results by which this method could be compared with the previous method of treatment.

Reagents Used:

Soda carbonate.....	12.0 lb./ton added to ball mill
Acid coal-tar creosote.....	0.35 " " " "
Copper sulphate.....	1.5 " 3 minutes' contact
Xanthate.....	0.4 " added to cells
Pine oil No. 5.....	0.02 " " "

Results:

Product	Weight, per cent	Assays				Per cent of values			
		Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
Flotation concen- trate.....	11.0	5.03	50.52	0.18	50.04	37.3	81.8	70.4
Flotation middling	5.3	4.43	17.26	0.12	24.16	15.7	13.4	16.3
Tailing.....	83.7	0.83	0.39	tr.	1.24	47.0	4.8	13.3

Test No. 7

The poor recovery of lead obtained in the previous tests indicated that it was oxidized. This test was run using sodium sulphide to sulphidize the oxidized lead. The ore at -14 mesh was ground for 15 minutes in a ball mill with the following reagents:

Soda carbonate.....	12.0 lb./ton
Acid coal-tar creosote.....	0.35 " "

Then 2 pounds sodium sulphide per ton, calculated as Na_2S , were added, the pulp density being kept as thick as possible. It required 8 minutes to consume the sodium sulphide. Since free sodium sulphide is detrimental to flotation when present in the pulp, the pulp was tested with lead acetate until no free sulphide was found. A selective flotation was then made, advantage being taken of the selective action of sodium sulphide. A lead product was floated which was not cleaned; cleaning, however, would be necessary in practice. A zinc concentrate was then floated by the addition of:

Copper sulphate.....	2.0 lb./ton.	3 minutes' time contact
Xanthate.....	0.4 "	added to cells
Pine oil, approximately.....	0.02 "	to froth.

Results:

Product	Weight, per cent	Assays				Per cent of values			
		Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
Lead concentrate..	5.0	13.38	16.65	0.12	64.94	47.7	12.4	35.8	48.3
Zinc " ..	10.4	2.57	53.14	0.10	23.08	19.0	81.7	35.6
" middling.....	3.5	2.21	4.53	0.01	9.07	5.6	2.4	4.8
Tailing.....	81.1	0.48	0.29	tr.	0.94	27.7	3.5	11.3

Test No. 8

In this test the ore was ground much finer before floating, the object being to determine whether finer grinding increased the recovery of the lead. The same reagents were used, and the same procedure followed as in test No. 7.

Results:

Product	Weight, per cent	Assays				Per cent of values			
		Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
Lead concentrate..	11.9	8.35	16.68	0.10	44.34	68.8	29.4	72.0
Zinc " ..	7.7	1.51	54.6	0.04	15.64	8.1	63.2	16.7
" middling.....	3.3	1.91	7.11	0.04	8.20	4.5	3.5	3.7
Tailing.....	77.1	0.35	0.34	tr.	0.72	18.6	3.9	7.6

Test No. 9

This test is a duplicate of Test No. 7.

Results:

Product	Weight, per cent	Assays				Per cent of values			
		Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
Lead concentrate..	6.5	11.57	10.25	0.03	42.36	49.6	9.6	36.6
Zinc " ..	10.7	2.62	54.18	0.04	31.26	18.7	84.0	44.9
" middling.....	2.8	3.22	6.90	0.03	18.79	6.1	2.8	7.2
Tailing.....	80.0	0.48	0.31	0.002	1.00	25.6	3.6	11.3

Test No. 10

This test is a duplicate of test No. 8.

Results:

Product	Weight, per cent	Assays				Per cent of values			
		Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
Lead concentrate..	6.2	11.87	10.25	0.08	42.48	49.7	9.3	37.8
Zinc " ..	10.7	2.67	55.33	0.04	30.52	19.2	86.1	46.8
" middling.....	3.1	1.96	2.69	0.03	8.17	4.1	1.2	3.6
Tailing.....	80.0	0.50	0.39	0.002	1.02	27.0	3.4	11.8

Test No. 11

The ore for this test was ground for 20 minutes in the ball mill. The object of the test was to determine the effect of using sodium sulphide when making a bulk concentrate which in practice would be tabled to separate the lead and zinc.

Reagents Used:

Soda carbonate.....	12.0 lb./ton added to ball mill
Acid coal-tar creosote.....	0.35 " "
Na ₂ S (sulphidizing period).....	2.5 " (approx. 8 lb. commercial salt)
	8 minutes' time contact
Copper sulphate.....	2.6 " 2 minutes' time contact
Xanthate.....	0.4 " added to cells.
Pine oil.....	0.02 " "

A bulk concentrate was made and cleaned with the following results:

Product	Weight, per cent	Assays				Per cent of values			
		Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
Flotation concentrate.....	12.0	0.30	51.77	0.26	40.54	55.8	88.5	72.2
" middling.....	5.5	4.43	7.11	0.22	15.32	18.2	5.6	12.7
" tailing.....	82.5	0.43	0.50	0.025	1.23	26.0	5.9	15.1

NOTE.—This test probably salted from high-grade gold ore.

SUMMARY

Tests Nos. 1, 4, and 5 are selective flotation tests. Test No. 5 represents an attempt to gather the lead into a small weight of concentrate by taking advantage of the natural property of lead to float more readily than zinc. This product, even under ideal conditions would run high in zinc and would have to be tabled for the lead. The advantage to be gained by such practice is that the main bulk of the zinc is floated afterwards and does not have to be tabled. Test No. 6 is a straight bulk flotation test, the object being to determine the grade of concentrate and recovery which should be readily obtained in practice. Tests Nos. 7 to 10 are selective flotation tests in which the oxidized lead was sulphidized by the

use of sodium sulphide. In test No. 10 the ores were ground finer than in the other three tests. Test No. 11 is a bulk flotation test in which the oxidized lead was sulphidized and then floated in bulk with the zinc.

CONCLUSIONS

It can readily be seen that selective flotation is not feasible on these tailings. The principal reason that they do not respond, is the oxidized condition of the lead. Test No. 6 represents ordinary bulk flotation practice and the results can be used as a basis of comparison when considering the results of other tests. This test shows that a 50 per cent zinc concentrate can easily be produced by one cleaning. The recovery of the zinc is high (81.8 per cent in the concentrate and 13.4 per cent in the middling), which in practice is continuously returned to the feed to the cells. This means that a 90 per cent recovery of the zinc can be obtained in practice. The silver recovery is high showing 70.5 per cent in the bulk concentrate and 16.4 per cent in the middling. It is not safe to assume over a 50 per cent recovery of the silver in the middling, so that 78 per cent total recovery is about the maximum. The lead recovery is low, 37.3 per cent in the concentrate and 15.7 per cent in the middling. The total recovery can be assumed at 45 per cent in a bulk concentrate assaying about 5 per cent lead, 50 per cent zinc, with 50 ounces of silver. This bulk concentrate would have to be tabled to separate the lead, and the table losses cannot be determined by small-scale tests. As this is the practice at present used on this material the operators can form a close estimate of what they will be.

By comparing the results of tests in which sodium sulphide was used for a sulphidizer, it will be observed that in every test the recoveries of the three metals are higher. Tests Nos. 7 and 9 are duplicate tests. A selective separation was attempted in these tests by the use of sodium sulphide sulphidizing for the oxidized lead. These tests show rather promising possibilities. There is a slight variation in the results of the two tests. More of the silver reports in the lead product in test No. 7 than in test No. 9. Test No. 9 gives the better figure to use in any calculations made, to determine the commercial possibilities of this separation. Tests Nos. 8 and 10 are duplicate tests and are similar to tests Nos. 7 and 9, with the exception that the feed was ground much finer. Test No. 10 represents the better practice of the two on account of the flotation of the lead product in test No. 8 being carried too far, which forced considerable zinc into it. The results of test No. 10 are practically the same as those of test No. 9, which shows that finer grinding is not necessary.

The operator can use the results obtained in tests Nos. 7, 9, and 10 to calculate whether the production and marketing of a lead product by this method will increase the value of his ore sufficiently to compensate for the cost of the additional equipment required. It should be kept in mind that the lead in this concentrate is really recovered, while the lead recovered in the bulk lead and zinc concentrate of tests Nos. 6 and 11 will be subjected to further loss when tableing to produce a lead concentrate. The additional equipment required would consist of a surge tank to give a time contact of 10 minutes for the addition of sodium sulphide; extra cell

capacity for flotation of the lead rougher concentrate, and a cleaner cell for cleaning the lead concentrate. Ample capacity should be allowed when figuring on size of lead cells as the material floats slowly.

Test No. 11 represents the use of sodium sulphide sulphidization for the production of a bulk lead zinc concentrate. While the writer has no cost figures available to estimate the returns from such a product sent direct to the smelter, this practice looks the most promising. The additional equipment required would be a surge tank for a 10-minute contact of sodium sulphide, and additional cell capacity for cleaning the concentrate. If Callow cells are used for the cleaners, they should be short, not over 8 feet in length, and preferably of the flat-bottom type (this applies to cleaner cells only). For the production of the low tailing obtained in these tests the present cell capacity for the production of the rougher concentrate should at least be doubled.

It will be observed that all the tests were made on the crude sample. It is understood that the practice used at present is to jig the crude material and collect a concentrate through the hutches of the jig, which is then reground and floated. It would appear that there must be considerable loss of lead by this practice because a large part of the lead in the crude material is present as slime.

In conclusion, the writer wishes to state that it is impossible to make any definite recommendations without a careful study of all the costs of milling and the freight and treatment charges on the products.

Report No. 262

EXPERIMENTAL TESTS ON A SPECIAL SAMPLE OF CYANIDE TAILINGS FROM DOME MINES LIMITED, SOUTH PORCUPINE, ONT.

J. S. Godard

Shipments. A shipment of 350 pounds was received December 10, 1926.

Characteristics of Sample. The sample was of a special high-grade cyanide tailing from the Dome Mine mill. It consisted of about 10 per cent sulphides as pyrite, and assayed 0.027 ounce gold per ton.

Purpose of Tests. The purpose of these tests is to ascertain if the pyrite, which contained the gold, could be concentrated by flotation.

EXPERIMENTAL TESTS

Two small-scale flotation tests were made in a laboratory Ruth machine.

Results:

Test No.	Product	Weight, per cent	Assay, Au oz./ton	Per cent of values
1	Concentrate.....	13.1	0.18	57.6
	Tailing.....	86.9	0.02	42.4
2	Concentrate.....	11.1	0.20	55.5
	Tailing.....	88.9	0.02	44.5

Reagents Used:

Test No. 1—

Soda ash.....	2.0 lb./ton, ground 15 minutes in ball mill.
Xanthate.....	0.15 " " "
Xylidine.....	2 drops " "
Copper sulphate.....	0.40 lb./ton added to cell.
Pine oil.....	0.10 " "

Test No. 2—

Soda hydroxide.....	1.0 lb./ton	} All ground 15 minutes in mill.
Soda ash.....	1.0 "	
Xanthate.....	0.20 "	
Copper sulphate.....	0.50 "	
Pine oil No. 5.....	0.05 "	

Test No. 3 was made on 180 pounds of the tailings. This was fed to a small grinding unit and a rougher concentrate made in a Callow machine. The rougher concentrate was cleaned in batch lots in a small Ruth machine.

Results:

Test No.	Product	Weight, per cent	Assay, Au oz./ton	Per cent of values
3	Concentrate.....	9.4	0.14	49.3
	Middling.....	13.4	0.03	15.1
	Clean-up.....	10.4	0.04	15.6
	Tailing.....	66.8	0.008	20.0

Reagents:

Soda ash.....	.50%	} 2.0 lb./ton added to mill.
Caustic soda.....	.50%	
Copper sulphate.....	0.5 "	" "
Xanthate.....	0.2 "	" " head of cell.
Pine oil.....	0.1 "	" "
Xanthate.....	0.1 "	" " for cleaning.
Pine oil.....	0.05 "	" " "

CONCLUSIONS

The flotation of these tailings was successful in that a concentrate assaying 0.14 ounce per ton and a concentration ratio of 10:1 was obtained. The final tailing was 0.008 ounce per ton. In continuous operation with the return of the middling to the regrinding circuit, a recovery of 65 per cent should be obtained.

The addition of soda ash is at present considered an essential reagent for the flotation of a cyanide tailing. It serves two purposes. First, it acts as a precipitant for the lime, and lime is detrimental to flotation producing a voluminous white froth that films the sulphide particles and inhibits their flotation; and, second, it aids in brightening the sulphides, thus permitting the other reagents to function.

III

REPORT OF INVESTIGATIONS: HYDROMETALLURGICAL
LABORATORY

R. J. Traill

General Introduction. The experimental work of this laboratory during the year comprised the completion of the tests on ilmenite treatment (see Report of Investigations, 1925, page 81), further studies on the electro-deposition of iron on rotating cathode, electro-deposition of iron on plates, the roasting and leaching of chalcopyrite with the recovery of copper from chalcopyrite leach liquors. The latter work is new in character and the experimental work to date has been of a preliminary nature for the purpose of determining the best mode of procedure to be followed in the treatment of such ore. The report in connexion with this work contains only such data as would seem of interest and importance at this stage, and which might be looked upon as a basis for carrying on future work.

In the treatment of ilmenite, results show that by obtaining a high metallization in the sponge process a very high-grade titanium dioxide concentrate can be made, low in iron content and, therefore, much more satisfactory for pigment or other purposes.

Further tests carried out on rotating mandrel cathodes are reported and show effects directly traceable to conditions of the diaphragm, such as new diaphragm and the diaphragm under continuous use. Various surface conditions on the cathode have also been tried and cathodes of different base metals have been used and results reported. A difficulty that has not been overcome completely is stripping of the deposits. This seems to be a purely mechanical operation, as it is practised quite successfully at Niagara Falls, N.Y., and at Grenoble, France. The equipment of the mechanical laboratory does not lend itself to a duplication of the methods employed at the above-mentioned centres.

