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)

1927

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OTTAWA
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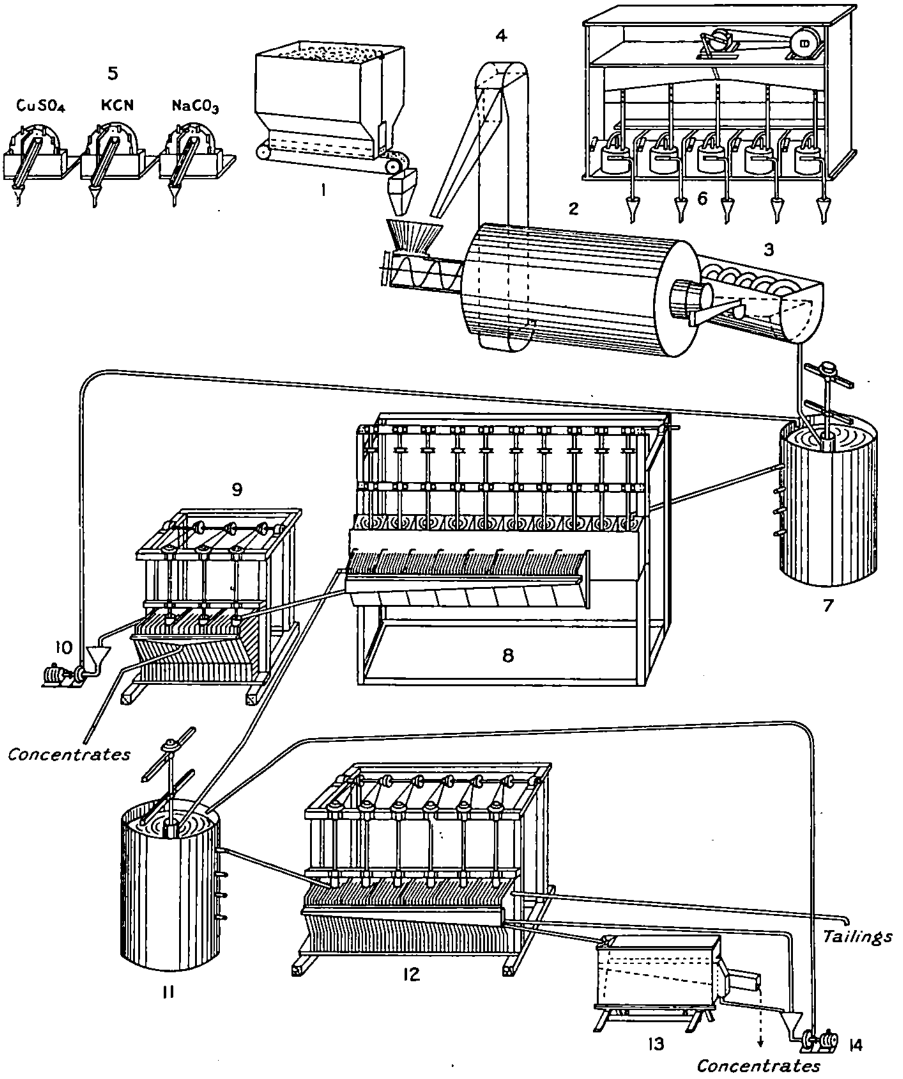
1928

No. 695

Frontispiece



1 ORE FEEDER
2 BALL OR ROD MILL
3 AKINS CLASSIFIER
4 ELEVATOR
5 REAGENT FEEDER
6 REAGENT FEEDER
7 CONDITIONING TANK
8 M.S. 8-CELL ROUGHER
9 D-SUB-A 3-CELL CLEANER
10 CENTRIFUGAL PUMP
11 CONDITIONING TANK
12 D-SUB-A 6-CELL ROUGHER
13 CALLOW CLEANER
14 CENTRIFUGAL PUMP

CONTINUOUS GRINDING AND FLOTATION UNIT:- CAPACITY-100 LBS. PER HOUR

ORE TESTING LABORATORIES, MINES BRANCH, DEPARTMENT OF MINES, OTTAWA

67672

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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.

MINES BRANCH INVESTIGATIONS IN
ORE DRESSING AND METALLURGY, 1927

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm

Chief of Division

During the year 1927 the experimental test and research work on the treatment of ores increased fully fifty per cent over that of the previous year as the result of the increased activity in the mining industry. The various laboratories of the Division were taxed to the limit to meet the demands of the mining public for such work. Increased laboratory space and better facilities are urgently required.

New equipment installed included a continuous grinding and flotation unit having a capacity of about 100 pounds per hour. In the experimental test work on the flotation of ores the unit has been found satisfactory as an intermediate step between the small laboratory machines and the large-scale, 1,000-pound per hour, unit. This intermediate unit requires shipments of two to three tons of ore, whereas carload lots are necessary for the large unit.

Co-operative arrangements were made with the Base Metal Extraction Company, Limited, and the Cassel Cyanide Company of Canada. The Base Metal Extraction Company built and equipped a laboratory for experimental test and demonstration of a hydrometallurgical process, developed in England, for the treatment of mixed (bulk) concentrates from base metal sulphide ores. The Mines Branch conducted the concentration and roasting tests on a number of ores, delivering to the company the calcines for leaching, electrolytic, and other tests for the recovery of the metals. The Cassel Cyanide Company placed in the laboratories, under Mines Branch direction and supervision, a research fellow to carry out research on the use of cyanide for the selective flotation of base metal sulphide ores, especially those containing copper and zinc minerals, to determine whether a better separation of the copper and zinc minerals could be obtained from such ores, of which large bodies have been discovered in recent years in western Quebec, northern Ontario, and northern Manitoba and Saskatchewan. A further research was to be undertaken on the use of cyanide salts for the extraction of the precious metal values in refractory gold ores.

Following up the investigations in the laboratories on the refractory gold ores of the Kirkland Lake area, flotation of the cyanide tailings was demonstrated at the mills of the Wright-Hargreaves Mines, Limited, and the Sylvanite Gold Mines, Limited. It was shown that the tailing losses could be materially reduced by the introduction of flotation, with the production of a commercial pyritic-gold concentrate.

During the year an investigation was commenced on the concentration of Canadian graphite ores for the production of crucible flake. Arrangements were made with the Morgan Crucible Co., Ltd., Battersea Works, London, England, to make standard crucibles from the Canadian flake produced. The crucibles will be tested out against similar ones made from Madagascar, Ceylon, and American flake. The object of the investigation is to determine whether Canadian flake is suitable for crucible purposes, and to obtain a comparison with other flakes used for this purpose.

Arrangements were made during the latter part of the year to conduct large-scale, continuous tests on ilmenite ore to determine the suitability of the rotary kiln type of metallizing furnace for the metallization of the iron content as a preparation of the ores for the manufacture of titanium pigment. The results of small-scale laboratory tests, given in the Report of Investigations for 1925 and 1926, indicated the commercial feasibility of a treatment process for the utilization of Canadian ilmenite ores for pigment and other purposes. The pyrometallurgical laboratory is equipped with a metallizing furnace of the rotary kiln type, having a capacity of 1,500 to 2,000 pounds daily, for the purpose of conducting direct reduction tests on iron oxide ores.

A large part of the investigatory work in the ore-dressing laboratory was on the selective flotation of base metal sulphide ores and the treatment of refractory gold ores; in the pyrometallurgical laboratory, on the roasting of bulk concentrates obtained from the flotation of base metal sulphide ores; in the hydrometallurgical laboratory, on the treatment of copper concentrates for the recovery of the iron content as electrolytic iron and the sulphur as elemental sulphur; in the non-metallic laboratory, on the dressing and preparation of such non-metallics as graphite, garnet, china clay, silica sands, sandstones, calcite, gypsum, etc.

The investigations were carried out under the direction of W. B. Timm, Chief Engineer, Division of Ore Dressing and Metallurgy.

Section II contains the results of investigations on the dressing of metallic ores conducted by C. S. Parsons, A. K. Anderson, and J. S. Godard.

Section III contains the results of the investigations on non-metallics conducted by R. K. Carnochan and R. A. Rogers.

Section IV contains the results of the investigations in hydrometallurgy conducted by R. J. Traill, W. R. McClelland, and J. D. Johnston, and a summary of the results of investigations conducted by the Base Metal Extraction Co., Ltd.

Section V contains the work and investigations in the chemical laboratory conducted by B. P. Coyne, R. A. Rogers, H. L. Beer, and L. Lutes, under the supervision of H. C. Mabee, chief chemist.

Section VI contains two papers prepared for the annual meeting of the Canadian Institute of Mining and Metallurgy held at Quebec in March, 1928; one a brief summary of the investigations on the concentration of the ores of western Quebec, and the other, on custom concentrators.

The operation and maintenance of the laboratories were performed by an operating and mechanical staff under the immediate supervision of B. M. Derry, mill foreman.

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 number, class of ore or product, source of shipment, name and address of shipper, and the weight of shipment.

Report No.	Page	Ore or product	Source of shipment	Shipper and address	Weight lb.
263	4	Gold	Sylvanite mine, Kirkland Lake, Ont.	Sylvanite Gold Mines, Ltd., Kirkland Lake, Ont.	700
264	22	Gold	Blue Quartz mine, Painkiller lake, Ont.	Blue Quartz Gold Mines Ltd., Toronto, Ont.	475
265	29	Gold-silver-lead	Planet mine, Stump lake, B.C.	Planet Mines and Reduction Co. of Nicola, Ltd., Vancouver, B.C.	720
266	30	Lead-zinc	Yankee Girl mine	Yankee Girl, Ltd., Ymir, B.C.	200
267	37	Gold	Wright-Hargreaves mine.	Wright-Hargreaves Mines, Ltd., Kirkland Lake, Ont.	60,000
268	42	Copper-zinc	Amulet mine, Rouyn, Que.	Amulet Gold Mines, Ltd., Rouyn, Que.	70,000
269	64	Gold	Francœur property, Boischatel tp., Que.	Towagmac Exploration Co., Rouyn, Que.	126
270	67	Gold	Cooper gold mine, Michipicoten, Ont.	Huronian Belt Co., Toronto, Ont.	195
271	73	Lead-zinc-gold	Premier mine, Premier, B.C.	Premier Gold Mining Co., Ltd., Premier, B.C.	190
272	83	Copper-zinc	Sherritt-Gordon mine, Man.	Sherritt-Gordon Mines, Ltd., Toronto, Ont.	1,089
273	98	Gold	Kirkland Premier mine, Kirkland Lake, Ont.	Kirkland Premier Mines, Ltd., Kirkland Lake, Ont.	220
274	102	Gold	Kirkland Lake, Ont.	Kirkland Lake Gold Mining Co., Kirkland Lake, Ont.	400
275	109	Copper-gold	Archean mine, Clericy, Que.	Alderson, MacKay, and Armstrong, Montreal, Que.	113
276	111	Lead-zinc	Stirling mine, C.B.	British Metal Corporation (Canada), Ltd., Montreal, Que.	100
277	115	Gold	Reno mine, B.C.	E. P. Crawford, Ymir, B.C.	110
278	119	Silver	Toric mine, B.C.	Toric Mines, Ltd., Vancouver	120
279	122	Lead-silver	Confederation group, Cambourne, B.C.	Lardeau Mines Exploration Co., Vancouver, B.C.	179
280	123	Calcite	Galetta, Ont	Kingdon Mining, Smelting, and Mfg. Co., Galetta, Ont.	700
281	124	Sand	Silico, Ltd	Silico, Ltd., Montreal, Que.	63
282	125	Garnet	Langlade, Que	Langlade Garnet Syndicate, Quebec.	526
283	129	Epsom salts		Purchased locally	100
284	130	Graphite	Buckingham, Que	Dominion Mine, Buckingham, Que.	10,000
285	131	Asbestos	Nicolet mine, Danville, Que.	Selective Treatment, Ltd., Montreal, Que.	6,777
286	134	Sandstone	E. Templeton, Que.	Canada Glass Products, Ltd., Ottawa, Ont.	7,956
287	136	Gypsum	Falkland, B.C.	*Manitoba Gypsum Co., Winnipeg, Man.	475
288	136	Quartz pebbles	Cypress hills, Alberta.	Mines Branch	239
289	136	Silica sand	Lake Winnipeg, Man.	Mines Branch	18,000
290	137	Diatomite	Chertsey, Que	Mines Branch	52

*Now part of the amalgamation known as Canada Gypsum Company.

II

REPORTS OF INVESTIGATIONS: ORE DRESSING AND METALLURGICAL LABORATORIES

Report No. 263

EXPERIMENTAL TESTS ON GOLD ORE FROM THE SYLVANITE GOLD MINE, KIRKLAND LAKE, ONTARIO

J. S. Godard

Shipment.—One shipment weighing 700 pounds was received November 18, 1926, from the Sylvanite Gold Mines, Limited, Kirkland Lake, Ont.

Characteristics and Analysis.—The ore is a high-grade gold ore. The gold is free and associated with small quantities of a number of sulphides the chief of which is iron pyrite, and some tellurides. The gangue is a reddish feldspar porphyry. Analysis of head sample:—

Gold	2·00 oz./ton	Molybdenite	0·14 per cent
Silver	0·14 "	Insoluble	87·40 "

Purpose of Experimental Tests.—The higher grade ores of Kirkland Lake are refractory and difficulty has been experienced in making high recoveries by cyanidation. The purpose of the experimental tests was to determine if cyanidation could be supplemented by flotation as a means of increasing the recovery of gold, and whether flotation could be more advantageously applied before cyanidation.

EXPERIMENTAL TESTS

Flotation and Tabling the Flotation Tailings

Test No.	Product	Weight, per cent	Au, oz./ton	Per cent of values
1	Flotation concentrate	5·4	27·30	77·1
	Table concentrate	9·2	1·70	8·2
	" middling	30·6	0·42	6·7
	" tailing	33·6	0·384	6·7
	" slimes	21·2	0·12	1·3
	Table tailing +100	4·7	2·36	28·9
	+150	12·4	0·56	18·1
	+200	9·7	0·44	11·1
	−200	73·2	0·22	41·9
	Average		0·384	
	Flotation tailing from products		0·46	

Flotation and Tabling the Flotation Tailings—Concluded

Test No.	Product	Weight, per cent	Au, oz./ton	Per cent of values
2	Flotation concentrate	4·5	28·36	68·7
	Table concentrate	2·4	10·10	13·0
	“ middling	8·0	1·08	4·7
	“ tailing	60·7	0·352	11·5
	“ slimes	24·4	0·16	2·1
	Table tailing +65	0·5		
	+100	8·7	0·54	13·4
	+150	18·5	0·48	25·2
	+200	16·8	0·36	17·2
	−200	55·5	0·28	44·2
	Average		0·352	
	Flotation tailing from products		0·61	
3	Flotation concentrate	4·2	42·37	79·0
	Table concentrate	6·5	3·13	8·8
	“ middling	26·8	0·50	5·8
	“ tailing	39·4	0·292	5·0
	“ slimes	23·1	0·14	1·4
	Table tailing +65	0·6	0·47	1·0
	+100	5·4	0·64	11·8
	+150	15·4	0·44	23·2
	+200	11·7	0·34	13·6
	−200	66·9	0·22	50·4
	Average		0·292	
	Flotation tailing from products		0·51	
4	Flotation concentrate	5·1	26·10	69·6
	Table concentrate	3·3	4·42	7·6
	“ middling	30·4	0·79	12·6
	“ tailing	45·6	0·376	8·9
	“ slimes	15·6	0·16	1·3
	Table tailing +65	2·1	0·70	3·9
	+100	9·7	0·64	16·6
	+150	17·4	0·64	29·7
	+200	11·8	0·48	15·1
	−200	59·0	0·22	34·6
	Average		0·376	
	Flotation tailing from products		0·61	
5	Flotation concentrate	6·9	27·80	82·1
	Table concentrate	6·1	4·49	11·7
	“ tailing	50·9	0·199	4·3
	“ slimes	36·1	0·12	1·9
	Table tailing +200	14·4	0·31	22·5
	−200	85·6	0·18	77·5
	Average		0·199	
	Flotation tailing from products		0·45	

Flotation Reagents—

Test No. 1.—Soda ash 2·0, thiocarbanilide 0·10 lb./ton to ball mill. Pine oil 0·05 lb./ton to cell.

Test No. 2.—Lime 2 lb./ton to ball mill. Xanthate 0·1, pine oil 0·05 lb./ton to cell.

Test No. 3.—Soda ash 2 lb./ton to ball mill. Xanthate 0·1, pine oil 0·05 lb./ton to cell.

Test No. 4.—Lime 2·0, thiocarbanilide 0·1 lb./ton to ball mill. Pine oil 0·05 lb./ton to cell.

Test No. 5.—Soda ash 1 lb./ton to ball mill. Xanthate 0·1, pine oil 0·05 lb./ton to cell.

Conclusions.—From 70 to 80 per cent of the gold is recovered in the flotation concentrate with a concentration ratio of 1 : 20. When soda ash, instead of lime, was used to provide the alkalinity higher recoveries were made and the froth had a bright mineralized appearance. When lime was used the froth was whitish in colour and no mineralization was apparent as the sulphides were covered with a film of lime. Tabling the flotation tailing added to the recovery. The table concentrates were comprised of the coarser sands that contained some unfreed gold carrying sulphides as well as free gold that was too coarse to be floated.

Flotation Tests

Test No.	Product	Weight, per cent	Au, oz./ton	Per cent of values
6	Concentrate	3·6	36·30	61·9
	Tailing	96·4	0·84	38·1
	Tailing +100	7·6	3·06	27·6
	−100	92·4	0·66	72·4
	Average		0·84	
7	Concentrate	6·0	27·92	71·8
	Tailing	94·0	0·70	28·2
	Tailing +100	8·9	2·52	32·1
	−100	91·1	0·52	67·9
	Average		0·70	
8	Concentrate	4·6	35·88	73·5
	Tailing	95·4	0·63	26·5
	Tailing +200	36·4	1·30	75·6
	−200	63·6	0·24	24·4
	Average		0·63	
9	Concentrate	6·9	23·40	78·4
	Tailing	93·1	0·48	21·6
	Tailing +200	43·3	0·84	76·2
	−200	56·7	0·20	23·8
	Average		0·48	
10	Concentrate	6·5	23·76	75·2
	Tailing	93·5	0·54	24·8
	Tailing +200	36·7	1·10	74·4
	−200	63·3	0·22	25·6
	Average		0·54	
11	Concentrate	2·9	45·02	63·0
	Tailing	97·1	0·79	37·0
	Tailing +150	15·8	2·54	50·9
	−150	84·2	0·46	49·1
	Average		0·79	

Reagents—

Test No. 6.—Lime 2·0, salt 2·0, xanthate 0·1 lb./ton to ball mill. Pine oil 0·05 lb./ton to cell.

Test No. 7.—Soda ash 2·0, salt 2·0 lb./ton to ball mill. Xanthate 0·1, pine oil 0·05 lb./ton to cell.

Test No. 8.—Soda ash 0·5 lb./ton to ball mill. Xanthate 0·1, pine oil 0·075 lb./ton to cell.

Test No. 9.—Xanthate 0·1 lb./ton, 5 minutes contact in ball mill. Pine oil 0·075 lb./ton to cell.

Test No. 10.—Soda ash 0·5, xylidine 0·15, xanthate 0·1 lb./ton to ball mill.

Test No. 11.—Ammonium carbonate 2·0, xylidine 0·1, xanthate 0·1 lb./ton to ball mill. Pine oil 0·05 lb./ton to cell.

Conclusions.—Recoveries varying from 62 to 78 per cent were made in the flotation concentrates which assayed from 23 to 36 ounces per ton gold. The average ratio of concentration was 1 : 18. As in the previous tests, better recoveries were made when soda ash was used instead of lime. In Test No. 9, where no alkaline reagent was used, the results were good. Salt was used in Tests Nos. 6 and 7, but its addition made no improvement either in the recoveries or in the appearance of the froth. The screen tests revealed the presence of a large amount of free gold in the coarser sizes. Where this free gold would report in actual milling conditions can only be determined by larger scale tests where the grinding would be done in a ball mill in closed circuit with a classifier.

Test No. 12

In this test the rougher concentrates from eight tests were combined and cleaned.

Product	Weight, per cent	Au, oz./ton	Per cent of values
Concentrate	1·65	88·38	74·0
Middling	3·13	4·58	7·3
Tailing	95·22	0·39	18·7

The concentrate assayed: gold 88·38 oz./ton, silver 4·70 oz./ton, copper 0·23%, lead 0·80%, zinc 0·25%, iron 28·67%, arsenic 0·42%, nickel and cobalt 0·32%, tellurium 0·47%, bismuth nil, molybdenite 0·88%, sulphur 31·55%.

Reagents.—Xanthate 0·1, pine oil 0·05 lb./ton to cell. No alkaline reagent was used.

Conclusions.—Seventy-four per cent of the gold was recovered in a high-grade concentrate with a ratio of concentration of 1 : 60. The high iron content of the concentrate shows that iron pyrite is the predominating sulphide.

Test No. 13

Thirty small-scale flotation tests were made, and the concentrates combined and treated by amalgamation, roasting, and cyanidation.

Flotation Results—

Product	Weight, per cent	Au, oz./ton	Per cent of values
Concentrate	7·1	25·44	78·4
Tailing	92·9	0·532	21·6
Tailing +200	28·6	1·41	75·8
−200	71·4	0·18	24·2
Average		0·532	

Amalgamation.—A sample of 1,200 grammes was cut from the combined concentrates and amalgamated to remove the free gold.

Results—

Head 25·44 oz./ton. Tailing 16·60 oz./ton. Recovery 34·7 per cent.

Roasting.—The concentrates, after amalgamation, were roasted for 3½ hours at a temperature of 700°C maximum.

Results—

Weight before roasting........................	800·0 grammes.	
" after "	755·4 "	
Loss during roasting........................	44·6 "	= 5·6 per cent

Assay of roasted concentrate—gold, 16·64 oz./ton; sulphur, 2·12 per cent.
Gold loss, mechanical and roasting, 5·48 per cent. This included some small spills that occurred while rabbling.

Cyanidation of the roasted concentrates.—The roasted concentrates were ground in a 1 : 1 pulp, 0·05 per cent KCN, and lime equivalent to 5 pounds per ton. The pulp was washed into a small agitator and agitated for 90 hours in 0·10 per cent KCN. The pulp density during agitation was 1 : 3·5.

Results—

Assay		Extraction, per cent	Reagent consumption in lb./ton			
			Concentrate		Ore	
Head	Tailing		KCN	CaO	KCN	CaO
16·64	0·12	99·3	3·46	22·2	0·25	1·7

Conclusions.—Flotation was satisfactory although the +200 mesh tailing was high due to some free gold.

By amalgamation 34·7 per cent of the gold was recovered. The amalgamation was done to minimize the gold loss during roasting and to remove the free gold which has a tendency to settle in the small agitators and give erratic results.

No attempt was made to secure a dead roast. The roasted product contained 2·12 per cent sulphur. The temperature during roasting was about 600° C, but was raised to 700° C at the finish to break up the sulphates. Some gold loss occurred during the roasting but it was due more to small spills while rabbling than to volatilization or being carried off with the sulphur fumes.

Cyanidation of the roasted concentrates was satisfactory as a good extraction was made with small cyanide consumption.

Test No. 14

Amalgamation, Flotation, and Tabling the Flotation Tailing (Four parts—A, B, C, and D)

Part	Product	Weight, per cent	Au, oz./ton	Per cent of values
A	*Table tailing screened on 200 mesh—*			
	+200	21·8	0·28	
	−200	78·2	0·21	
	Average		0·225	
	Concentration—			
	Flotation concentrate	4·8	21·18	81·9
	Table concentrate	5·1	1·08	4·4
	Table tailing	51·4	0·225	9·3
	Slimes	38·7	0·14	4·4
	Amalgamation tailing from products		1·242	
B	*Table tailing screened on 200 mesh—*			
	+200	9·9	0·24	
	−200	90·1	0·16	
	Average		0·168	
	Concentration—			
	Flotation concentrate	7·4	13·73	
	Table concentrate	3·9	1·01	
	Table tailing	54·9	0·168	
	Slimes	33·8	0·11	
	Amalgamation tailing from products		1·195	
C	*Table tailing screened on 200 mesh—*			
	+200	33·4	0·33	
	−200	66·6	0·20	
	Average		0·243	
	Concentration—			
	Flotation concentrate	5·9	15·54	79·7
	Table concentrate	3·7	1·16	3·7
	Table tailing	62·6	0·243	13·2
	Slimes	27·8	0·14	3·4
	Amalgamation tailing from products		1·151	
D	*Table tailing screened on 200 mesh—*			
	+200	5·6	0·18	
	−200	94·4	0·14	
	Average		0·142	
	Concentration—			
	Flotation concentrate	7·1	12·80	86·3
	Table concentrate	9·2	0·54	4·7
	Table tailing	44·9	0·142	6·1
	Slimes	38·8	0·08	2·9
	Amalgamation tailing from products		1·053	

Summary.—Distribution of gold in per cent, based on 2·00 ounces per ton head.

Part	Amalgamation	Flotation concentrate	Table concentrate	Table tailing	Slimes	Recovery, per cent
A	37·9	50·9	2·7	5·8	2·7	91·5
B	40·8	50·8	2·0	4·6	1·8	93·6
C	42·5	45·7	2·2	7·6	2·0	90·4
D	47·4	45·3	2·5	3·2	1·6	95·2

Conclusions.—Satisfactory results were obtained on this grade of ore with the above treatment.

By amalgamation, 38 to 47 per cent of the gold is recovered. The amount of gold recovered increased on finer grinding.

Flotation was satisfactory. Gold that in former tests reported in the concentrate was amalgamated. A concentrate assaying about one ounce in gold and weighing about 5 per cent of the ore was obtained by tabling the flotation tailing.

Test No. 15

Flotation and Cyanidation of Flotation Tailing

Product	Weight, per cent	Au, oz./ton	Per cent of values
Concentrate	5·9	26·20	70·3
Tailing	94·1	0·696	29·7
Tailing +200	32·8	1·67	78·8
−200	67·2	0·33	21·2
Average		0·696	
Cyanide tailing screened on 200 mesh—			
+200	9·5	0·84	26·9
−200	90·5	0·24	73·1
Average		0·297	

The flotation tailing was reground before cyanidation.

Time of agitation	48 hours
KCN consumption	0·29 lb./ton

The −200 cyanide tailing was re-cyanided for 48 hours using 1 : 2·5 pulp and 0·075 per cent KCN.

Results—	
Head assay	0·24 oz./ton
Tailing assay	0·06 "
Extraction	75 per cent
Recovery: Flotation and cyanidation (1st only)	87·3 "

Test No. 16

Flotation and Cyanidation of Reground Tailing

Product	Weight, per cent	Au, oz./ton	Per cent of values
Concentrate	6·2	25·24	84·6
Tailing	93·8	0·304	15·4
Head from products		1·85	
Tailing +200	26·0	1·70	51·3
−200	74·0	0·20	48·7
Average		0·304	
Cyanide tailing screened on 200 mesh—			
+200	1·3	0·18	2·3
−200	98·7	0·10	97·7
Average		0·101	

Recovery—Flotation and cyanidation	94·8 per cent
Cyanide consumption	0·20 lb./ton

Test No. 17

Flotation and Cyanidation of the Flotation Tailing
(Six parts—A to F)

Part	Product	Weight, per cent	Au, oz./ton	Per cent of values	Head from products, Au, oz/ton
	Flotation—				
A	Concentrate	6·2	27·73	84·0	2·05
	Tailing	93·8	0·35	16·0	
B	Concentrate	5·4	26·52	68·7	2·08
	Tailing	94·6	0·69	31·3	
C	Concentrate	4·6	38·54	87·3	2·03
	Tailing	95·4	0·27	12·7	
D	Concentrate	6·8	23·68	83·2	1·94
	Tailing	93·2	0·35	16·8	
E	Concentrate	6·2	25·70	76·9	2·07
	Tailing	93·8	0·51	23·1	
F	Concentrate	6·7	25·28	71·6	2·37
	Tailing	93·3	0·72	28·4	

Part	Product	Weight, per cent	Au, oz./ton	Per cent of values	Average tailing, Au. oz./ton
	Flotation tailings screened on 200 mesh—				
A	+200	24·6	0·84	59·0	0·35
	−200	75·4	0·19	41·0	
B	+200	22·6	2·07	67·6	0·69
	−200	77·4	0·29	32·4	
C	+200	11·1	1·07	44·0	0·27
	−200	88·9	0·17	56·0	
D	+200	10·7	1·89	57·5	0·35
	−200	89·3	0·17	42·8	
E	+200	17·8	1·90	66·2	0·51
	−200	82·2	0·21	33·8	
F	+200	20·7	2·64	75·9	0·72
	−200	79·3	0·22	24·1	
	Cyanide tailings screened on 200 mesh—				
A	+200	34·2	0·23	44·3	0·177 amalgamated before assaying
	−200	65·8	0·15	55·7	
B	+200	34·9	0·64	75·8	0·295
	−200	65·1	0·11	24·2	
C	+200	21·0	0·20	27·5	0·153 amalgamated before assaying
	−200	79·0	0·14	72·5	
D	+200	14·4	0·90	71·6	0·181
	−200	85·6	0·06	28·4	
E	+200	29·0	0·88	76·6	0·333
	−200	71·0	0·11	23·4	

Test No. 17—Concluded

F. The flotation tailing from test F was divided into three equal parts and each part was cyanided in Winchesters in place of agitators. Time 48 hrs., KCN 0·075 per cent, pulp density 1 : 2·5.