A start has also been made on plate deposition tests, wherein higher current densities are used than are ordinarily employed in diaphragm cells.

The possibility of developing a hydrometallurgical process for the treatment of high copper-iron ores or concentrates using the ferric chloride leaching principle has been suggested and preliminary results in this work are reported. The results are quite encouraging, showing high extractions of iron and copper and high recoveries of copper and sulphur. The report contains only the practical results, the details of operation are omitted as being of less importance at present. In the recovery of copper by cementation it will be observed that electrolytic iron scrap was used instead of sponge iron as generally used and suggested. This was done for two reasons; first, because quite a quantity was on hand, and,

second, because the grade of copper obtained was so much higher and might be better suited for subsequent proposed tests relative to refining as electrolytic copper. Using sponge iron, as made in the laboratory, having approximately 20 per cent non-metallic material, the highest grade copper obtainable was approximately 64 per cent copper with about 15 per cent iron and the remainder insoluble carbon and silica. The procedure to be followed in commercial practice would, of course, be governed by the relative costs of sponge iron and electrolytic iron scrap at the plant.

Report No. 239

A PROCESS FOR THE TREATMENT OF ILMENITE FOR THE RECOVERY OF ELECTROLYTIC IRON AND TITANIUM OXIDE CONCENTRATE FOR PIGMENT AND OTHER PURPOSES

R. J. Traill and W. R. McClelland

The 1925 Summary Report (page 81) describes in detail the various steps in this process and gives the results of the experimental work carried out to show that the process is applicable from a metallurgical standpoint. Briefly, the process consists of treating an ore such as ilmenite with carbon at a temperature around 1,000° C. whereby the iron content is metallized and the titanium content unaltered from the oxide. The metallic iron is then leached with ferric chloride, the residue separated and subsequently treated for recovery of pure titanium dioxide, and the ferrous chloride resulting from the leach reaction electrolysed for recovery of pure iron. In the metallization tests obtained and reported on last year, results were not so encouraging as had been hoped for, the metallization being around 80 to 83 per cent. This, of course resulted in considerable iron oxide, 22 to 25 per cent being found in the leach residue, titanium dioxide content was lower than desired, namely, 53 to 56 per cent. Further tests were conducted in an effort to improve the grade of titanium-dioxide concentrate and by getting higher metallization in the sponge this was readily accomplished. The results clearly demonstrate what may be expected by way of higher grade titanium-dioxide concentration when a high-grade metallized sponge product is leached. The following tests are examples:

A sponge product made in the usual way showed the following analysis after separation of the carbon and non-metallics:

Metallic iron.....	44.39 per cent
Total iron.....	47.58 "
Metallization.....	93.3 "

This product was leached in the usual manner with ferric-ferrous chloride solution, with results as follows:

Iron extraction.....	97.4 per cent
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Analysis of Residue:

Titanium dioxide.....	67.10 per cent
Metallic iron.....	0.26 "
Total iron.....	10.09 "

A still higher grade sponge product was obtained, showing on analysis:

Metallic iron.....	44.39 per cent
Total iron.....	45.34 "
Metallization.....	97.9 "

On leaching this product with ferric ferrous chloride an extraction of 96 per cent iron was obtained and the residue had the following analysis:

Titanium dioxide.....	76.15 per cent
Total iron.....	3.65 "

These results are decidedly better than those¹ obtained in the initial experimental work and substantiate the opinion previously expressed that with proper design in furnace construction high metallization should be obtained with resultant high-grade titanium-dioxide concentrate.

DEPOSITION TESTS ON ROTARY CATHODE

A number of iron depositions were carried out using steel mandrel cathodes of various grades and surface coatings to determine the influence on grade of deposits and the adherence of deposit to base metal. Although the results did not prove conclusive they furnished some interesting data. It is not possible to go into the details of the various tests, but a brief summary of the more important results follows: The mandrel cathodes used were (1) cold rolled low-carbon steel; (2) cast steel, hard; (3) cast steel, case-hardened; (4) high-speed steel; (5) high-speed steel, case-hardened; (6) steel surfaced with electrolytic iron; and (7) cast iron.

The electrolyte used was a purified ferrous chloride electrolyte from ilmenite sponge and varied in concentration from 163 to 169 grammes ferrous iron per litre. The acidity of the electrolyte ranged from pH3.7 to pH3.9.

The cell used was the same as described and used in tests reported in 1925, namely, a diaphragm cell having two anolyte and one catholyte compartments. In all cases, with the exception of the case-hardened cathodes, the mandrels were given a fine machine-finished surface in the lathe. The cathode coatings varied in the following respects: (1) machined, surface plain; (2) machined, surface vaselined; (3) machined, surface oxidized; (4) machined, surface oxidized and vaselined. The cold rolled low-carbon steel, the cast steel, and the electrolytic iron-surfaced steel proved about the same, giving satisfactory deposits. The cast steel case-hardened, the high-speed steel and the high-speed steel case-hardened proved satisfactory in the nature of the metal deposited, but in all cases the deposits peeled off the cathode within the first twenty to sixty minutes of deposition. Better results were obtained when these surfaces were oxidized previous to deposition. The cast iron cathode proved quite unsatisfactory, the deposit being badly pitted, probably due to the presence of graphitic carbon on the surface of the cathode.

With respect to coatings, all worked satisfactorily under standard conditions, the oxidized coating without vaseline showing more consistent results than the others.

Diaphragm Effects

In these tests two points were noted with respect to diaphragm effects. These were the effect on deposition of a fresh or new diaphragm (asbestos cloth) and the behaviour and effect of a diaphragm used for several tests. Using a new diaphragm, it was consistently found that the deposit peeled off during the initial half hour of deposition. The reason for this was not definitely traced, but it is presumed to be due to some compound present in the asbestos cloth. Attempts were made to overcome this effect by treating the cloth with a hot ferrous or ferrous-ferric chloride solution and by treating in the cell under the usual operating conditions for a short period, taking the precaution to stop the process as soon as peeling occurred, to prevent the loosened deposit from adhering to the diaphragm. The latter method proved the more efficient and satisfactory, the main objection to the former procedure being time, the soaking process requiring several hours.

It was also observed that with a new diaphragm a higher voltage was required in relation to the average current density employed, the difference being about 0.4 volts at 90 to 100 amperes. By treatment of the diaphragm as outlined, the normal voltage was obtained.

Deposition tests were usually run for a period of seven hours. After each test the cell was emptied of liquor and then refilled prior to the next test with hot ferrous electrolyte liquor. As soon as the cathode was covered by electrolyte the current was applied and deposition started. The same diaphragm was used for as many as seven tests. As deposition proceeds there is always a slight hydrolysis of liquor occurring, and the precipitate of iron oxychloride so formed adheres to and impregnates the diaphragm. As this process continues, it is accompanied by a gradual increase in the voltage.

It is thought that this condition may be more pronounced in the present method of operating than it would be in a system of continuous operation, or a system wherein the cell did not require emptying for removal of mandrel. In the latter case the diaphragm has time to dry and assumes a hard and rigid form, and the precipitate may change physically and become more resistant to the passage of the current. These diaphragms have been cleaned by a hydrochloric acid wash and have proved satisfactory in subsequent depositions.

The above observations are based on thirty-two depositions, made up as follows:

Excellent.....	14
Very good.....	2
Good.....	5
Fair.....	2
Poor.....	2
Peeled deposits.....	7

PLATE DEPOSITION TESTS

Several tests on plate deposition of iron were carried out for the purpose of determining the maximum current density that could be used with a catholyte liquor movement or flow. Former experience has shown that with stationary electrodes, slow feed and overflow of electrolyte, and current density around and above 15 amperes per square foot cathode surface, marked chlorination occurred in the anolyte compartment. When a rotating cathode was used, current densities up to 125 amperes per square foot cathode surface were obtained with little or no chlorination. In the stationary electrode cell, deposits usually tree badly, whereas in the rotating cathode cell deposits are much smoother.

It was thought, therefore, that by causing a rapid flow of liquor in the catholyte conditions closely resembling the rotating cathode would result. The results which follow, while only preliminary in their nature, would indicate that with a flow of electrolyte in the catholyte compartment current densities up to around 35 amperes per square foot can be used with no noticeable chlorination in the anolyte. The deposits of iron have varied somewhat due to several reasons, the purity of electrolyte being, perhaps, the most important, lead being present in small amounts, probably around 0.02 gramme per litre, due to an accidental addition of an impure liquor.

The cell used was an ebony asbestos one, 12 inches long, 12 inches deep, and $7\frac{1}{4}$ inches wide. It was divided into two sections, an asbestos cloth diaphragm in the form of a bag serving as the catholyte in the centre of the cell. The catholyte compartment measured about $10\frac{1}{2}$ inches deep, 10 inches long, and 2 inches wide; the anolyte being approximately 10 inches deep, 12 inches long, and 5 inches wide, considering both sides. One cathode and two anodes were used. The effective cathode area was 1.16 to 1.30 square feet, and the distance between electrodes about $1\frac{3}{4}$ to 2 inches. Steel plates and monel metal plates were used for cathodes, and graphite plates for anodes. The liquor was heated in a storage feed tank to 100 to 105° C. and fed at a regulated rate through a glass tube into the catholyte compartment of the cell. The feed entered at the bottom of the catholyte bag and the overflow came out at the top on the opposite side. The overflow electrolyte was pumped back to the storage feed tank. The anolyte overflow was regulated to suit conditions, depending on the current density used and the rate of formation of ferric chloride. The average temperature of cell liquor in all tests was 82 to 86° C. Eight depositions were made and the following table gives details of the various conditions under which the tests were made and shows the efficiency of the deposition. Remarks on each individual test follow the table.

Test	Cathode	Acidity, pH	Electro- lyte concen- tration, gm./l	Rate of flow, l/hour	Voltage	Amps. per sq. ft.	Weight deposit, grms.	Time, hours	Cathode current efficiency
1	Steel.....	3.7	167.5	40	2.75	44.7	398	7	95.8
2	"	3.7	182	36	3.65	67.8	552	7	96.3
3	"	3.7	194.6	20-35	2.99	34.8	280	6	97.2
4	"	4.1	179	80	2.97	30.7	7
5	Monel.....	3.9	172	73.5	2.36	37.2	337	7	95.6
6	"	3.9	170.6	115	2.31	36.3	354	7½	97.4
7	"	3.8	179	108	2.55	36.2	7
8	"	3.8	185	108	2.78	35.9	7

REMARKS

Test 1. No chlorination was noticeable. The deposit was silvery grey in colour with some small threads on the surface and slight trees at one end and bottom of plate. Deposit removed as two plates, and somewhat malleable as compared to tube deposits at higher current densities. Distance between electrodes was 2 inches. Anode efficiency based on iron deposited was 95.2 per cent. A concentration of the catholyte occurs due to steam evaporation, the overflow liquor containing 185 grammes ferrous per litre. The acidity also increases slightly to pH 3.6.

Test 2. During the first two hours, no chlorination occurred; about the third hour a trace of chlorine was noticed and after four hours the chlorine from anolyte was quite noticeable. The deposit was badly treed but easily removed as plates. Distance between electrodes was $1\frac{3}{4}$ inches. The anode efficiency based on iron deposited was 95.8 per cent. From the nature of the deposit it would appear that the current density here employed was too high.

Test 3. A very slight chlorination was noticed towards the end of this test. The deposit was treed and somewhat badly pitted. A new diaphragm was used and the catholyte flow was unsatisfactory due to occasional blocking, which conditions probably account for poor deposit. Distance between electrodes was $1\frac{3}{4}$ inches. The anode current efficiency showed 98.3 per cent, being higher than the cathode current efficiency. No explanation can be given for this condition.

Test 4. No chlorination occurred in this test. The deposit was smooth with slight treeing at base. The deposit could not be completely removed and the base metal was not weighed through oversight, so that cathode current efficiency was not obtained. The distance between electrodes was $1\frac{3}{4}$ inches. The general character of the deposit seemingly justified the increased flow to catholyte.

Test 5. Chlorination was absent. The deposit was smooth, free from treeing, dull grey in colour, but showed a few pit holes at top and bottom of plate. The deposit was easily stripped to within an inch or so of the bottom, where it was removed with difficulty. Some anode liquor

was lost in this test and anode current efficiency could not be determined. Distance between electrodes was $1\frac{1}{2}$ inches.

Test 6. Chlorination was absent. The deposit was quite smooth, dull grey in colour, but showed a large number of small circular holes, with iron deposition in centre adhering to base metal when deposit was removed. Very slight fine treeing occurred along bottom edge of plate. Deposit was easily stripped. Distance between electrodes was $1\frac{1}{2}$ inches. The anode current efficiency in this test from the data collected would appear to be well over 100 per cent, but it is evident that an error in readings or analyses must have occurred.

Test 7. No chlorination occurred. The deposit was smooth, dull grey, and slightly pitted. This deposit was only partially removed from plate and no weight was obtained. Distance between electrodes, $1\frac{1}{2}$ inches.

Test 8. No chlorination occurred. The deposit was smooth and even, though badly pitted. Pits as large as $\frac{1}{16}$ inch were observed. This deposit was not removed except in small portions.

No satisfactory explanation for the results obtained in the last two depositions can be offered at present.

SUMMARY

The results so far obtained would indicate the practicability of operating a plate cell at about 35 amperes per square foot, without causing undue chlorine evolution at the anode, by arranging a rapid flow of electrolyte through the catholyte compartment of the cell. It is hardly to be expected that smooth matt deposits can be obtained by plate deposition, to compare with those obtained in rotating cathode practice, but it is conceivable that, with a carefully designed cell permitting flow or circulation of liquor in both anolyte and catholyte, deposits free from trees, pits and other outstanding defects can be obtained.

The tests here reported constitute a preliminary study of this phase of deposition, and it is intended to carry out further tests in an endeavour to find a successful procedure for plate deposition of iron at higher current densities than are generally used.