Part	Product	Weight, per cent	Au, oz./ton	Per cent of values	Average tailing, Au, oz./ton
(1)	+200	17·2	0·18	34·1	0·088
	−200	82·8	0·07	65·9	
(2)	+200	18·5	0·20	43·1	0·086
	−200	81·5	0·06	56·9	
(3)	+200	34·2	0·17	55·8	0·104
	−200	65·8	0·07	44·2	

Summary—

Part	Head	Tailings		Recoveries, per cent		
		Flotation	Cyanidation	Flotation	Cyanidation	Total
A	2·05	0·35	0·177	84·0	7·3	91·3
B	2·08	0·69	0·295	68·7	17·3	85·8
C	2·03	0·27	0·153	87·3	5·2	92·5
D	1·94	0·35	0·181	83·2	7·6	90·8
E	2·07	0·51	0·333	76·9	3·1	84·0
F	2·37	0·72	0·093	71·6	24·5	96·1

NOTE.—In A and C, the cyanide tailings were amalgamated before assaying. This was done for the purpose of removing the free gold. In F the cyanide tailing, 0·093 oz./ton, is the average of parts 1, 2, and 3.

Test No. 18

Cyanidation of the Flotation Tailing

Three lots, 1,000 grammes each, of flotation tailing from Test No. 13 assaying 0·532 ounce per ton, were ground and cyanided. Pulp density, 1 : 2·5; KCN, 0·075 per cent. Time of agitation, 48 hours. The cyanide tailings were screened on 200 mesh.

Results—

Part	Mesh	Weight, per cent	Au, oz./ton	Per cent of values	Average tailing	Extraction, per cent
A	+200	0·03	95·84	32·3	0·089	83·4
	−200	99·97	0·06	67·7		
B	+200	0·03	9·47	5·8	0·064	88·0
	−200	99·97	0·06	94·2		
C	+200	0·01	12·50	1·8	0·071	86·6
	−200	99·99	0·07	98·2		

Summary.—Tests Nos. 13 and 18.

Part	Head	Tailing, Au oz./ton		Recovery, per cent		
		Flot.	Cyan.	Flot.	Cyan.	Total
A	2·29	0·532	0·089	78·4	18·0	96·4
B	2·29	0·532	0·064	78·4	19·0	97·4
C	2·29	0·532	0·071	78·4	18·8	97·2

Test No. 19

Amalgamation and Cyanidation of Tailings from Test No. 13

The test consisted of four parts, each of 1,000 grammes of the flotation tailings from Test No. 13. Head, 0·532 ounce per ton.

Part A—Amalgamation—cyanidation—no regrinding.
" B—Ground 20 minutes, amalgamated, cyanided.
" C— " 40 " " "
" D— " 60 " " "

Agitation 48 hours, pulp density, 1 : 2; KCN, 0·10 per cent. The cyanide tailings were screened on 200 mesh.

Part	Mesh	Weight, per cent	Au, oz./ton	Per cent of values	Average tailing	Extraction per cent
A	+200	33·1	0·15	59·7	0·083	84·5
	−200	66·9	0·05	40·3		
B	+200	7·8	0·15	20·2	0·058	89·1
	−200	92·2	0·05	79·8		
C	+200	3·5	0·15	9·9	0·054	90·1
	−200	96·5	0·05	90·1		
D	+200	1·3	0·32	7·8	0·054	90·1
	−200	98·7	0·05	92·2		

Summary.—(Tests Nos. 13 and 19)—

Part	Head	Tailing		Recovery, per cent		
		Flot.	Cyan.	Flot.	Cyan.	Total
A	2·29	0·532	0·083	78·4	17·7	96·1
B	2·29	0·532	0·058	78·4	18·8	97·2
C	2·29	0·532	0·054	78·4	19·0	97·4
D	2·29	0·532	0·054	78·4	19·0	97·4

Conclusions.—Satisfactory results were obtained in these tests using a combination of flotation and cyanidation of the flotation tailings when amalgamation preceded the cyanidation. This step was found necessary as the free gold was too coarse to be attacked by the cyanide, and reported in the +200 mesh tailing, as in Test No. 18. In practice, this step would not be necessary as the regrinding of the flotation tailing would be done in a closed circuit and the gold returned until sufficiently fine to be dissolved by the cyanide.

Test No. 20

Cyanidation and Flotation of the Cyanide Tailing

Cyanidation: KCN, 0·075 per cent. Pulp density, 1 : 2·5. Agitation, 40 hours.

Part	Assay		Extraction, per cent	Reagents consumed, lb./ton	
	Head	Tailing		KCN	CaO
A	2·00	0·73	63·5	0·28	3·4
B	2·00	0·69	65·5	0·28	3·4

Flotation of cyanide tailing—

Part	Product	Weight, per cent	Au, oz./ton	Per cent of values
A	Concentrate	5·6	9·88	76·2
	Tailing	94·4	0·183	23·8
B	Concentrate	5·7	10·04	82·7
	Tailing	94·3	0·127	17·3

Screening flotation tailing on 200 mesh—

Part	Mesh	Weight, per cent	Au, oz./ton	Per cent of values	Average tailing
A	+200	1·7	5·02	46·5	0·183
	−200	98·3	0·10	53·5	
B	+200	1·3	3·69	37·8	0·127
	−200	98·7	0·08	62·2	

Reagents—

A.—Soda ash 1·0, xanthate 0·15, xylidine 0·12 lb./ton to ball mill, 10 minutes contact. Pine oil 0·04 lb./ton to cell.

B.—Sodium silicate 1·0, soda ash 1·0, xanthate 0·15, xylidine 0·08 lb./ton to ball mill, 10 minutes contact. Pine oil 0·08 lb./ton to cell.

Summary—

Part	Head	Tailing		Recovery, per cent		
		Cyan.	Flot.	Cyan.	Flot.	Total
A	2·00	0·73	0·183	63·5	26·7	90·2
B	2·00	0·69	0·127	65·5	28·5	94·0

Test No. 21

Cyanidation and Flotation of Cyanide Tailing

Cyanidation: KCN, 0·075 per cent. Pulp density, 1 : 2·5. Agitation, 48 hours.

Part	Head	Tailing	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
A	2·00	0·47	71·5	0·21	2·0
B	2·00	0·46	72·0	0·21	2·0

Flotation of cyanide tailing.

Part	Product	Weight. per cent	Au, oz./ton	Per cent of values
A	Concentrate	5·3	7·64	85·6
	Tailing	94·7	0·072	14·4
	Tailing +200	1·2	0·23	3·9
	−200	98·8	0·07	96·1
	Average		0·072	
B	Concentrate	5·7	7·00	87·2
	Tailing	94·3	0·062	12·8
	Tailing +200	0·8	0·25	3·2
	−200	99·2	0·06	96·7
	Average		0·062	

Reagents—

A.—Soda ash 3·0, coal tar (40 per cent) and coal-tar creosote (60 per cent) 0·1, copper sulphate 1·0 lb./ton to ball mill, 15 minutes contact. Pine oil 0·05 lb./ton to cell.

B.—Soda ash 2·0, xylidine 0·08, kerosene 0·1, coal tar (40 per cent) and coal-tar creosote (60 per cent) 0·05 lb./ton to ball mill, 15 minutes contact. Xanthate 0·2, pine oil 0·06 lb./ton to cell.

Summary—

Part	Head	Tailing		Recovery, per cent		
		Cyan.	Flot.	Cyan.	Flot.	Total
A	2·00	0·47	0·072	71·5	24·4	95·9
B	2·00	0·46	0·062	72·0	24·4	96·4

Test No. 22

Cyanidation and Flotation of Cyanide Tailing

Cyanidation: KCN, 0·075 per cent. Pulp density, 1 : 2·5. Time of agitation, 48 hours.

Part	Head	Tailing	Extraction, per cent	KCN, lb./ton
A	2·00	0·546	72·7	0·33
B	2·00	0·513	74·4	

Flotation of cyanide tailing.

Part	Product	Weight, per cent	Au, oz./ton	Per cent of values
A	Concentrate	7·2	6·80	89·7
	Tailing	92·8	0·06	10·3
B	Concentrate	8·2	5·80	92·8
	Tailing	91·8	0·04	7·2

Reagents (A and B).—

Soda ash 2·0, kerosene 0·10, coal-tar mixture 0·10, xylidine 0·08 lb. /ton to ball mill, 15 minutes contact. Xanthate 0·1, pine oil 0·08 lb. /ton to cell.

Summary—

Part	Head	Tailing		Recoveries, per cent		
		Cyan.	Flot.	Cyan.	Flot.	Total
A	2·00	0·546	0·06	72·7	23·5	96·2
B	2·00	0·513	0·04	74·4	23·7	98·1

Test No. 23

Cyanidation and Flotation of Cyanide Tailing

In the cyanidation test 160 pounds of ore were crushed dry to —48 mesh and fed to a small rod mill in closed circuit with a classifier. The grinding was done in 1 : 1 pulp, 0·05 per cent KCN. The classifier overflow was emptied into a small Pachuca tank and agitated 24 hours with 0·034 per cent KCN. The cyanide strength was then increased to 0·075 per cent KCN and agitation continued for another 24 hours. The pulp density during agitation was about 1 : 2·5. The tailing was filtered in a frame press and after washing was repulped and filtered. Tailings were screened on 200 mesh.

Mesh	Weight, per cent	Au, oz./ton	Per cent of values	Average tailing
+200	6·0	0·22	6·0	0·22
−200	94·0	0·22	94·0	

A number of flotation tests were made on the cyanide tailing.

Flotation Series A—

Part	Product	Weight, per cent	Au, oz./ton	Per cent of values
A-1	Concentrate	8·2	2·08	73·7
	Tailing	91·8	0·066	26·3
	Tailing +200	16·4	0·15	37·0
	−200	83·6	0·05	63·0
A-2	Concentrate	15·2	1·27	77·7
	Tailing	84·8	0·065	22·3
	Tailing +200	11·8	0·18	32·4
	−200	88·2	0·05	67·6
A-3	Concentrate	8·3	2·49	79·3
	Tailing	91·7	0·059	20·7
	Tailing +200	13·5	0·18	41·3
	−200	86·5	0·04	58·7
A-4	Concentrate	8·6	2·72	80·9
	Tailing	91·4	0·06	19·1
	Tailing +200	14·5	0·18	43·3
	−200	85·5	0·04	56·7

Reagents—

A-1.—Soda ash 2·0 lb./ton, agitated 15 minutes, added Flotagen S 0·1, pine oil 0·05 lb./ton.

A-2.—Soda ash 4·0 lb./ton, agitated 15 minutes, added Flotagen S 0·1, pine oil 0·05 lb./ton.

A-3.—Soda ash 2·0 lb./ton, agitated 15 minutes, copper sulphate 1·0 lb./ton, agitated 5 minutes, Flotagen S 0·1, pine oil 0·05 lb./ton.

A-4.—Soda ash 4·0 lb./ton, agitated 15 minutes, copper sulphate 1·0 lb./ton, agitated 5 minutes, Flotagen S 0·1, pine oil 0·05 lb./ton.

Flotation Series B—

Part	Product	Weight, per cent	Au, oz./ton	Per cent of values
B-1	Concentrate	6·8	3·38	79·6
	Tailing	93·2	0·063	20·4
	Tailing +200	14·2	0·20	45·3
	−200	85·8	0·04	54·7
B-2	Concentrate	8·2	2·28	59·0
	Tailing	91·8	0·142	41·0
	Tailing +200	9·1	0·16	10·3
	−200	90·9	0·14	89·7
B-3	Concentrate	7·8	3·22	81·4
	Tailing	92·2	0·062	18·6
	Tailing +200	9·9	0·17	27·2
	−200	90·1	0·05	72·8

Reagents:

In Series B, Flotagen replaced Flotagen S in all tests. Other reagents and conditions the same as in Series A.

Flotation Series C—

Part	Product	Weight, per cent	Au, oz./ton	Per cent of values
C-1	Concentrate	5·4	2·96	83·7
	Tailing	94·6	0·033	16·3
	Tailing +200	8·4	0·12	30·6
	−200	91·6	0·025	69·4
C-2	Concentrate	7·8	2·28	87·0
	Tailing	92·2	0·029	13·0
	Tailing +200	10·1	0·11	38·1
	−200	89·9	0·02	61·9
C-3	Concentrate	5·9	2·87	79·0
	Tailing	94·1	0·048	21·0
	Tailing +200	4·9	0·21	21·3
	−200	95·1	0·04	78·7
C-4	Concentrate	8·2	1·79	82·9
	Tailing	91·8	0·033	17·1
	Tailing +200	3·9	0·22	26·4
	−200	96·1	0·025	73·6

Reagents—

In Series C, xanthate replaced the Flotagen S in all tests. Other reagents and conditions the same as in corresponding tests in Series A.

Flotation Series D—

Part	Product	Weight, per cent	Au, oz./ton	Per cent of values
D-1	Concentrate	8·4	1·93	86·3
	Tailing	91·6	0·039	13·7
	Tailing +200	8·5	0·14	30·2
	−200	91·5	0·03	69·8
D-2	Concentrate	9·5	2·06	81·6
	Tailing	90·5	0·049	18·4
	Tailing +200	9·7	0·13	25·8
	−200	90·3	0·04	74·2
D-3	Concentrate	5·0	4·19	80·6
	Tailing	95·0	0·053	19·4
	Tailing +200	12·5	0·14	33·3
	−200	87·5	0·04	66·7
D-4	Concentrate	6·4	2·63	75·9
	Tailing	93·6	0·057	24·1
	Tailing +200	11·4	0·19	38·0
	−200	88·6	0·04	62·0
D-5	Concentrate	6·1	2·85	64·9
	Tailing	93·9	0·10	35·1
	Tailing +200	12·3	0·22	27·8
	−200	87·7	0·08	72·2

Reagents—

D-1.—Soda ash 1·0, NaOH 1·0 lb./ton, agitated 15 minutes, Flotagen 0·1, pine oil 0·05 lb./ton to cell.
D-2.—Soda ash 1·0, NaOH 1·0 lb./ton, agitated 15 minutes, copper sulphate 1 lb./ton, agitated 5 minutes, Flotagen 0·1, pine oil 0·05 lb./ton to cell.
D-3.—Soda ash 2 lb./ton, agitated 5 minutes, coal-tar mixture 0·05, kerosene 0·05 xylidine 0·05 lb./ton, agitated 10 minutes, pine oil, 0·05 lb./ton to cell.
D-4.—Soda ash 2 lb./ton, agitated 5 minutes, coal-tar mixture 0·05, kerosene 0·05, xylidine 0·05 lb./ton, copper sulphate 1 lb./ton, agitated 10 minutes, pine oil 0·05 lb./ton to cell.
D-5.—Sulphuric acid 3·75, copper sulphate 1 lb./ton, agitated 5 minutes, xanthate 0·1, pine oil 0·05 lb./ton to cell.

Flotation Series E—

Part	Product	Weight, per cent	Au, oz./ton	Per cent of values
E-1	Concentrate	14·0	1·34	79·9
	Tailing	86·0	0·055	20·1
	Tailing +200	4·2	0·16	12·3
	−200	95·8	0·05	87·7
E-2	Concentrate	8·0	2·35	82·0
	Tailing	92·0	0·045	18·0
	Tailing +200	3·4	0·18	13·6
	−200	96·6	0·04	86·4
E-3	Concentrate	7·1	2·68	76·5
	Tailing	92·9	0·063	23·5
	Tailing +200	2·9	0·17	7·7
	−200	97·1	0·06	92·3
E-4	Concentrate	10·3	1·98	85·0
	Tailing	89·7	0·04	15·0
	Tailing +200	3·1	0·18	14·2
	−200	96·9	0·035	85·8
E-5	Concentrate	10·1	2·06	80·8
	Tailing	89·9	0·055	19·2
	Tailing +200	3·5	0·18	11·5
	−200	96·5	0·05	88·5
E-6	Concentrate	11·3	1·59	78·6
	Tailing	88·7	0·055	21·4
	Tailing +200	3·6	0·18	80·8
	−200	96·4	0·05	19·2
E-7	Concentrate	10·3	1·98	77·7
	Tailing	89·7	0·065	22·3
	Tailing +200	3·5	0·20	10·8
	−200	96·5	0·06	89·2
*E-8	Concentrate	3·5	5·20	82·4
	Middling	9·2	0·04	1·8
	Tailing	87·3	0·04	15·8
	Tailing +200	7·1	0·16	29·0
	−200	92·9	0·03	71·0

*In this test eight rougher concentrates were made and these were combined, dewatered, reground, and cleaned.

Reagents—

E-1.—Soda ash 2·0 lb./ton to ball mill, 15 minutes contact. Copper sulphate 1·0, Flotagen S. 0·1, pine oil 0·05 lb./ton to cell.
E-2.—Soda ash 2·0 lb./ton to ball mill, 15 minutes contact. Flotagen 0·1, pine oil 0·05 lb./ton to cell.
E-3.—Soda ash 2·0 lb./ton to ball mill, 15 minutes contact. Copper sulphate 1·0, Flotagen 0·1, pine oil 0·05 lb./ton to cell.
E-4.—Soda ash 2·0 lb./ton to ball mill, 15 minutes contact. Copper sulphate 1·0, xanthate 0·1, pine oil 0·05 lb./ton to cell.
E-5.—Soda ash 2·0, coal-tar mixture 0·05, kerosene 0·05, xylidine 0·05 lb./ton to ball mill, 15 minutes contact. Pine oil 0·05 lb./ton to cell.
E-6.—Soda ash 2·0, coal-tar mixture 0·05, kerosene 0·05, xylidine 0·05 lb./ton to ball mill, 15 minutes contact. Copper sulphate 1·0, pine oil 0·05 lb./ton to cell.
E-7.—Soda ash 2·0, thiocarbanilide 0·1 lb./ton to ball mill, 15 minutes contact. Copper sulphate 1·0, pine oil 0·05 lb./ton to cell.
E-8—For rougher concentrate—Soda ash 2·0 lb./ton, agitated 10 minutes: Sodium sulphide 0·4 lb./ton, agitated 5 minutes, xanthate 0·1 lb./ton, pine oil 0·05 lb./ton to cell.
For cleaning—Soda ash 1·0 lb./ton to ball mill, 15 minutes contact. Xanthate 0·2, copper sulphate 1·0, pine oil 0·075 lb./ton to cell.

Summary—

Head	Tailing		Recoveries, per cent		Total
	Cyan.	Flot. (Av.)	Cyan.	Flot. (Av.)	
2·00	0·22	0·057	89·1	8·6	97·7

Conclusions.—No difficulty was experienced in the flotation of the cyanide tailings. Good recoveries were made in the concentrates which had a well mineralized appearance. The grade of flotation concentrates is shown in Test No. 23, E-8, where the concentrate assayed over 5 ounces per ton in gold with a recovery of 82 per cent, and a concentration ratio of 1 : 28.

In Test No. 20 the +200 mesh flotation tailing was high in free gold. In Tests Nos. 21 and 22 this was overcome by introducing amalgamation on the cyanide tailing before flotation. The cyanidation recoveries were then based on the amalgamation tailing. In Tests Nos. 20–21–22, the cyanidation tailings were reground to brighten up the sulphides and obtain a thorough mixing of the flotation reagents which were of the insoluble type. In Test No. 23, where 160 pounds of the ore were used and the grinding done in a small rod mill in closed circuit with the classifier, no trouble from free gold was experienced. Soluble flotation reagents were used and no regrinding was done previous to flotation. The amount of soda ash used was from 2 to 4 pounds per ton, and a few soluble flotation reagents were tried. Two pounds per ton of soda ash were found to be sufficient and 0·1 pound per ton xanthate with 1 pound per ton copper sulphate and a little pine oil gave the best results and the most satisfactory froth.

GENERAL CONCLUSIONS

On an ore of this type flotation may be practised in conjunction with cyanidation. Flotation increases the recovery considerably, as the more refractory part of the ore is obtained in the form of a flotation concentrate which may be sold to a smelter or otherwise treated. Whether flotation be practised previous to cyanidation or on the cyanidation tailing will depend on local conditions as the recoveries are about the same in both cases. If flotation be practised previous to cyanidation a high-grade flotation concentrate will be obtained, which may be roasted and cyanided with a good recovery and a low-cyanide consumption, or it may be sold to a smelter, whichever practice is the more economical. By following this procedure the greater part of the gold will be in the form of flotation concentrates, and the feed to the cyanide plant will be much lower in grade with a smaller output in bullion from the main cyanide section of the mill. As the flotation will be conducted in about 1 : 2·5 pulp density, the tailing will require thickening previous to entering the cyanide agitators. This will involve a certain loss of cyanide because of the excess of cyanide solution which must be run to waste. For satisfactory flotation results the ore must be ground as fine as for cyanidation and the advantage of grinding in cyanide solution, where the greatest dissolution of the gold takes place, will be lost.

Should cyanidation precede flotation, the chief return will be in the form of bullion. This procedure will allow of grinding being done in cyanide solution and the filter cake may be diluted to the required density for flotation without any additional equipment. This flotation concentrate has proved very refractory to cyanidation. Attempts have been made to cyanide it after regrinding, but the extractions were low though strong cyanide solutions and bromo-cyanide were used, and in some tests various washes were tried previous to cyanidation. It may prove amenable to cyanidation after roasting, and a number of tests are planned along these lines.

To obtain consistent satisfactory recoveries in the concentrate after cyanidation certain conditions are at present thought to be essential: The flotation feed should be free from cyanide and soluble lime, as both of these have a depressing influence on the iron pyrite with which the gold in the cyanide tailing is mainly associated, and which constitutes by far the greater part of the concentrate. If cyanide be present in appreciable amounts the iron pyrite will not float and the presence of lime may be detected by the whitish colour and limy nature of the froth. These conditions may be avoided by good filtration with water washes only, applied on the back half of the second filter, thereby ensuring that the flotation feed is almost free of both these chemicals. The addition of soda ash is necessary as it has a scouring action on the sulphide particles and brightens them previous to flotation as well as acting as a precipitant for any small quantities of soluble lime that may be present, but some time should be allowed to elapse before the addition of the other reagents. Good results have been obtained by adding this reagent in the dry form, the more conveniently to the discharge of the second filter where the pulp should be diluted to the proper density for flotation. After the addition of the soda ash the pulp should be pumped into an agitation tank of sufficient capacity to allow about fifteen minutes contact, and the other reagents added here. This ensures a thorough mixing of the pulp and reagents before entering the cells.

The point where flotation may be economically practised on the cyanide tailing can be determined only by pilot tests and will depend largely on the costs of the subsequent treatment of the concentrates.

Report No. 264

EXPERIMENTAL TESTS ON ORE FROM THE BLUE QUARTZ GOLD MINE, PAINKILLER LAKE, ONT.

J. S. Godard

Shipment.—One shipment of ore, weight 475 pounds, was received December 15, 1926, from the Blue Quartz Gold Mines, Ltd., Toronto, from their mine at Painkiller lake, near Matheson, Ont.

Characteristics of the Ore.—The valuable constituent of this ore is gold which occurs free and associated with the sulphides. The sulphides present are white pyrites which occur in veins, arsenopyrite, and chalcopyrite. The gangue is mainly siliceous containing small stringers of calcite.

Analysis.—The head sample contained—

Gold	0·38 oz./ton	Copper	0·08 per cent
Iron	15·29 per cent	Arsenic	0·97 "
Insoluble	56·54 "	Lime	7·43 "

Test No. 1

Amalgamation at 100 mesh

Screen test on amalgamation tailing

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+ 65	0·8	0·52	1·5
+100	8·9	0·22	7·1
+150	10·6	0·20	7·7
+200	9·2	0·22	7·3
−200	70·5	0·30	76·4
Average		0·28	
Head sample		0·38	

Recovery, per cent 26·3

Test No. 2

Amalgamation at 15 per cent +200 mesh

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+200	15·1	0·24	16·2
−200	84·9	0·22	83·8
Average		0·22	
Head sample		0·38	

Recovery, per cent 42·1

Test No. 3

Flotation and Tabling

Product	Weight, per cent	Au, oz./ton	Per cent of values	Remarks
Flotation concentrate	26·6	1·28	80·4	Ore ground 10 per cent + 100 mesh for flotation.
Table concentrate	4·4	0·42	4·4	
Table tailing	44·8	0·10	10·6	
Slimes	24·2	0·08	4·6	
Head from products		0·42		

Test No. 4

Flotation and Tabling

Product	Weight, per cent	Au, oz./ton	Per cent of values	Remarks
Flotation concentrate	33·6	0·94	84·6	Ore ground 10 per cent +200 mesh for flotation. Flotation concentrate, Cu 0·26 per cent, As 2·18 per cent. Table concentrate, Cu 0·04 per cent, As 0·42 per cent.
Table concentrate	2·0	0·39	2·1	
" middling	7·8	0·14	2·9	
" tailing	23·8	0·08	5·1	
" slimes	32·8	0·06	5·3	
Head from products		0·37		

Test No. 5

Flotation and Tabling

Product	Weight, per cent	Au, oz./ton	Per cent of values	Remarks
Flotation concentrate	24·3	1·38	79·1	Ore ground 10 per cent +150 mesh. Flotation concentrate, Cu 0·31 per cent, As 2·85 per cent Table concentrate, Cu 0·04 per cent, As 0·54 per cent
Table concentrate	4·3	0·40	4·1	
" tailing	35·6	0·10	8·4	
" slimes	35·8	0·10	8·4	
Head from products		0·42		

Tests Nos. 6, 7, and 8

Flotation at 10 per cent +200 mesh

Test	Product	Weight, per cent	Au, oz./ton	Per cent of values
6	Concentrate	23·8	1·55	82·9
	Tailing	76·2	0·10	17·1
	Head from products		0·45	
7	Concentrate	25·0	1·37	85·1
	Tailing	75·0	0·08	14·9
	Head from products		0·40	
8	Concentrate	25·2	1·69	90·4
	Tailing	74·8	0·06	9·6
	Head from products		0·47	

Test No. 9

Screening −50 +150, tabling the +150 material, regrinding table middling and tailing before flotation, tabling the flotation tailing. Flotation of the −150 mesh material and tabling flotation tailing.

Product	Weight, per cent	Au, oz./ton	Per cent of values
−50+150 mesh—			
Table concentrate	19·5	0·96	47·9
Flotation concentrate	9·9	1·55	39·3
Table concentrate from flotation tailing	1·9	0·43	2·1
Table tailing	48·1	0·07	8·6
Table slimes	20·6	0·04	2·1
−150 mesh—			
Flotation concentrate	29·7	1·34	87·8
Table concentrate	2·0	0·58	2·6
" tailing	37·1	0·09	7·4
" slimes	31·2	0·03	2·2
SUMMARY			
Concentrates—			
Table −50+150	30·9	0·96	24·7
Flotation −50+150	15·7	1·55	20·2
Table from flotation tailing	3·1	0·43	1·1
Flotation −150	47·1	1·34	52·5
Table −150	3·2	0·58	1·5
Average all concentrates		1·203	
Tailings—			
Table − 50+150	56·4	0·07	50·2
−150	43·6	0·09	49·8
Average all tailings		0·079	
Slimes—			
Slimes − 50+150	39·7	0·04	46·8
−150	60·3	0·03	53·2
Average all slimes		0·034	
General—			
Concentrates	31·5	1·203	90·0
Tailings	42·5	0·079	8·0
Slimes	26·0	0·034	2·0
Head from products		0·4215	

Test No. 10

Tabling and Flotation

A sample of 2,100 grammes of ore at −20 mesh was screened on 100 mesh. The − 20 + 100 product was tabled, the middling and tailing reground and floated, and the flotation tailing tabled. The −100 mesh product from the screen was floated and the tailing tabled.

Product	Weight, per cent	Au, oz./ton	Per cent of values
Table concentrate +100	17·5	0·64	31·7
Flotation concentrate	9·2	1·86	48·3
Table concentrate from flotation tailing	4·4	0·46	5·7
Table tailing	46·4	0·09	11·8
Slimes	22·5	0·04	2·5
Flotation concentrate −100	26·5	1·47	83·3
Table concentrate	5·4	0·44	5·1
Table tailing	34·6	0·118	8·7
Slimes	33·5	0·04	2·9

Test No. 10—Concluded

Product	Weight, per cent	Au, oz./ton	Per cent of values
SUMMARY			
Concentrates—			
Table +100	37·3	0·64	22·0
Flotation +100	19·7	1·86	33·8
Table from flotation	0·3	0·46	4·0
Flotation −100	28·0	1·47	37·9
Table −100	5·7	0·44	2·3
Average all concentrates		1·085	
Tailings—			
+100	73·1	0·09	67·5
−100	26·9	0·118	32·5
Average tailing		0·098	
Slimes—			
From +100	57·6	0·04	57·6
−100	42·4	0·04	42·4
Average slimes		0·04	
General—			
Concentrates	31·4	1·085	86·8
Tailings	42·5	0·09	10·6
Slimes	26·1	0·04	2·6
Head from products		0·393	

Test No. 11

Cyanidation

Five small-scale cyanidation tests were made in the following manner: For each test 750 grammes of ore at −20 mesh were used. Each lot was ground in a 1 : 1 pulp, 0·05 per cent KCN. The pulp was agitated for 48 hours in 1 : 2·5 pulp density. Cyanide strength was maintained at 0·075 per cent KCN by additions of cyanide twice daily.

Test	+ 200 mesh, per cent	Head, Au, oz./ton	Tailing, Au, oz./ton			Extraction, per cent	Reagents, lb./ton	
			+200	−200	Average		KCN	CaO
11A	15·5	0·43	0·11	0·09	0·093	78·3	1·70	5·15
11B	4·5	0·43	0·10	0·07	0·071	83·5	5·01	6·56
11C	1·4	0·43	0·10	0·06	0·061	85·7	2·09	6·16
11D	0·6	0·43	0·10	0·07	0·072	83·3	2·42	6·35
11E	0·3	0·43	1·87	0·04	0·046	89·2	2·41	7·04

Test No. 12

Cyanidation

Two small-scale cyanidation tests were made in the following manner: 900 grammes of the ore at −20 mesh were ground in a 1: 1 pulp in a pebble jar. The pulp was dewatered and cyanided in two Winchester bottles for 48 hours in 1 : 2·5 pulp, 0·075 per cent KCN. The solution from the dewatering was used for cyanidation.

Test	+200 mesh, per cent	Head Au, oz./ton	Tailing, Au, oz./ton			Extraction, per cent	Reagents, lb./ton	
			+200	−200	Average		KCN	CaO
12A	11·1	0·43	0·05	0·08	0·079	81·7	1·65	4·25
12B	11·4	0·43	0·11	0·07	0·072	83·3	1·65	4·25

Test No. 13

Amalgamation, Flotation, and Cyanidation of Flotation Concentrate

A batch of 132 pounds of ore at –48 mesh was ground in 1 : 1·25 pulp in a small rod mill in closed circuit with a classifier. The classifier overflow passed through an amalgamator and over a 12-foot amalgamation plate. Some of the amalgamation tailing was floated in batch lots in a small Ruth machine. Rougher concentrates were made first, then two rougher concentrates were combined and cleaned. Three cleaner concentrates were made. These were sampled and reground in 1 : 1 pulp, 0.05 per cent KCN, then agitated for 48 hours in a 1·3 pulp, 0·05 per cent KCN. Head sample for test 0·43 ounce per ton.

(*A*) *Sampling of Amalgamation Plate Tailings*

Sample	Mesh	Weight, per cent	Au, oz./ton	Per cent of values
No. 1	+200	2·7	0·17	2·9
	–200	97·3	0·16	97·1
	Average		0·16	
No. 2	+200	3·7	0·19	2·9
	–200	96·3	0·24	97·1
	Average		0·238	

(*B*) *Flotation of Amalgamation Tailing*

Product	Weight, per cent	Au, oz./ton	Per cent of values
Concentrate	21·8	1·07	83·7
Middling	10·9	0·145	5·8
Tailing	67·3	0·043	10·5
Amalgamation tailing from products		0·274	

A grab sample of 1,020 grammes of amalgamation tailing was floated in a small Ruth machine.

Product	Weight, per cent	Au, oz./ton	Per cent of values
Concentrate	25·4	0·76	86·2
Tailing	74·6	0·041	13·8
Tailing—			
+200	2·7	0·09	5·8
–200	97·3	0·04	94·2
Average		0·041	
Amalgamation tailing from products		0·22	

Head sample, 0·43 oz./ton; average amalgamation tailing, 0·213 oz./ton. Per cent of gold amalgamated, 50·4.

(*C*) *Cyanidation of Flotation Concentrate*

No.	Head, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagents, lb./ton		Reagents, lb./ton of original ore	
				KCN	CaO	KCN	CaO
1.......	1·08	0·22	79·5	5·0	13·1	1·09	2·86
2.......	1·04	0·22	78·8	5·82	15·1	1·27	3·29
3.......	1·04	0·19	81·7	6·44	16·6	1·40	3·62

Remarks—

No. 1—Concentrates ground 15 minutes before agitation.
" 2— " " 30 " "
" 3— " " 60 " "

Test No. 14

Amalgamation and Cyanidation

A sample of 135 pounds of ore at −48 mesh was ground and amalgamated as in Test No. 13. Lime equivalent to one pound per ton was added during grinding. The amalgamation tailing was dewatered to about 60 per cent solids and cyanided in a Pachuca tank for 48 hours. The strength was maintained at 0·05 per cent KCN. The pulp dilution was about 1 : 3.

Sampling of Amalgamation Tailing—

Sample	Mesh	Weight, per cent	Au. oz./ton	Per cent of values
No. 1..	+200	3·8	0·27	5·1
	−200	96·2	0·20	94·9
	Average....		0·20	
No. 2..	+200	3·4	0·24	3·6
	−200	96·6	0·23	96·4
	Average....		0·23	
No. 3..	+200	3·8	0·15	2·5
	−200	96·2	0·23	97·5
	Average....		0·227	
Average tailing, 3 samples......................			0·22	
Head sample....................................			0·43	

Recovery, per cent.. 48·8

Sampling Cyanide tailing—

Sample	Mesh	Weight, per cent	Au. oz./ton	Per cent of values
No. 1..	+200	1·5	0·07	1·8
	−200	98·5	0·06	98·2
	Average....		0·06	
No. 2..	+200	5·2	0·07	5·95
	−200	94·8	0·06	94·05
	Average....		0·0605	
Head sample....................................			0·43	

Recovery, per cent.. 86

Test No. 15

Cyanidation

A batch of 113·5 pounds of ore at −35 mesh was ground in a small rod mill in closed circuit with a classifier. The grinding was done in 1 : 1 pulp 0·024 per cent KCN. Lime equivalent to 3 pounds per ton of ore was fed to the rod mill. The ground ore was emptied into a Pachuca tank and agitated 24 hours in 1 : 3 pulp maintaining the cyanide strength at 0·02 per cent KCN. Cyanide strength was then raised to 0·046 per cent and agitation continued for 24 hours. Tailings were filtered, washed, and repulped before sampling.

Sample	Mesh	Weight, per cent	Au, oz./ton	Per cent of values
No. 1	+200	6·8	0·09	8·6
	−200	93·2	0·07	91·4
	Average		0·0713	
No. 2	+200	9·3	0·09	11·7
	−200	90·7	0·07	88·3
	Average		0·0719	
No. 3	+200	7·6	0·08	8·6
	−200	92·4	0·07	91·4
	Average		0·0708	
Average tailing, 3 samples			0·0713	
Head sample			0·42	

Recovery, per cent .... 83·1

CONCLUSIONS

Twenty-six per cent of the gold was recovered by amalgamation at 10 per cent +100 mesh, and 42 per cent at 15 per cent +200 mesh. The amount of gold recovered by amalgamation increased with finer grinding.

Thirty per cent of the gold was the maximum recovery by concentration. This was obtained in Test No. 8 by flotation only and in Test No. 9 by tabling and flotation. The ratio of concentration was low, 1 : 4, and the grade of concentrate unsatisfactory, 1·69 ounce per ton being the best. As in amalgamation the recovery increases with finer grinding. The tailings obtained by flotation compared favourably with those by cyanidation, but the cyanidation of the concentrate from flotation more than offset this. The concentrate has no constituent other than gold that would make a desirable smelter product, the copper content being too low and the presence of arsenic would probably incur a penalty.

The cyanidation test showed that about 85 to 90 per cent of the gold may be recovered by this method provided the grinding is sufficiently fine. About 2 pounds KCN and 6 pounds lime per ton of ore would be required. Where amalgamation preceded cyanidation a small increase in recovery is noted, but in large-scale operations this would probably be negligible.

Report No. 265

EXPERIMENTAL WORK ON A GOLD-SILVER-LEAD ORE FROM THE PLANET MINES, NICOLA, B.C.

C. S. Parsons

Shipment.—A shipment of 4 sacks consisting of 46 samples, having a gross weight of 720 pounds, was received December 28, 1926, from the Planet Mines and Reduction Co. of Nicola, Ltd., Vancouver, B.C. The ore was from the company's mine at Stump lake, Merritt, B.C.

Analysis.—A composite sample was made up from selected samples on which to conduct the experimental work. Analysis of composite sample:

Copper................	0·28 per cent	Gold..................	0·22 oz./ton
Lead..................	1·89 "	Silver.................	6·50 "
Zinc...................	1·16 "		

Purpose of Experimental Tests.—The purpose of the tests was to determine an economic method of concentrating the ore.

EXPERIMENTAL TESTS

After a careful examination of the ore flotation was decided on as being the most practical method of concentration, owing to the fine dissemination of the mineral values in the ore. The reagents used were:

Acid coal-tar creosote..	1·25 lb./ton
Xanthate..	0·15 "
Steam-distilled pine oil..	To froth.

Results of Test

Product	Weight, per cent	Assay					Per cent of values			
		Pb, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Fe, per cent	Pb	Zn	Au	Ag
Concentrate.......	15·0	12·07	7·08	1·38	40·62	14·44	95·7	78·6	92·4	95·2
Tailing...........	85·0	0·10	0·34	0·02	0·36		4·3	21·4	7·6	4·8

SUMMARY AND CONCLUSIONS

It will be observed that only one test was made on this ore. Owing to the good results obtained it was not thought necessary to run any more. The ore is a remarkably clean one, free from any interfering gangue matter or soluble salts, and if the sample truly represents the ore-body no difficulty should be experienced in concentration by flotation.

Report No. 266

EXPERIMENTAL WORK ON THE LEAD-ZINC ORE FROM THE YANKEE GIRL MINE, YMIR, BRITISH COLUMBIA

C. S. Parsons and J. S. Godard

Shipment.—A shipment of 200 pounds of ore was received March 15, from Yankee Girl, Limited, Ymir, B.C. This sample was from the Yankee Girl mine at Ymir, B.C.

Characteristics and Analysis.—The sample received was obtained by taking the discards from the mechanical sampling of the shipments of ore made to the Trail smelter during the month of February. The sample represents about 1,600 tons of ore. The ore shipped during February contained a lower gold content than the average for the past year's shipments. Analysis:

Lead	3·12 per cent	Gold	0·35 oz./ton
Zinc	4·60 "	Silver	2·48 "

Purpose of Tests: The following tests were requested by the shipper:

1. Cyanidation followed by flotation of tailing and recovery of the lead and zinc, or the lead only.
2. Flotation, making a lead concentrate and a zinc concentrate, followed by cyanidation of tailing to recover remaining gold and silver, if any.
3. Cyanidation only, recovering the gold and silver and wasting the lead and zinc.
4. Flotation only, recovering the lead and zinc in concentrates for shipment to smelter.

EXPERIMENTAL TESTS

The results of the experimental work will be grouped under the above headings and considered in the order given.

Cyanidation and Flotation of Cyanide Tailing

Three tests were made, 1,000 grammes of ore being taken for each test. In the first two the ore was taken at −14 mesh and ground 40 minutes in a ball mill with cyanide solution containing 0·025 per cent KCN and 3 pounds lime per ton. The grinding was approximately 75 per cent −200 mesh. The pulp from the ball mill was diluted to 1:2 and agitated for 48 hours. The cyanide solutions were brought up to the following strengths for agitation:

Test No.	Cyanide	Lime
1	0·05 per cent	0·01 per cent
2	0·075 "	0·01 "
3	0·25 "	

The solutions were kept up to strength by the addition of cyanide and lime from time to time as required.

Reagent Consumption—

Test No.	Cyanide	Lime
1	1·04 lb./ton	4·16 lb./ton
2	1·24 "	3·54 "
3	3·20 "	

In Test No. 3 the ore was ground dry to 75 per cent −200 mesh and cyanided direct. The purpose of the stronger solution was to increase, if possible, the recovery of the silver.

Results of Cyanidation—

Test	Assay, head		Assay, tailing		Recoveries, per cent	
	Au, oz./ton	Ag, oz./ton	Au, oz./ton	Ag, oz./ton	Au, oz./ton	Ag, oz./ton
1	0·355	2·43	0·06	1·64	83·0	32·5
2	0·355	2·43	0·06	1·52	83·0	37·4
3	0·355	2·43	0·055	1·33	84·5	45·2

The cyanide tailings were filter-pressed and washed before flotation.

Test No. 1—Reagents:

Soda ash	3 lb./ton, 4 minutes contact
Aerofloat	3 drops

It was observed that the zinc immediately floated so that selective flotation in this test was abandoned and a bulk concentrate floated with the addition of:

Copper sulphate	1·5 lb./ton
Xanthate	0·3 "

Test No. 2—Reagents:

Soda ash	3·0 lb./ton, 15 minutes contact
Cyanide	0·10 " 15 " "
Zinc sulphate	2·0 " 15 " "

Lead flotation with Aerofloat.

Zinc Flotation with—

Copper sulphate	1·5 lb./ton
Xanthate	0·2 "
Pine oil	

Test No. 3—Reagents:

Soda ash	3·0 lb./ton, 20 minutes contact
Cyanide	0·3 " 20 " "
Zinc sulphate	2·0 " 20 " "

Acid coal-tar creosote was used to float the lead. Zinc came up with the lead in large amount and no selective action between the lead and zinc was obtained. A bulk concentrate had to be made by the addition of:—

Copper sulphate	1·5 lb./ton
Xanthate	0·3 "

The iron sulphides did not show much tendency to float.

Results of Flotation of Cyanide Tailings—

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	Concentrate	12·3	17·20	31·47	0·32	8·88	70·9	86·9	63·6	66·8
	Middling	6·6	6·34	5·18	0·22	3·96	13·9	7·6	23·3	15·9
	Tailing	81·1	0·56	0·30	0·01	0·35	15·2	5·5	13·1	17·3
2	Lead concentrate	12·5	20·82	11·37	0·32	9·68	86·5	30·9	70·0	79·5
	Zinc concentrate	6·3	1·58	37·74	0·07	1·53	3·3	51·6	7·7	6·3
	Zinc middling	3·8	1·69	3·69	0·03	0·81	1·7	3·0	2·0	2·0
	Tailing	77·4	0·33	0·86	0·015	0·24	8·5	14·5	20·3	12·2
3	Concentrate	10·4	15·53	33·39	0·32	6·88	52·0	78·0	59·9	43·8
	Middling	5·4	8·00	8·82	0·10	3·42	14·1	10·7	9·8	32·2
	Tailing	82·4	1·25	0·60	0·02	0·51	33·9	11·3	30·3	14·0

Summary.—For discussion of cyaniding refer to heading "Straight Cyanidation." The reason for repulping in fresh water and the addition of soda ash should be carefully noted. This was to eliminate the lime and cyanide mixture which, if not removed, would interfere and prevent flotation by causing a dirty pasty froth to form in the cells. The soda ash precipitates the lime as carbonate and also cleans up the surface of the sulphides and deflocculates the gangue slimes. It will be observed in Test No. 1 that zinc immediately floated, showing more active flotation properties than the lead. It was found necessary as shown in Test No. 2 to give the cyanide tailing, after washing and repulping, a time contact with cyanide and zinc sulphate. A fair selective flotation was then obtained, but in Test No. 3, where a stronger solution of cyanide had been used during cyanidation, the selective flotation property of the lead was entirely killed and the zinc showed much more active flotation properties. The addition of copper sulphate in this case acted on the galena as well as on the zinc blende increasing the recovery of both minerals.

These erratic and contrary results give a warning that should be considered, namely, that the flotation of lead and zinc from cyanide tailing can be expected to give varying results and success will depend largely on the uniform operation of the cyanide plant. The controlling factors apparently are the strength of cyanide solution used, amount of lime, and the washing and repulping of the cyanide tailing before flotation. In other words, it can be done, but will require careful control.

Selective Flotation and Cyanidation of Flotation Tailing

For each test 1,000 grammes of ore at −14 mesh were ground for 40 minutes in a ball mill.

Test No. 1—Reagents—

Soda ash	6·0 lb./ton	Ground with ore.
Cyanide	0·2 "	"
Coal-tar creosote	0·32 "	"
Zinc sulphate	2·0 "	10 minutes contact for lead flotation.
Pine oil	1 drop	For lead flotation.
Lime	5·0 lb./ton	10 minutes contact for zinc flotation.
Copper sulphate	1·5 "	5 minutes contact
Xanthate	0·1 "	1 " "
Pine oil	1 drop	

Test No. 2—Reagents—

Soda ash	6·0 lb./ton	Ground with ore.
Coal-tar creosote	0·32 "	Not ground with ore.
Cyanide	0·2 "	Zinc cyanide, 30 minutes contact.
Zinc sulphate	2·0 "	
Pine oil	1 drop	To float lead.
Lime	5·0 lb./ton	15 minutes contact for zinc flotation.
Copper sulphate	1·5 "	3 minutes contact.
Xanthate	0·1 "	1 " "
Pine oil	1 drop	

For iron flotation—dewatered tailing and added 10 lb./ton H_2SO_4, coal-tar creosote, and pine-tar oil.

Flotation Results—

Test No.	Product	Wt., per cent	Analysis					Per cent of values			
			Pb, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Fe, per cent	Pb	Zn	Au	Ag
1	Lead concentrate	8·0	34·61	10·15	2·94	21·60	......	86·6	17·8	69·4	75·0
	Zinc concentrate	6·4	2·57	51·60	0·50	3·18	......	5·2	72·6	9·5	8·8
	Zinc middling	2·5	2·87	7·47	0·86	3·58	......	2·2	4·1	6·3	3·9
	Tailing	83·1	0·23	0·3	0·06	0·34	......	6·0	5·5	14·8	12·3
2	Lead concentrate	8·6	32·49	13·17	3·32	22·06	12·57	90·7	24·6	82·4	81·6
	Zinc concentrate	6·0	1·43	51·89	0·40	2·46	8·85	2·8	68·5	7·0	6·4
	Zinc middling	2·3	1·62	6·42	0·40	2·68	13·51	1·2	3·3	2·7	2·7
	Iron concentrate	6·4	0·74	0·77	0·19	0·97	33·12	1·5	1·1	3·5	2·7
	Tailing	76·7	0·15	0·15	0·02	0·20	9·13	3·8	2·5	4·4	6·6

The tailings from these two tests were cyanided separately under the following conditions:

Pulp dilution 1 : 2. Solution strength 0·05 per cent KCN. Cyanide consumption: Test No. 1, 1·3 lb./ton; Test No. 2, 1·2 lb./ton.

Results—

Test No.	Assay, head		Assay, tailing		Recovery, per cent	
	Au, oz./ton	Ag, oz./ton	Au, oz./ton	Ag, oz./ton	Au	Ag
1	0·06	0·34	0·016	0·133	73	59
2	0·02	0·20	0·011	0·056	50	72

Screen Analysis of Cyanide Tailing—

Test	Mesh	Weight, per cent	Au, oz./ton	Ag, oz./ton
1	+200	13·0	0·06	0·22
	−200	87·0	0·01	0·12
2	+200	9·3	0·025	0·115
	−200	90·7	0·01	0·05

Discussion of Results.—By cyaniding the flotation tailing from Test No. 1 the gold is reduced from $1.20 to 32 cents with an extraction of 59 per cent of the silver and 73 per cent of the gold. In Test No. 2 an iron product was floated leaving a flotation tailing of 40 cents. The iron product contained 0·19 ounce gold per ton or $3.80 and represented 6·4 per cent of the ore. It is apparent that the better practice would be to float out an iron product and cyanide it instead of cyaniding the whole of the flotation tailing. The fact that cyanide gives a good extraction on the tailing from Test No. 1, which contains the sulphides, leaves no doubt that the iron concentrate obtained in Test No. 2 can be successfully cyanided.

Straight Cyanidation

Four tests were made each on 1,000 grammes of crude ore, at 14 mesh, ground 40 minutes in ball mill in cyanide solution of strength approximately 0·025 per cent KCN (0·20 grammes NaCN) and CaO 3 pounds per ton. The pulp from the ball mill was run into agitators and diluted to 1 : 2. The solutions were brought up to the following strengths in the agitator:—

Test No.	Cyanide	Lime
1	0·05 per cent	0·01 per cent
2	0·05 "	0·01 "
3	0·075 "	0·009 "
4	0·075 "	0·01 "

Time of agitation, 48 hours. Strength of solutions kept up by addition of cyanide and lime from time to time.

Reagent Consumption—

Test No.	Cyanide	Lime
1	1·04 lb./ton	4·16 lb./ton
2	0·88 "	3·52 "
3	1·24 "	3·54 "
4	1·24 "	3·80 "

Results—

Test No.	Assay, head		Assay, tailing		Recovery, per cent	
	Au, oz./ton	Ag, oz./ton	Au, oz./ton	Ag, oz./ton	Au	Ag
1	0·355	2·43	0·06	1·64	83	32·5
2	0·355	2·43	0·07	1·57	80	35·4
3	0·355	2·43	0·06	1·52	83	37·4
4	0·355	2·43	0·06	1·50	83	38·25

Discussion of Results.—The ore cyanided readily but with only fair recoveries. The consumption of reagents for small-scale tests is not excessive and there seems to be no tendency for the solution to foul. It will be observed that in Test No. 4 no advantage is gained by using a stronger cyanide solution and that the consumption of cyanide is increased by so doing.

Selective Flotation

Test No. 1—

1,000 grammes ore at −14 mesh, ground 30 minutes with:—

Soda ash	6·0	lb./ton	
Cyanide	0·2	"	15 minutes contact only

Lead floated with Aerofloat.

Zinc floated with:—

Copper sulphate	2·0	lb./ton
Xanthate	0·2	"
Pine oil	1	drop

Test No. 2—

1,000 grammes ore at −14 mesh, 25 minutes in ball mill with:—

Soda ash	6·0	lb./ton	
Thiocarbanilide	0·1	"	15 minutes contact only

Lead floated with zinc cyanide (0·24 lb. cyanide and 2·0 lb. zinc sulphate) added 6 drops coal-tar creosote (acid) to float.

Zinc floated with:—

Copper sulphate	2·0	lb./ton,	3 minutes contact
Xanthate	0·3	"	
Pine oil	1	drop	

NOTE.—Iron tended to float at end of test.

Test No. 3—

1,000 grammes ore at −14 mesh, 25 minutes in ball mill with:—

Soda ash	6·0	lb./ton
Sodium cyanide	0·5	"

Lead floated with:—

Zinc sulphate	0·6	lb./ton.	Added to cells.
Xanthate	0·2	"	"

Pine oil to froth

Zinc floated with:—

Copper sulphate	0·5	lb./ton
Xanthate	0·2	"

Pine oil to froth.

NOTE.—Zinc sulphate was added to form zinc xanthate to increase selection between lead and zinc.

Test No. 4—

1,000 grammes ore at −14 mesh, 40 minutes in ball mill with:—

Soda ash	6·0	lb./ton
Cyanide	0·2	"
Acid coal-tar creosote	0·32	"

Lead floated with:—

Zinc sulphate	2·0	lb./ton	Added to cells.
Pine oil	1	drop	"

Zinc floated with:—

Lime	5·0	lb./ton,	10 minutes contact before flotation
Copper sulphate	1·5	"	5 " " "
Xanthate	0·1	"	1 " " "
Pine oil	1	drop	

Test No. 5—

1,000 grammes ore at −14 mesh, 40 minutes in ball mill with :—

Soda ash	6·0 lb./ton	Given time contact but not ground with ore
Acid coal-tar creosote	0·32 "	
Cyanide	0·2 "	Emulsion zinc cyanide, 30 minutes time contact
Zinc sulphate	2·0 "	
Pine oil	1 drop	

Zinc floated with:—

Lime	5·0 lb./ton,	15 minutes contact before flotation
Copper sulphate	1·5 "	3 " " "
Xanthate	0·1 "	1 " " "
Pine oil	1 drop	

Iron flotation: Dewatered tailing and added 10 lb./ton H_2SO_4, coal-tar creosote, and pine-tar oil.

Results—

Test No.	Product	Weight, per cent	Analysis					Per cent of values			
			Pb, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Fe, per cent	Pb	Zn	Au	Ag
1	Lead concentrate	10·8	25·55	13·57	2·42	14·80	......	87·5	30·2	75·8	76·6
	Zinc concentrate	10·8	2·34	29·28	0·48	2·76	......	8·0	65·0	15·1	14·4
	Tailing	78·4	0·18	0·30	0·04	0·24	......	4·5	4·8	9·1	9·0
2	Lead concentrate	10·5	27·15	14·27	2·60	17·06	......	89·4	32·0	84·5	81·2
	Zinc concentrate	7·6	1·56	35·75	0·30	2·14	......	4·0	58·3	7·1	7·4
	Zinc middling	3·9	1·40	4·63	0·40	1·84	......	1·7	3·9	4·8	3·3
	Tailing	77·9	0·20	0·36	0·015	0·23	......	4·9	5·8	3·6	8·1
3	Lead concentrate	8·5	29·07	11·93	2·54	19·40	......	84·7	22·4	73·5	78·4
	Zinc concentrate	4·9	2·44	40·62	0·50	2·80	......	4·9	52·9	10·1	7·9
	Zinc middling	2·8	1·65	12·40	0·30	2·10	......	3·4	7·8	2·9	2·9
	Iron concentrate	10·8	0·88	4·98	0·23	1·41	23·8	3·2	11·7	8·5	7·3
	Tailing	72·0	0·18	0·33	0·02	0·10	......	4·8	5·2	4·9	3·4
4	Lead concentrate	8·0	34·61	10·15	2·94	21·70	......	86·6	17·8	69·4	75·0
	Zinc concentrate	6·4	2·57	51·60	0·50	3·18	......	5·2	72·6	9·5	8·8
	Zinc middling	2·5	2·87	7·47	0·86	3·58	......	2·2	4·1	6·3	3·9
	Tailing	83·1	0·23	0·3	0·06	0·34	......	6·0	5·5	14·8	12·3
5	Lead concentrate	8·6	32·49	13·17	3·32	22·06	12·57	90·7	24·6	82·4	81·6
	Zinc concentrate	6·0	1·43	51·89	0·40	2·46	8·85	2·8	68·5	7·0	6·4
	Zinc middling	2·3	1·62	6·42	0·40	2·68	13·51	1·2	3·3	2·7	2·7
	Iron concentrate	6·4	0·74	0·77	0·19	0·97	33·12	1·5	1·1	3·5	2·7
	Tailing	76·7	0·15	0·15	0·02	0·20	9·13	3·8	2·5	4·4	6·6

Discussion of Results.—A gradual improvement is shown from test to test in the selective separation between the lead, zinc, and iron sulphides. The principal difficulty in the separation was in keeping the iron pyrite from floating with the zinc. It was also necessary to float part of the pyrite with the lead in order to recover the gold and silver.

Tests Nos. 4 and 5 give methods of obtaining this separation. In Test No. 4 the reagent cyanide was ground with the ore and the zinc sulphate added later and given a time contact before flotation of the lead. In Test No. 5 the cyanide and zinc sulphate were added as an emulsion and given a time contact in a conditioning tank before the flotation of the lead. The latter method apparently allows more gold and silver to be recovered in the lead concentrate. It will be observed that lime is used in the zinc flotation and that the pulp is given a time contact with the lime in a conditioning tank before the flotation of the zinc. Fresh unslaked lime was used, being added to a large volume of water and ground before adding to conditioning tank.