PROCESS FOR THE HYDROMETALLURGICAL TREATMENT OF HIGH IRON-COPPER SULPHIDE CONCENTRATES: RECOVERY OF IRON BY ELECTROLYSIS, COPPER AS CEMENT COPPER, AND SULPHUR BY DISTILLATION

R. J. Traill and W. R. McClelland

Present-day practice in the metallurgical treatment of high-copper sulphides consists of the well-known process of smelting and converting. The iron content of the ore is lost in slag and the sulphur lost in waste gases to the air. Sensing the possible recovery of these two useful elements, it was felt that an attempt to develop a process with this purpose in view would be worth while.

The experience gained and the measure of success obtained in the experiments on the hydrometallurgical treatment of pyrrhotite (see Reports of Investigations 1924 and 1925) in this laboratory suggested a basis in the method of treatment.

Among the copper-bearing ores of Canada, chalcopyrite is probably the most common variety, and it was decided to start experiments with this ore. This mineral is generally recognized by the formula Cu_2S , Fe_2S_3 , and is frequently associated with pyrite FeS_2 . In reaction with ferric chloride it was expected that this mineral would behave as an intermediate between pyrrhotite and pyrite. This expectation seemed to be borne out in initial tests where it was found that the copper leached fairly readily with ferric chloride solution, while only half of the iron was leached out. To obtain a higher iron extraction therefore, it would be necessary to convert the iron to a more soluble form, such as monosulphide or artificial pyrrhotite, by roasting in a non-oxidizing atmosphere.

PRELIMINARY TESTS

The ore used in these tests was copper concentrate from a chalcopyrite ore from Eustis, Quebec, the concentrate averaging 60 to 65 per cent chalcopyrite and 22 to 29 per cent pyrite, or 21 to 23 per cent copper and 30 to 34 per cent iron.

Roasting

Roasting was first attempted in an air-tight rotary type furnace, having a quartz tube about 4 inches in diameter by 4 feet long, heated by resistance wire elements. Difficulty was experienced in maintaining a steady feed, the ore being too fine for slow feed, and becoming sticky with a tendency to build up in the hot zone of the tube. With a more rapid feed, the ore passed through the hot zone too quickly to be properly roasted, and it became necessary to pass the ore through the furnace several times to bring about any definite change in its composition. This procedure seemed to cause marked oxidation of the ore particles resulting in increased hydrolysis during the subsequent leaching of the roasted product. Little or no sulphur was recovered from these runs, and the composition of the product varied too much from test to test to be of any material value in subsequent experiments. After several trials this method was abandoned in favour of retort roasting. The retort method gave more consistent results, the product varying about one per cent in total iron with an almost consistent loss in weight of about 17 per cent. The latter roasts were made on 200- and 300-gramme samples. In the sulphur condenser nothing more than a film of flowers of sulphur was obtained, most of the sulphur apparently being oxidized. The sulphur that would be set free amounts to only about 9 per cent of the charge, and it is plain to see that this would be a rather small amount to permit of recovery. An average analysis of the raw concentrate and of the roast product is as follows:

	Concentrate, per cent	Roast product, per cent
Iron.....	33.66	38.88
Copper.....	23.10	26.86
Sulphur.....	35.47	31.77

The product here obtained would apparently conform approximately to the formula $\text{Cu}_2\text{S} + \text{FeS}$.

Leaching

Leaching tests were carried out on the raw concentrate and on the roast product in 1- and 4-litre volumes of ferric ferrous chloride in the usual manner. The leach liquor used averaged 64 to 70 grammes per litre ferric and about 70 grammes per litre ferrous. The liquors were kept agitated and the temperature maintained as close as possible to 95°C .

For the purpose of brevity and comparison, the results are shown in tabulated form below. The column marked ratio $\text{Cu} + \text{Fe}$ to ferric shows the amount, in grammes, of ferric chloride leach in terms of iron used to treat one gramme total metal content of concentrate.

Test No.	Material	Leach volume, litres	Time, hours	Ratio $\text{Cu} + \text{Fe}$ ferric	Iron extraction, per cent	Copper extraction, per cent	Residue analysis		Residue weight, per cent
							Fe per cent	Cu per cent	
3	Raw.....	1	5	1.455	57.80	88.03	26.27	5.23	52.2
4	".....	1	5	1.455	58.40	90.79	26.88	4.19	50.5
12	".....	4	6	1.45	58.25	94.38	26.16	3.89	51.2
13	".....	4	6	1.45	56.42	89.90	26.57	5.75	52.0
14	".....	4	6	1.45	54.92	89.23	27.21	5.86	53.3
1	Rotary.....	1	5	1.54	81.08	80.50	15.06	6.0	43.6
2	Furnace.....	1	5	1.54	83.30	92.30	14.38	4.55	42.5
6	Roasted.....	4	5	1.52	84.60	89.60	13.14	6.04	43.0
8	Retort.....	1	3	1.70	84.80	95.17	14.38	3.34	40.0
9	Roasted.....	1	4	1.60	88.67	91.92	11.28	5.52	40.0
10	".....	1	4	1.58	82.86	96.74	16.94	2.20	41.0
11	".....	1	5½	1.49	85.50	90.60	14.01	6.30	41.0

REMARKS

The outstanding feature in these tests is the increase in iron extraction in the roast products over the raw concentrate. The copper extraction is much the same in roasted and raw concentrate. It should be said that in the case of the roast products, each leach test is made on a different roast and undoubtedly there would be a slight variance in the composition of each product, which may account for the variance in extraction of copper and iron. In some tests hydrolysis seemed to have occurred to a greater extent than in others; there was also a variance in the volume at the end of the leach, due to the difficulty experienced in controlling gas-heating to maintain an even temperature. In the residues it was found that in the raw concentrate residue the free sulphur amounted to about 50 per cent, and in the roast residue the free sulphur amounted to about 60 per cent. On the distillation of the sulphur under non-oxidizing conditions a residue would be obtained of sufficient iron and copper content to warrant further leach treatment. Such a scheme was carried out and the results appear below.

(A) LEACHING WITH RESIDUE RE-TREATMENT

The residues from tests Nos. 12, 13, and 14 were combined and roasted in a non-oxidizing atmosphere by retort method at 700 to 800° C. for two hours. The loss in weight in roasting amounted to 50 per cent. The roast product had the following analysis: iron 50.68, copper 10.17, and sulphur 34.20 per cent. This approximates closely $\text{Cu}_2\text{S} + \text{FeS}$. The roasted product was leached with a ferrous-ferrie liquor for 6 hours under normal conditions. The weight of residue was 48 per cent of the treated roast product, or 52 per cent of the product was leached. The analysis of the residue was: iron 17.66, copper 2.91, and sulphur 68.95 per cent. This would show an extraction of iron 83.3, and copper 86.3 per cent.

Sulphur Recovery. The secondary leached residue thus obtained was roasted in an alundum tube fitted at the open end with a water-cooled galvanized pipe for sulphur recovery test. The temperature employed was 700 to 750°C. The roasted product showed a loss in weight of 67 grammes, and 48 grammes of sulphur were recovered in the condenser. This would show a sulphur recovery based on loss in weight of 71.6 per cent. The analysis of this final roast product was: iron 43.13, copper 3.54, and sulphur 25.37 per cent, and gold 0.12 ounce per ton. The final roast was slightly oxidized during cooling.

Copper Recovery. The copper was recovered by cementation using electrolytic iron scrap as the precipitant. This gives a high-grade cement copper. The liquors were heated to about 75°C. and the scrap added. The following table shows the details.

Test	Quantity electrolytic iron used	Time, hours	Weight of copper precipitate	Analysis copper, per cent	Iron unused	Recovery from ore, per cent
	grms.		grms.			
12.....	65.0	3	64.3	99.0	1.8	89.8
13.....	72.5	3	63.0	98.98	5.0	84.4
14.....	80.0	$\frac{3}{4}$	66.0	95.76	9.0	86.0
Secondary leach.....	40.0	$\frac{3}{4}$	23.0	98.64	21.0	93.0

Test No. 14 and the secondary leach test were run with agitation, while tests Nos. 12 and 13 were run without agitation. The copper remaining in the liquors was 0.776, 0.428, 0.36, and 0.40 gramme per litre respectively.

Summary. The following shows an interesting summing up of the results obtained in the tests here described:

Total concentrate in 3 primary leaches.....	981 grms.
Final roast residue.....	45 "
Iron extraction.....	93.9 per cent
Copper extraction.....	99.2 "
Copper recovery.....	98.0 "
Total metal extraction.....	96.0 "

Added to these a sulphur recovery of probably over 70 per cent might be expected and a factor of interest is the concentration of gold taking place in the final residue. The final residue would probably be added to a

new lot of raw concentrate and the process continued until the gold content had worked up to quite high. The results here obtained suggest that a possible method of treatment would be to combine the primary roast residue with fresh concentrate and leach in the usual manner.

(B) LEACHING WITH EXCESS FERRIC, AND LEACHING ROAST RESIDUE

In this series of tests the raw concentrate was leached with a ferric-ferrous chloride solution containing sufficient ferric to dissolve all the iron and copper present in the charge. This naturally gave an excess of ferric iron in the primary leaching. The primary leach residue after drying was roasted in a non-oxidizing atmosphere, and the roast residue thus obtained added to the filtrate liquor of the primary leach to completely reduce same. The secondary leach residue thus obtained was roasted for recovery of sulphur. The following tabulation shows in brief the results obtained by this method of procedure on a 4-litre leach:—

Charge to primary leach.....	319 grms.
Weight of primary leach residue.....	162 "
Per cent leached.....	49.2
Weight of primary roast product.....	83 grms.
Per cent loss of weight in roasting.....	48.1
Sulphur recovered.....	69 grms.
Per cent recovery sulphur based on weight loss.....	87.3
Leaching primary roast product	
Weight of secondary leach residue.....	41 grms.
Per cent leached.....	50.6
Weight of secondary roast product.....	17 grms.
Per cent loss in weight.....	54.1
Sulphur recovered.....	13 grms.
Per cent recovery sulphur based on weight loss.....	54.1
Analysis secondary roast residue: Iron.....	42.56 per cent
Copper.....	6.02 "
Sulphur.....	19.61 "

Cementation of Copper:

Electrolytic iron scrap used.....	90 grms.
" " recovered.....	43 "
" " consumed.....	47 "
Weight of cement copper precipitate.....	71.8 "
Copper content.....	96.82 per cent
Cemented liquor analysis: Iron.....	152.88 grm./l.
Copper.....	0.044 "

Recoveries and Extractions:

Iron extraction.....	93.1 per cent
Copper extraction.....	98.55 "
Copper recovery from charge.....	98.47 "
Sulphur recovery.....	71.4 "

(C) 50-LITRE TEST ON METHOD B

This test was a duplication of test B, but on a larger scale, 50 litres of ferric-ferrous chloride liquor being used for leaching. The wash water from the primary leach amounting to approximately 10 litres was retained and treated for copper recovery by cementation. The following data show the results obtained at the various stages.

Raw concentrate charge.....	3,950 grms.
(6 hours leaching at 92-96° C.)	
Weight primary leach residue.....	2,186 "
Per cent leached.....	44.66
Analysis primary leach residue: Iron.....	26.02 per cent
Copper.....	9.0 "
Sulphur.....	56.67 "

Primary roast carried out in alundum tube, 3½ inches diameter by 24 inches long, with metal tube, 30 inches long, fitted to open end and water-cooled.

Weight of primary roast.....	1,302 grms.
Per cent loss in weight.....	40.2
Weight of sulphur recovered.....	822 grms.
Per cent recovery based on weight loss.....	93.4
Analysis roast residue: Iron.....	42.93 per cent
Copper.....	14.80 "
Sulphur.....	33.41 "

Secondary leach was made on this residue using the primary leach filtrate liquor and leaching for 4½ hours at 96° C.

Weight of secondary leach residue.....	662 grms.
Per cent leached.....	49.1

This secondary leach residue was roasted in alundum tube for sulphur recovery.

Weight of roast residue.....	267 grms.
Per cent loss in weight.....	59.6
Sulphur recovered.....	338 grms.
Per cent recovery based on weight loss.....	93.2
Analysis final roast residue: Iron.....	30.57 per cent
Copper.....	9.72 "
Sulphur.....	24.34 "

Copper Recovery. The copper was cemented out with electrolytic iron scrap, the wash water being treated separately from the main electrolytic liquor.

Total electrolytic iron used.....	987 grms.
Electrolytic iron recovered.....	355 "
Electrolytic iron consumed.....	632 "
Weight cement copper.....	907 "
Per cent copper.....	91.91
Copper recovered.....	834 grms.

An interesting feature here is that 834 grammes of copper are precipitated by 632 grammes of iron, showing the copper-precipitating power of electrolytic iron under these conditions to be about 1:1.32.

Extractions and Recoveries Summary:

Loss in weight by leaching and roasting.....	93.2 per cent
Iron extraction.....	93.72 "
Copper extraction.....	97.03 "
Copper recovery.....	95.37 "
Sulphur recovery.....	95.60 "
Electrolyte obtained for iron deposition: Iron.....	156.0 grm./l.
Copper.....	0.015 "
Lead.....	0.02 "
Sulphur.....	6.02 "

These results are very satisfactory as regards extractions and recoveries. However, the method of procedure is, perhaps, a little complicated and something simpler, such as a single roast and single leach treatment, would be much more satisfactory. Such a scheme is suggested in the next series.

(D) SINGLE ROAST AND SINGLE LEACH METHOD

The results of the tests already described show that by using a roast product for leaching, higher extractions of iron and copper can be obtained than are obtained when using raw concentrate. Tests on roasting the raw concentrate and leaching the product have shown that the product exhibits a slight tendency toward oxidation, which causes hydrolysis in the leaching liquor. The roasting of raw concentrate seems difficult to control probably because of the small excess of sulphur present in the charge, only about 9 per cent over a mono-sulphide, while a leach residue product with its high sulphur content roasts well and is more consistent in composition. It would appear that the reason for this difference was due to a deficiency of sulphur in the roasting, i.e. that the roasting is more efficient with an excess of sulphur gas present. It was decided therefore to add to the raw concentrate sufficient sulphur to ensure an excess in the roasting atmosphere. To start with, raw concentrate was roasted with free sulphur, the product leached and the leach residue added to a fresh charge of raw concentrate. The product leached and the leach residue again used with a fresh charge. This procedure can be carried on indefinitely almost, or at least until a stage is reached where the gangue material has built up to a great extent. Where gold is present in any amount, it will gradually build up in the residue and when of sufficient value can be readily recovered by any standard method. A single test embracing this procedure was carried out and the results which follow would seem to justify the assumptions made.

Experimental. The preliminary test was not carried out in as direct a sequence as subsequent tests will be run, but, nevertheless, it shows results that are encouraging. The capacity of the small-scale leaching equipment necessitated making more batch leaches than would otherwise be required, so that there were ten leaches for five roasts. The first three leaches were really trial leaches to determine, principally, the time and charge necessary. The leach tests are numbered Nos. 23 to 32 inclusive and the roasts Nos. 20 to 25.

Roast test No. 20 consisted of 200 grammes of concentrate with 40 grammes of sulphur, and roast No. 21 was 400 grammes concentrate with 80 grammes sulphur. These roast products were leached in leach tests Nos. 23, 24, 25, and 26. In tests Nos. 23 and 24 the charge was calculated on the copper being present as Cu_2S and forming cuprous chloride in the leach. This, however, proved a wrong assumption, as the cuprous chloride apparently reacts with ferric chloride to form cupric chloride and therefore requires ferric chloride equivalent to a CuS basis (see summary).

Each of the following roasts, tests Nos. 22, 23, and 24, consisted of 400 grammes of concentrate plus the leach residue from the preceding leach, and the roast product was leached in two lots, roast No. 22 serving leaches

Nos. 27 and 28, and so on. The final roast No. 25 consisted of leach residues alone. In the leaching tests it is apparent that longer time gives higher extraction in copper and iron.

In the roasting tests it will be observed that the roast product has a fairly constant composition. The sulphur recovery was low, but this was due to losses through a leak in the connexion between the condenser and the retort. The condenser apparently was smaller than should be employed and the sulphur coming off fairly rapidly was not cooled in the condenser sufficiently fast to avoid a pressure of sulphur gas in the apparatus. The following table shows details of the leaches and roasts.

Leaching Tests

Test No.	Leach, litres	Time, hrs.	Charge, grms.	Per cent dissolved	Analysis			Extraction	
					Fe per cent	Cu per cent	S per cent	Fe per cent	Cu per cent
23.....	1	4½	70	51.43	19.02	10.05	54.49	75.8	81.43
24.....	1	4½	70	51.43	19.56	10.03	62.60	75.2	81.40
25.....	1	6	55	58.20	13.08	6.19	76.02	80.1	90.20
26.....	4	6½	220	56.80	13.67	6.89	74.10	84.6	88.6
27.....	4	6	210	59.70	9.33	4.55	70.08	90.3	93.0
28.....	1	8½	50	61.60	6.76	2.56	82.30	93.8	96.2
29.....	4	7½	225	59.60	9.11	4.02	79.12	90.3	93.7
30.....	4	8½	223	60.50	8.27	3.82	78.04	91.4	94.2
31.....	4	8½	220	61.30	5.24	1.83	82.26	94.4	97.2
32.....	2½	8	126	60.30					

Roasting Tests

Test No.	Fresh conc. charge, grms.	Composition of charge	Loss in weight, per cent	Analysis			Sulphur recovery, grms.
				Fe per cent	Cu per cent	S per cent	
20.....	200	+40 grms. S.....	22.0	38.32	26.31	32.76	20
21.....	400	+ 80 " S.....	22.0	38.38	26.32	33.12	72
22.....	400	+120 " res....	31.5	38.86	26.5	33.12	59
23.....	400	+ 80 " res....	31.4	38.11	25.88	33.3	50
24.....	400	+151 " res....	36.1	36.18	25.35	32.35	84
25.....	214 " res....	83.6	30.37	12.47	26.45	145

Total sulphur recovery 60 per cent. Gold 0.13 oz./ton.

Extractions and Recoveries

Raw concentrate charge.....	1,800 grms.
Weight of last roast residue.....	35 "
Per cent leached.....	98
Iron extraction.....	98 per cent
Copper extraction.....	98.8 "
Sulphur recovered.....	60.0 "
Gold content last residue.....	0.13 oz./ton

SUMMARY

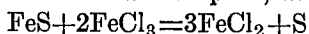
The results obtained in the tests here reported are more or less satisfactory in the application of the ferric chloride leaching method of treatment for these high copper-iron concentrates. The last test, series D,

looks the most encouraging of any. In this test the extractions of copper and iron should be very close to one hundred per cent, except for mechanical losses, as the residue from each leach is utilized wholly in subsequent roasts, at the same time acting in a beneficial manner in the roasting process.

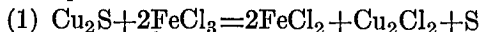
It has been shown that for satisfactory extractions of iron and copper recourse must be had to roasting at some stage in the process. This, of course, applies particularly to chalcopyrite-pyrite mixtures and the same may not be true with respect to bornite or certain other sulphides. No definite attempt has been made to determine the chemical reactions occurring, but the following would seem to be a reasonable assumption.

From the analyses of several roast products, it is fairly evident that the copper is present as cuprous sulphide, Cu_2S , and the iron as mono-sulphide, FeS to Fe_8S_9 . In the roasting, therefore, it is apparent that the copper combination is not altered, but the iron combination Fe_2S_3 loses part of its sulphur, and Cu_2S , $\text{Fe}_2\text{S}_3 = \text{Cu}_2\text{S}$. $2\text{FeS} + \text{S}$ probably expresses the reaction.

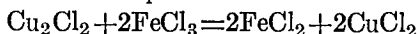
In the leaching process the mono-sulphide of iron combines with ferric chloride, forming ferrous chloride and sulphur, as



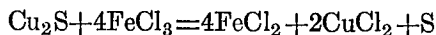
In the case of the copper sulphide, the reaction is probably of a double nature and may be expressed as follows:—



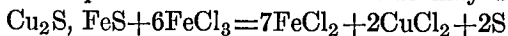
the cuprous chloride thus formed, and being readily oxidized, probably reacts with ferric chloride still present in solution as follows:



This assumption is confirmed to some extent by the results obtained in leaching. It has been found in calculating the amount of ferric chloride required to leach a roast product, that when the calculations were based on copper as Cu_2S (the iron always being taken as FeS) low extraction results were obtained in both iron and copper and no excess ferric was present in the leach at end of test, whereas when the copper was calculated as CuS the extractions were very much higher. Therefore the reaction might properly be written as follows:



and combining this reaction with the iron sulphide reaction, the chemical reaction of the roast product with ferric chloride may be expressed as:



To check this up by chemical analysis has proved somewhat difficult by reason of the complexity of the leach liquor, as in most cases qualitative test has shown the presence of cuprous and cupric chlorides, ferrous and traces of ferric chlorides. The reactions, therefore, suggested are based entirely on the experience in leaching and partly substantiated through the cementation process by the amount of iron consumed in the precipitation of the copper.

It is hoped to check and corroborate the results obtained in method D by a further series of tests during the coming year.

IV

REPORT ON THE WORK AND INVESTIGATIONS OF THE
CHEMICAL LABORATORY OF THE DIVISION

H. C. Mabec, Chief Chemist

The main work of the chemical laboratory during the year consisted, as in previous years, in the determination of the metal constituents of ores, minerals, and their concentration products from test operations conducted in the research laboratories of the Ore Dressing and Metallurgical Division. This work of the usual diversified character was performed by B. P. Coyne, R. A. Rogers, and L. Lutes. The first two were employed on chemical analyses and the latter on the work of fire assaying. During the year, 1,243 samples were received from the research laboratories on which over 4,000 determinations were reported. The ores and minerals from which these samples, as well as samples from their concentration products, were obtained are indicated in the following list:

Arsenical.....	21	Gypsum.....	15
Calcite.....	5	Galena.....	38
China clay.....	17	Lead-silver.....	5
Colloidal clay (bentonite)....	4	Lead-silver-zinc.....	233
Cyanite (aluminium silicate)..	3	Molybdenite.....	7
Copper.....	17	Nickel-copper.....	33
Copper-zinc.....	121	Sandstone.....	17
Gold-arsenic.....	125	Titaniferous.....	1
Gold-copper.....	152	Zinc.....	34
Gold.....	322	Zinc-gold.....	9
Graphite.....	64		

The analytical work represented in the above list does not include that involved in connexion with numerous control samples of ores and products from special investigational work in progress in the electrochemical and hydrometallurgical laboratories, the results of which will be found in detail under Section III.

In order to ascertain the cause of certain discrepancies to which our attention had been directed in connexion with the arsenic content reported in what were said to represent duplicate samples of arsenical ore, B. P. Coyne, in co-operation with the writer, conducted a series of investigations with a view to studying the details of the chemical method by which these erratic results were produced, and further, with the use of reliable standard methods, determine the exact arsenic content present by means of comparative results.

The sample of arsenical ore on which this investigation was conducted was submitted by the Nova Scotia Technical College at Halifax, together with their method of analysis. The result of the investigation showed very clearly that in following the method as indicated, and failing to observe certain necessary precautions in the elimination of all interfering elements, this method yielded a consistently high result, averaging about

one-half per cent on a sample containing 3.30 per cent arsenic. For the determination of this element in ores or metallurgical products, it was found that the regular arsenic sulphide method as described in common reference, or the distillation method, similar to that referred to in the "Official Methods of Analysis of the Association of Agricultural Chemists" are generally applicable and reliable.

Laboratory accommodation was again provided the technical staff of the National Research Council during the year for a continuation of their work on the beneficiation of magnesite ores.

V

SELECTIVE FLOTATION AS APPLIED TO CANADIAN ORES

C. S. Parsons

Canada possesses many deposits of complex ores, such as lead-zinc-iron sulphides, copper-zinc-iron sulphides, copper-iron sulphides, and other combinations of these minerals, many of which contain values in both gold and silver.

The Mines Branch of the Department of Mines, through its Ore Dressing and Metallurgical Laboratories at Ottawa, has made numerous investigations into the treatment of such Canadian ores by selective flotation. The purpose of this review is to bring more directly to the attention of the mining public the information which has been acquired and the progress which has been made in the treatment of ores by this method, and also to point out the assistance the Government is rendering the mining industry by maintaining at Ottawa fully equipped ore dressing and metallurgical laboratories for research on the treatment of ores and metallurgical products. A brief description of the flotation equipment in these laboratories is given in order that the reader may better realize the type of work which can be performed.

The small-scale laboratory equipment consists of a 1000-gramme Ruth flotation machine, a Callow pneumatic batch unit, a Janney machine and a continuous fine-grinding flotation unit which will treat 20 pounds of ore an hour, grinding 200 mesh. The large-scale tonnage equipment consists of two different types of grinding mills, viz., one 4½-foot Hardinge conical mill and one 6 by 3-foot Marcy rod mill. Both can be operated in closed circuit with a standard Dorr classifier having an interchangeable bowl attachment. Flotation equipment comprises a Greenawalt 8-cell flotation machine and a Callow pneumatic flat-bottom unit with a capacity of one ton per hour. Tables, classifiers, and screens can be cut in as desired, and continuous flow-sheets can be built up by using various combinations of these machines. The addition of a unit which will treat 200 pounds of ore an hour is contemplated.

RECENT FLOTATION PRACTICE

The tonnage of complex ores separated by flotation has rapidly increased. The use of chemical reagents to modify the surface of the ore particles, the discovery of soluble chemical flotation reagents, and the economic solution of the fine-grinding problem are the principal factors responsible for the increase. The remarkably successful operation of a number of all-flotation custom plants in treating complex ores is unquestionably due to their efficient research organizations.

A general effort is being made to raise the grade of flotation concentrates and to determine for each mill the point at which the greatest profit can be made, when the lowered freight and smelter charges on the concentrates are compared with the extra cost of concentration and greater loss in tailing. A careful study of this question has shown, in a number of cases, that it would pay to regrind either the flotation concentrate or middling, as the case may be, and subject it to a second flotation. It is quite obvious that a flotation concentrate, for the most part already ground to -200 mesh, is difficult to regrind unless it is carried as an extremely thick pulp to the grinding mill and a high circulating load maintained in the classifier-ball mill circuit, consequently concentrate or middling to be ground generally goes to a desliming classifier of the drag type; sands from this classifier go direct to the mill, while the overflow goes to a thickener, from which the thickened discharge also goes to the regrinding mill. An alternative arrangement, and one that has proved satisfactory, is the use of the Genter thickener, in which case the product to be ground goes direct to the thickener. The mill, as a rule, is operated in closed circuit with a bowl classifier, and very high circulating loads, up to 1000 per cent, are recommended.

The types of flotation machines are numerous, but when compared to a few well-known types of standard machines, the majority have little merit. In the flotation of lead-zinc ores practice seems to favour the use of sub-aeration machines for lead and standard spitz type machines for zinc. The pneumatic type machine has come very much to the fore with the development of a new type of cell known as the MacIntosh, which uses a revolving blanket. Blanket troubles caused by banking and clogging are eliminated by this type of construction. The cell is especially adaptable for selective separations.