Report No. 267

FLOTATION OF CYANIDE TAILINGS FROM WRIGHT-HARGREAVES ORE

C. S. Parsons and J. S. Godard

Shipment.—One shipment consisting of about 30 tons wet weight was received March 30, 1927, from the Wright-Hargreaves Mines, Ltd., Kirkland Lake, Ont., consisting of tailings discharged from the Oliver filters in the company's cyanide mill.

Characteristics and Assays.—The tailings assayed about 0·075 ounce per ton in gold, largely associated with a very fine pyrite and small amounts of the sulphides of other base metals. Screen analysis showed that the tailings had been ground to approximately 80 per cent −200 mesh.

Purpose of Experimental Tests.—The purpose of these tests was to determine if the gold remaining in the cyanide tailings could be concentrated by flotation.

SMALL-SCALE TESTS

Three small-scale laboratory tests were made on the tailings.

Results—

Test No.	Product	Weight, per cent	Au, oz./ton	Per cent of values
1	Concentrate	10·1	0·44	63·8
	Tailing	89·9	0·028	36·2
	Tailing +200	16·9	0·07	41·5
	−200	83·1	0·02	58·5
	Average		0·028	
2	Concentrate	4·2	1·00	54·9
	Tailing	95·8	0·036	45·1
	Tailing +200	28·8	0·05	40·2
	−200	71·2	0·03	59·8
	Average		0·036	
3	Concentrate	2·8	1·60	60·6
	Tailing	97·2	0·03	39·4
	Tailing +200	27·8	0·06	53·7
	−200	72·2	0·02	46·3
	Average		0·03	

Reagents—

Test 1.—Soda ash 2·0, thiocarbanilide 0·1 lb./ton to ball mill, 15 minutes contact. Copper sulphate 1·0, xanthate 0·2, pine oil 0·08 lb./ton to cell.

Test 2.—Soda ash 2·0 lb./ton agitated 10 minutes. Copper sulphate 1 lb./ton agitated 5 minutes, xanthate 0·2, pine oil 0·08 lb./ton to cell.

Test 3.—Soda ash 1·0, sodium sulphate 1·0 lb./ton agitated 10 minutes. Xanthate 0·2, pine oil 0·1, xylidine 0·02 lb./ton to cell.

LARGE-SCALE TESTS

Test No. 4

The cyanide tailings were fed to a 4½-foot x 13-inch Hardinge ball mill containing 300 pounds of balls. The ball mill discharge was elevated to a small contact tank, which discharged into an 8-cell Greenawalt flotation machine. Concentrates were taken from the first 3 cells and the middling returned to the ball mill.

Results—

Head	0·075 oz./ton	Recovery	68·25 per cent
Tailing (average)	0·0255 "	Ratio of concentration	1 : 15·2
Concentrate (average)	0·78 "		

Screen Test on Flotation Tailing—

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+150	5·6	0·07	14·4
+200	21·7	0·04	32·0
−200	72·7	0·02	53·6
Average		0·0255	

Reagents—

Soda ash 1·7 lb./ton to ball mill. Copper sulphate 0·53, coal-tar creosote 0·17 lb./ton to contact tank. Xanthate 0·17, pine oil 0·08 lb./ton to head of cells.

Test No. 5

The cyanide tailings were fed to the Hardinge mill, then elevated to a launder leading to a 3-cell, flat-bottom Callow flotation machine. The middling was returned to the launder leading to the rougher cell.

Results—

Head	0·075 oz./ton	Recovery	48·01 per cent
Concentrate (average)	1·36 "	Ratio of concentration	1 : 37·7
Tailing (average)	0·04 "		

Reagents—

Soda ash 1·93 lb./ton to ball mill. Copper sulphate 0·7, coal-tar creosote 0·45, xanthate 0·18 lb./ton to ball mill discharge. Pine oil 0·05 lb./ton to head of rougher cell.

Test No. 6

The flow-sheet was the same as in Test No. 5.

Results—

Head	0·075 oz./ton	Recovery	48·4 per cent
Concentrate (average)	1·59 "	Ratio of concentration	1 : 61·5
Tailing (average)	0·05 "		

Reagents—

Soda ash 1·7 lb./ton to ball mill. Sodium sulphide 0·49, coal-tar creosote 0·44, xanthate 0·19 lb./ton to ball mill discharge. Pine oil 0·045 lb./ton to head of rougher cell.

Test No. 7

The flow-sheet was the same as in Test No. 5, except that the flotation middling was returned to the ball mill and the flotation tailing tabled on a large Wilfley table, slime deck.

Results—

Head	0·075 oz./ton	Recovery	50·6 per cent
Concentrate (average)	1·54 "	Ratio of concentration	1 : 40·6
Tailing (average)	0·038 "		

Screen Test on Table Concentrate from Flotation Tailing—

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+100	28·3	0·18	43·2
+150	18·3	0·12	18·7
+200	22·3	0·09	17·0
−200	31·1	0·08	21·1
Average		0·12	

Flotation Tailing Screened on 200 Mesh—

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+200	20·2	0·07	37·1
−200	79·8	0·03	62·9
Average		0·038	

Table Tailing Screened on 200 Mesh—

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+200	18·2	0·10	47·2
−200	81·8	0·025	52·8
Average		0·0386	

Reagents—

Soda ash 2·05 lb./ton to ball mill. Copper sulphate 0·84, coal-tar creosote 0·08, xanthate 0·14 lb./ton to ball mill discharge. Pine oil 0·044 lb./ton to head of cells.

Test No. 8

The flow-sheet was the same as in Test No. 4. Hardinge mill charged with 2,300 pounds of balls and flotation tailing tabled on a large Wilfley table, slime deck.

Results—

Head	0·075 oz./ton	Recovery	65·2 per cent
Concentrate (average)	1·26 "	Ratio of concentration	1 : 25·9
Tailing (average)	0·0274 "		

Screen Test on Table Concentrate from Flotation Tailing—

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+100	12·3	0·20	29·6
+150	17·8	0·10	21·5
+200	27·5	0·07	23·3
−200	42·4	0·05	25·6
Average		0·083	

Flotation Tailing Screened on 200 Mesh—

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+200	8·7	0·08	16·6
−200	91·3	0·025	83·4
Average		0·0274	

Table Tailing Screened on 200 Mesh—

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+200	6·68	0·08	22·1
−200	93·32	0·02	77·9
Average		0·0274	

Reagents—

Soda ash 2·4 lb./ton to ball mill. Copper sulphate 0·86, coal-tar creosote 0·08, xanthate 0·14 lb./ton to ball mill discharge. Pine oil 0·09 lb./ton to head of cells.

Test No. 9

The flow-sheet was the same as in Test No. 7. Hardinge mill charged with 2,300 pounds of balls.

Results—

Head	0·075 oz./ton	Recovery	61·6 per cent
Concentrate (average)	1·19 "	Ratio of concentration	1 : 25·8
Tailing (average)	0·030 "		

Screen Test on Table Concentrate from Flotation Tailing—

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+100	11·3	0·17	26·0
+150	18·2	0·10	24·6
+200	20·9	0·08	22·6
−200	49·6	0·04	26·8
Average		0·074	

Flotation Tailing Screened on 200 Mesh—

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+200	9·8	0·08	25·6
−200	90·2	0·025	74·4
Average		0·03	

Reagents—

Soda ash 2·7 lb./ton to ball mill. Copper sulphate 0·88, coal-tar creosote 0·075, xanthate 0·2 lb./ton to ball mill discharge. Pine oil 0·09 lb./ton to head of cells.

Flotation Tests made in the Mill of the Wright-Hargreaves Mines, Ltd., at Kirkland Lake

Flow-sheet.—The tailings from the Oliver filters were discharged into a Dorr agitator until sufficient had been obtained for a test of from four to six hours' duration. The pulp from the agitator was pumped into a launder leading to a MacIntosh cell of one ton per hour capacity.

Results—

Date	Head	Concentrate	Tailing	Recovery, per cent	Ratio of concentration
July 13	0·096	1·075	0·032	68·8	1 : 16·3
" 16	0·070	2·190	0·029	59·4	1 : 52·7
" 19	0·062	3·407	0·031	50·6	1 : 109
" 20	0·057	1·695	0·0235	59·6	1 : 49·8

SUMMARY OF FLOTATION RESULTS

1. The results obtained show conclusively that the gold remaining in the cyanide tailing from the mill of the Wright-Hargreaves mine can be recovered in a sulphide concentrate by flotation.
2. The screen analyses of the different products including the products from table tests on the flotation tailings, show that fine grinding is essential if a low flotation tailing is to be produced.
3. Proper conditioning of the pulp preceding flotation is of vital importance, and for this purpose violent mechanical agitation in the conditioning tank for a period of not less than ten minutes is recommended.
4. The design of the conditioning tank is also important. It should be so constructed as to give a minimum of short circuiting with continuous feed and discharge. A conditioner with a submerged feed pipe, preferably in the centre of the tank, which will carry the feed to the bottom of the tank, and with the discharge from the top of the tank is recommended.
5. The reagents are simple and inexpensive. They are soda ash (sodium carbonate) and copper sulphate as conditioning reagents; xanthate, acid coal-tar creosote, and steam-distilled pine oil for the flotation reagents. The soda ash and copper sulphate require time contact in

the conditioning tank. They should be, however, added at separate places and not allowed to come into contact with each other before entering the pulp owing to the chemical reaction between them. It is better to add the soda ash before the pulp enters the conditioner.

6. The conditioning should be done in about a 1 : 2 pulp, and the flotation in a 1 : 2·5 or 1 : 3 pulp.

7. It is believed that flotation on tailings from a filter will give much better results than on tailings from a decantation plant.

8. A lower tailing and better grade of concentrate can be obtained if the cyanide tailing is reground before flotation, but the economy of such an extra step is doubtful.

9. The functions of the different reagents used are as follows: The soda ash deflocculates the gangue slimes and in so doing chemically precipitates the lime. The proper dispersion of the slimes is essential, otherwise the sulphides would remain enclosed in floccules of slimes, thus preventing their being oiled and also from presenting a surface to the air bubbles. A short period of time is required to complete the deflocculation of the pulp and for this reason a conditioning tank is used and violent mechanical agitation recommended to accelerate the action. Copper sulphate cleans the surface of the sulphides; after the slimes have been dispersed its action is probably fairly rapid. The xanthate and creosote require only sufficient time to allow them to become thoroughly mixed with the pulp before entering the flotation machine. The best place to add them can be determined by experiment after the plant is in operation. Pine oil is used as a frothing reagent.

Report No. 268

CONCENTRATION OF A COPPER-ZINC ORE FROM THE AMULET MINE, ROUYN, QUEBEC

C. S. Parsons and A. K. Anderson

Shipment.—A carload of ore was received April 27, 1927, from the Amulet Gold Mines, Limited, Rouyn, Quebec.

Characteristics of Ore.—The ore was a heavy sulphide type containing intimate mixture of sulphides of iron, copper, and zinc, and values in both gold and silver. The iron was present chiefly in the form of pyrite; the copper as chalcopyrite, and the zinc as a zinc blende having a high content of iron. The shipment contained some surface material consisting of masses of iron oxide formed from the weathering of the sulphides. The free acid content of the sample was high and tests made for soluble acid salts showed a sulphuric acid content of 9 pounds per ton of ore.

Purpose of Tests.—During the last two years a large number of small-scale tests have been conducted both in this laboratory and at other laboratories on samples of this ore. The results of these tests indicated that a remarkably good separation could be obtained between the copper and zinc and marketable grades of both minerals produced. The successful operation of a flotation concentrator handling a complex mixture of sulphides depends to a large extent on the proper control of conditions in the flotation pulp.

The object of the tests was to demonstrate that the small tests could be duplicated on a scale in which the conditions would, for all practical purposes, be similar to those obtaining in actual commercial operations.

Sampling and Analysis.—A head sample was taken for each individual test, but an approximate analysis of the whole car would be:—

Copper..................	3·85 per cent	Gold....................	0·20 oz./ton
Zinc....................	8·75 "	Silver..................	3·78 "
Free acid as H_2SO_4......	9·0 lb./ton		

Experimental Tests

Series of individual tests were made so that the conditions of control with different reagents could be studied in order to determine under what conditions and with what reagents the best results could be obtained.

Problem of the Gold and Silver Recovery

The small-scale tests previously made brought to light certain difficulties in effecting a good recovery of the gold and silver. In order to obtain a maximum return, the gold and silver should be recovered with the copper concentrate. If a part of the gold reports in the zinc concentrate there is not sufficient gold to be paid for by the zinc smelter, and, therefore, the gold and silver not recovered with the copper must be considered as lost. It was observed from the small-scale tests, which have been previously reported, that at times the recovery of the gold in the copper concentrate was much more than at others. It was definitely determined that when lime was used to produce an alkaline condition in the pulp during grinding and flotation of copper, the recovery of gold was low, but when soda ash was used in place of lime, the recovery was much higher. The grade of the concentrate produced was found to have an important bearing on the recovery, but for a given grade a higher recovery of gold was obtained with soda ash than with lime. It was evident that the lime, which is one of the best inhibiting reagents for pyrite and pyrrhotite known, was either preventing the gold-bearing pyrite from floating, or, that the use of soda ash allowed a selective action to take place in the presence of the cyanide whereby the gold-bearing pyrite was floated in preference to other pyrite. This action can be explained by assuming that there are two different classes of pyrite present, or it can also be explained by assuming that a certain amount of gold is set free during grinding, either as metallic gold or in some other form, which is floated better with soda ash than with lime. It was also found that unless an excessive amount of cyanide was used during flotation of the copper and a high alkalinity maintained, in the subsequent flotation of the zinc the sulphides of iron tended to float, preventing a commercial grade of zinc concentrate being produced. The small-scale tests indicated that the best separation and recovery of gold with copper could be obtained by using a small amount of cyanide and floating the copper in a pulp made slightly alkaline with soda ash, then adding lime to the pulp after the flotation of the copper and conditioning the pulp in a conditioning tank equipped with agitators to produce agitation before the flotation of zinc. The lime acting on the sulphides of iron prevents them from floating with the zinc. This procedure was attempted in Tests Nos. 1, 2, and 3, but was not found to be feasible owing to the time required to build up a complete circuit through the conditioning tank. To do this practically the whole carload would be required to build up the circuit. Thus, if unforeseen troubles were encountered, all the ore would have been used up before the separation could have been demonstrated on a large scale. It was therefore decided to eliminate the conditioning tank and the use of lime, and endeavour to balance the amount of cyanide and soda ash so that a maximum recovery of the gold with the copper would be obtained.

Test No. 1

This test was made on approximately 4 tons of ore. The ore was crushed to 4 mesh and fed to a Hardinge ball mill in closed circuit with a Dorr classifier. The Dorr overflow went to a 30 x 36-inch contact tank or agitator at an approximate density of 30 per cent solids. From the contact tank the pulp went to an 8-cell flotation machine, mechanical agitation type, where the copper was removed. The first 3 cells produced concentrate and the last 5, middlings which were returned to the ball mill for regrinding. The tailing from the copper cells went to a 30 x 36-inch contact tank, and was then pumped to a Callow unit consisting of two rougher cells and two cleaner cells, the tailing from the first rougher being returned to the second rougher. The concentrate from both passed to one cleaner and the cleaned concentrate from it was again cleaned in the second cleaner. Both middling products from the cleaners were returned with the feed. Rate of feed was between 800 and 900 pounds per hour.

Copper Reagents—

Sodium carbonate (soda ash). This was varied between 8 and 12 pounds to the ton and was fed dry to the ball mill.
Sodium cyanide was fed to the ball mill at the rate of ½ lb./ton.
Zinc sulphate was fed to the contact tank at rate of 2 lb./ton.
Aerofloat No. 15, 0·05 to 0·1 lb./ton fed direct to flotation cells.
XY. During latter part of test, XY was substituted for Aerofloat—separate samples were taken.

Zinc Reagents—

Lime. This was varied during test but about 3·5 lb./ton produced the most satisfactory conditions. The lime was fed to a contact tank in form of finely ground quicklime, where theoretically a 20-minute contact should have been obtained.
Copper sulphate was fed with lime to the contact tank. The amount was 1·5 lb./ton.
Xanthate was fed directly to the flotation cells. The amount was 0·15 to 0·3 lb./ton.
Frothing. No frothing reagents were required.

Densities—

Ball mill discharge	50 to 55 per cent solids
Classifier overflow	40 " "
Copper cells	35 " "

Copper Reagents—

Time	Sodium carbonate, lb./ton	Cyanide, lb./ton	Zinc sulphate, lb./ ton	Aerofloat No. 15, lb./ton	XY, lb./ton
9.00 a.m.	12·0	0·5	2·0	0·05	
11.00	12·0	0·5	2·0	0·05	
12.00	12·0	0·5	2·0	0·10	
12.30 p.m.	11·4	0·5	2·0	0·10	
1.30	11·0	0·5	2·0	0·10	
2.30	8·0	0·5	2·0	0·10	
3.00	12·0	0·5	2·0		0·20
3.30	12·0	0·5	2·0		0·20
4.30	12·0	0·5	2·0		0·20

Zinc Reagents—

Time	Lime, lb./ton	Copper sulphate, lb./ton	Xanthate lb./ton
9.00 a.m.	10·0	1·2	0·3
11.00	10·0	1·2	0·3
12.00	10·0	1·2	0·3
2.30 p.m.	3·5	1·2	0·3
3.30	3·5	1·2	0·3
4.30	3·5	1·2	0·3

Densities—Solids to Solution:

Time	Discharge ball mill, per cent	Classifier overflow
9.45 a.m.	50	1 : 2·5
11.30	51	1 : 2·8
12.30 p.m.	55	1 : 1·5
2.30	51	1 : 1·5

Sampling Periods—

Copper flotation—	
Sample No. 1	11 a.m. to 3.25 p.m.
" No. 2	3.25 p.m. to 4.35 p.m.
Zinc flotation—	
Sample No. 1	12.30 to 2.40 p.m.
" No. 2	2.40 to 4.35 p.m.

Total running time, 7·25 hours.

Analyses of Samples—

Head	Copper, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton
Head	4·35	9·90	0·24	3·46
No. 1.—Copper concentrate	20·98	8·12	0·62	12·92
" tailing	0·38	7·63	0·04	1·14
No. 2.—Copper concentrate	18·58	9·30	0·60	11·4
" tailing	0·32	7·95	0·04	1·18
No. 1.—Zinc concentrate	1·14	32·56	0·04	0·96
" tailing	0·30	0·81	0·02	0·82
No. 2.—Zinc concentrate	0·71	52·99	0·04	0·54
" tailing	0·26	1·73	0·09	1·0

Test No. 2

The flow-sheet used in this test was the same as in Test No. 1, with the exception that the contact tank before the copper flotation was omitted, the modifying reagents being added to the ball mill and ground with the ore.

Amount of feed, 8,240 pounds. Time, 15·25 hours. Feed per hour, 540 pounds.

Copper Reagents—

Sodium carbonate.—Added in dry state by a mechanical feeder to the ball mill and ground with the ore.

Sodium cyanide and zinc sulphate.—These two reagents added to the ball mill, the streams of the two reagents being brought together and a zinc cyanide emulsion formed.

Aerofloat, No. 15.—Used for the flotation of the copper and added to the head of the flotation cells.

Xanthate.—Used as a substitute in the place of Aerofloat during the latter part of the run.

Zinc Reagents—

Lime.—Added at the discharge of the copper cells in the form of milk of lime. The pulp containing the lime received a time contact in tank 30 x 36 inches before the flotation of the zinc. The milk of lime was formed by grinding quicklime in a large volume of water.

Copper sulphate.—Fed into the pulp at the same time as lime and given the same contact. Later this reagent was fed to a pump which discharged the ore pulp direct to the zinc cells.

Xanthate.—Fed to the pump which was discharging direct to the zinc flotation cells.

Copper Reagents—

Time	Sodium carbonate, lb./ton	Cyanide, lb./ton	Zinc sulphate, lb./ton	Oils, etc., lb./ton	Feed, lb./hr.	Alkalinity of ball mill discharge, lb./ton solution
May 31—						
9.30 a.m.	10·0	0·44	1·18	0·10	840	5·7
10.00	9·8					5·0
10.30		0·44	1·18	0·10	900	5·2
11.00	6·8	0·44	1·18	0·10	900	2·4
11.30	9·8	0·44	1·18	0·10		1·8
12.00	4·5	0·44	1·18	0·10		
1.00 p.m. (Breakdown).						
June 1—						
9.45 a.m.	7·0	0·44	1·18	0·10		2·1
10.30		0·44	1·18	0·12		
11.00	5·0	0·44	1·18	0·12		
12.00		0·44	1·18	0·11		
12.30 p.m.		0·44	1·18	0·10		2·2
1.30		0·36	0·88	0·10		
2.50	5·6	0·36	0·88	0·10		1·6
3·00		0·36	0·88	0·25 Xanthate		
4.00		0·36	0·88	0·25 Xanthate		

Zinc Reagents—

Time	Lime, lb./ton	Copper sulphate, lb./ton	Xanthate, lb./ton
May 31—			
10·30 a.m.	....	0·68	....
11.00	....	0·70	0·29
11.30	....	0·8	0·55
12.00	7·7	0·8	0·55
12.45 p.m.	....	1·2	....
June 1—			
9.45 a.m.	7·7	1·7	....
10.30	7·7	1·7	0·6
11.00	....	1·2	0·6
12.30 p.m.	7·7	1·2	0·6
1.30	8·0	1·2	0·29
2.00	9·0	1·2	0·29
3.00	9·0	1·2	0·29
4.00	9·0	1·2	0·29

Densities—

Time	Ball mill discharge, per cent	Classifier overflow, per cent	Copper cell discharge, per cent	Zinc cell discharge, per cent
May 31—				
9.45	65	8	8	....
10.15	70	25	....	....
10.45	68	37	20	....
11.30	68	42	22	....
12.00	61	32	28	17
12.30	60	33	27	17
June 1—				
10.00	53	32	....	....
11.00	49	18	18	12
11.30	60	24	....	....
12.00	70	47	....	....
12.30	....	48	20	....
1.00	67	36	31	5
2.00	55	35	38	5
3.00	61	43	31	23
4.00	62	36	36	28

Sampling Periods—

Copper flotation—

No. 1	11·15 a.m. to 1.40 p.m., May 31.
No. 2	10·50 a.m. to 2·50 p.m., June 1.
No. 3	2.50 p.m. to 4.30 p.m., "

Zinc flotation—

No. 1	11.25 a.m. to 1.40 p.m., May 31.
No. 2	1.40 p.m. to 3.25 p.m., June 1.
No. 3	3.25 p.m. to 4.30 p.m., "

Analyses of Samples—

—	Copper, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton
Head	3·97	8·85	0·24	3·42
No. 1.—Copper concentrate	25·36	6·82	0·80	7·34
" tailing	0·91	9·25	0·06	1·69
No. 2.— " concentrate	21·56	8·47	0·76	13·38
" tailing	0·43	9·00	0·06	1·28
No. 3.— " concentrate	20·92	10·50	0·60	11·48
" tailing	0·42	8·58	0·06	1·42
No. 1.—Zinc concentrate	2·27	50·97	0·12	1·66
" tailing	0·68	4·90	0·04	1·47
No. 2.— " concentrate	1·30	41·37	0·08	2·12
" tailing	0·31	2·68	0·05	1·17
No. 3.— " concentrate	1·16	39·12	0·08	1·80
" tailing	0·35	4·42	0·04	1·25

Test No. 3

The flow-sheet during the first part of the run was as in Test No. 2, but in the latter half of the run the conditioning tank was eliminated, the copper tailing being pumped direct to the zinc flotation cells by a centrifugal pump.

Amount of feed, 6,160 pounds. Time, 7·5 hours. Feed per hour, 820 pounds.

Copper Reagents—

Sodium carbonate was added in dry state to ball mill and ground with ore.
Sodium cyanide, a 10 per cent solution of this was fed to ball mill and ground with ore.
Zinc sulphate, a 20 per cent solution was fed to ball mill and brought in contact with the cyanide, the resultant zinc cyanide solution was ground with the ore. It was used only during first half of test.
Xanthate was used during the first half of test while zinc sulphate was in use. When zinc sulphate was discontinued xanthate was replaced with Aerofloat.
Aerofloat No. 15 was fed directly to the cells.

Zinc Reagents—

Sodium sulphite replaced lime which was used in previous tests to inhibit the iron sulphides. Added to contact tank conditioning copper tailing.
Sodium cyanide was used as a substitute for sodium sulphite; added to the conditioning tank.
Copper sulphate added both to the contact tank and to discharge of contact tank.
Xanthate fed to the flotation cells.

Sampling Periods—

Copper flotation—	
No. 1	11.05 a.m. to 2.25 p.m.
No. 2	2.25 p.m. to 4.10 p.m.
No. 3	4.10 p.m. to 5.15 p.m.
Zinc flotation—	
No. 1	4.00 p.m. to 5.15 p.m.

Special zinc concentrate sample taken during last 15 minutes.

Analyses of Samples—

——	Copper, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton
Head	3·85	9·25	0·24	3·28
No. 1.—Copper concentrate	17·00	11·47	0·56	9·88
" tailing	0·32	8·31	0·07	1·47
No. 2.— " concentrate	22·60	7·93	0·72	13·24
" tailing	0·33	9·33	0·05	1·30
No. 3.— " concentrate	17·14	8·04	0·64	10·58
" tailing	0·36	9·54	0·05	1·31
No. 1.—Zinc concentrate	0·74	41·49	0·08	1·54
" tailing	0·27	5·68	0·06	1·28
Special zinc concentrate	0·68	52·74	0·06	1·32

Copper Reagents—

Time	Sodium carbonate, lb./ton	Cyanide, lb./ton	Zinc sulphate, lb./ton	Xanthate, lb./ton	Aerofloat, lb./ton	Alkalinity of ball mill discharge, lb./ton solution
10.00 a.m.	9·6	0·5	1·75	0·13	0·10	
10.30		0·5	1·75	0·13	0·10	4·2
10.45		0·5	1·75	0·13	0·10	
11.15		0·5	1·75	0·16	0·10	
12.00		0·5	1·75	0·16	0·10	1·5
12.30 p.m.		0·5	1·75	0·16	0·10	
1.00		0·5	1·75	0·16	0·10	
1.30	8·6	0·5	1·75	0·16		3·2
2.00		0·5				
2.30		0·5				
3.00	9·8	0·5				
3.30		0·25				
4.00	9·6	0·25				
4.30		0·25				3·7
5.00		0·25				

Zinc Reagents—

Time	Sodium sulphite, lb./ton	Cyanide, lb./ton	Copper sulphate, lb./ton	Xanthate, lb./ton
10.30 a.m.	1		0·75	0·2
10.45	1		0·75	
11.15	1		0·75	
11.45	1		0·75	
12.00	1		0·75	
12.30 p.m.	1		0·75	0·2
1.00	1		1·00	0·16
1.20			0·25	0·16
1.30	1		0·25	0·16
2.00	1		0·60	0·16
2.20		0·23	0·60	
2.30		0·23	0·60	
3.00		0·23	0·60	
3.30			0·60	0·30
4.00			0·75	0·30
4.30			0·88	0·30
5.00			0·88	0·30

Densities—

Time	Ball mill discharge	Classifier discharge	Copper cell discharge	Zinc cell discharge
10.30 a.m.	1 : 0·75	1 : 1·5	....	....
10.45	1 : 0·85	1 : 1·86	....	....
11.15	1 : 0·96	1 : 1·6	1 : 1·8	....
11.45	1 : 0·96	1 : 2·1	1 : 1·8	....
12.00	1 : 1·08	1 : 1·86	1 : 2·0	....
12.30 p.m.	1 : 1·13	1 : 2·2	1 : 2·0	....
1.00	1 : 1·04	1 : 2·3	1 : 2·3	1 : 4·8
1.30	1 : 0·92	1 : 1·9	1 : 2·3	....
2.00	1 : 0·85	1 : 1·3	1 : 2·1	1 : 13·3
2.30	1 : 0·96	1 : 1·5	1 : 1·7	1 : 13·3
3.00	1 : 0·96	1 : 1·7	1 : 1·7	1 : 10·1
3.30	1 : 1·0	1 : 1·7	1 : 1·9	1 : 11·5
4.00	1 : 1·13	1 : 2·2	1 : 1·9	1 : 9·0
4.30	1 : 0·96	1 : 1·5	1 : 2·1	1 : 13·3

Test No. 4

The flow-sheet used in this test was similar to that used in the latter part of the previous test, that is, no conditioning tank was used between the copper and the zinc flotation. In all the previous tests the copper was floated in a mechanical agitation machine, and zinc in Callow cells. The tailing from the copper cells had to be pumped to the zinc cells and an irregular and too dilute feed was obtained for the zinc cells. This was changed in this test and the copper being floated in Callow cells, the tailing from which was air-lifted to the zinc cells. A more regular feed with no dilution of pulp was obtained. No conditioning tanks were used, the cyanide being fed to the ball mill which gave ample contact.