A number of new flotation reagents of great merit have been recently developed. Cresylic acid reconstructed with phosphorous pentasulphide is a new reagent which is highly selective in the presence of sulphides of iron and can be used on copper-zinc or lead-zinc ores. "Flotagen S" and "Flotagen" are two other new reagents, the former functions with cyanide, sodium sulphite, and also with sodium thiosulphate. Flotagen, which should be added as a solid, can be used only in alkaline pulps and requires sufficient agitation before flotation to allow it to go into solution, although violent mechanical agitation is not necessary. Flotagen S, on the other hand, goes into solution readily and should not be in contact with the ore pulp for more than one minute before flotation. It is not advisable to allow Flotagen to remain in contact any longer than necessary, as there is often a marked decrease in efficiency if the time is prolonged. The use of xanthate, phospho-cresylic acid, and other soluble flotation agents has been found to have a beneficial effect on the filtration of the flotation concentrate. By the substitution of these reagents for oils it has been found, in some cases, that the tonnage handled by the filters can be doubled.

The length of time that the modifying reagent is allowed to remain in contact with the ore before flotation is a most important factor and until recently has been more or less disregarded. The general

practice has been to take care of the time element by adding the reagents to the ball mill. This was effective in most cases but took care only of the flotation of the first mineral. It has been found advisable to provide time contact for certain reagents used for the flotation of the second mineral. Improved results have been obtained and the consumption of reagents lessened by increasing the time of contact before the flotation of the first mineral by the addition of conditioning tanks equipped with agitating devices. Present practice uses up to $2\frac{1}{2}$ hours' contact before the lead separation and up to one hour between the lead and zinc separations. The time factor, of course, depends entirely on the ore being treated, and some ores require only a few minutes. The claim that violent agitation is necessary during the time contact is, in the writer's opinion, a very mistaken assertion.

It is the experience of those in charge of concentrators for the flotation of heavy sulphide ores that to obtain maximum recovery the ore must be freshly broken, and that only a small supply of broken ore can be carried in the stopes, as the slightest oxidation interferes with the flotation, and heavy losses result. This applies particularly to the taking of samples which are to be used for experimental work. The author has frequently found it necessary to request that new samples be taken from freshly mined ore because of the difficulty in obtaining any separation on previous samples submitted. The second samples, taken from freshly mined ore, responded readily to selective flotation.

Oxidation may also take place in the grinding mills when heavy sulphides are being treated. Serious difficulty has been experienced in some mills with this problem. The condition is undoubtedly caused by the heat generated at the point of contact by the impact of the balls striking the lumps of sulphide ore. The action can be overcome by increasing the classifier capacity and building up a high circulating load to the mill so as to keep the pulp in the mill only a very short time, by the addition of modifying reagents in the proper amount, or by carrying a more dilute pulp in the grinding mill at the sacrifice of capacity.

There are two different methods of applying selective flotation to an ore: The first, and the one which is more commonly used, is a straight flotation separation made on the original pulp. The second method is by bulk flotation of the two or more minerals and the subsequent separation of them by selective flotation. Some misunderstanding seems to exist regarding the use of these two methods, and attempts are frequently made to compare their respective merits without taking into consideration the particular ore on which they are to be used. Although each method has a distinct field of its own, there is generally little room left for any argument as to which method should be used, provided that sufficient information on the character and behaviour of the ore has been obtained by previous experimental work. An ore containing 2.25 per cent lead, and 6 per cent zinc, providing there are no interfering soluble salts present, calls for a straight selective float. But if the lead was around 1 per cent or less, a bulk float of both the lead and zinc with a selective separation on the bulk concentrate would probably be the better method. In the latter case, if possible, only a part of the zinc

should be cut with the lead. This would be possible, in the majority of cases, because of the better floating properties of the lead, and by not adding the copper sulphate reagent used to promote the flotation of zinc until after the lead had been floated. By this method the tonnage of bulk concentrate to be treated in the time contact tanks would be much less than if the whole of the zinc had been floated. The ratio of lead to zinc would also be higher which would assist the separation to be obtained between the lead and zinc in the subsequent selective separation. In an ore containing soluble copper salts, a bulk flotation of the zinc and lead is practically always necessary. The reason for this is the amount of modifying reagents necessary to overcome the promoter action of the copper salts on the zinc.

In the general practice of bulk flotation the zinc concentrate is obtained as a tailing from the flotation of the lead. In the case of an ore containing a large proportion of pyrites, considerable trouble may be expected from the iron which will tend to float in the bulk concentrate. As a rule it is found impossible to reactivate the zinc mineral after the bulk concentrate has been subject to the time-contact treatment with the reagents required to effect the separation. The iron which had floated would remain with the zinc preventing possibly a shipping grade of zinc concentrate being produced.

Heavy sulphide copper ores frequently contain small amounts of zinc. Such an ore having an approximate analysis of copper 1.8, zinc 0.6, and iron 20 per cent or higher, and also containing considerable soluble sulphates, could be concentrated for the copper content alone, the zinc being discarded. The copper concentrate, however, would probably carry 5 per cent zinc. This amount would be objectionable to the smelter. The reagent costs would be too high, owing to the presence of soluble salts, to attempt to retard the zinc by the ordinary methods. The flotation of a bulk concentrate containing the copper, zinc, and part of the iron would in all probability give a product which it would pay to work on for the production of a marketable zinc concentrate.

On the other hand, a heavy sulphide ore containing copper 2, and zinc 6 per cent, lends itself much better to a straight selective flotation. Another point worth drawing attention to is that should the proportion of zinc in the above ore be raised to 10 per cent, the copper concentrate would carry a much greater amount of zinc and the final recovery of zinc might be lower than from the first example.

Some ores contain soluble salts, such as sulphates of iron, alumina, magnesium, copper, etc., in such quantities that even the addition of large amounts of soda ash or lime will not prevent the modifying reagents being destroyed as quickly as they are added, thus preventing them from fulfilling their function. There are in general two ways of attacking such a problem: (1) to endeavour to make a bulk concentrate with the idea of subsequently making the selective separation on the bulk concentrate, and (2) to wash the ore by crushing and partly dewatering and again bringing the pulp to the required density by the addition of fresh alkaline water. Ores of the above characteristic present a research problem which will tax the ingenuity of the metal-

lurgist to solve. Fortunately, mining companies which have been successful in overcoming such difficulties are showing a very fair attitude by publishing their methods.

Owing to separate problems involved in the work from which the information contained in this article has been obtained, the method of presentation will be confined to giving examples of the separations obtained on different types of ore, and the methods and reagents used to obtain them.

MODIFYING REAGENTS

In order that the reader may better understand the methods used in obtaining these separations, a brief description of the uses of some of the more common modifying reagents will be given. For a description of the properties of oils and flotation reagents, see Memorandum Series No. 11: "Selective Flotation as Applied to Canadian Ores."

Lime. This reagent was one of the first to be used in flotation. It requires a time contact with the ore and, if possible, should be added in the grinding mills. It is used to overcome the deleterious effect of soluble salts which are frequently present in ores, but its principal use is as a depressant for iron sulphides in the selective separation of lead and zinc, copper and zinc, and copper and iron sulphides. It also has a marked depressing action on zinc sulphides in their separation from lead and copper. If added in too large amounts it will also prevent the lead from floating. Chalcopyrite will float in a strongly alkaline pulp, but chalcocite and some of the other copper minerals are affected by the least excess. Therefore, lime should not be used in their flotation. As a general rule, much better results can be obtained in the flotation of these minerals with a combination of soda ash and cyanide than with lime.

Soda Ash (Sodium Carbonate). The discussion of the functions of soda ash is difficult. To begin with, soda ash will revive pyrite under certain conditions. We have just completed a successful 20-ton flotation test on a tailing from a cyanide mill. In the mill the ore had been ground to 75 per cent —200 mesh in cyanide solution and agitated in pachuca for 48 hours, lime being used to maintain the alkalinity, then vacuum filtered, repulped, and filtered again. The final filter cake was the product floated. A high-grade sulphide concentrate was obtained by the use of soda to revive the pyrite which contained the remaining gold values. The flotation conditions obtained during the test were excellent. It will be observed that in this case lime is coupled with cyanide, and it has been our general experience that when lime and cyanide are coupled the subsequent addition of soda ash will promote the flotation of pyrite which has been deadened by the combined action of these two reagents. On the other hand, we have found that soda ash alone has a slight deadening action on pyrite, and quite a decided deadening effect on pyrrhotite.

In the flotation of copper-nickel ores high ratios of concentration have been obtained, dropping both pyrite and pyrrhotite, by using soda ash as the only addition reagent. Also a similar separation, copper

sulphate being used as well, on ores containing sphalerite, pyrite, and pyrrhotite. But then again, we have frequently had the opposite experience of obtaining better results by using soda ash when endeavouring to float pyrite, for example, a gold ore in which the values are contained in the sulphides. From these apparently contradictory results, we have drawn the general conclusion that the effect of soda ash is to increase the difference between the relative flotation properties of the minerals.

Sodium Bicarbonate. The use of this reagent is preferred in the selective separation of lead and zinc when the ore contains only a minor amount of pyrite or pyrrhotite.

Sodium Cyanide. This is one of the most valuable of all modifying reagents, since by its action the surfaces of the zinc sulphides and iron sulphides are modified so that they do not float. By taking advantage of this action, lead and copper sulphides may be separated from them. As a rule, cyanide is used with either soda ash or lime, generally with soda ash. It is frequently used in combination with zinc sulphate, in which case the solutions of the two salts are brought together just as they enter the flotation pulp so as to form an emulsion of zinc cyanide. Time contact is important when using cyanide, some ores requiring one hour contact, others only a few minutes' contact.

Sodium Sulphite and Thiosulphate. These two salts have similar properties. Sometimes one is the more effective, sometimes the other, depending entirely on the ore. They are used for the same purpose as cyanide and in a number of cases have been found just as effective.

Zinc Sulphate. This is used as a depressant for zinc in separations of lead and zinc. It is possible that its use might be extended to the separation of copper and zinc, but the writer has never had occasion to try it.

Sodium Acid Phosphate. This is used occasionally as a depressant for zinc in lead and zinc separations.

Sodium Hydrosulphite. This can be used also as a depressant for zinc.

Sodium Dichromate. This salt is used in the selective separation of lead and zinc with cyanide, on ores which contain large amounts of iron sulphides. It seems to have a beneficial action. It is seldom necessary.

Copper Sulphate. This is added in quantities of one-half pound to two pounds per ton, as a decided promoter for the flotation of zinc.

ORES CONSISTING OF COMPLEX MIXTURES OF GALENA, SPHALERITE, PYRITE AND PYRRHOTITE, CONTAINING GOLD AND SILVER

The problems involved in the treatment of such ores are varied although the recovery of the gold and silver is, in most cases, the major problem. In connexion with the recovery of silver some very interesting results were obtained on an ore received from the Slocan district, B.C., where the recovery of silver is the major problem. The particular sample was sent in from a mill which was recovering only 50 per cent of the silver in

the feed, by a combination of gravity separation and flotation. The typical flow-sheet of the district includes jigs and tables for concentrating the lead. The tailings containing the zinc and a large proportion of the silver are dewatered, reground, and floated, the products being a low-grade lead product, high in silver, and a high-grade zinc concentrate.

In order to determine the association of the silver, pieces of galena, blende, and gangue were picked out and assayed for silver. The pieces were approximately $\frac{3}{8}$ inch in size. The results were as follows:

	Assay silver
Galena, crystallized in large cubes.....	100.7 oz./ton
Galena, finely disseminated, aggregates of crystals.....	102.0 "
Zinc blende (Pb 0.16, Zn 63.30 per cent).....	7.5 "
Gangue (Pb trace, Zn 1.66 per cent).....	1.18 "

This shows conclusively that in place the silver, in some form, is associated almost exclusively with the galena. Samples of the ore were crushed to about 10 mesh and a lead concentrate made by tabling. The concentrate contained 70 per cent lead but only 67 ounces silver per ton, as against 100 ounces per ton in the coarse unbroken pieces of pure galena. Other tests showed that the finer the galena was crushed the less silver the lead concentrate contained. It was obvious, therefore, that the silver was being freed by crushing. This was confirmed by the results of the flotation tests, as it was found that the largest proportion of the silver generally reported in the zinc concentrate. There was abundant evidence to show that the silver which was freed from the galena by crushing was in an extremely fine form. This was further borne out by some work on the products from one of the operating, combined gravity and flotation mills which showed that 75 per cent of the silver in the zinc flotation concentrate was in the —200-mesh material, and that the losses in the tailing were due to very finely divided silver mineral.

It follows that owing to the finely divided state of a part of the silver values, high recovery could not be expected in a mill using gravity concentrators, nor would a high recovery of silver be obtained by the use of jigs or tables. It would also follow that a large amount of the fine silver would be lost when the jig and table tailings were dewatered preparatory to regrinding for flotation of the zinc. The fact that the tailings from the flotation department treating the reground tailings and fines from the jigs and tables contain high values in silver shows that the fine silver is very hard to collect in the flotation concentrate. This is in accordance with personal experience of the writer who has observed in a number of cases that very fine particles of minerals by themselves are very difficult to oil and collect in the froth. It had also been observed that in a flotation pulp containing relatively coarse grains of minerals, the fine grains exhibited a marked tendency to cluster and cling around the larger ones. This characteristic suggested that in order to collect and recover the finely divided silver values a greater proportion of large grains must be present so that by the bulk action of the flotation of the coarse grains, the fine grains would be gathered in and collected.

For these reasons separation of the lead and zinc from the original mill pulp by selective flotation was regarded as a solution of the problem. Later, after a method of separating the lead and zinc had been worked out, it was found that very high recoveries of the silver were obtained, the largest proportion reporting in the zinc concentrate.

In general, the selective separation of lead and zinc is obtained by the addition to the ore pulp of some reagent that will temporarily deaden the floating properties of the zinc, thus permitting the recovery of a high-grade lead concentrate. In the case of the majority of Canadian ores, the separations are made more difficult by the presence of large amounts of iron sulphides, both pyrite and pyrrhotite, and some reagent must be used which will permanently deaden these two sulphides so that they will not float with either the lead or zinc. The only reagents used which perform both these functions are sodium cyanide, sodium sulphite, and sodium thiosulphate, but the following have been used with some success in temporarily deadening the zinc: zinc sulphate, sodium acid phosphate, and sodium hydrosulphite. So far as has been observed by the writer, these reagents when used separately have no effect in preventing the iron sulphides from floating. The action of zinc sulphate is much more effective when used with sodium cyanide, the two being mixed by bringing the streams of the two reagents together just as they enter the flotation pulp.