Amount of feed, 7,800 pounds. Time, 8·25 hours. Feed per hour, 945 pounds.

Copper Reagents—

Sodium carbonate.—Fed direct to ball mill.
Cyanide.—Fed direct to ball mill.
Aerofloat No. 15.—Used during first part of test.
Aerofloat No. 25.—Used during latter part of test—fed at the classifier overflow.

Zinc Reagents—

Copper sulphate.—Fed to the air-lift raising copper tailing to zinc cells.
Xanthate.—Fed direct to the head of the zinc cells.
Pine oil.—Some steam-distilled pine oil used to produce extra frothing.

Sampling Periods—

Copper flotation—

No. 1	12.20 p.m. to 4.20 p.m.
No. 2	4.20 p.m. to 5.20 p.m.

Zinc flotation—

No. 1	12.20 p.m. to 4.20 p.m.
No. 2	4.20 p.m. to 5.20 p.m.
Special zinc middling sample	4.40 p.m. to 5.30 p.m.

Analyses of Samples—

—	Copper, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton
Head sample	3·85	9·25	0·24	3·28
No. 1.—Copper concentrate	15·16	11·74	0·56	9·88
" tailing	1·10	9·08	0·04	1·40
No. 2.— " concentrate	10·20	11·32	0·40	7·66
" tailing	0·51	8·81	0·04	0·90
No. 1.—Zinc concentrate	0·70	40·69	0·08	2·88
" tailing	0·37	4·70	0·05	1·15
No. 2.— " concentrate	1·90	46·63	0·04	1·46
" tailing	0·31	2·01	0·03	0·87
Special zinc middling	1·88	18·64	0·10	3·22

Copper Reagents—

Time	Sodium carbonate, lb./ton	Cyanide, lb./ton	Xanthate, lb./ton	Aerofloat, lb./ton	Feed, lb./hour	Ball mill alkalinity, lb./ton solution
9.30 a.m.	12·3	0·181		No. 15 0·057	1,100	
10.00	10·0	0·181		0·057		
10.30		0·181		0·057		1·6
11.00		0·181		0·057		
11.30	10·1	0·181		0·057		4·1
12.00		0·181		0·057		
12.30 p.m.		0·301		0·057		
1.00		0·424	0·169	No. 25 0·089	780	1·9
1.30		0·339	0·169	0·089		
1.45		0·339		0·089		
2.00	11·0	0·339		0·089		
2.30	10·5	0·305		0·089		1·1
3.00		0·271		0·089	880	
3.30	8·0	0·225		0·089		0·8
3.45	9·9	0·105		0·089		
4.00		0·103		0·095	900	
4.30	10·0	0·103		0·095		2·0
5.00		0·088		0·095		
5.30		0·103		0·095		

Zinc Reagents—

Time	Copper sulphate, lb./ton	Xanthate, lb./ton
9.30 a.m.	1·21	0·181
10.00	1·21	0·121
10.21	0·72	0·121
10.30	0·72	0·121
11.00		0·121
12.00	0·95	0·121
12.30 p.m.	0·95	0·121
1.00	1·35	0·170
1.30	1·35	0·170
2.00	1·35	0·170
2.30	1·35	0·170
3.00	1·51	0·052
3.30	1·51	0·075
4.00	1·03	0·074
4.30	1·03	0·074
5.00	1·03	0·074
5.30	1·00	0·074

Densities—Ratio Solids to Solution—

Time	Ball mill discharge	Classifier discharge	Copper cell discharge	Zinc cell discharge
9.30 a.m.	1 : 0·78	....	....	....
10.00	1 : 0·75	1 : 1·38	....	....
10.30	1 : 0·75	1 : 1·56	1 : 1·32	1 : 4·88
11.00	1 : 0·88	1 : 3·16	1 : 1·70	1 : 2·44
11.30	1 : 1·12	1 : 4·55		
12.00	1 : 0·92	1 : 4·88	1 : 4·26	1 : 2·33
12.30 p.m.	1 : 0·92	1 : 4·0	1 : 2·84	1 : 5·25
1.00	1 : 1·00	1 : 5·25	1 : 2·44	1 : 6·69
1.30	1 : 1·96	1 : 6·14	1 : 5·66	1 : 8·09
2.00	1 : 0·92	1 : 4·55	1 : 3·76	1 : 8·09
2.30	1 : 1·04	1 : 4·88	1 : 2·57	1 : 3·54
3.00	1 : 1·0	1 : 3·54	1 : 2·12	1 : 8·09
3.30	1 : 1·04	1 : 4·26	1 : 2·44	1 : 4·26
4.00	1 : 1·0	1 : 6·14	1 : 2·33	1 : 5·25
4.30	1 : 0·96	1 : 4·26	1 : 4·26	1 : 9·00
5.00	1 : 1·08	1 : 6·69	1 : 7·33	1 : 13·28
5.30	1 : 0·92	1 : 4·00	1 : 6·14	1 : 9·0

Test No. 5

The flow-sheet was similar to that of Test No. 4.

Amount of feed, 6,400 pounds. Time, 7·75 hours. Feed per hour, 845 pounds.

Copper Reagents—

Sodium carbonate.—Fed to ball mill.
Sodium cyanide.—Fed to ball mill.
Aerofloat No. 25.—Fed to classifier.
Zinc sulphate.—Small amount fed to cleaner cell.

Zinc Reagents—

Copper sulphate.—Fed to the well of an air-lift elevating copper tailing to zinc circuit.
Xanthate.—Fed partly at the head of the cell and partly to middle cell of flotation machine.

Sampling Periods—

Copper flotation—

No. 1	10.55 a.m. to 1.45 p.m.
No. 2	1.45 p.m. to 3.30 p.m.
No. 3	4.35 p.m. to 4.50 p.m.

Zinc flotation—

No. 1	11.05 a.m. to 1.45 p.m.
No. 2	1.45 p.m. to 3.30 p.m.
No. 3	4.35 p.m. to 4.50 p.m.

Analysis of Samples—

—	Copper, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton
Head sample	3·85	9·25	0·24	3·28
No. 1.—Copper concentrate	9·98	11·86	0·36	6·98
" tailing	0·44	6·63	0·02	0·83
No. 2.— " concentrate	9·54	12·62	0·34	6·50
" tailing	0·48	6·09	0·04	0·86
No. 3.— " concentrate	13·58	12·08	0·58	9·62
" tailing	0·39	7·88	0·03	1·03
No. 1.—Zinc concentrate	1·79	52·44	0·04	1·22
" tailing	0·36	1·50	0·03	0·78
No. 2.— " concentrate	2·25	47·74	0·06	1·80
" tailing	0·32	0·92	0·02	0·72
No. 3.— " concentrate	1·63	52·55	0·04	1·40
" tailing	0·27	3·07	0·02	0·85

Copper Reagents—

Time	Soda ash, lb./ton	Cyanide, lb./ton	Aerofloat No. 25, lb./ton	Ball mill alkalinity, lb./ton solution
9.00 a.m.	12·5	0·12		4·9
10.00				2·0
10.30	10·0	0·12		
11.00		0·08		2·1
11.30	9·9	0·066	0·16	
12.00			0·18	
12.30 p.m.		0·09		
1.45		0·06		
2.15	12·5	0·06		1·8
2.45		0·08		
3.45		0·14	0·12	2·2
4.30	12·5	0·16		3·8

Zinc Reagents—

Time	Copper sulphate, lb./ton	Xanthate, lb./ton	Rate of feed, lb./hour	
10.30 a.m.	0·8		1,100	9.40–10.40
11.00	0·8		800	10.40–11.40
11.40	0·8	0·2	740	11.40–12.40
12.30	0·8		800	12.40– 1.40
1.45	0·8			
2.10	0·8		800	
2.45	0·8	0·2	700	1.40– 2.40
3.45	0·8			
4.30	1·1	0·2		

Densities—

Time	Ball mill discharge, per cent	Classifier overflow, per cent	Copper cell discharge, per cent	Zinc cell discharge, per cent
9.20 a.m.	54	28		
10.00	48	15		
10.45	46	7	5	5
11.45	49	9	7	5
2.00 p.m.	51			
2.30	45	6	5	5
3.00	53	20	5	5
3.30	50	17		
4.00	52	19	18	10

Test No. 6

The flow-sheet was similar to that of Tests Nos. 4 and 5.

Amount of feed, 5,940 pounds. Time, 7·75 hours. Feed per hour, 764 pounds.

Copper Reagents—

Sodium carbonate.—Fed to ball mill.
Sodium cyanide.—Fed to ball mill.
Zinc sulphate.—During first part of test fed to cleaner cell. Latter part of test to ball mill with the cyanide as emulsion of zinc cyanide.
Aerofloat No. 15.—Fed to classifier.

Zinc Reagents—

Copper sulphate.—Added to well of air-lift elevating the copper tailing to zinc flotation circuit.
Xanthate.—Fed direct to flotation cells, partly with the feed and partly at the middle cell of the machine.

Sampling Periods—

Copper flotation—	
No. 1	11.15 a.m. to 1.30 p.m.
No. 2	1.30 p.m. to 3.30 p.m.
No. 3	3.30 p.m. to 4.50 p.m.
Zinc flotation—	
No. 1	11.15 a.m. to 1.30 p.m.
No. 2	1.30 p.m. to 3.30 p.m.
No. 3	3.30 p.m. to 4.50 p.m.

Analysis of Samples—

—	Copper, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton
Head sample	3·85	9·25	0·24	3·28
No. 1.—Copper concentrate	14·32	9·78	0·58	8·78
" tailing	0·44	8·92	0·03	1·06
No. 2.— " concentrate	14·24	11·10	0·66	9·74
" tailing	0·32	8·43	0·04	0·98
No. 3.— " concentrate	20·16	9·36	0·88	12·84
" tailing	0·44	8·60	0·06	1·44
No. 1.—Zinc concentrate	1·73	51·58	0·04	1·46
" tailing	0·36	2·48	0·05	1·13
No. 2.— " concentrate	1·18	52·87	0·04	1·16
" tailing	0·38	2·00	0·03	0·92
No. 3.— " concentrate	1·48	52·01	0·06	1·44
" tailing	0·34	1·25	0·04	1·08

Copper Reagents—

Time	Soda ash, lb./ton	Cyanide, lb./ton	Aerofloat, lb./ton	Pine oil, lb./ton	Zinc sulphate, lb./ton to cleaner	Feed, lb./hr.	Ball mill alkalinity lb./ton solution
9.10 a.m.	11·0	0·112	0·053		0·634		
9.45	8·7					1,000	
10.00		0·156			0·875		
10.15	12·5	0·156	0·073			725	4·1
11.30	10·0	0·148	0·070		0·208	760	2·0
12.00		0·174	0·085		0·771		
12.30 p.m.		0·174	0·085		0·771		
1.00		0·114	0·085		to ball mill		
1.30		0·116	0·085			740	
2.00	12·5	0·116	0·085		1·27		2·4
2.30		0·190	0·085		1·36	700	
3.00		0·190	0·085	0·082	1·36		
3.30	13·0	0·223	0·085	0·082	1·36	710	
4.00		0·223	0·074	0·074	1·56		4·4
4.30		0·223	0·059	0·074	1·56		

Zinc Reagents—

Time	Copper sulphate, lb./ton	Xanthate, lb./ton	Feed, lb./hr.
9.15 a.m.			1,000
9.30	1·05	0·053	
10.00	1·05	0·053	
10.30	1·45	0·073	725
11.00			760
11.30	1·39	0·243	
12.00	1·74	0·243	
12.30 p.m.	2·09	0·243	
1.00	2·44	0·243	
1.30	2·50	0·321	740
2.00	2·50	0·321	
2.30	3·03	0·340	700
3.00	3·03	0·302	
3.30	2·98	0·428	710
4.00	2·98	0·447	
4.30	2·98	0·447	

Densities—

Time	Ball mill discharge	Classifier discharge	Copper cell discharge	Zinc cell discharge
9.30 a.m.	1 : 0·809	1 : 2·03		
10.00	1 : 1·128	1 : 4·26	1 : 6·69	1 : 11·5
10.30	1 : 0·809	1 : 3·55	1 : 6·69	1 : 6·92
11.00	1 : 0·724	1 : 2·03	1 : 4·26	1 : 1·35
11.10	1 : 0·809			
11.30	1 : 0·724	1 : 3·35	1 : 4·26	1 : 5·67
12.00	1 : 0·809	1 : 3·55	1 : 8·09	1 : 8·09
12.30 p.m.	1 : 0·852	1 : 3·76	1 : 4·26	1 : 19·0
1.00	1 : 0·809	1 : 4·0	1 : 1·15	1 : 15·67
1.30	1 : 0·809	1 : 3·76	1 : 11·5	1 : 15·67
2.00	1 : 0·809	1 : 4·0	1 : 9·0	1 : 15·67
2.30	1 : 0·887	1 : 4·0	1 : 6·69	1 : 11·5
3.00	1 : 0·852	1 : 4·56	1 : 10·11	1 : 10·11
3.30	1 : 0·786	1 : 4·0	1 : 6·14	1 : 10·11
4.00	1 : 0·754	1 : 2·7	1 : 5·25	1 : 8·09
4.30	1 : 0·754	1 : 2·85	1 : 4·56	1 : 7·33

Test No. 7

The flow-sheet used was similar to that of the previous test, No. 6.

Amount of feed, 6,360 pounds. Time, 8 hours. Feed per hour, 795 pounds.

Copper Reagents—

Soda ash.—This reagent was fed to the ball mill as in the previous test.
Cyanide.—This reagent was fed to the ball mill as in the previous test.
Zinc sulphate.—This reagent was fed to the ball mill with the cyanide, an emulsion of zinc cyanide being formed.
Xanthate.—This reagent was fed to the classifier.
Pine oil.—Steam-distilled pine oil was fed with the xanthate to the classifier.

Zinc Reagents—

Copper sulphate.—This reagent was added to the air-lift between the copper and zinc flotation circuits.
Xanthate.—This reagent was added to the head of the zinc cells and part to the middle cell of the machine.

Sampling Periods—

Copper samples—

No. 1	11.00 a.m. to 2.20 p.m.
No. 2	2.20 p.m. to 5.00 p.m.

Zinc samples—

No. 1	11.00 a.m. to 5.00 p.m.
No. 2	11.00 a.m. to 5.00 p.m.
Special sample	End of run.

Analysis of Samples—

—	Copper, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton
Head	3·85	9·25	0·24	3·28
No. 1.—Copper concentrate	23·0	7·82	0·72	10·96
" tailing	0·69	8·87	0·16	1·79
No. 2.— " concentrate	19·54	9·68	0·68	11·26
" tailing	0·51	8·65	0·10	1·44
No. 1.—Zinc concentrate	1·61	49·66	0·08	1·60
" tailing	0·47	3·56	0·13	1·45
No. 2.— " concentrate	1·61	49·66	0·08	1·60
" tailing	0·47	3·56	0·13	1·45
Special tailing	0·03	1·06	0·11	1·39

Copper Reagents—

Time	Soda ash, lb./ton	Cyanide, lb./ton	Flotation reagents		Zinc sulphate, lb./ton	Feed, lb./hour	Ball mill alkalinity, lb./ton solution
			Pine oil, lb./ton	Xanthate, lb./ton			
9.15 a.m.	....	0·161	0·059	....	2·01		
9.30	....	0·161	....	0·129	2·03		
10.00	....	0·202	....	0·129	2·01	820	
10.30	11	0·193	0·015	0·113	1·93		
11.00	....	0·161	0·012	0·107	1·61	990	
11.30	....	0·161	....	0·080	1·61		
12.00	....	0·161	....	0·094	1·61		
12.30 p.m.	....	0·203	0·016	0·102	2·04	780	
1.00	....	0·203	....	0·102	2·04		
1.30	....	0·254	0·064	0·136	2·44		
2.00	13	0·238	....	0·119	2·37		
2.30	....	0·238	....	0·102	2·17		
3.00	....	0·238	0·019	0·204	2·24		
3.30	....	0·238	....	0·204	Off		
4.00	....	0·203	....	0·102	"	....	6·1
4.30	....	0·246	....	....	"		

Zinc Reagents—

Time	Copper sulphate, lb./ton	Xanthate, lb./ton	Feed, lb./hour
10.10 a.m.	2·58	0·384	820
10.30	2·58	0·452	
11.00	2·11	0·161	990
11.30	2·14	0·214	
12.00	2·14	0·187	
12.30 p.m.	2·71	0·271	780
1.00	2·71	0·203	
1.30	2·71	0·169	
2.00	2·71	0·057	
2.30	2·71	0·305	
3.00	2·71	0·203	
3.30	2·71	0·373	
4.00	2·71	0·271	
4.30	2·71	0·492	

Test No. 8

In the flow-sheet used in this test the classifier overflow went direct to the mechanical agitation machine for the flotation of the copper. The copper flotation tailing then went to an air-lift with a large well, which was built for this particular test in order that a certain amount of contact would be obtained between the copper sulphate and the pulp before entering the zinc flotation cells. The zinc was floated in a set of Callow cells, the rougher concentrate being recleaned twice, and the middlings from both cleaner cells returned with the feed to rougher cells.

Amount of feed, 5,400 pounds. Time, 7 hours. Feed per hour, 772 pounds.

Copper Reagents—

Sodium carbonate.—This reagent was added in the dry state to the ball mill.

Sodium cyanide.—The point of introduction of this reagent to the circuit was changed. Instead of adding it to the ball mill it was introduced between the ball mill discharge and the boot of an elevator lifting the pulp to the Dorr classifier.

Zinc sulphate.—This reagent was introduced immediately after the cyanide, but these were not brought together before entering the pulp.

Aerofloat No. 15.—This reagent was used to float the copper and was added to the head of the flotation machine.

Zinc Reagents—

Copper sulphate.—This reagent was added to the well of the air-lift between the copper and zinc flotation machine.

Xanthate was added to the air-lift, but care was taken not to allow it to mix with the copper sulphate before reaching the pulp.

Pine oil.—A little steam-distilled pine oil was used at times to increase frothing.

Sampling periods—

Copper samples—

No. 1	12.30 p.m. to 3.50 p.m.
No. 2	3.50 p.m. to 4.50 p.m.

Zinc samples—

No. 1	12.30 p.m. to 3.50 p.m.
No. 2	3.50 p.m. to 4.50 p.m.

Analyses of Samples—

—	Copper, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton
Head sample	3·85	9·25	0·24	3·28
No. 1.—Copper concentrate	21·66	9·56	0·82	12·54
" tailing	0·40	8·70	0·04	1·44
No. 2.— " concentrate	21·80	8·49	0·96	13·12
" tailing	0·40	9·25	0·06	1·55
No. 1.—Zinc concentrate	1·07	52·86	0·06	1·32
" tailing	0·36	2·98	0·05	1·40
No. 2.— " concentrate	1·37	52·22	0·04	1·46
" tailing	0·30	2·07	0·05	1·43

Copper Reagents—

Time	Sodium carbonate, lb./ton	Cyanide, lb./ton	Zinc sulphate, lb./ton	Aerofloat, lb./ton
10.15 a.m.	13	0·251	1·07	
11.00		0·251	1·07	
11.30	11	0·305	1·31	0·045
12.00		0·305	0·541	0·045
12.30 p.m.		0·305	0·541	
1.00		0·270	0·541	
1.30		0·270	0·541	
2.00				
2.30	13	0·301	0·568	
3.30			0·602	0·045
4.00		0·328		
4.30				

Zinc Reagents—

Time	Copper sulphate, lb./ton	Xanthate, lb./ton	Feed, lb./hour
11.30 a.m.	1·69	....	780
12.00	1·86	0·407	
12.30 p.m.	1·83	0·187	
1.00	1·83	0·339	
1.30	1·97	0·339	
2.00	2·10	0·254	730
3.00	2·08	0·245	700
4.00	2·33	0·282	680
4.30	....	....	720

Densities—

Time	Ball mill discharge		Classifier discharge		Copper cell discharge		Zinc cell discharge	
	Solids, per cent	Ratio	Solids, per cent	Ratio	Solids, per cent	Ratio	Solids, per cent	Ratio
10.30 a.m.	47	1 : 1·12	24	1 : 3·16				
11.00	56	1 : 0·78	35	1 : 1·85				
12.00	65	1 : 0·53	41	1 : 1·41	16	1 : 5·25	9	1 : 10·11
12.30 p.m.	63	1 : 0·58	42	1 : 1·38	24	1 : 3·16	10	1 : 9·00
1.00	49	1 : 1·04	23	1 : 3·24	28	1 : 2·57		
1.30	52	1 : 0·92	....	....	23	1 : 3·34	13	1 : 6·69
2.00	52	1 : 0·92	30	1 : 2·33				
2.25	52	1 : 0·92	24	1 : 3·16				
2.30	53	1 : 0·88	25	1 : 3·00				
3.00	48	1 : 1·08	34	1 : 1·94	23	1 : 3·34	8	1 : 11·5
3.30	53	1 : 0·88	30	1 : 2·33				
4.00	57	1 : 0·75	31	1 : 2·22				
4.30	53	1 : 0·88	31	1 : 2·22				

Test No. 9

The flow-sheet used in this test was similar to that of Test No. 8. A change, however, was made in the method of feeding the ore to a ball mill. A belt conveyer was used and a much more uniform feed was obtained.

Amount of feed, 5,872·5 pounds. Time, 7·25 hours. Feed per hour, 810 pounds.

Copper Reagents—

Sodium carbonate was added as in the preceding test but the quantity used was increased.

Sodium cyanide was added at the discharge of the ball mill as in the preceding test.

Zinc sulphate was used during the first part of the test and was added as in the preceding test to the discharge of the ball mill.

Xanthate was used in place of Aerofloat and was continued after the zinc sulphate was taken off.

Zinc Reagents—

Copper sulphate was added to an air-lift pumping the copper cell tailing to the zinc flotation cells.

Xanthate was added to the air-lift and at times an additional amount was used at the head of the second rougher.

Sampling Periods—

Copper flotation—

No. 1	1.00 p.m. to 3.00 p.m.
No. 2	3.00 p.m. to 4.55 p.m.

Zinc flotation—

No. 1	1.00 p.m. to 3.00 p.m.
No. 2	3.00 p.m. to 4.55 p.m.

Analysis of Samples—

—	Copper, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton
Head	3·85	9·25	0·24	3·28
No. 1.—Copper concentrate	23·78	9·57	0·52	11·38
" tailing	0·45	8·65	0·14	2·02
No. 2.— " concentrate	23·78	10·82	0·58	11·60
" tailing	0·43	8·16	0·13	1·92
No. 1.—Zinc concentrate	0·90	50·50	0·14	2·54
" tailing	0·35	1·25	0·12	1·86
No. 2.— " concentrate	1·33	51·47	0·32	2·90
" tailing	0·32	0·87	0·08	1·74

Recoveries—

Copper	90 per cent	Gold	57·0 per cent
Zinc	75 "	Silver	46·7 "

Copper Reagents—

Time	Sodium carbonate, lb./ton	Cyanide, lb./ton	Xanthate, lb./ton	Zinc sulphate, lb./ton	Feed, lb./ton
10.15 a.m.	13	0·33	0·15	0·54	
10.30		0·33	0·15	0·54	780
11.00	12	0·29	0·14	0·52	810
11.30		0·36	0·14	0·52	
12.00		0·37	0·14	0·39	
12.30 p.m.	19	0·36	0·11	0·49	
1.00	13	0·32	0·11	0·52	
1.30		0·50	0·16	Off	790
2.00	14	0·45	0·15		
2.30		0·50	0·22		
3.00		0·31	0·16		810
3.30		0·51	0·14		
4.00		0·42	0·14		810
4.30		0·52	0·14		

Zinc Reagents—

Time	Copper sulphate, lb./ton	Xanthate, lb./ton
10.30 a.m	1·42	0·271
11.00	1·37	0·261
11.30	1·63	0·278
12.00	1·51	0·343
12.30 p.m	1·37	0·229
1.00	1·57	0·196
1.30	1·34	0·251
2.00	1·24	0·218
2.30	2·07	0·218
3.00	1·89	0·179
3.30	2·12	0·425
4.00	1·70	0·213
4.30	1·47	0·229

Densities—

Time	Ball mill discharge,		Classifier discharge		Copper cell discharge		Zinc cell discharge	
	Solids, per cent	Ratio	Solids, per cent	Ratio	Solids, per cent	Ratio	Solids, per cent	Ratio
10.00 a.m.	54	1 : 0·85	42	1 : 1·38	....	....	....	....
10.30	59	1 : 0·69	30	1 : 2·33	....	....	....	....
11.00	59	1 : 0·69	31	1 : 2·23	....	....	....	....
11.15	57	1 : 0·75	37	1 : 1·70	....	....	....	....
11.30	57	1 : 0·75	35	1 : 1·86	....	....	....	....
12.00	56	1 : 0·79	33	1 : 2·03	....	....	....	....
12.30 p.m.	57	1 : 0·75	31	1 : 2·23	27	1 : 2·70	17	1 : 4·88
1.30	62	1 : 0·61	38	1 : 1·63	30	1 : 2·33	18	1 : 4·56
2.00	58	1 : 0·72	33	1 : 2·03	30	1 : 2·33	18	1 : 4·56
2.30	57	1 : 0·75	30	1 : 2·33	27	1 : 2·70	16	1 : 5·25
3.00	60	1 : 0·67	30	1 : 2·33	24	1 : 3·16	11	1 : 8·09
3.30	57	1 : 0·75	31	1 : 2·23	27	1 : 2·70	10	1 : 9·00
4.00	59	1 : 0·69	30	1 : 2·33	24	1 : 3·16	13	1 : 6·69
4.30	58	1 : 0·72	30	1 : 2·33	25	1 : 3·00	9	1 : 10·11

Test No. 10

The same flow-sheet was used as in Test No. 9.

Copper Reagents—

Soda ash.—Fed to the ball mill.
Sodium cyanide.—Fed to ball mill. It will be observed that in the previous test this reagent was added at the ball mill discharge.
Zinc sulphate.—Fed to ball mill discharge.
Aerofloat No. 15.—Fed to copper cells in place of the xanthate used in Test No. 9.

Zinc Reagents—

Copper sulphate.—Fed to air-lift between copper and zinc flotation circuits.
Xanthate.—Fed to air-lift between copper and zinc flotation circuits.
Pine oil.—A little steam distilled added to increase amount of froth.

Sampling Periods—

Copper flotation—

No. 1	10.30 a.m. to 1.20 p.m.
No. 2	1.20 p.m. to 2.40 p.m.
No. 3	2.40 p.m. to 5.20 p.m.

Zinc flotation—

No. 1	11.35 a.m. to 1.20 p.m.
No. 2	1.20 p.m. to 2.40 p.m.
No. 3	2.40 p.m. to 5.20 p.m.

Analyses of Samples—

—	Copper, per cent	Zinc, per cent	Gold, oz./ton	Silver, oz./ton
Head sample	3·85	9·25	0·24	3·28
No. 1.—Copper concentrate	20·64	10·76	0·62	11·94
“ tailing	0·46	8·84	0·06	1·48
No. 2.— “ concentrate	24·00	8·53	0·88	13·60
“ tailing	0·48	10·35	0·07	1·43
No. 3.— “ concentrate	24·32	8·22	0·90	13·84
“ tailing	0·53	9·67	0·07	1·51
No. 1.—Zinc concentrate	1·30	52·42	0·10	1·60
“ tailing	0·26	1·87	0·05	1·16
No. 2.— “ concentrate	1·32	49·40	0·10	2·14
“ tailing	0·35	1·51	0·05	1·29
No. 3.— “ concentrate	1·65	52·83	0·10	1·78
“ tailing	0·40	1·40	0·07	1·43

Recoveries—

Copper, 88 per cent. Zinc, 77 per cent. Gold, 76·8 per cent. Silver, 72 per cent.