These modifying reagents are used in an alkaline pulp, either with soda ash or lime. Lime has a permanent deadening effect on pyrite and pyrrhotite, and also affects galena in the same way but to a lesser degree. Cases have been reported of sodium sulphide being used to effect a separation between lead and zinc, but, with the exception of one instance, no success has been obtained with it in our laboratories. By the use of copper sulphate, the action of these modifying reagents is destroyed, and the flotation properties of the zinc minerals is revived. The length of time the modifying reagent is in contact with the ore is important. As a general rule it may be stated that the effect of a small amount in contact for a longer period approaches the effect of a larger amount in contact for a shorter period. Mills are now erected in which surge tanks are used to increase the time the modifying reagents are in contact with the ore.

The problems which are met with in ore treatment are so complicated by marketing conditions, such as requirements of smelter, freight and treatment charges, etc., that it is difficult to write concisely on the subject when treated in the broader sense.

An example is given of an ore on which the test work has just been completed. The ore was a lead-zinc containing both gold and silver—lead 3.1 per cent, zinc 4.6 per cent, gold 0.35 ounce per ton, silver 2.5 ounces per ton, iron 12 per cent (present as pyrite). In this ore the zinc shows a marked persistence to float with the lead in spite of the use of both cyanide and zinc sulphate, and the inhibiting effect on the pyrite was not sufficient to allow a marketable zinc concentrate to be produced. That is to say, when the flotation of zinc was attempted the iron floated in excess. The fact that the effect of the cyanide was not sufficient to prevent the pyrite from floating with the zinc did not allow a marketable zinc concen-

trate being made. To add to this complication was the fact that part of the gold was associated with the pyrite and in order to obtain the maximum returns from the smelter it was necessary to obtain as much gold as possible with the lead concentrate.

It was advisable, therefore, to endeavour to float at least a part of the pyrite with the lead and at the same time prevent the zinc from reporting there also. Lime could not be used as it deadened the pyrite completely, and 65 per cent of the gold reported in the final tailing. It was found that by grinding the ore with soda ash, 6 pounds per ton, which was sufficient to keep the pulp alkaline, and by adding an emulsion of cyanide, 0.25 pound per ton, and zinc sulphate, 2 pounds per ton, to a time contact tank for a contact of 20 minutes before the flotation, sufficient iron floated so that a lead concentrate could be obtained containing 82 per cent of the gold values, 81 per cent of the silver, and ran 32 per cent lead with a recovery of 90 per cent, and assayed 13 per cent zinc. By subjecting the lead tailing to a time contact with 5 pounds lime and 1.5 pounds copper sulphate per ton, and using xanthate, a 52 per cent zinc concentrate was produced with a recovery of 68 to 70 per cent. It should be explained that tests were also run using cyanide alone, and no difference was noticed as far as the inhibiting effect on the pyrite was concerned, but the effect on the sphalerite was greatly increased by the addition of the zinc sulphate. The reason for adding these two reagents as an emulsion was that apparently better results were obtained in that it inhibited more of the zinc and less of the pyrite from floating than if the two were added separately.

The use, wherever possible, of pilot tables for indicating the work the flotation machines are doing is recommended. If the lead and zinc separations are not perfect enough the flotation concentrate may be tabled without causing undue losses. The action of flotation coagulates the fine mineral grains so that they settle much more rapidly and there is not the same tendency for the fine mineral to be lost in the overflow of the thickeners.

EXAMPLES OF SELECTIVE SEPARATION ON LEAD-ZINC ORES

1. The ore consisted of galena and sphalerite with silver, the sulphides being disseminated throughout the gangue, requiring fine grinding.

Analysis:

Lead.....	7.10 per cent
Zinc.....	14.06 "
Silver.....	3.04 oz./ton
Gold.....	trace

The ore was ground to 50 mesh for flotation. A small-scale test only was made.

Results:

Product	Weight, per cent	Assays			Per cent of values		
		Pb per cent	Zn per cent	Ag oz./ton	Pb	Zn	Ag
Lead concentrate.....	13.6	48.90	8.45	9.06	77.0	8.5	45.6
Zinc concentrate.....	24.5	6.25	46.95	5.30	17.7	84.7	47.9
Tailing.....	61.9	0.75	1.49	0.29	5.3	6.8	6.6

Reagents Used:

Soda ash.....	4.0 lb./ton	added and ground with ore
Cyanide.....	0.5	" " "
Thiocarbamilide.....	0.2	" " "
Zinc sulphate.....	0.5	" " "
Pine oil.....	0.1	" " to lead cell
Copper sulphate.....	1.0	" " to zinc cell
Xanthate.....	0.3	" " "
Pine oil.....	0.1	" " "

Both cyanide and zinc sulphate were used to modify the flotation properties of the zinc and pyrite. On this particular ore, the addition of zinc sulphate gave a sharper separation between the lead and zinc than was obtained with cyanide alone. The length of time the ore was kept in contact with the cyanide was found to be an important factor. A mill is under construction based on the results of these tests.

2. The ore was a heavy sulphide type containing gold, silver, galena, sphalerite, pyrite, and pyrrhotite, with very little gangue. It required grinding to approximately 65 mesh for flotation.

Analysis:

Copper.....	0.73 per cent
Lead.....	18.47 "
Zinc.....	23.54 "
Gold.....	0.42 oz./ton
Silver.....	19.42 "

A small-scale test only was made.

Results:

Product	Weight, per cent	Assays				Per cent of values			
		Pb per cent	Zn per cent	Au oz./ton	Ag oz./ton	Pb	Zn	Au	Ag
Lead concentrate.....	32.7	52.31	10.97	0.80	46.3	93.0	13.2	63.9	81.3
Zinc ".....	41.1	2.31	47.69	0.16	5.74	5.2	83.3	16.1	12.7
Table ".....	5.2	0.90	9.80	11.4	2.7
Tailing.....	21.0	1.25	1.25	0.17	2.93	1.3	1.4	8.6	3.3

Reagents Used:

Soda ash.....	6.0 lb./ton	added and ground with ore.
Thiocarbamilide.....	0.2	" " "
Sodium cyanide.....	0.3	" " five minutes' contact.
Cresylic acid.....	0.2	" " added to lead cells.
Copper sulphate.....	1.25	" " zinc cells.
Xanthate.....	0.4	" " "
Pine oil.....	0.1	" " "

A mill, based on the results of these tests, has been built and is in successful operation.

3. The following separation was obtained on dump middlings consisting of sphalerite, pyrite, and pyrrhotite, and only 2 to 3 per cent gangue.

Analysis:

Lead.....	2.45 per cent
Zinc.....	23.00 "

The following are results of a 5-ton test:

Product	Assays		Approx. recoveries	
	Pb per cent	Zn per cent	Pb per cent	Zn per cent
Lead concentrate.....	16.50	12.40	85
Zinc ".....	0.30	51.62	91
Tailing.....	0.45	0.85

Reagents Used:

Soda ash.....	18.0 lb./ton added and ground with ore.
Cyanide.....	0.26 " " to lead cells.
Acid coal-tar creosote.....	0.21 " " and ground with ore.
Water-gas tar.....	0.21 " " " "
Copper sulphate.....	2.0 " " to zinc cells.
Xanthate.....	0.4 " " " "

A mill, based on these results, has been built and is in operation.

4. The sulphides in this ore consisted of sphalerite, galena, pyrrhotite and a little pyrite and chalcopyrite, also values in gold and silver.

Analysis:

Lead.....	1.96 per cent
Zinc.....	5.52 "
Iron.....	7.42 "
Gold.....	0.02 oz./ton
Silver.....	3.28 "

The following are results of a 5-ton test:

Product	Assays					Approx. recoveries	
	Zn per cent	Pb per cent	Fe per cent	Au oz./ton	Ag oz./ton	Pb per cent	Zn per cent
Lead concentrate.....	4.17	51.09	2.60	0.30	40.0	89.4
Zinc ".....	47.18	1.15	9.0	94.8
Tailing.....	0.30	0.15	9.4

Reagents Used:

Soda ash.....	2.0 lb./ton added and ground with ore.
Acid creosote.....	0.2 " " " "
Water-gas tar.....	0.2 " " " "
Sodium cyanide.....	0.15 " " to lead cells.
Cresylic acid.....	0.4 " " " "
Copper sulphate.....	1.0 " " to zinc cells.
No. 2 neutral creosote.....	0.4 " " " "
TT mixture.....	0.2 " " " "
Pine oil No. 5.....	0.2 " " " "

5. The ore consisted of sulphides of iron, copper, lead, and zinc, finely crystalline and intimately associated. It also carried values in silver and gold.

Analysis:

Copper.....	2.35 per cent
Arsenic.....	0.76 "
Lead.....	1.26 "
Zinc.....	14.25 "
Gold.....	0.03 oz./ton
Silver.....	3.91 "
Antimony.....	trace

A small-scale test only was made.

Results:

Product	Weight percent	Assays					Per cent of values				
		Cu percent	Pb percent	Zn percent	Au oz./ton	Ag oz./ton	Cu	Pb	Zn	Au	Ag
Cu-Pb concentrate..	17.0	12.32	6.62	13.86	0.08	4.76	90.3	93.3	16.3	41.6	70.0
Zinc concentrate..	23.0	0.56	0.25	49.6	0.04	3.0	5.5	4.2	80.5	29.1	19.6
Tailing.....	60.0	0.16	0.05	0.7	0.015	0.56	4.1	2.5	3.0	28.3	9.5

Reagents Used:

Soda ash.....	10.0 lb./ton added and ground with ore.
Thiocarbamide.....	0.25 " " " "
Sodium cyanide.....	0.24 " " " "
Pine oil.....	0.10 " " to lead cells.
Copper sulphate.....	1.5 " " to zinc cells.
Xanthate.....	0.3 " " "
Pine oil.....	0.1 " " "

6. This sample consisted of material taken from a tailing dump produced from the concentration of a lead-zinc ore.

Analysis:

Lead.....	0.07 per cent.
Zinc.....	10.06 "

The following are results of a 4-ton test:

Product	Assays			Approx. recoveries	
	Pb per cent	Zn per cent	Cu per cent	Pb per cent	Zn per cent
Lead concentrate.....	22.81	10.41	4.23	74
Zinc ".....	0.14	49.09	85
Tailing.....	0.06	0.40

Reagents Used:

Soda ash.....	2.0 lb./ton added and ground with ore.
Acid coal-tar creosote.....	0.3 " " " "
Sodium cyanide.....	0.09 " " to lead cells.
Cresylic acid.....	0.6 " " "
Copper sulphate.....	1.24 " " to zinc cells.
Xanthate.....	0.25 " " "
Pine oil.....	0.25 " " "

7. This sample was a lead-zinc ore carrying silver, the galena and sphalerite were intimately disseminated with pyrite and pyrrhotite.

Analysis:

Lead.....	3.35 per cent.
Zinc.....	7.08 "
Iron.....	4.82 "
Silver.....	9.5 oz./ton.

Small-scale test only was made.

Results:

Product	Weight, per cent	Assays			Per cent of values		
		Pb per cent	Zn per cent	Ag oz./ton	Pb	Zn	Ag
Lead concentrate.....	6.0	46.53	2.74	134.0	80.9	2.2	72.2
Lead middling.....	5.9	9.02	6.88	28.5	15.5	5.6	15.2
Zinc concentrate.....	10.9	0.32	55.32	4.5	1.0	82.7	4.4
Zinc middling.....	3.5	0.80	15.16	7.44	0.8	7.4	2.4
Tailing.....	73.7	0.08	0.21	0.88	1.7	2.1	5.8

Reagents Used:

Soda ash.....	5.0 lb./ton added and ground with ore.
Thiocarbamide.....	0.2 " " "
Sodium cyanide.....	0.24 " " to lead cells.
Cresylic acid.....	0.75 " " "
Copper sulphate.....	1.0 " " to zinc cells.
No. 2 neutral creosote.....	0.5 " " "
Xanthate.....	0.1 " " "

ORES CONSISTING OF INTIMATE MIXTURE OF CHALCOPYRITE, PYRITE AND PYRRHOTITE, WITH AND WITHOUT APPRECIABLE AMOUNTS OF GOLD

The successful separation of chalcopyrite from the iron sulphides, both pyrrhotite and pyrite, requires the solution of two problems. The first problem is to free the copper mineral from the iron. Fine grinding is nearly always necessary owing to the intimate association of chalcopyrite with the pyrite. The development within the last few years in the art of fine grinding makes it possible to free economically the two minerals in the finest grained ores. The second problem is to prevent the iron from floating with the copper. In the case of chalcopyrite ore containing no gold, this can generally be done by maintaining a strongly alkaline pulp with lime. Lime has a permanent deadening effect on pyrite and pyrrhotite and very high ratios of concentration can be obtained.

As a general rule, copper ores containing gold present an entirely different problem, that of recovering the gold in the copper concentrate. The use of lime to retard the iron, except in rare instances, tends to throw the greater part of the gold with the iron tailing. Soda ash or caustic soda is used because it has not the detrimental effect on the gold, but unfortunately it does not deaden the pyrite. Its effect is sufficient to keep the pyrrhotite from floating, but pyrite floats readily. Soda disperses or

deflocculates the gangue slimes whereas lime coagulates them. However, in the case of soda, there also seems to be a tendency to coagulate the sulphides.

Sodium cyanide added to a pulp, the alkalinity of which is maintained with soda ash, can be used to permanently deaden at least a part of the pyrite, but it also has a modifying effect in some cases on the chalcopyrite as well. The chalcopyrite can be reactivated by the addition of a limited amount of copper sulphate. By the use of cyanide under these conditions, it is often possible to obtain a higher ratio of concentration without lowering the recovery of the gold.