Copper Reagents—

Time	Soda ash, lb./ton	Cyanide, lb./ton	Aerofloat No. 15, lb./ton	Zinc sulphate, lb./ton	Alkalinity classifier overflow, lb./ton solution
9.15 a.m.		0·3		0·52	
10.00		0·3	0·065	0·52	
10.30	17·0	0·3		0·52	8·3
11.00		0·3	0·07	0·25	
11.30		0·3	0·05	0·55	
12.00		0·3	0·05	0·55	
12.30 p.m.		0·25	0·05	0·55	
1.00		0·25	0·04	0·55	
1.30		0·25	0·04	0·55	
2.00		0·25	0·04	0·50	
2.30		0·25	0·05		3·1
2.45				0·31	5·6
3.00	13·5	0·25	0·04	0·31	
3.30	16·0	0·25	0·04	0·31	
4.00		0·25	0·04	0·31	
4.30		0·25	0·04	0·31	

Zinc Reagents—

Time	Copper sulphate, lb./ton	Xanthate, lb./ton	Pine oil, lb./ton
10.30 a.m.	1·6	0·01	....
11.00	2·5	0·01	....
11.30	2·5	0·02	0·03
12.00	1·3	0·02	....
12.30 p.m.	2·5	0·02	....
1.00	1·6	0·01	0·01
1.30	1·7	0·01	0·01
2.00	1·9	0·02	0·03
2.30	1·6	0·01	0·03
3.00	1·6	0·02	0·03
3.30	1·9	0·02	0·03
3.45	1·9	0·03	0·03
4.00	1·9	0·01	0·03
4.30	1·6	0·02	0·03

Densities—Solids to Solution

Time	Ball mill discharge	Classifier overflow	Copper cell discharge	Zinc cell discharge
10.00 a.m.	1 : 1·2	1 : 4·882	....	....
10.00	1 : 1·0	....	....	....
10.30	1 : 0·8	1 : 2·226	1 : 5·66	....
10.45	1 : 0·7	1 : 1·703	....	....
11.00	1 : 0·7	1 : 1·778	....	....
11.30	1 : 0·7	1 : 1·94	1 : 2·70	1 : 5·66
11.45	1 : 0·7	1 : 2·030	....	....
12.00	1 : 0·7	1 : 2·226	....	....
12.30 p.m.	1 : 0·6	1 : 2·125	....	....
1.00	1 : 0·6	1 : 2·448	1 : 2·52	1 : 8·09
1.30	1 : 0·6	1 : 2·226	....	....
2.00	1 : 0·6	1 : 2·704	1 : 2·70	1 : 6·69
2.30	1 : 0·1	1 : 2·704	1 : 3·34	1 : 5·25
3.00	1 : 0·7	1 : 3·348	....	....
3.30	1 : 0·7	1 : 3·0	1 : 3·76	1 : 8·04
4.00	1 : 0·7	1 : 3·167	1 : 3·76	1 : 7·33
4.30	1 : 0·7	1 : 2·846	1 : 3·34	1 : 6·69

The following screen tests were made as an indication of grinding obtained:

Screen test on Test No. 1			Screen test on Test No. 9		
	grammes	per cent		grammes	per cent
+ 65	0·4	0·08	+ 65	0·5	0·10
+100	2·8	0·56	+100	4·2	0·84
+150	20·1	4·02	+150	15·5	3·10
+200	32·5	6·50	+200	39·0	7·80
−200	444·2	88·84	−200	440·8	88·16

SUMMARY AND CONCLUSIONS

Owing to the oxidized condition of the ore the separation proved much more difficult than was expected. The principal difficulty encountered was in the flotation of the zinc. The copper could be floated and good recoveries made under a wide range of conditions, but the zinc was very difficult to float. The inhibiting effects of the reagents added to prevent it from floating during the copper flotation apparently persisted after the ordinary promoters, such as copper sulphate and xanthate, were used. It was only under certain conditions of very limited range that both a good recovery and grade of concentrate could be made. As far as could be determined the alkalinity of the pulp played an important part in this condition. No doubt the large amount of soluble sulphates equivalent to 9 pounds of H_2SO_4 per ton of ore was largely responsible for this difficulty. Small-scale tests made on fresh unweathered ore gave no trouble in this way, but similar tests made on old samples of ore left about the laboratory for a number of months gave the same trouble as was experienced in these large tests. The reason advanced for this is that the precipitation of complex basic cyanides and hydroxides, which are generally supposed to film the sphalerite and prevent its floating, is much heavier than usual owing to the excess of soluble sulphates present, thus retarding the action of the promoter.

It will be observed that in Tests Nos. 7 and 9 the gold recovery in the copper concentrate is much lower than in Tests Nos. 8 and 10, and that in each case where the gold recovery is low, xanthate was used in combination with zinc sulphate during the flotation of copper, whereas when Aerofloat was used, as in Tests Nos. 8 and 10, the gold recovery is much higher. Special attention is drawn to the results obtained in Tests Nos. 7, 8, 9, and 10. Although the results in Tests Nos. 9 and 10 show a very satisfactory separation between the copper and zinc it must be pointed out that a great deal of further experimental work must be performed before the most economical flow-sheet and reagent combination can be determined. In order to determine these conditions, the installation of a pilot plant of at least 50 tons capacity at the mine is recommended. This should be operated for a sufficient length of time to thoroughly study the effects of every change in character of the ore from different parts of the mine.

Report No. 269

EXPERIMENTAL TESTS ON GOLD ORE FROM THE FRANCŒUR PROPERTY, BOISCHATEL TOWNSHIP, QUEBEC

A. K. Anderson

Shipment.—Three bags of ore, gross weight 126 pounds, were received May 2, 1927, from the Towagmac Exploration Company, Rouyn, Quebec. The ore was from the Francœur property, Boischatel township, Quebec.

Characteristics of the Ore.—The ore consisted of fine-grained, pink feldspar porphyry containing finely divided sulphides of iron and arsenic carrying gold and a small amount of silver.

Purpose of Tests.—The tests were conducted with the object of obtaining a concentration of the gold and silver in a product suitable for smelting.

Sampling and Analysis.—The shipment was crushed to −16 mesh and passed through a Jones riffle sampler, cutting out a sample for assay. This head sample showed the following:—

Gold	0·37 oz./ton
Silver	0·10 "

EXPERIMENTAL TESTS

In each of the following tests, portions of the −16 mesh material, weighing 1,000 grammes, were ground for 40 minutes in a ball mill with an equal weight of water. This grinding gave the following screen analysis:—

− 65+100	0·9 per cent
−100+150	5·3 "
−150	93·8 "

A microscopic examination of the product showed the sulphide particles to be free of gangue material at this fineness.

Test No. 1—Flotation

Reagents.—Coal-tar creosote 0·5, tar oil 0·5, pine oil 0·1 lb./ton.

Product	Weight, per cent	Assay		Per cent of values	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Flotation concentrate	11·98	2·41	0·50	86·7	88·2
Flotation tailing	88·02	0·05	0·01	13·3	11·8
Head from products		0·33	0·04		

Test No. 2—Amalgamation

A portion of the ore was ground in a ball mill and then agitated in a 1 : 1 pulp with 10 per cent by weight of mercury.

—	Au, oz./ton	Ag, oz./ton	Per cent of values	
			Au	Ag
Amalgamation tailing	0·11	0·05	29·5	50·0
Amalgamated			70·5	50·0

Test No. 3—Flotation and Amalgamation

The ore was floated and the tailings amalgamated.

Reagents.—Coal-tar creosote 0·5, tar oil 0·5, pine oil 0·10 lb./ton, mercury 10 per cent by weight.

—	Weight, per cent	Au, oz./ton	Ag, oz./ton	Per cent of values	
				Au	Ag
Flotation concentrate	8·16	3·03	0·74	66·7	74·0
Gold recovered from amalgam				23·8	

Test No. 4—Flotation and Tabling

Flotation in an alkaline circuit, and tailing tabled.

Reagents.—Coal-tar creosote 0·5, tar oil 0·5, pine oil 0·1, and soda ash 2·0 lb./ton.

Product	Weight, per cent	Assay		Per cent of values	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Flotation concentrate	9·5	2·54	0·47	81·6	51·1
Table concentrate	2·5	1·05	0·32	8·8	9·0
" middling	3·0	0·11	0·03	1·1	1·0
" tailing	85·0	0·03	0·04	8·5	38·9
Head from products		0·30	0·087		

Test No. 5—Flotation and Tabling

Flotation in neutral circuit, and tailing tabled.

Reagents.—Coal-tar creosote 0·5, tar oil 0·5, pine oil 0·10 lb./ton.

Product	Weight, per cent	Assay		Per cent of values	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Flotation concentrate	11·3	2·40	0·36	79·2	45·9
Table concentrate	2·8	0·92	0·19	7·5	5·7
" middling	2·0	0·18	0·07	1·1	1·1
" tailing	83·9	0·05	0·05	12·2	47·3
Head from products		0·344	0·087		

CONCLUSIONS

From the above tests it would appear that flotation of the iron sulphides followed by concentration of the tailing on Wilfley tables to recover values present as free gold and gold associated with arsenopyrite, will yield a 90 per cent or better recovery of the gold in a concentrate acceptable for smelting.

Report No. 270

EXPERIMENTAL TESTS ON GOLD ORE FROM THE COOPER GOLD MINE, MICHIPICOTEN AREA, ONTARIO

J. S. Godard

Shipments.—One shipment of 195 pounds was received May 25, 1927, from the Huronian Belt Company, Toronto. It consisted of two lots of ore, 145 pounds from the 1st level and 50 pounds from the 3rd level.

Characteristics and Analysis.—The gold is free and associated with small amounts of pyrite, pyrrhotite, and chalcopyrite in a quartz gangue.

Analysis of head sample of ore from the 1st level was:

Gold	1·33 oz./ton
Copper	0·12 per cent
Iron	1·42 "

Analysis of head sample of ore from the 3rd level was:

Gold	0·40 oz./ton
Copper	Trace
Iron	2·75 per cent

Purpose of Experimental Tests.—Mr. Dorfman, of the Huronian Belt Co., requested the following test work on both ores: amalgamation; amalgamation and flotation; and amalgamation, flotation, and tabling; also the same tests and cyanidation tests on a sample consisting of 75 per cent 1st level ore and 25 per cent 3rd level ore.

Experimental Tests on Ore from 1st Level

Test No. 1—Amalgamation. Results of screen test on amalgamation tailings:—

Product	Weight, per cent	Au, oz./ton	Per cent of values
+100	4·3	0·26	5·2
+150	12·4	0·24	12·8
+200	19·3	0·26	23·1
−200	64·0	0·20	58·9
Average tailing		0·217	

Recovery, per cent .... 83·4

Test No. 2—Amalgamation. Results of amalgamation tailing screened on 200 mesh:—

Product	Weight, per cent	Au, oz./ton	Per cent of values
+200	12·9	0·26	15·5
−200	87·1	0·21	84·5
Average tailing		0·216	

Recovery, per cent .... 83·8

Test No. 3—Amalgamation, Flotation, and Tabling—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
Flotation concentrate	14·5	1·14	68·0
Table concentrate	10·2	0·31	13·0
" middling	5·4	0·08	1·8
" tailing	52·5	0·063	13·6
" slimes	17·4	0·05	3·6
Table tailing screened on 200 mesh—			
+200	15·6	0·08	19·8
−200	84·4	0·06	80·2
Average		0·063	
Amalgamation tailing from products		0·243	

Recovery by amalgamation, per cent .... 81·6
Total recovery, table middling included in the tailing, per cent .... 96·5

Test No. 4—Flotation and Tabling—

Product	Weight, per cent	Assay		Per cent of values
		Au, oz./ton	Cu, per cent	
Flotation concentrate	12·1	5·63	0·98	79·1
Table concentrate	11·7	1·22		16·6
" middling	11·5	0·10		1·3
" tailing	52·7	0·042		2·6
" slimes	12·0	0·03		0·4
Table tailing screened on 200 mesh—				
+200	24·0	0·05		28·3
−200	76·0	0·04		71·7
Average		0·042		

Recovery in concentrate, per cent .... 95·7

Test No. 5—Flotation—

Product	Weight, per cent	Assay		Per cent of values
		Au, oz./ton	Cu, per cent	
Concentrate	12·2	8·86	0·92	80·7
Tailing	87·8	0·294		19·3
Tailing screened on 200 mesh—				
+200	18·7	1·27		80·7
−200	81·3	0·07		19·3
Average		0·294		

Experimental Tests on Ore from 3rd Level

Test No. 1—Amalgamation. Results of screen test on amalgamation tailing:—

Product	Weight, per cent	Assay, Au. oz./ton	Per cent of values
+100	7·6	0·09	8·2
+150	15·1	0·12	21·7
+200	16·1	0·09	18·6
−200	61·2	0·07	51·5
Average		0·083	

Recovery, per cent.......... 79·3

Test No. 2—Amalgamation. Results of screening tailing on 200 mesh:

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
+200	16·7	0·08	18·7
−200	83·3	0·07	81·3
Average		0·072	

Recovery, per cent.......... 82·0

Test No. 3—Amalgamation, Flotation, and Tabling—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
Flotation concentrate	5·4	0·86	39·0
Table concentrate	4·2	1·05	37·1
" middling	7·5	0·07	4·5
" tailing	54·4	0·032	14·6
" slimes	28·5	0·02	4·8
Table tailing screened on 200 mesh—			
+200	14·0	0·04	17·8
−200	86·0	0·03	82·2
Average		0·032	
Amalgamation tailing from products		0·119	

Recovery by amalgamation, per cent.......... 70·2
Total recovery, table middling included with the tailings, per cent.......... 93·0

Test No. 4—Flotation and Tabling—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
Flotation concentrate	3·0	5·82	46·1
Table concentrate	10·7	1·58	44·7
" middling	10·1	0·07	1·9
" tailing	54·2	0·043	6·1
" slimes	22·0	0·02	1·2
Table tailing screened on 200 mesh—			
+200	30·7	0·05	35·7
−200	69·3	0·04	64·3
Average		0·043	

Recovery in concentrates, per cent.......... 90·8

Test No. 5—Flotation—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
Flotation concentrate	4·0	5·16	47·4
Flotation tailing	96·0	0·239	52·6
Tailing screened on 200 mesh—			
+200	17·8	1·02	76·0
−200	82·2	0·07	24·0
Average		0·239	

Experimental Tests on the Composite Sample

Test No. 1—Amalgamation. Results of screen test on tailing:—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
+100	11·7	0·20	12·9
+150	15·3	0·22	18·6
+200	18·1	0·20	20·0
−200	54·9	0·16	48·5
Average		0·18	
Head, composite sample		1·10	

Recovery, per cent.. 83·5

Test No. 2—Amalgamation. Tailing screened on 200 mesh:—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
+200	13·9	0·54	35·2
−200	86·1	0·16	64·8
Average		0·213	

Recovery, per cent.. 80·6

Test No. 3—Amalgamation, Flotation, and Tabling—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
Flotation concentrate	9·4	1·32	58·6
Table concentrate	6·5	0·80	24·6
" middling	7·7	0·12	4·3
" tailing	58·4	0·033	9·1
" slimes	18·0	0·04	3·4
Table tailing screened on 200 mesh—			
+200	16·6	0·05	24·9
−200	83·4	0·03	75·1
Average		0·033	

Recovery by amalgamation, per cent.................................. 80·7
Total recovery, table middling included with tailing, per cent.......... 96·7

Test No. 4—Flotation and Tabling—

Product	Weight, per cent	Assay		Per cent of values
		Au, oz./ton	Cu, per cent	
Flotation concentrate	10·4	9·02	0·92	72·1
Table concentrate	6·4	5·10		25·2
" middling	7·9	0·11		0·7
" tailing	53·3	0·043		1·7
" slimes	22·0	0·02		0·3
Table tailing screened on 200 mesh—				
+200	33·9	0·05		39·2
−200	66·1	0·04		60·8
Average		0·043		

Recovery in concentrates, per cent.................................... 97·3

Test No. 5—Flotation—

Product	Weight, per cent	Assay		Per cent of values
		Au, oz./ton	Cu, per cent	
Concentrate	12·1	4·80	0·80	59·7
Tailing	87·9	0·447		40·3
Tailing screened on 200 mesh—				
+200	15·9	2·39		84·9
−200	84·1	0·08		15·1
Average		0·447		

Test No. 6—Two-cycle Cyanidation Test. Two charges each of 750 grammes of the composite sample were ground in a pebble mill in 1 : 1 pulp 0·025 per cent KCN and lime equivalent to 4 pounds per ton. The pulps were washed into agitators and diluted to 1 : 2·5 and agitated 48 hours using 0·05 per cent KCN solution. During agitation the cyanide solutions were sampled and strengthened to 0·05 per cent KCN twice daily. The protective alkalinity was maintained at 0·02 per cent CaO. The tailings were filtered, washed, repulped, filtered, and screened on 200 mesh. The solutions, exclusive of washings, were combined and analysed for copper.

Another lot of 750 grammes of composite sample was ground as in Cycle No. 1, using the combined solutions from Cycle No. 1, which contained 0·034 per cent KCN. The pulp was treated under the same conditions as in Cycle No. 1. The tailings were treated as in Cycle No. 1, and the solution, exclusive of washings, analysed for copper.

Results—Tailings screened on 200 mesh—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
Cycle No. 1, part 1:			
+200	2·5	9·25	85·6
−200	97·5	0·04	14·4
Average		0·27	
Cycle No. 1, part 2:			
+200	2·1	13·52	66·0
−200	97·9	0·15	34·0
Average		0·43	
Cycle No. 2:			
+200	2·8	9·12	55·5
−200	97·2	0·21	44·5
Average		0·46	

Summary—

Cycle	Au, oz./ton		Extraction, per cent	Consumption	
	Head	Tailing		Cyanide, lb./ton	Lime, lb./ton
No. 1, part 1	1·10	0·27	75·4	2·07	6·6
No. 1, part 2	1·10	0·43	60·8	2·17	6·4
No. 2	1·10	0·46	58·0	2·20	5·8

Analysis of cyanide solutions for copper—

Cycle No. 1	0·038 gramme Cu per litre
Cycle No. 2	0·073 " "

Test No. 7—Large-scale Cyanidation Test.—A lot of 116 pounds of the composite sample at −20 mesh was fed to a small rod mill in closed circuit with a classifier. The classifier overflow was pumped into a Pachuca tank and agitated for 36 hours in 0·05 per cent cyanide solution. The pulp density was about 1 : 3·25 and an average protective alkalinity of 0·02 per cent CaO was maintained. The grinding was done in 1 : 1 pulp, 0·03 per cent KCN. The tailings were filtered, washed, repulped, sampled in duplicate, and the samples screened on 200 mesh:—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
+200	14·7	0·05	17·8
−200	85·3	0·04	82·2
Average tailing		0·042	
+200	12·3	0·05	15·0
−200	87·7	0·04	85·0
Average tailing		0·041	
Head sample		1·12	
Tailing		0·042	

Extraction per cent .... 96·3

CONCLUSIONS

Amalgamation showed that about 80 per cent of the gold is recoverable by this method. The results are nearly the same at 5 per cent +100 as at 15 to 20 per cent +200 mesh grinding.

Flotation alone showed erratic results, the +200 mesh material being very high in gold on all three samples. This is largely due to the free gold being pounded out in the grinding rather than ground, as would be the case in large-scale operations. It is somewhat difficult to remove all the pyrrhotite by flotation. When flotation is succeeded by tabling, the gold that reported in the +200 flotation tailing reports in the table concentrate and a satisfactory recovery is obtained. Flotation preceded by amalgamation and followed by tabling gives a good recovery, but the grade of concentrate is low.

The small-scale cyanide tests were good indicators of the reagent consumption, and the amount of cyanide-soluble copper, but very unsatisfactory from the standpoint of extraction, especially in the +200 mesh material. The free gold was in too coarse a state to be attacked by the cyanide. This would not be the case in large-scale operations, as shown in Test No. 7, where 116 pounds of ore were used in the cyanide unit.

The cyanide consumption was about two pounds per ton, and although check results were obtained it is a little high, largely because the ore was over-ground, only 2·5 per cent remaining on 200 mesh. The soluble copper is almost negligible. After the first cycle it amounted to 0·038 gramme per litre, and at the end of the second cycle 0·073 gramme per litre, this latter quantity amounting to 0·146 pound per ton of solution should not present any serious trouble.

Report No. 271

THE CONCENTRATION AND RECOVERY OF VALUES CONTAINED IN A LEAD-ZINC-GOLD ORE FROM THE PREMIER GOLD MINING COMPANY, LTD., PREMIER, B.C.

A. K. Anderson

Shipment.—Two bags of Premier mill ore, gross weight 190 pounds, were received at the laboratories June 11, 1927.

Characteristics of Ore.—The shipment consisted of about 30 per cent mineral as heavy sulphides of lead, zinc, and iron intimately associated, carrying values in gold and silver.

Purpose of Tests.—Due to the increasing amounts of sulphides now being encountered in the Premier ore-bodies, bulk flotation produces a concentrate of too low a grade to be satisfactory as a lead or zinc concentrate and also too low in grade and bulky to be shipped for the recovery of the contained precious metals. The object of the following tests was to determine the adaptability of this ore to selective flotation to increase the ratio of concentration by dropping barren iron sulphides and to make a concentration of the gold and silver in a product which will have a commercial value in lead and zinc, and to indicate whether other methods of recovery will be of benefit.

Sampling and Analysis.—The two bags as received were crushed and screened through a 14-mesh screen, passed through a Jones riffle sampler and a head sample cut out which gave the following:—

Copper	0·18 per cent	Iron	9·68 per cent
Zinc	3·69 "	Gold	0·34 oz./ton
Lead	1·33 "	Silver	8·08 "

EXPERIMENTAL TESTS

For each of the following flotation tests 1,000 grammes of the −14 mesh material were ground in a porcelain jar containing iron balls and a volume of water equal in weight to the ore. As indicated, various reagents were added to the ball mill. Due to the intimate association of the sulphides and to the finely divided condition of the gold, the ore for all tests was ground to pass 150 mesh. Screen analyses made show the necessity of grinding to at least this fineness to free the minerals. The cyanide tests were conducted on portions weighing 750 grammes. These were ground to pass 200 mesh in a porcelain jar containing flint pebbles and a volume of cyanide solution equal in weight to the ore. For all tests the solution was freshly prepared and was equivalent in strength to 3 pounds KCN per ton. After grinding the pulp was transferred to a small mechanical agitator where it was diluted to 1: 2 ·5 and agitated for 24 hours. It was found necessary to wash the cyanide tailing prior to flotation by filtering and washing in a pressure filter. The filter cake was then repulped with water and passed to a small Ruth flotation machine in which all tests were conducted.

In Tests Nos. 1-7, bulk flotation of the lead and zinc sulphides was attempted under varying conditions. Table concentration also was incorporated in several of these.

Test No. 8 is a selective flotation of the lead and zinc sulphides. It also shows the association of the precious metals.

Test No. 9 shows the result of table concentration of the flotation tailing.

Tests Nos. 10-14 show results obtained by cyanidation and cyanidation followed by flotation.

Test No. 1

Reagents—

To ball mill—		To flotation machine—	
Barrett's No. 634	0·12 lb./ton	Copper sulphate	1·0 lb./ton
Soda ash	5·0 "	Pine oil	0·08 "

The object of this test was to produce a bulk concentrate containing the precious metals.

Product	Weight, per cent	Assay		Per cent of values	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Concentrate	27·67	1·00	19·84	88·5	88·1
Tailing	72·33	0·05	1·03	11·5	11·9

Ratio of concentration, 3·6 : 1.

This test shows that bulk flotation with the reagents used gives a low-grade bulky concentrate.

Test No. 2

Reagents—

To ball mill—		To flotation machine—	
Barrett's No. 634	0·12 lb./ton	Copper sulphate	1·0 lb./ton
Soda ash	5·0 "	Xanthate	0·2 "
Sodium cyanide	0·1 "	Pine oil	0·08 "

The object of this test was to determine the effect of cyanide as a depressant of the iron pyrite, and the action of xanthate as a lifter for sulphides reactivated by copper sulphate.

Product	Weight, per cent	Assay		Per cent of values	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Concentrate	27·33	0·94	18·86	85·4	84·6
Tailing	72·67	0·06	1·30	14·6	15·4

Ratio of concentration, 3·7 : 1.

This test shows that no benefit is derived by the addition of cyanide.

Test No. 3

Reagents—

To ball mill—		To flotation machine—	
Barrett's No. 634	0·12 lb./ton	Copper sulphate	1·0 lb./ton
Soda ash	5·0 "	Xanthate	0·2 "
		Pine oil	0·08 "

This test was run to determine the effect of the removal of part of the sulphides as a table concentrate. The ore was crushed dry to pass 60 mesh, 15 per cent remaining on 80 mesh. This coarser material was passed over a Wilfley table and the tailing product with the original −80 mesh material ground in the mill with the reagents.

Product	Weight, per cent	Assay		Per cent of values	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Flotation concentrate	24·62	1·00	23·52	69·9	79·0
Table concentrate	7·21	1·20	11·12	24·4	10·2
Tailing	68·17	0·03	1·09	5·7	10·8

Ratio of concentration, 3·1 : 1.

The results obtained show that table concentration of the coarser products increases the recovery of the precious metals over that secured by flotation alone. The two concentrates combined, however, give a product low in gold and silver.

Test No. 4

This test was similar to Test No. 3, with the exception that a greater percentage of the ore was taken for tabling, 38 per cent remaining on 80 mesh.

Product	Weight, per cent	Assay		Per cent of values	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Flotation concentrate	20·57	1·04	25·64	68·81	77·52
Table concentrate	5·78	1·17	10·85	21·86	9·17
Tailing	73·65	0·04	1·25	9·33	13·31

Ratio of concentration, 3·8 : 1.

The results obtained are similar to those of Test No. 3, showing that table concentration, while increasing recovery, is not entirely satisfactory.

Test No. 5

This test has for its object the checking of Test No. 8, run at an earlier date, to establish the fact that the iron sulphides carry appreciable amounts of gold, also to note any difference between the results obtained when using soda ash and lime as alkaline reagents.

Reagents—

To ball mill—

Barrett's No. 634	0·12 lb./ton
Sodium cyanide	0·15 "
Zinc sulphate	0·10 "
Soda ash	5·0 "

To flotation machine—

Copper sulphate	4·0 lb./ton
Xanthate	0·2 "
Pine oil	0·08 "

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
Concentrate	25·3	4·78	12·58	0·87	23·13	94·49	88·09	81·49	82·28
Tailing	74·7	0·10	0·57	0·07	1·69	5·51	11·91	18·51	17·72

Ratio of concentration, 3·9 : 1.

These results confirm those of Test No. 8. A high percentage of the gold remains with the iron pyrite in the tailing.

Test No. 6

This test was run to observe the effect of Aerofloat on the nature of the bulk concentrate produced by its use. The zinc and iron sulphides were depressed by the addition of lime and cyanide during grinding, the zinc then reactivated by copper sulphate, and Aerofloat used to float the sulphides.

Reagents—

To ball mill—		To flotation machine—	
Lime	4·0 lb./ton	Copper sulphate	1·0 lb./ton
Sodium cyanide	0·5 "	Aerofloat	0·08 "

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
Concentrate	9·86	11·57	24·23	2·16	55·82	84·08	74·03	68·17	72·85
Tailing	90·14	0·24	0·93	0·11	2·27	15·92	25·97	31·83	27·15

Ratio of concentration, 10·2 : 1.

This test shows that Aerofloat used as a flotation reagent produces a better grade of concentrate than that secured in previous tests. More gold and silver, however, remain in the tailing, due to the selective action of this reagent on zinc sulphides and on iron sulphides.

Test No. 7

This test is identical with No. 6, with the exception that soda ash was used in place of lime, at the rate of 4·0 pounds per ton.

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
Concentrate	13·20	10·11	24·33	2·07	47·01	96·90	98·67	84·93	82·92
Tailing	86·80	0·05	0·05	0·06	1·45	3·10	1·33	15·07	17·08

Ratio of concentration, 7·6 : 1.

This test shows that the use of soda ash gives a higher recovery of all the metals due to the less drastic depressing action of this reagent as compared with lime. The ratio of concentration, however, is lower, due to more of the iron pyrite being carried into the concentrate.

Test No. 8

The test was undertaken with the object of isolating, if possible, the gold and silver with one or another of the three sulphides present, those of lead, zinc, and iron. Selective flotation of the lead from the zinc and iron was followed, depressing the latter two by means of lime, cyanide, and zinc sulphate. The zinc was then reactivated by copper sulphate and floated, leaving the iron in the tailing.

Reagents—

To ball mill—

Barrett's No. 634......	0·12 lb./ton
Sodium cyanide.......	0·10 "
Zinc sulphate.........	0·20 "
Lime.................	4·0 "

To flotation machine—

Copper sulphate.......	2·0 lb./ton
Xanthate.............	0·2 "
Pine oil..............	0·04 "

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	7·16	13·29	9·26	1·96	52·86	69·9	17·9	46·5	52·9
Zinc concentrate.	9·76	3·43	24·73	1·04	20·66	24·3	65·3	33·5	28·1
Tailing..........	83·08	0·10	0·75	0·07	1·63	5·8	16·8	20·0	19·0

These results indicate that the gold and silver are associated with all three sulphides, the depressed iron pyrite carried into the tailing contained $1.40 per ton in gold. The concentrates are low in lead and zinc.