EXAMPLES OF SELECTIVE SEPARATIONS ON COPPER-IRON ORES, WITH AND WITHOUT GOLD VALUES

1. This ore was a complex sulphide consisting of chalcopyrite intimately associated with pyrite. The chalcopyrite was disseminated throughout the pyrite, the sulphides constituting 80 per cent of the ore. The ore was ground to pass 100 mesh, 75 per cent being —200 mesh.

Analysis:

Copper.....	3.0 per cent.
Iron.....	40.0 "
Insoluble.....	7.5 "
Other gangue.....	4.5 "
Sulphur.....	45.0 "

Results of Tonnage-scale Test:

Copper concentrate.....	22.77 per cent copper.
Tailing.....	0.12 " "
Recovery.....	96 per cent.

Reagents Used:

Lime.....	10.0 lb./ton	added to ball mill.
Barrett's No. 634.....	0.6 "	" "
TT mixture.....	0.25 "	" to cells.

2. The ore was a copper-gold ore from the Rouyn district, Quebec.

Analysis:

Copper.....	4.60 per cent.
Iron.....	19.83 "
Silica.....	50.64 "
Gold.....	0.04 oz./ton.
Silver.....	0.22 "

Small-scale test only was made.

Results:

Product	Weight, per cent	Assays		Per cent of values	
		Cu per cent	Au oz./ton	Cu	Au
Copper concentrate.....	35.3	12.14	0.11	95.0	81.1
Copper middling.....	8.5	1.39	0.04	2.6	7.1
Tailing.....	56.2	0.19	0.01	2.4	11.8

Reagents Used:

Soda ash.....	4.0 lb./ton added to ball mill
Xanthate.....	0.07 " " cells
Pine oil.....	0.05 " " "

3. This is a copper-gold ore from the Rouyn district, Quebec.

Analysis:

Copper.....	1.6 per cent
Gold.....	0.28 oz./ton

Small-scale test only was made.

Results:

Test	Product	Weight, per cent	Assays		Per cent of values	
			Cu, per cent	Au oz./ton	Cu	Au
A	Copper concentrate.....	31.0	4.77	0.76	90.4	85.2
	Tailing.....	69.0	0.11	0.06	9.6	14.8
B	Copper concentrate.....	22.9	6.50	0.98	94.1	80.06
	Tailing.....	77.1	0.12	0.07	5.9	19.94

Reagents Used:

(A) Soda ash.....	6.0 lb./ton added to ball mill
No. 4 creosote (Barrett's).....	0.4 " " "
Cresylic acid.....	0.4 " " cells
(B) Soda ash.....	6.0 " " ball mill
Cyanide.....	0.2 " " "
No. 4 creosote.....	0.2 " " "
Water-gas tar.....	0.05 " " "
Xanthate.....	0.05 " " cells
Copper sulphate.....	0.4 " " "

NOTE.—The use of copper sulphate in test B was found necessary in order to reactivate the chalcopryite which had been affected by cyanide.

ORES CONSISTING OF INTIMATE MIXTURES OF CHALCOPRYITE, SPHALERITE, AND IRON SULPHIDES

This type of ore probably presents the most difficult of all selective flotation problems. The use of cyanide to temporarily deaden the zinc and to permanently deaden the iron sulphides seems essential. However, separations have been obtained in our laboratory by floating the copper in alkaline pulp by the addition of lime. For the flotation of the zinc the copper tailing was dewatered to eliminate a part of the lime and the pulp then brought up to the required density by addition of fresh water. At this point soda ash was added and the zinc floated in a soda ash pulp, copper sulphate being used to help reactivate the zinc.

EXAMPLES OF SELECTIVE SEPARATION OF COPPER AND ZINC FROM PYRITE AND PYRRHOTITE GANGUES

(1) This was a copper-zinc ore from the Rouyn district, Quebec. The sample contained sphalerite and chalcopyrite together with a little gold and silver associated in a heavy sulphide gangue. The copper and zinc sulphides were finely disseminated and fine grinding was required.

Analysis:

Copper.....	4.15 per cent
Zinc.....	10.0 "
Gold.....	0.18 oz./ton
Silver.....	3.02 "

Small-scale test only was made.

Results:

Product	Weight, per cent	Assays				Per cent of values			
		Cu per cent	Zn per cent	Au oz./ton	Ag oz./ton	Cu	Zn	Au	Ag
Copper concentrate.....	17.3	20.66	6.91	0.56	11.96	85.5	11.8	50.3	65.7
Zinc concentrate.....	14.6	1.96	48.57	0.32	1.39	6.9	69.8	24.3	6.4
Zinc middling.....	8.1	1.30	11.37	0.30	3.80	2.6	9.1	12.7	9.8
Tailing.....	61.0	0.35	1.56	0.04	0.93	5.0	9.3	12.7	18.1

Reagents Used:

Soda ash.....	10.0 lb./ton added to ball mill
Cyanide.....	0.5 "
Thiocarbamide.....	0.2 " "
Cresylic acid.....	0.2 " " copper cells
Copper sulphate.....	1.0 " " zinc cells
Xanthate.....	0.3 " "

(2) This was a copper-zinc ore from the Rouyn district, Quebec. The sample contained sphalerite and chalcopyrite together with a little gold and silver in a heavy sulphide gangue. The copper and zinc sulphides were finely disseminated and fine grinding was required to free them. The erection of a mill is contemplated in the near future on the basis of the results of these tests.

Analysis:

Copper.....	7.36 per cent
Zinc.....	6.53 "
Lead.....	trace
Iron.....	35.40 "
Insoluble.....	18.6 "
Gold.....	0.03 oz./ton
Silver.....	2.45 "

Small-scale test only was made.

Results:

Product	Weight, per cent	Assays				Per cent of values			
		Cu per cent	Zn per cent	Au oz./ton	Ag oz./ton	Cu	Zn	Au	Ag
Copper concentrate.....	28.0	23.74	3.68	0.80	7.04	93.0	16.1	75.4	76.0
Zinc concentrate.....	9.8	1.22	47.38	0.01	0.31	1.7	72.9	3.4	3.1
Zinc middling.....	9.7	1.07	3.68	0.01	0.61	1.4	5.6	3.4	2.3
Tailing.....	52.5	0.53	0.65	0.01	0.92	3.9	5.4	17.8	18.6

Reagents Used:

Soda ash.....	4.0 lb./ton added to ball mill
Sodium cyanide.....	0.3 " " "
Thiocarbamide.....	0.2 " " "
Cresylic acid.....	0.08 " " "
Copper sulphate.....	0.4 " " copper cells
Copper sulphate.....	1.5 " " zinc cells
Xanthate.....	0.3 " " "
Pine oil No. 5.....	0.08 " " "

(3) The ore contained gold 0.099 and silver 1.36 ounces per ton; copper 1.89, zinc 4.61, lead 0.45 per cent. The ore was ground to 95 per cent —200 mesh for the test work. A large amount of research work has been carried out on this ore by different people. The best results obtained so far are with the following methods, two of which will be described.

Method No. 1. Lime, 3 pounds per ton, was added to the ball mill while grinding. Agitated with salt, 1 pound per ton, for 3 minutes. Floated copper with xanthate, 0.12 pound per ton, and 1 drop pine oil. Reclean concentrate with salt, $\frac{1}{2}$ pound per ton; xanthate, 0.02 pound per ton; and 1 drop pine oil. Agitated with lime, 1 pound per ton, and copper sulphate, $\frac{1}{4}$ pound per ton. Floated zinc with xanthate, 0.12 pound per ton, and 1 drop cresylic acid. Cleaned concentrate with copper sulphate, 0.1 pound per ton; lime, 1 pound per ton; and xanthate, 0.02 pound per ton.

Method No. 2. Agitated with soda ash, 3 pounds per ton; sodium sulphite, 1 pound per ton; and sodium cyanide, $\frac{1}{4}$ pound per ton. Floated copper with xanthate, 0.12 pound per ton, and 1 drop pine oil. Recleaned with soda ash, 1 pound per ton; sodium sulphite, $\frac{1}{2}$ pound per ton; and xanthate, 0.01 to 0.02 pound per ton. Agitated with soda ash, 1 pound per ton; copper sulphate, $\frac{1}{4}$ pound per ton; and sodium sulphite, 1 pound per ton. Floated zinc with xanthate, 0.12 pound per ton, and 1 drop cresylic acid. Cleaned with soda ash 1 pound per ton; sodium sulphite, $\frac{1}{2}$ pound per ton; xanthate, 0.01 pound per ton, and 1 drop pine oil.

The following results were obtained from the first method, and in order to approximate continuous operation a series test was made. The copper was floated as a rougher concentrate which was cleaned, the tailing from the cleaner being added to another batch of ore before this batch was floated for the copper. The tailing from the copper flotation was floated

for zinc, and a rougher zinc concentrate made. This was cleaned and the tailing from cleaning added to the copper tailing from the second copper flotation. The test was run continuously on 15 charges of ore, 1,500 grammes to the charge. The results follow:

—	Copper concentrate			Zinc concentrate			Tailing	
	Assay	Metal contained, per cent	Ratio of concentration	Assay	Metal contained, per cent	Ratio of concentration	Assay	Metal contained, per cent
Cu.....	17.0	83.0	10.85	1.1	5.05	11.3	0.28	11.95
Zn.....	3.0	5.97	43.5	83.2	0.6	10.83
Au.....	0.53	48.5	0.03	26.4	0.06	25.10
Ag.....	4.90	33.1	1.6	10.4	0.04	56.5
Pb.....	4.4	90.0	0.0	0.0	0.05	10.0

EXAMPLE OF SEPARATION OF CHALCOPYRITE AND GALENA

These two minerals are being successfully separated in actual mill operation. Previous to the operation of the selective separation, the two minerals were collected in the one concentrate and shipped to the smelter. The operation of the selective separation to produce separate lead and copper concentrates has increased the net returns per ton of crude by the amount of \$4.65.

The separation is effected by floating a bulk concentrate containing the lead and copper. This concentrate is subjected to a time contact with the required modifying reagents until sufficient action has taken place to prevent the copper from floating. The pulp is then floated to produce a lead concentrate, a copper concentrate being obtained as a tailing. The ore contains approximately: lead 6.9, copper 2.36, iron 25.0, and insoluble 24.4 per cent. An average day's run gives:

—	Pb per cent	Cu per cent	Fe per cent	Insol. per cent
Feed.....	6.0	1.75
Pb concentrate.....	51.7	2.7	4.1	21.6
Cu concentrate.....	3.9	22.7	24.1	15.5
Tailing.....	0.7	0.3

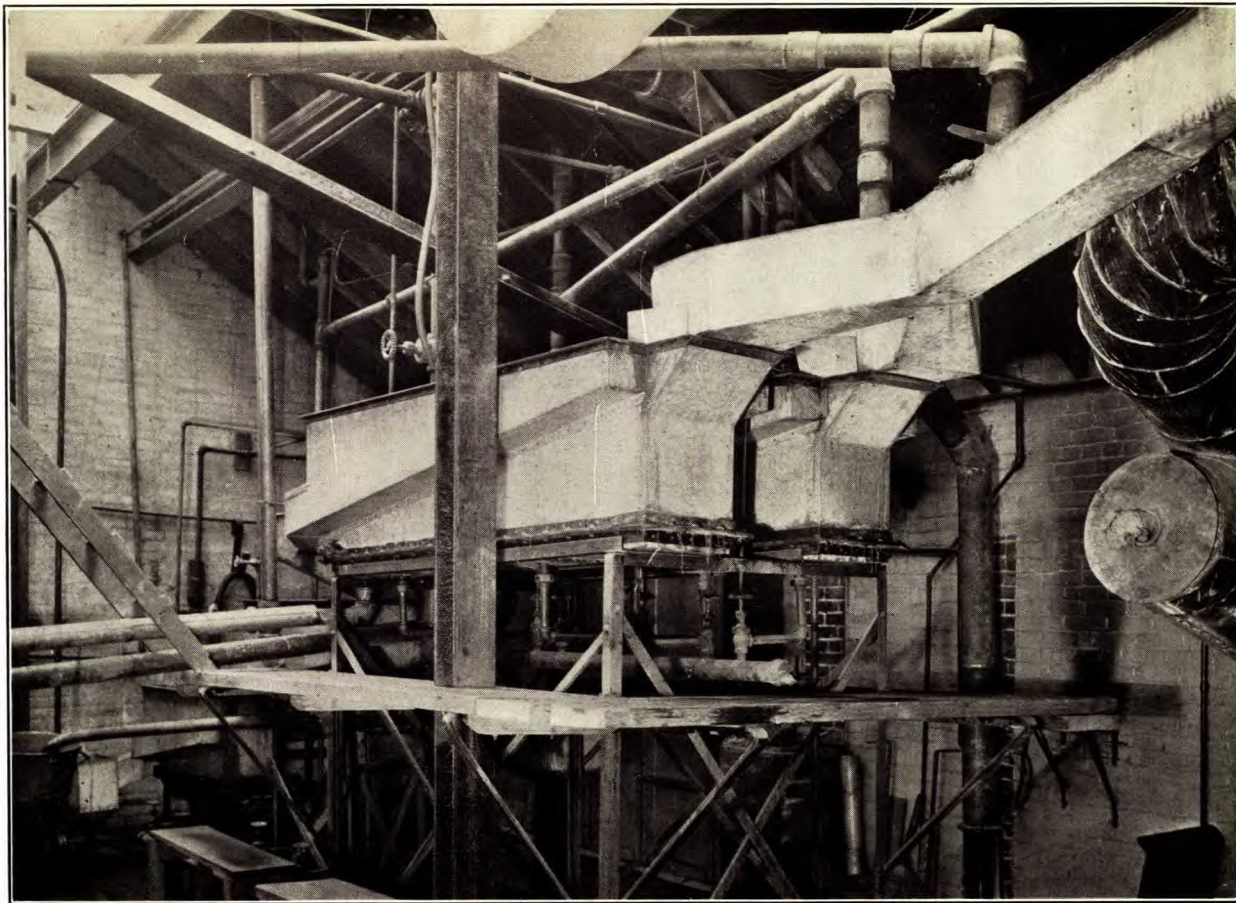
The flow-sheet of this separation is as follows: The feed goes to a 10-foot MacIntosh cell, tailing from this to a second 10-foot cell. The froth from the second is returned to the first; the rougher froth goes to a cleaner and recleaner. The recleaner froth, which is a bulk concentrate containing lead and copper, goes to the time contact tanks where reagents are added to effect selective separation. The pulp then comes back to a MacIntosh cell for the lead-copper separations. The tailing from this cell is the copper concentrate. The froth is recleaned in another MacIntosh cell

which makes the final lead concentrate. The tailing from this cell is returned to the cleaner which cleans the bulk concentrate and the tailing from the bulk concentrate cleaner goes back to the head of the circuit. The reagents used are approximately as follows:

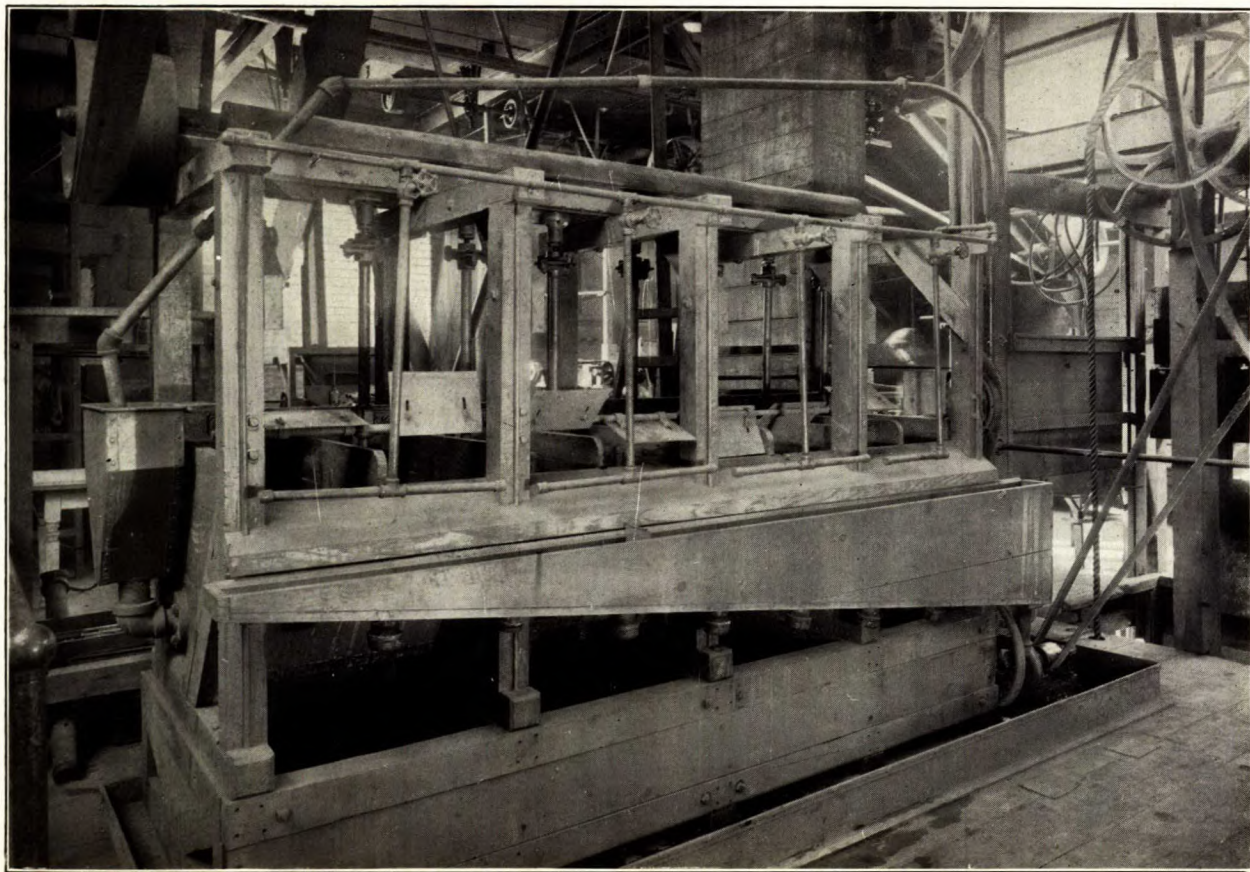
Ore ground with: soda ash 5.0, sodium cyanide 0.5, zinc sulphate 2.0, thiocarbamilide 0.1, and xanthate 0.1 pounds per ton. The ore is then floated with the addition of xanthate 0.16 pound per ton, added to the cells. This produces a bulk concentrate which is given a time contact of 15 minutes with, lime 1.0, cyanide 0.5 pound per ton, and then floated for the lead concentrate with TT or T & A mixtures.

The results of the experiments given here are not confined to laboratory experiments; they can, under proper control, be obtained in actual milling operations, and in general they can be improved upon. It was pointed out at the beginning of this article that selective flotation is no longer an uncertain operation, and no mining company should hesitate to adopt it on this account. The increasing number of customs flotation plants in the United States is the proof of this statement.

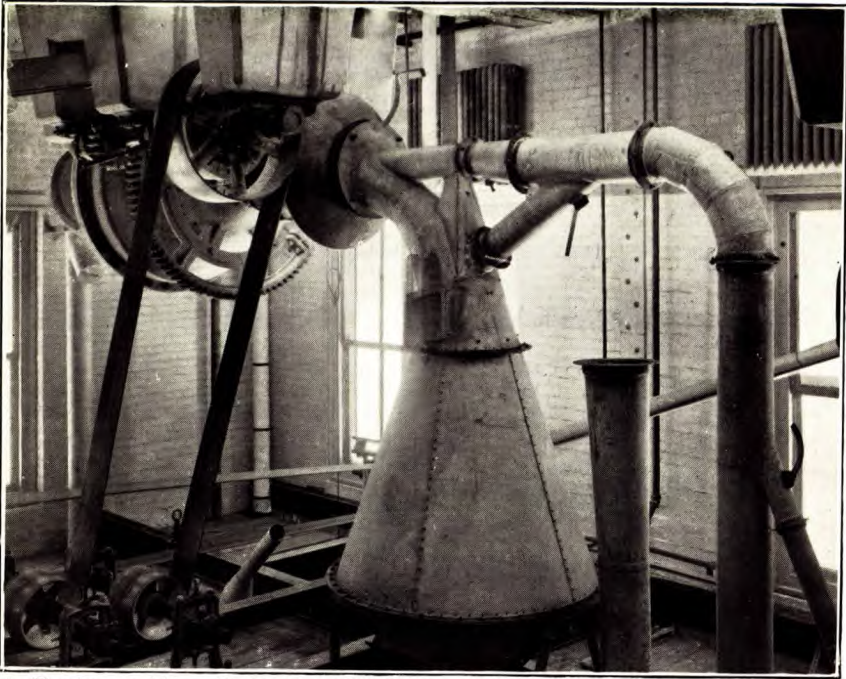
If possible, a small pilot plant should be erected at first to develop the proper methods of control. The controlling condition must not only be known, but it must also be known how to maintain it in actual operations. Some idea of the delicate control required is obtained when it is considered that often twelve or more different reagents are being used in quantities varying from one-tenth of a pound to ten pounds per ton. If it is not possible to build a pilot plant, then the next best procedure is to have tonnage-scale tests made at some laboratory equipped to perform them. In this connexion it is desirable for the company to have its own engineer representative present during the tests.



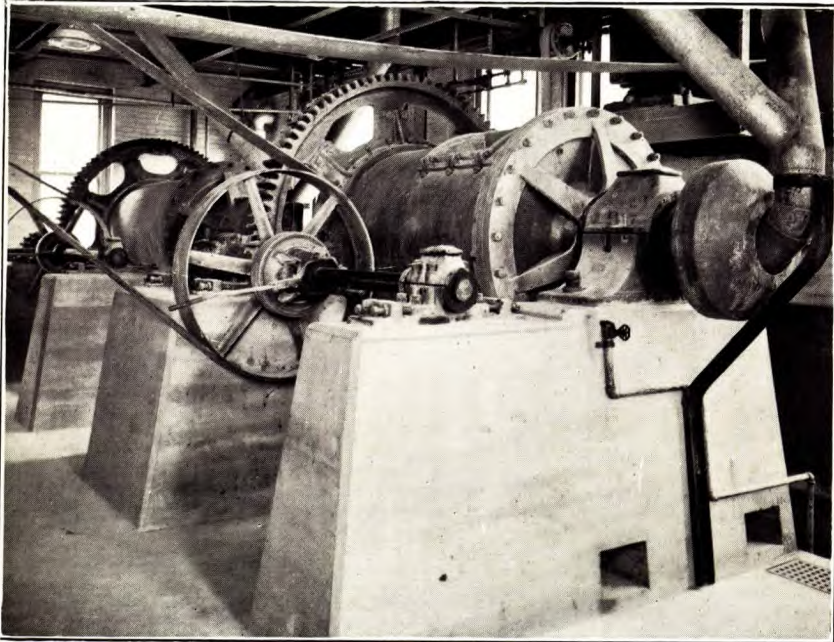
Callow pneumatic flotation unit. This unit consists of two rougher and two cleaner cells and is used for large-scale or tonnage check tests for comparative tests with the mechanical agitation type. Both types are used for selective flotation tests, one mineral being floated on one and a second mineral on the other. It has a capacity of 1,000 pounds per hour.



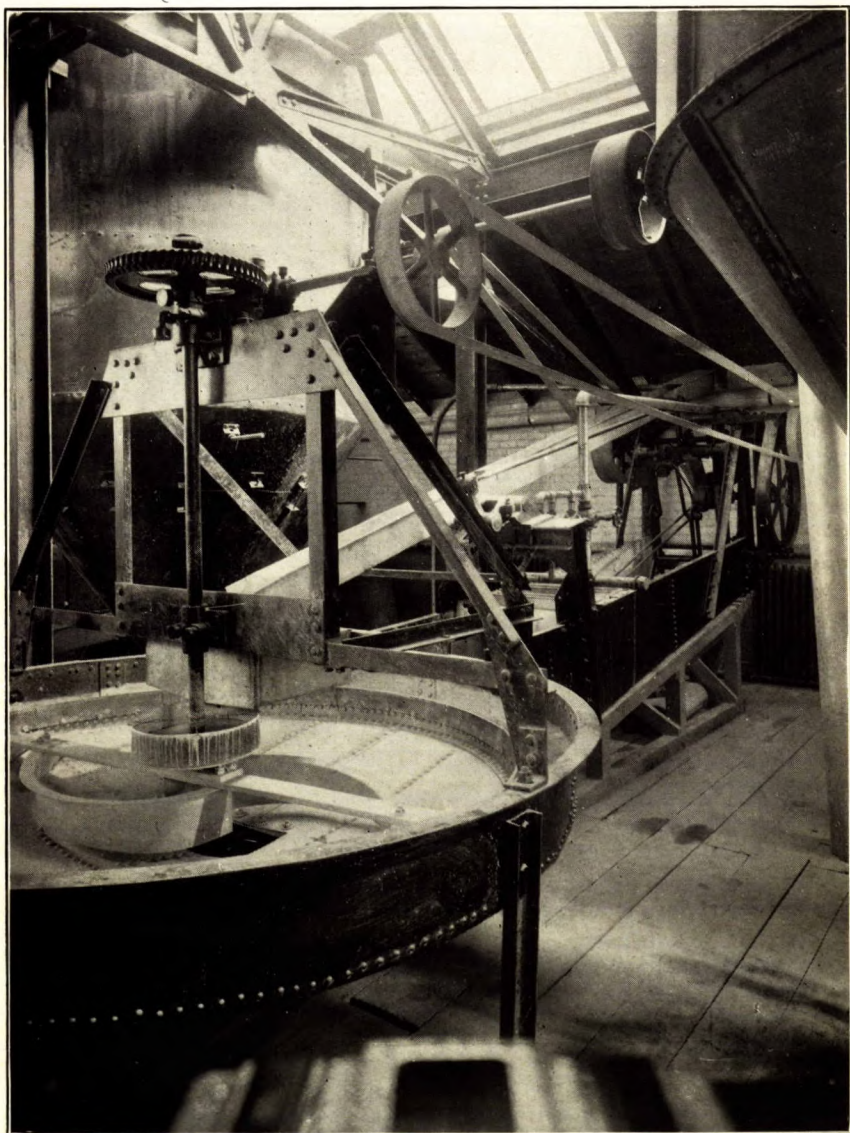
Greenawalt mechanical agitation flotation machine. This machine has eight 12-inch cells and is used for large-scale or tonnage flotation tests. It has a capacity of about 1,000 pounds per hour.



A. Hardinge pebble mill (4' 6" x 18") lined with Silex, and with air separator equipment for the grinding of non-metallies. This mill has in circuit with it an Akins classifier when wet grinding is required.



B. Traylor pebble mill (4' x 6") lined with Silex, and pebble mill (4' x 8') lined with porcelain, for the grinding of non-metallies. The smaller mill for dry grinding has an 8-foot, Sturtevant air separator in circuit, and the larger mill an 8-foot, Gayco air separator. For wet grinding the smaller mill has a Dorr simplex classifier (15" x 11' 8") with 3-foot bowl attachment in circuit, and the larger mill a 7-foot bowl, Dorr classifier (2' 3" x 16' 4").



Seven-foot bowl, Dorr classifier used in the wet grinding on non-metallics to obtain a fine water-floated product. The overflow of the bowl goes to a two-tray (12' x 12'), Dorr thickener and thence to filters. In the background are the 8-foot, Sturtevant air separator on the right, and the 8-foot Gayco separator on the left.