Test No. 9

This test was run to determine the result of passing the tailing from the flotation machine over concentrating tables.

Reagents—

To ball mill—

Barrett's No. 634......	0·12 lb./ton
Sodium cyanide.......	0·15 "
Zinc sulphate.........	0·10 "
Soda ash..............	10·0 "

To flotation machine—

Copper sulphate.......	4·0 lb./ton
Xanthate.............	0·20 "
Pine oil..............	0·08 "

The concentrates from the flotation machine and table were cleaned, a small percentage being taken as a finished concentrate to determine if a product high in gold could be obtained.

Product	Weight, per cent	Assay				Per cent of values			
		Pb, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Pb	Zn	Au	Ag
Flotation concentrate....	6·61	10·97	24·33	1·12	38·42	57·04	49·92	23·20	35·97
Flotation middling......	14·07	2·83	7·67	0·74	18·04	31·31	33·49	32·61	35·97
Table concentrate....	1·09	0·78	6·27	3·22	19·47	0·62	2·12	10·98	2·97
Table middling.	4·13	0·70	1·95	2·03	12·01	2·27	2·52	26·32	7·08
Tailing..........	74·10	0·15	0·52	0·03	1·71	8·76	11·95	6·89	18·01

The flotation concentrate thus produced is no higher in gold than that from preceding tests, although the ratio of concentration, 15·1 : 1, is higher. An additional 37 per cent of the gold is contained in the table concentrate and middling, weighing 5·2 per cent of the total. The combined flotation and table concentrates give a low-grade product.

Test No. 10

This test was run to determine what extraction of the gold and silver would result from the cyanidation of the ore. No attempt was made to recover the lead and zinc. The ore after grinding in cyanide solution, (3 pounds KCN per ton of solution) together with 3 pounds lime per ton of ore was diluted to 1 : 2½ with cyanide solution of the same strength and agitated for 24 hours.

Cyanide consumption 2·2, lime consumption 2·1 pounds per ton.

Assay of tailing		Recovery, per cent	
Au, oz./ton	Ag, oz./ton	Au	Ag
0·08	4·06	76·5	49·7

A screen analysis of the tailing showed 36·3 per cent remaining on 150 mesh, indicating that finer crushing might be advantageous.

Test No. 11

To investigate the effect of fine grinding, this test was run under same conditions as Test No. 10. A screen analysis of tailing showed 98·5 per cent passing 200 mesh. After cyanidation the tailing was passed over a table to determine if an increased recovery could be made by tabling.

Product	Weight, per cent	Assay		Per cent of values	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Table concentrate	10·3	0·14	5·80	2·94	7·42
Table tailing	89·7	0·01	3·41	2·65	37·86
Cyanidation				94·41	54·72

As shown, finer grinding yields a recovery of 94 per cent of the gold and 54·7 per cent of the silver. Tabling of the cyanide tailing yields a low-grade product assaying $2 in gold and 5·8 ounces silver per ton.

Test No. 12

Having found that the gold can be extracted by cyanidation, this test was conducted to determine if a recovery of the zinc and lead contained in the ore can be effected by flotation. The cyanide procedure was identical with that of Test No. 11. After agitation the tailing was filtered, washed with water, repulped with water, and a selective float made, using the following reagents:—

Sodium cyanide	0·2 lb./ton
Copper sulphate	2·0 "
Aerofloat	0·14 "

Consumption of cyanide	2·8 lb./ton
Consumption of lime	3·7 "

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	2·22	44·35	10·08	0·40	72·60	66·71	6·77	2·59	19·92
Lead middling..	9·12	2·84	4·67	0·04	10·28	18·82	12·87	1·06	11·63
Zinc concentrate.	3·53	0·93	58·94	0·04	7·44	2·39	62·89	0·41	3·22
Zinc middling...	5·88	1·21	8·62	0·04	7·60	5·16	15·32	0·68	5·57
Tailing..........	79·25	0·12	0·09	0·02	0·98	6·92	2·14	4·64	0·28
Cyanidation.....	....	....	....	....	....	....	....	90·62	50·38

This test shows that the gold and part of the silver can be recovered by cyanidation. A marketable lead concentrate can be made which contains an additional 2·5 per cent of the gold and 20 per cent of silver. A concentrate high in zinc can be obtained also.

Test No. 13

To determine whether it is necessary to wash the tailing after cyanidation this test was run under conditions similar to the previous one, but omitting the washing of the filtered tailing.

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	11·97	10·46	26·32	0·33	25·95	87·25	94·97		
Zinc concentrate.	11·82	0·91	0·84	0·03	1·71	7·45	2·96		
Tailing..........	76·21	0·10	0·09	0·05	0·69	5·30	2·07		

The resulting froth was very heavy, white in colour, and continued to rise as long as agitation was continued. There was no selective action, the lead concentrate containing most of the zinc. It appears that it is necessary to wash the tailing from cyanidation before attempting flotation.

Test No. 14

This test is a duplication of Test No. 12. Grinding —200 mesh. Consumption of cyanide 2·6, and lime 4·05 pounds per ton.

Flotation Reagents—

Sodium cyanide	0·20 lb./ton
Aerofloat	0·14 "
Copper sulphate	2·0 "

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	4·19	17·71	46·29	0·03	49·47	52·81	53·36	0·29	25·62
Lead middling..	5·96	7·92	20·15	0·04	16·58	33·59	33·40	0·59	12·25
Zinc concentrate.	2·92	1·86	3·43	0·01	3·94	3·84	2·78	....	1·36
Zinc middling...	21·03	0·40	0·91	0·02	1·12	5·98	5·31	1·18	2·97
Tailing..........	65·90	0·08	0·28	0·01	0·45	3·73	5·15	1·76	3·58
Cyanidation.....	....	....	....	....	....	....	....	96·18	54·22

This test, although run under conditions similar to those of Test No. 12, did not yield the same results by flotation. It will be noted that whereas in the former test high-grade lead and zinc concentrates were obtained, this test gave a lead concentrate high in zinc and carrying 25·6 per cent of the total silver. The recovery of gold by cyanidation bears out the former conclusion that the metal can be recovered by this method. As an explanation of the variations in flotation results may be offered the fact that considerable time elapsed between the tests, and during this period the ore was oxidizing rapidly as shown by the increased consumption of lime in the cyaniding. It has been found from tests on other sulphide ores that results secured on freshly ground material can not be duplicated once the ore has been exposed to the air for any length of time.

Comparison of Results

In calculating the per cent recovery in all these tests no attempt was made to eliminate the middling product. From tests on other ores, at least 50 per cent of the contained values in the middling may be recovered.

Test	Ratio of concentration	Assay		Recovery		Reagents
		Au, oz./ton	Ag, oz./ton	Au, per cent	Ag, per cent	
No. 1	3·6 : 1	1·00	19·84	88·5	88·1	Oils, Potassium xanthate, Pine oil.
No. 2	3·7 : 1	0·94	18·86	85·4	84·6	
No. 3, Flotation	3·1 : 1	1·00	23·52	69·9	79·0	
Table		1·20	11·12	24·4	10·2	
No. 4, Flotation	3·8 : 1	1·04	25·64	68·8	77·5	
Table		1·17	10·85	21·9	0·2	
No. 5	3·9 : 1	0·87	23·13	81·5	82·3	Aerofloat.
No. 6	10·2 : 1	2·16	55·82	68·2	72·8	
No. 7	7·6 : 1	2·07	47·01	84·9	82·9	

In the above tests it appears that a high concentration of the gold and silver cannot be secured in a bulk flotation concentrate. The ratio of concentration is low due to the gold being associated with all three sulphides, those of lead, zinc, and iron. This is shown in Test No. 8 in which 46·5, 33·5, and 22 per cent of the gold is contained in the lead concentrate, zinc concentrate, and tailing respectively.

Table concentration combined with flotation, as shown in Tests Nos. 3, 4, and 9, gives a higher recovery than flotation alone. In Test No. 9, the tailing from the flotation machine passed over a concentrating table yields a concentrate 1·1 per cent of the total weight, assaying 3·22 ounces gold per ton and containing 11 per cent of the total gold. This combined with the concentrate secured by the previous flotation operations yields a low-grade product.

Test Nos. 10 and 11 show the advantage of fine grinding.

Test	Grinding	Recovery, per cent	
		Au	Ag
No. 10	63·7 per cent—150 mesh	76·5	49·7
No. 11	98·5 " —200 "	94·4	54·7

Tests Nos. 11, 12, and 14 show that the gold and silver can be recovered.

Test	Recovery, per cent					
	By cyanidation		By flotation		Total	
	Au	Ag	Au	Ag	Au	Ag
No. 11	94·4	54·7				
No. 12	90·6	50·4	2·59	19·9	93·2	70·3
No. 14	96·2	54·2	0·30	25·6	95·5	79·8

These tests show that a good recovery of the gold can be secured by cyanidation. Most of the silver not extracted by cyanide is recovered in a flotation concentrate high enough in lead and zinc to have a marketable value.

SUMMARY AND CONCLUSIONS

It is apparent that a bulk concentrate yielding the highest recovery of gold and silver cannot be expected. To recover these values it is necessary to float a large part of the iron pyrites which carries appreciable amounts of the precious metals. The iron pyrites passing into the concentrate along with the lead and zinc sulphides lowers the grade of the product. The resulting concentrate is therefore bulky and low in all metal contents.

The brief study of cyanidation shows encouraging results. Fine grinding appears to be necessary. No difficulty was encountered in securing a recovery of from 94 to 96 per cent of the gold content and approximately 50 per cent of the silver. Flotation of the residues from cyanidation indicates possibilities of a further recovery of from 25 to 30 per cent of the silver. It also reveals that the lead and zinc can be recovered in a concentrate of commercial value. One point quite apparent is the absolute necessity of washing the residues from the cyanide treatment prior to flotation. In a commercial operation, this would entail double filtration, repulping the discharge of the first filter in water, and refiltering. If this point be neglected, a voluminous low-grade froth will result which will be exceedingly difficult to control. As indicated in the flotation of the cyanide tailing, conditions giving best results must be carefully established. This will include a study of the cyanide practice in order to produce a residue suitable for the selective flotation of lead and zinc.

To establish these conditions it is recommended that a small pilot plant of about 50 tons capacity be operated on the property. The practice indicated would include fine grinding to pass 200 mesh, cyanidation with various strengths of cyanide solution, washing of residues by double filtration, followed by flotation under conditions to produce suitable grades of lead and zinc concentrates.

Report No. 272

THE SELECTIVE FLOTATION OF A COPPER-ZINC ORE FROM THE SHERRITT-GORDON MINES LIMITED, MANITOBA

A. K. Anderson

Shipments.—Three lots of ore, designated as Shipments Nos. 1, 2, and 3, were received as follows: Shipment No. 1, gross weight 200 pounds, August 25, 1927. Shipment No. 2, consisting of three lots, total weight 109 pounds, arrived from the Temiskaming Laboratories, Cobalt, on October 25, 1927. Shipment No. 3, also from the Temiskaming Laboratories, arrived November 18, 1927. This had a gross weight of 780 pounds and consisted of twenty-two separate lots.

Characteristics of Ore.—All three shipments contained a high percentage of sulphides of copper, zinc, and iron. The ore, when received, had been crushed to a fairly fine state of division, thus hastening oxidization. Shipment No. 1, on which the major number of tests were performed, appeared to have been crushed for some time. The three lots of Shipment No. 2, weighing 36, 36, and 37 pounds respectively, were distinctive in appearance; the first was quite brown and contained a high percentage of zinc and iron sulphides; the second and third lots were of the characteristic colour of chalcopyrite. The twenty-two individual lots of Shipment No. 3, similar in appearance and sulphide content to Shipment No. 1, were combined and thoroughly mixed together.

Purpose of Tests.—The object of the tests was to separate the copper and zinc by selective flotation. A copper concentrate containing above 22 per cent copper, and one of zinc above 48 per cent zinc were desired, with a minimum loss in the tailing.

Sampling and Analysis.—Each shipment as received was crushed and screened to pass 14 mesh, thoroughly mixed, passed through a Jones riffle sampler, and a head sample cut out. Analyses of the lots are as follows:—

Shipment	Cu, per cent	Zn, per cent	Pb, per cent	Fe, per cent	Au, oz./ton	Ag, oz./ton
No. 1	3·12	4·63	Nil	....	0·02	0·72
No. 2 (combined)	7·64	13·15	....	25·8	0·05	3·17
No. 2, Lot No. 1	0·53	21·39	....	....	tr.	0·88
No. 2, Lot No. 2	10·45	8·14	....	....	0·08	3·74
No. 2, Lot No. 3	11·61	8·77	....	....	0·06	3·92
No. 3	5·55	7·02	....	....	0·05	2·30

Most of the test work was on Shipment No. 1. For each test 1,000 grammes of the −14 mesh material were ground to pass 100 mesh in a porcelain jar containing iron balls and a weight of water equal to that of the ore. As indicated in the tests, various reagents were also added while grinding. The pulp was then transferred to a small Ruth laboratory flotation machine in which all tests were made.

Shipment No. 1: Tests Nos. 1 to 7 were conducted to investigate the effect of various reagents on the recovery of copper, and the results secured when using potassium xanthate for the recovery of zinc. Tests Nos. 8 to 17 were to determine the effect of Aerofloat on the recovery of copper and zinc, using various conditioning reagents.

Shipment No. 2: Tests Nos. 1 to 3 were run to determine if this shipment were amenable to the same treatment as Shipment No. 1.

Shipment No. 3: Tests Nos. 1 to 3 had for their object the investigation of the behaviour of this ore under conditions similar to those of preceding tests. The tests in detail follow:—

SHIPMENT NO. 1

Test No. 1

This test was run to determine the results of depressing the zinc and iron sulphides by the use of cyanide in a circuit made alkaline with soda ash and floating the copper with Aerofloat. Lime was then added and the zinc reactivated by copper sulphate and floated with potassium xanthate and pine oil.

Reagents—

To ball mill—		To flotation machine—	
Sodium cyanide......	0·15 lb./ton	Lime..................	2·0 lb./ton
Aerofloat.............	0·12 "	Copper sulphate.......	5·0 "
Soda ash.............	5·0 "	Xanthate.............	0·3 "
		Pine oil...............	0·06 "

Results—

Product	Weight, per cent	Analysis				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate....	12·42	19·06	7·20	0·12	2·92	78·64	20·14	78·95	56·35
Copper middling.	16·42	0·98	4·31	0·02	0·58	5·35	15·93	15·79	14·75
Zinc concentrate.	11·80	2·55	21·16	0·01	0·53	10·10	56·26	5·26	9·63
Zinc middling...	8·43	0·86	2·28	tr.	0·29	2·39	4·32	....	3·74
Tailing..........	50·93	0·21	0·29	tr.	0·20	3·52	3·35	....	15·53
Heads..........	....	3·12	4·63	0·02	0·72				

This test indicates that a selective separation of the copper and zinc is possible.

Test No. 2

This test had for its object the observing of the effect of lime added to the pulp prior to the floating of copper. Other conditions were the same as in Test No. 1.

Reagents—

To ball mill—		To flotation machine—	
Sodium cyanide......	0·15 lb./ton	Copper sulphate.......	5·0 lb./ton
Soda ash.............	5·0 "	Xanthate.............	0·3 "
Lime.................	2·0 "	Pine oil...............	0·06 "
Aerofloat.............	0·12 "		

Results—

Product	Weight, per cent	Analysis				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate....	11·41	18·50	5·55	0·12	1·93	68·21	15·23	82·35	41·51
Copper middling.	16·86	4·00	4·72	0·02	0·91	21·78	19·16	17·65	28·30
Zinc concentrate.	8·55	0·77	26·98	trace	0·42	2·13	55·52	....	7·55
Zinc middling...	8·24	0·82	3·68	trace	0·30	2·19	7·31	....	3·77
Tailing..........	54·94	0·32	0·21	trace	0·19	5·69	2·78	....	18·87
Head...........	....	3·12	4·63	0·02	0·72				

This test shows that lime when used in the circuit before the copper has been removed has a depressing action on that mineral. The copper concentrate is low-grade and a large amount of middling product is also produced. The zinc concentrate is slightly higher than in the preceding test and contains less copper.

Test No. 3

It will be noted that the gold and the greater part of the silver is associated with the copper sulphides. As the ore contains considerable iron sulphides, it is apparent that to raise the grade of the copper and zinc concentrates it is necessary to eliminate as much of the barren iron sulphide as possible. This test was run under conditions which normally produce such results, depressing the zinc and iron sulphides by lime and cyanide and using thiocarbanilide and Aerofloat to float the copper.

Reagents—

To ball mill—		To flotation machine—	
Lime................	8·0 lb./ton	Aerofloat............	0·1 lb./ton
Sodium cyanide......	0·15 "	Copper sulphate......	5·0 "
Thiocarbanilide......	0·10 "	Xanthate............	0·2 "
		Pine oil..............	0·06 "

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu per cent	Zn per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate....	8·62	24·20	4·05	0·16	3·72	67·20	7·99	73·68	8·51
Copper middling.	4·95	4·84	5·81	0·05	1·45	7·72	6·57	10·53	19·15
Zinc concentrate.	11·99	4·50	27·59	0·02	0·90	17·36	75·73	10·54	28·72
Zinc middling...	7·50	1·48	4·05	0·01	0·58	3·54	6·96	5·25	11·71
Tailing..........	66·94	0·20	0·18	....	0·18	4·18	2·75	....	31·91

In this test a satisfactory grade of copper concentrate was secured, although the recovery was not high, 17 per cent being contained in the zinc concentrate, due to the depressing action of lime on the copper sulphides. The recovery of zinc in a low-grade concentrate is higher than in previous tests.

Test No. 4

This test was run to note the effect of zinc sulphate combined with cyanide to form zinc cyanide. An increased amount of lime was used so that the effect of higher alkalinity might be observed. Sodium dichromate also was added to see if that reagent would benefit the grade of zinc concentrate.

Reagents—

To ball mill—			To flotation machine—		
Sodium cyanide......	0·2	lb./ton	Aerofloat............	0·1	lb./ton
Lime................	12·0	"	Copper sulphate......	3·0	"
Thiocarbanilide......	0·1	"	Xanthate............	0·2	"
Zinc sulphate.........	0·15	"	Pine oil............	0·06	"
			Sodium dichromate..	0·1	"

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate....	7·94	24·08	3·84	0·18	3·52	62·44	7·00	70·0	53·04
Copper middling.	4·97	3·33	5·71	0·04	0·86	5·42	6·46	10·0	8·17
Zinc concentrate.	19·38	3·82	16·68	0·02	0·52	24·16	73·66	20·0	19·20
Zinc middling...	20·05	0·91	2·49	....	0·18	5·95	11·36	....	6·85
Tailing..........	47·66	0·13	0·14	....	0·14	2·03	1·52	....	12·74

The results from this test were similar to Test No. 3, a satisfactory grade of copper concentrate was obtained. The higher alkalinity due to lime caused an increased amount of copper in the zinc concentrate. A lower grade zinc concentrate and a lower recovery also resulted. No noticeable advantage was derived from the use of sodium dichromate.

Test No. 5

The object of this test was to note the difference in results caused by the use of soda ash instead of lime.

Reagents—

To ball mill—			To flotation machine—		
Soda ash.............	12·0	lb./ton	Aerofloat............	0·08	lb./ton
Sodium cyanide......	0·4	"	Copper sulphate......	3·0	"
Zinc sulphate.........	0·3	"	Xanthate............	0·2	"
Thiocarbanilide......	0·2	"	Pine oil..............	0·06	"

Results—

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Zn, per cent	Cu	Zn
Copper concentrate........................	14·18	19·26	7·64	91·46	24·27
Copper middling...........................	16·87	0·35	4·98	1·97	18·82
Zinc concentrate...........................	4·63	1·00	42·04	1·54	43·62
Zinc middling..............................	3·45	0·81	5·71	0·91	4·42
Tailing....................................	60·87	0·20	0·65	4·09	8·87

This test showed that soda ash gives a higher recovery of copper as copper concentrate. However, it is lower in grade and contains more zinc than that secured when lime was used as the alkaline reagent. A large weight of middling product was produced. The zinc concentrate was much higher in grade than any previously obtained. However, due to the zinc in the copper concentrate, the recovery is low.

Test No. 6

This test was run with a higher alkalinity of soda ash, to note any variations caused.

Reagents—

To ball mill—		To flotation machine—	
Soda ash.............	16·0 lb./ton	Aerofloat.............	0·08 lb./ton
Zinc sulphate.........	0·3 "	Copper sulphate......	3·0 "
Sodium cyanide......	0·4 "	Xanthate............	0·2 "
Thiocarbanilide.......	0·2 "	Pine oil..............	0·06 "

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate....	13·34	20·32	6·49	0·14	2·91	89·62	19·82	82·61	74·33
Copper middling.	14·69	0·84	4·41	0·03	0·31	4·07	14·85	17·39	8·62
Zinc concentrate.	4·73	0·97	50·09	nil	0·32	1·52	54·29	....	2·87
Zinc middling...	2·79	1·03	7·61	nil	0·37	0·95	4·86	....	1·91
Tailing..........	64·45	0·18	0·42	nil	0·10	3·84	6·18	....	12·27

These results show a higher recovery of zinc than was obtained in previous tests. The concentrate was of a suitable grade, but a larger percentage of the zinc passed out with the tailing, probably due to the increase in alkalinity. The copper recovery was lower than in the preceding test, more mineral passing into the middling.

Test No. 7

In the previous tests Aerofloat and thiocarbanilide were used in the flotation of copper. The object of this test was to observe if Aerofloat was necessary.

Reagents—

To ball mill—		To flotation machine—	
Soda ash.............	16·0 lb./ton	Copper sulphate.......	3·0 lb./ton
Zinc sulphate.........	0·4 "	Potassium xanthate....	0·2 "
Sodium cyanide......	0·5 "	Pine oil...............	0·06 "
Thiocarbanilide......	0·3 "		

Results—

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Zn, per cent	Cu	Zn
Copper concentrate	10·74	23·64	4·67	85·28	11·51
Copper middling	7·38	1·70	5·81	4·19	9·84
Zinc concentrate	6·50	1·73	42·95	3·76	64·02
Zinc middling	10·76	0·86	4·67	3·10	11·51
Tailing	64·62	0·17	0·21	3·67	3·12

This test shows that although a satisfactory grade of copper concentrate was made more of the copper was left to be floated with the zinc. Aerofloat apparently is of benefit. A higher recovery of zinc was obtained, but in a lower grade concentrate.

Test No. 8

The object of this test was to determine whether the addition of lime to the flotation pulp after the copper had been removed, would depress the iron pyrite which in previous tests lowered the grade of zinc concentrate. It was observed in the foregoing tests that potassium xanthate caused large amounts of iron sulphides to float with the zinc. The use of that reagent was, therefore, discontinued and Aerofloat used.

Reagents—

To ball mill—		To flotation machine—	
Soda ash	16·0 lb./ton	Lime	6·0 lb./ton
Sodium cyanide	0·5 "	Copper sulphate	3·0 "
Zinc sulphate	0·4 "	Aerofloat	0·06 "
Thiocarbanilide	0·3 "		

Results—

Product	Weight, per cent	Analysis		Per cent of values	
		Cu, per cent	Zn, per cent	Cu	Zn
Copper concentrate	11·42	23·74	5·19	88·62	13·61
Copper middling	12·08	1·23	5·14	4·87	14·25
Zinc concentrate	6·17	1·00	46·01	2·03	65·16
Zinc middling	13·39	0·47	1·51	2·05	4·64
Tailing	56·94	0·13	0·18	2·43	2·34

This test shows that a high recovery of copper in a suitable grade of concentrate can be obtained. The use of Aerofloat as a reagent for floating the zinc gave slightly better results than in previous tests. A large amount of zinc middling, low in value, is produced. The copper concentrate was found to contain 0·05 per cent lead.

Test No. 9

This test is similar to Test No. 8, with the exception that no lime was added to the pulp after removal of the copper concentrate.

Reagents—

To ball mill—		To flotation machine—	
Soda ash	16·0 lb./ton	Copper sulphate	3·0 lb./ton
Sodium cyanide	0·5 "	Aerofloat	0·06 "
Zinc sulphate	0·4 "		
Thiocarbanilide	0·3 "		

Results—

Product	Weight, per cent	Analysis		Per cent of values	
		Cu, per cent	Zn, per cent	Cu	Zn
Copper concentrate	10·81	25·01	4·46	86·95	10·65
Copper middling	6·98	1·53	5·86	3·44	9·03
Zinc concentrate	7·19	1·52	44·28	3·50	70·32
Zinc middling	17·84	0·52	1·61	2·99	6·34
Tailing	57·18	0·17	0·29	3·12	3·66

The copper recovery was the same as in the previous test, any variation in results being due to differences in manipulation. The absence of lime appears to be detrimental as a lower grade of concentrate was obtained and a larger weight of middling produced. A higher total recovery of zinc was obtained due to the fact that less zinc floated with the copper.

Test No. 10

As somewhat over 10 per cent of the zinc reported with the copper middling, an attempt was made in this test to recover this. The middling product resulting from cleaning the rougher copper concentrate was treated with copper sulphate to reactivate the zinc mineral and floated with Aerofloat.

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate	9·36	24·82	3·89	0·11	3·35	78·06	8·23	52·63	58·18
Zinc concentrate	6·35	2·68	44·28	0·06	0·76	5·71	63·56	21·05	8·92
Concentrate from copper middling	2·47	3·71	5·39	0·03	0·89	3·09	3·00	5·26	4·09
Middling " "	3·49	4·26	6·28	0·06	0·82	5·00	4·95	10·53	5·39
Zinc middling	12·76	0·97	3·58	0·02	0·27	4·16	10·33	10·53	6·31
Tailing	65·57	0·18	0·67	nil	0·14	3·98	9·93	....	17·11

The zinc contained in the copper middling did not respond to selective flotation, possibly because it was too intimately associated with the copper mineral.

Test No. 11

TT mixture was used in place of thiocarbanilide in this test. The rougher concentrates were not cleaned.

Reagents—

To ball mill—		To flotation machine—	
Lime	4·0 lb./ton	TT mixture	0·06 lb./ton
Sodium cyanide	0·3 "	Copper sulphate	3·0 "
		Aerofloat	0·06 "

Results—

Product	Weight, per cent	Analysis		Per cent of values	
		Cu, per cent	Zn, per cent	Cu	Zn
Copper concentrate	16·47	16·24	4·94	87·36	18·48
Zinc concentrate	1·47	2·95	5·90	1·41	1·97
Tailing	82·06	0·42	4·27	11·23	79·55

The conditions were not right for good results. The zinc refused to float, only a thin black froth resulting after the copper had been removed.

Test No. 12

This test was run to note results when using soda ash instead of lime, under conditions similar to those of Test No. 11. Soda ash, 16 pounds per ton, was added to ball mill.

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate	7·20	26·06	3·34	0·08	5·58	63·83	5·37	26·09	57·14
Copper middling	9·47	7·15	5·97	0·06	1·36	23·04	12·64	26·09	18·57
Zinc concentrate	4·93	1·42	50·19	0·06	0·63	2·38	55·34	13·05	4·28
Zinc middling	19·96	1·00	4·98	0·04	0·40	6·77	22·23	34·77	11·43
Tailing	58·44	0·20	0·34	Nil	0·11	3·98	4·42	....	8·58

The effect of soda ash is quite apparent. A clean copper concentrate of good grade was secured. Although the recovery was not so high as in previous tests, the copper in the copper middling could doubtless be recovered under correct conditions. The grade of zinc concentrate was satisfactory, although recovery was low. A large quantity of zinc middling was again produced.

In all tests performed the chief difficulty was in making a high-grade concentrate with a good recovery of the zinc. To secure a tailing low in zinc it was necessary to float a large bulk of mineral and this contained a high percentage of iron sulphide. On cleaning this product a large quantity of middling was produced.

Test No. 13

This test was run to note the effect of using Aerofloat in place of TT mixture.

Reagents—

To ball mill—		To flotation machine—	
Lime	4·0 lb./ton	Aerofloat	0·12 lb./ton
Zinc sulphate	0·3 "	Copper sulphate	3·0 "
Sodium cyanide	0·3 "		

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate	9·95	23·46	4·26	0·16	3·08	76·82	9·89	84·21	53·40
Copper middling...	4·96	3·46	4·57	0·04	0·86	5·63	5·28	10·53	7·33
Zinc concentrate...	5·70	2·99	49·68	0·01	1·05	5·59	66·21	5·26	10·47
Zinc middling.....	7·00	1·99	6·44	tr.	0·50	4·58	10·52	....	6·11
Tailing...........	72·39	0·31	0·48	Nil	0·18	7·38	8·10	....	22·69

This test bears out the conclusion previously arrived at that lime has a depressing action on the copper sulphides, as more reported in the tailing than in Test No. 12. The recovery as copper concentrate was higher as a greater weight was obtained than in the former test. A higher recovery of zinc also was effected with a lower grade, less bulky middling product, although the zinc content of the tailing was somewhat higher.

Test No. 14

To check previous observations that lime is detrimental to copper recovery, this test was run under conditions similar to Test No. 13, but with increased lime.

Reagents—

To ball mill—		To flotation machine—	
Lime	8·0 lb./ton	Aerofloat	0·12 lb./ton
Sodium cyanide	0·3 "	Copper sulphate	3·0 "
Zinc sulphate	0·3 "		

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate	8·76	20·62	3·94	0·16	2·92	58·33	7·81	70·00	53·44
Copper middling...	5·54	2·98	5·81	0·01	0·66	5·33	7·29	5·00	7·51
Zinc concentrate...	7·71	9·37	35·73	0·03	1·07	23·32	62·38	10·00	17·12
Zinc middling......	17·41	1·76	5·14	0·02	0·36	9·89	20·31	15·00	13·15
Tailing...........	60·58	0·16	0·16	Nil	0·07	3·13	2·21	....	8·78

This shows that flotation of this ore in a circuit high in lime is not the best practice. The copper was depressed and on being reactivated by copper sulphate floated with the zinc, thus lowering the grade of that product. These conditions also produced a large amount of zinc middling.

Test No. 15

In order to note any differences caused by a decreased amount of sodium cyanide, this test was run under conditions similar to Test No. 13.

Reagents—

To ball mill—		To flotation machine—	
Lime	4·0 lb./ton	Aerofloat	0·14 lb./ton
Sodium cyanide	0·1 “	Copper sulphate	1·0 “
Zinc sulphate	0·1 “		

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate	9·50	25·06	6·07	0·14	3·32	79·82	13·29	50·00	66·59
Copper middling	9·58	4·20	5·86	0·05	0·77	13·48	12·92	19·23	15·64
Zinc concentrate	6·08	0·76	43·06	0·02	0·30	1·54	60·32	3·84	3·81
Zinc middling	32·25	0·28	1·43	0·01	0·06	3·02	10·63	11·54	4·02
Tailing	42·59	0·15	0·29	0·01	0·11	2·14	2·84	15·39	9·94

The necessity of adding sufficient cyanide to depress the zinc minerals is clearly indicated. More zinc was floated with the copper and also a larger amount of iron floated with the zinc which, on cleaning, yielded a large amount of zinc middling.

Test No. 16

An attempt was made in this test to cut down the amount of iron floated with the zinc by the addition of lime after the copper had been floated.

Reagents—

To ball mill—		To flotation machine—	
Lime	4·0 lb./ton	Lime	4·0 lb./ton
Sodium cyanide	0·1 “	Copper sulphate	1·0 “
Zinc sulphate	0·1 “	Aerofloat	0·14 “

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate	10·94	23·40	6·33	0·14	3·04	83·88	16·26	55·56	69·37
Copper middling	11·62	2·72	5·24	0·05	0·56	10·35	14·30	22·22	13·54
Zinc concentrate	5·96	0·78	41·23	0·01	0·27	1·51	57·70	3·70	3·33
Zinc middling	21·49	0·28	1·35	trace	0·12	1·96	6·82	....	5·42
Tailing	49·99	0·14	0·42	0·01	0·08	2·30	4·92	18·52	8·34

This shows that lime added after copper has been floated decreased the bulk of zinc middling produced. However, the amount of zinc passing into the tailing was also increased.

Test No. 17

This test was run to note any benefit derived from an increase in the amount of cyanide added.

Reagents—

To ball mill—			To flotation machine—	
Lime	4·0 lb./ton		Copper sulphate	1·0 lb./ton
Sodium cyanide	0·4 "		Aerofloat	0·08 "
Zinc sulphate	0·3 "			

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate....	13·90	18·58	5·19	0·12	2·46	84·00	16·59	73·91	79·35
Copper middling.	5·45	3·17	4·62	0·04	0·48	5·63	5·79	8·70	6·03
Zinc concentrate.	8·68	1·39	32·17	0·01	0·28	3·93	64·24	4·35	5·56
Zinc middling...	26·89	0·45	1·56	0·01	0·08	3·93	9·64	13·04	4·88
Tailing..........	45·08	0·17	0·36	trace	0·04	2·51	3·74	....	4·17

No apparent benefit results from an increased addition of cyanide. From the results obtained in the last few tests may be drawn the conclusion that the zinc is in a condition which makes it difficult to obtain a good grade of concentrate with a corresponding high recovery. This is doubtless due to excessive oxidization taking place, as the acidity showed a consistent increase.

SHIPMENT NO. 2

Test No. 1

This test was run on Lot No. 1 of this shipment. The material was high in zinc and low in copper. The object was to determine if this ore behaved similarly to that of Shipment No. 1.

Reagents—

To ball mill—			To flotation machine—	
Lime	4·0 lb./ton		Aerofloat	0·10 lb./ton
Sodium cyanide	0·2 "		Copper sulphate	1·0 "
Zinc sulphate	0·2 "			

Results—

Product	Weight, per cent	Analysis		Per cent of values	
		Cu, per cent	Zn, per cent	Cu	Zn
Head..........	....	0·53	21·39		
Copper concentrate..........	0·83	23·32	10·70	34·11	0·40
Copper middling..........	4·51	6·03	18·13	47·80	3·75
Zinc concentrate..........	33·06	0·21	55·84	12·13	84·65
Zinc middling..........	12·10	0·20	18·44	4·20	10 24
Tailing..........	49·50	0·02	0·42	1·76	0·96

These results indicate that this class of material is similar to the first shipment. The copper minerals float quite readily and the zinc also can be recovered in a high-grade concentrate.

Test No. 2

This test was run on Lot No. 2 of this shipment, the material containing much more copper and less zinc than that of Lot No. 1. The object was to determine if this class of ore acts similarly to Shipment No. 1.

Reagents—

To ball mill—		To flotation machine—	
Lime...............	4·0 lb./ton	Aerofloat...............	0·1 lb./ton
Sodium cyanide.....	0·2 "	Copper sulphate.........	1·0 "
Zinc sulphate.......	0·2 "		

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Head............	....	10·45	8·14	0·08	3·74				
Copper concentrate....	27·76	30·86	2·92	0·10	6·02	82·28	9·72	36·37	57·78
Copper middling.	20·25	7·75	17·56	0·12	3·02	15·07	42·66	31·17	21·11
Zinc concentrate.	8·06	0·71	42·06	0·03	1·03	0·55	40·67	2·59	2·77
Zinc middling...	10·38	0·92	3·81	0·06	2·08	0·91	4·86	7·79	7·61
Tailing..........	33·55	0·37	0·52	0·05	0·93	1·19	2·09	22·08	10·73

These results indicate that this ore is of much the same nature as the preceding shipment. A selective flotation of the copper and zinc is possible.

Test No. 3

This test was run on the combined three lots comprising the shipment to determine what results could be obtained under practice similar to that employed on Shipment No. 1.

Reagents—

To ball mill—		To flotation machine—	
Lime.................	4·0 lb./ton	Copper sulphate.......	1·0 lb./ton
Sodium cyanide.......	0·2 "	Aerofloat.............	0·14 "
Zinc sulphate.........	0·2 "		

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Head............	....	7·64	13·15	0·05	3·17				
Copper concentrate....	20·22	28·50	7·46	0·18	8·02	72·72	11·46	56·25	62·08
Copper middling.	22·77	8·61	22·43	0·04	2·44	24·73	38·80	14·06	21·46
Zinc concentrate.	16·41	0·59	37·61	0·03	0·91	1·23	46·89	7·81	5·75
Zinc middling....	8·08	0·50	2·35	0·05	1·50	0·50	1·44	6·25	4·57
Tailing..........	32·52	0·20	0·57	0·03	0·51	0·82	1·41	15·63	6·14

On Lots Nos. 1 and 2 of this shipment, 4 pounds of lime per ton were found sufficient to give an alkaline reaction. However, the above test was found to be acid, showing that oxidation with the formation of sulphuric acid had set in. The results, therefore, were unsatisfactory, a large amount of zinc being found in the copper middling, a lower grade of zinc concentrate and a lower recovery also resulted.

Test No. 4

In this test the ore required 14 pounds lime per ton to give an alkaline reaction. Other conditions were similar to Test No. 3.

Reagents—

To ball mill—		To flotation machine—	
Lime	14·0 lb./ton	Copper sulphate	1·0 lb./ton
Sodium cyanide	0·2 "	Aerofloat	0·12 "
Zinc sulphate	0·2 "		

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate....	16·51	31·80	3·34	0·18	8·66	67·48	4·27	51·72	52·19
Copper middling.	11·69	18·12	12·74	0·10	5·84	27·23	11·45	20·69	24·82
Zinc concentrate.	15·33	0·72	54·80	0·02	0·67	1·57	65·87	5·17	3·65
Zinc middling...	4·40	3·04	16·21	0·07	2·32	1·72	5·52	5·17	3·65
Tailing..........	51·87	0·30	3·21	0·02	0·84	2·00	12·89	17·25	15·69

This test again shows the depressing action of lime on the copper minerals, a lower recovery being made with more copper reporting in the other products instead of in the copper concentrate. The zinc recovery was improved, although the final tailing was much higher than in Test No. 3.

SHIPMENT NO. 3

Test No. 1

This shipment, containing a mixture of twenty-two lots, was tested under conditions similar to those of previous shipments.

Reagents—

To ball mill—		To flotation machine—	
Soda ash	24·0 lb./ton	Copper sulphate	2·0 lb./ton
Sodium cyanide	0·5 "	Aerofloat	0·14 "
Zinc sulphate	0·4 "		

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Head............	....	5·55	7·02	0·05	2·30				
Copper concentrate....	15·67	29·28	2·74	0·18	8·32	83·52	6·39	68·29	69·15
Copper middling.	9·49	7·73	7·73	0·04	4·41	13·34	10·92	9·76	22·34
Zinc concentrate.	7·66	0·43	52·73	0·01	0·54	0·60	60·19	2·44	2·13
Zinc middling...	13·22	0·44	8·48	0·02	0·60	1·06	16·72	7·31	4·25
Tailing..........	53·96	0·15	0·72	0·01	0·47	1·48	5·78	12·20	2·13

This test shows that the ore of this shipment can be treated by selective flotation, a satisfactory grade of copper concentrate and a reasonable recovery being secured. The use of soda ash, while producing clean copper concentrate, allowed a large amount of iron to float with the zinc, thus producing a bulky middling.

Test No. 2

This test is similar to the preceding one, using lime in place of soda ash.

Reagents—

To ball mill—		To flotation machine—	
Lime	14·0 lb./ton	Copper sulphate	2·0 lb./ton
Sodium cyanide	0·5 "	Aerofloat	0·14 "
Zinc sulphate	0·4 "		

Results—

Product	Weight, per cent	Assay				Per cent of values			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Zn	Au	Ag
Copper concentrate....	6·74	32·04	1·72	0·30	9·30	38·66	1·71	46·53	18·64
Copper middling.	7·26	20·65	3·92	0·12	9·15	26·83	4·21	20·93	19·52
Zinc concentrate.	14·69	10·72	33·44	0·03	3·11	28·19	72·55	9·30	13·61
Zinc middling...	6·88	2·34	6·39	0·06	2·52	2·87	6·50	9·30	5·03
Tailing...... ...	64·43	0·30	1·58	0·01	2·27	3·45	15·03	13·94	43·20

This test again shows the detrimental effect of lime on copper flotation, resulting in a poor recovery as copper concentrate and throwing the remainder into the other products. The grade of zinc concentrate was low also, due to its copper content. The zinc tailing was again higher due to the rapid oxidization of the ore and also to the effect of lime.

SUMMARY AND CONCLUSIONS

In Tests Nos. 1 to 7, it should be noted that potassium xanthate when used for the flotation of zinc gives poor results, as a large weight of middling results due to iron pyrite being lifted. This reagent, therefore, was omitted as being too powerful to produce a satisfactory zinc concentrate. In all tests where soda ash was used instead of lime it will be noted that a copper concentrate higher in zinc is produced than when lime is used. The total copper floated, however, is in the copper concentrate. On the other hand, when lime is used in the circuit for copper flotation, the copper concentrate is lower in zinc and the total copper recovered in this product is lower. The copper not reporting with the copper concentrate is recovered when the zinc is floated, lowering the grade of that product. The use of lime benefits the grade of the zinc concentrate recovered due to the depressing action of this reagent on iron sulphides.

In Tests Nos. 1 to 8 a comparison of results with different reagents used for lifting the copper shows that best results are obtained when using thiocarbanilide. This must be added to the grinding circuit as it is insoluble in water. In Tests Nos. 11 to 12 the use of TT does not offer any inducement for its adoption, as it gives a lower recovery. Aerofloat, or phosphocresylic acid, appears to be suitable for the flotation of both copper and zinc. Results secured in Tests Nos. 1 to 7 show this reagent to be superior to potassium xanthate, especially in the flotation of zinc. In Tests Nos. 13 to 15, the necessity of the proper amount of cyanide being used is apparent, an insufficient quantity causing a low-grade bulky concentrate due to iron sulphides floating with the zinc.

In all tests, apart from desiring a suitable grade of concentrate together with a good recovery, the attempt was made to keep as much as possible of the iron sulphides in the tailing. Owing to the fact that the ore had been finely crushed for some time prior to the date received, oxidization had set in. This made it difficult to make a good recovery of the zinc in a clean concentrate. The mineral was hard to float, iron pyrite coming up with it. This condition either caused a concentrate low in zinc, or a very bulky middling product.

The majority of the tests show that the greater part of the gold and silver is associated with the copper sulphides. This is fortunate as these metals will be recovered in the copper refinery.

Tests Nos. 3 and 4 on Shipment No. 2, show the necessity of maintaining an alkaline circuit. The fourth test also shows the detrimental effect of too much lime, resulting as it does in poor recoveries of copper, and a higher loss of zinc in the tailing.

In calculating the recovery in all tests, no attempt was made to eliminate the middling product. From former experimental work it has been found that at least 50 per cent of the values can be recovered, thus raising the total recovery indicated in all tests.

It has been noticed in experimental work done on other ores of a similar nature that results secured on freshly crushed ore could not be duplicated after the material had been allowed to stand in contact with air for any length of time. Oxidization setting in makes the flotation action much slower, with a lower recovery. As these shipments had been crushed for some time prior to the tests it is to be expected that better results can be obtained on freshly mined ore.

The practice as indicated by these tests will be grinding with cyanide and zinc sulphate in a circuit alkaline with soda ash to a mesh fine enough to free the various minerals. The copper can then be floated either by the addition of thiocarbanilide to the grinding mill or Aerofloat to the classifier overflow or to the head of the copper flotation machine, where a rougher concentrate is taken off and cleaned in following cells.

Lime should then be added to the tailing from the copper cells, together with the requisite amount of copper sulphate, and passed to a mixing tank where a time contact is allowed prior to reaching the zinc flotation machines. The addition of Aerofloat to the mixing tank or at the head of the cells produces a rougher zinc concentrate which when passed to cleaner cells, yields the finished concentrate. The middling product from the copper flotation cells may be returned to the grinding circuit, to the head of the copper flotation machine, or treated in a separate unit. Which method would yield the best results is yet to be determined. The zinc middling also will require further treatment to recover the contained values.

No great difficulties are anticipated in the recovery of copper and zinc from freshly mined ore. Selective flotation will doubtless yield results well within the required limits.

Report No. 273

EXPERIMENTAL TESTS ON ORE FROM THE KIRKLAND PREMIER MINE, KIRKLAND LAKE, ONTARIO

J. S. Godard

Shipment.—A shipment of 220 pounds was received October 5, 1927, from the Kirkland Premier Mines, Ltd., Kirkland Lake, Ont.

Characteristics and Analysis.—The sample submitted was a gold ore in which the gold is partly free but mainly associated with small amounts of the sulphides of iron and copper. The gangue is siliceous. The analysis of the head sample was:—

Gold	0·39 oz./ton	Copper	0·19 per cent
Silver	0·33 "	Iron	5·38 "
		Insoluble	75·81 "

Purpose of Tests.—It was requested that small-scale flotation tests be made to determine whether the gold values could be recovered with the sulphides in a concentrate.

EXPERIMENTAL TESTS

Flotation

Test No. 1—

Product	Weight, per cent	Au, oz./ton	Per cent of values
Concentrate	12·1	2·97	88·6
Tailing	87·9	0·053	11·4
Tailing +200	20·7	0·14	54·9
Tailing −200	79·3	0·03	45·1
Average		0·053	

Test No. 2—

Product	Weight, per cent	Au, oz./ton	Per cent of values
Concentrate	17·6	2·07	92·2
Tailing	82·4	0·037	7·8
Tailing +200	18·1	0·07	34·0
Tailing −200	81·9	0·03	66·0
Average		0·037	

Test No. 3—

Concentrate	17·5	1·97	89·5
Tailing	82·5	0·049	10·5
Tailing +150	13·1	0·11	29·4
+200	17·1	0·08	28·0
−200	69·8	0·03	42·6

Test No. 4—

Concentrate	25·9	1·37	89·0
Tailing	74·1	0·059	11·0
Tailing +200	19·2	0·14	45·4
−200	80·8	0·04	54·6
Average		0·059	

Test No. 5—

Concentrate	7·7	4·22	85·0
Middling	9·8	0·18	4·6
Tailing	82·5	0·049	10·4
Tailing +200	26·5	0·10	54·6
−200	73·5	0·03	45·4
Average		0·049	

Reagents—

Test No. 1—Soda ash 3·0, coal-tar creosote 0·10 lb./ton added to ball mill.
Copper sulphate 1·0, xanthate 0·3, pine oil 0·08 lb./ton to cell.

Test No. 2—Soda ash 3·0, P.T. & T. Co. No. 1580, 0·10 lb./ton to ball mill.
Copper sulphate 1·0, xanthate 0·3, pine oil 0·08 lb./ton to cell.

Test No. 3—Soda ash 3·0, P.T. & T. Co. No. 80, 0·1 lb./ton to ball mill. Sodium sulphide 1·0 lb./ton to ball mill. Xanthate 0·3, pine oil 0·08 lb./ton to cell—froth deep and sticky.

Test No. 4—Soda ash 3·0, water-gas tar 0·12 lb./ton added to ball mill.
Xanthate 0·3, copper sulphate 0·4, pine oil 0·08 lb./ton to cell.

Test No. 5—Soda ash 2·5, coal-tar creosote 0·08 lb./ton added to ball mill.
Xanthate 0·4, pine oil 0·06 lb./ton added to cell. Concentrate cleaned using 0·04 lb./ton pine oil.

Flotation and Tabling the Flotation Tailing

Test No. 6—

Product	Weight, per cent	Au, oz./ton	Per cent of values
Flotation concentrate	15·4	2·49	91·3
Table concentrate	2·4	0·46	2·6
" middling	5·4	0·09	1·2
" tailing	35·3	0·041	3·4
" slimes	41·5	0·015	1·5
Table tailing +200	5·5	0·06	8·0
−200	94·5	0·04	92·0
Average		0·041	

Test No. 7—In this test the rougher concentrates from two tests were combined and cleaned. The tailing from one test was screened on 200 mesh and that from the other tabled and table tailing screened on 200 mesh.

Product	Weight, per cent	Au, oz./ton	Per cent of values
Flotation—			
Concentrate (Copper 2·64 per cent, Iron 36·45 per cent)	6·0	4·57	85·0
Middling	4·9	0·18	2·7
Tailing	89·1	0·045	12·3
Tabling the flotation tailing—			
Concentrate	2·2	0·52	26·2
Middling	1·9	0·17	7·4
Tailing	42·0	0·043	41·6
Slimes	53·9	0·02	24·8
Flotation tailing screened on 200 mesh—			
+200	7·8	0·10	17·5
−200	92·2	0·04	82·5
Average		0·045	
Table tailing screened on 200 mesh—			
+200	10·0	0·07	16·3
−200	90·0	0·04	83·7
Average		0·043	

Reagents—

Test No. 6—Soda ash 3·0, P.T. & T. Co. No. 1580, 0·10 lb./ton to ball mill.
Copper sulphate 1·0, xanthate 0·3, pine oil 0·08 lb./ton to cell.

Test No. 7—Soda ash 3·0, coal-tar creosote 0·08 lb./ton to ball mill.
Copper sulphate 1·0, xanthate 0·3, pine oil 0·08 lb./ton to cell.

Amalgamation and Flotation of Amalgamation Tailing

Test No. 8—Flotation—

Product	Weight, per cent	Au, oz./ton	Per cent of values
Concentrate	11·4	1·19	71·8
Tailing	88·6	0·06	28·2
Tailing +200	19·4	0·10	32·5
−200	80·6	0·05	67·5
Average	....	0·06	

Amalgamation tailing from flotation products.................... 0·19 oz./ton

Summary—

Assay			Recoveries, per cent		
Head	Amalgamation tailing	Flotation tailing	Amalgamation	Flotation	Total
0·39	0·19	0·06	51·2	35·0	86·2

Test No. 9—Flotation—

Product	Weight, per cent	Au, oz./ton	Per cent of values
Concentrate	13·2	1·00	78·0
Tailing	86·8	0·043	22·0
Tailing +200	7·5	0·08	14·0
−200	92·5	0·04	86·0
Average		0·043	

Amalgamation tailing from flotation products.................... 0·17 oz./ton

Summary—

Assay			Recoveries, per cent		
Head	Amalgamation tailing	Flotation tailing	Amalgamation	Flotation	Total
0·39	0·17	0·043	56·4	34·0	90·4

Reagents—

Tests Nos. 8 and 9—Soda ash 3·0, coal-tar creosote 0·12 lb./ton to ball mill.
Copper sulphate 1·0, xanthate 0·40, pine oil 0·08 lb./ton to cell.

CONCLUSIONS

The best recovery made by flotation was 92 per cent of the gold. With this recovery the flotation concentrate assayed 2 ounces per ton in gold and the ratio of concentration was 1 : 5·7. The grade of flotation concentrate may be increased to 4 ounces per ton by cleaning the rougher concentrate. This operation will increase the ratio of concentration to 1 : 13. Grinding to 80 to 85 per cent −200 mesh is necessary to free the sulphides. Better flotation results could be obtained on freshly broken ore as the sample was slightly oxidized. According to Mr. Osborn, superintendent of the mine, it had been mined for some time, and the addition of copper sulphate was necessary, as otherwise the froth was greyish black and the recoveries were low.

Tabling the flotation tailing increased the gold recovery by two to three per cent. This step might be advantageously adopted as it also serves to indicate the performance of the flotation cells.

Amalgamation recovered about 50 per cent of the gold on ore ground to 80 per cent −200 mesh, but as the gold appears in the flotation concentrate when this step is omitted the recovery is not increased.

Report No. 274

EXPERIMENTAL TESTS ON CYANIDE TAILINGS FROM THE KIRKLAND LAKE GOLD MINING COMPANY, LTD.

J. S. Godard

Shipments.—Two shipments of tailings, each of 200 pounds wet weight, were received on the following dates: No. 1, September 28; No. 2, October 7, 1927.

Characteristics.—The tailings were those discharged from the countercurrent decantation mill of the Kirkland Lake Gold Mining Company. They contained about 40 per cent moisture as dilute cyanide solution, and were ground to about 20 per cent +200 mesh. The gold values in the tailings from Lot. No. 1, average of 21 tests, was 0·051 ounce per ton; Lot No. 2, average of 5 tests, was 0·049 ounce per ton.

Purpose of Test.—The purpose of the tests was to ascertain what recovery of the gold remaining in the tailings could be obtained by flotation—*(a)* after washing the tailings in a filter to wash out the cyanide, and *(b)* on the tailings as received.

EXPERIMENTAL TESTS

Shipment No. 1—Series A

About one-half of Shipment No. 1 was agitated for one hour in a Pachuca tank and filtered in a plate and frame press. The tailings thus treated were used for Series A flotation tests.

Results—

Test No.	Product	Weight, per cent	Au, oz./ton	Per cent of values
1	Concentrate	8·8	0·22	41·4
	Tailing	91·2	0·03	58·6
	Tailing screened on 200 mesh—			
	+200	20·2	0·07	46·8
	−200	79·8	0·02	53·2
	Average		0·03	
2	Concentrate	7·7	0·26	34·6
	Tailing	92·3	0·041	65·4
	Tailing screened on 200 mesh—			
	+200	26·6	0·07	45·8
	−200	73·4	0·03	54·2
	Average		0·041	
3	Concentrate	11·6	0·21	50·6
	Tailing	88·4	0·0269	49·4
	Tailing screened on 200 mesh—			
	+200	13·9	0·07	36·0
	−200	86·1	0·02	64·0
	Average		0·0269	
4	Concentrate	7·6	0·29	47·1
	Tailing	92·4	0·0267	52·9
	Tailing screened on 200 mesh—			
	+200	16·7	0·06	37·4
	−200	83·3	0·02	62·6
	Average		0·0267	
5	Concentrate	11·4	0·20	42·0
	Tailing	88·6	0·0356	58·0
	Tailing screened on 200 mesh—			
	+200	11·2	0·08	25·3
	−200	88·8	0·03	74·7
	Average		0·0356	
6	Concentrate	6·9	0·36	42·4
	Tailing	93·1	0·0362	57·6
	Tailing screened on 200 mesh—			
	+200	15·4	0·07	29·8
	−200	84·6	0·03	70·2
	Average		0·0362	
7	Concentrate	8·0	0·30	46·0
	Tailing	92·0	0·0306	54·0
	Tailing screened on 200 mesh—			
	+200	21·3	0·07	48·7
	−200	78·7	0·02	51·3
	Average		0·0306	
8	(The rougher concentrates from five lots of ore were combined and cleaned.)			
	Concentrate	2·3	0·71	39·5
	Middling	4·9	0·01	1·2
	Tailing	92·8	0·0263	59·3
	Tailing screened on 200 mesh—			
	+200	21·0	0·05	39·9
	−200	79·0	0·02	60·1
	Average		0·0263	
	Tailing was tabled on a small Wilfley table—			
	Concentrate	4·6	0·08	15·2
	Middling	5·7	0·06	11·5
	Tailing	44·3	0·03	54·8
	Slimes	45·4	0·01	18·5

Results—Concluded

Test No.	Product	Weight, per cent	Au, oz./ton	Per cent of values
	(Three lots were treated as in Test No. 8.)			
9	Concentrate (Iron in concentrate 35·43 per cent)...	2·0	0·86	39·2
	Middling..	2·7	0·04	2·5
	Tailing...	95·3	0·0269	58·3
	Tailing screened on 200 mesh—			
	+200..	23·3	0·05	43·1
	−200..	76·7	0·02	56·9
	Average....................................		0·0269	
	Tailing tabled as in No. 8—			
	Concentrate................................	3·7	0·09	12·9
	Middling...................................	5·0	0·06	11·8
	Tailing....................................	40·6	0·0348	55·3
	Slimes.....................................	50·7	0·01	20·0
	Table tailing screened on 200 mesh—			
	+200..	24·1	0·05	34·6
	−200..	75·9	0·03	65·4
	Average....................................		0·0348	

Reagents—

Test No. 1—Soda ash 3·0 lb./ton, agitated 20 minutes. Coal-tar creosote 0·12, $Al_2(SO_4)_3$ 0·85, xanthate 0·2 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 2—Soda ash 3·0 lb./ton, agitated 20 minutes. Coal-tar creosote 0·12, $FeSO_4$ 0·85, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 3—Soda ash 3·0 lb./ton, $CuSO_4$ 1·7 lb./ton, agitated 20 minutes. Coal-tar creosote 0·12, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·15 lb./ton, added to cell.

Test No. 4—Soda ash 3·0. $Al_2(SO_4)_3$ 1·7 lb./ton, agitated 20 minutes. Coal-tar creosote, 0·12, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 5—Soda ash 3·0, $FeSO_4$ 1·7 lb./ton, agitated 20 minutes. Coal-tar creosote 0·12, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 6—Soda ash 3·0 lb./ton, agitated 20 minutes. Na_2S 0·85, coal-tar creosote 0·12, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 7—Soda ash 3·0 lb./ton, agitated 20 minutes. Na_2SO_4 0·85 lb./ton, coal-tar creosote 0·12, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 8—Soda ash 3·0, coal-tar creosote 0·16 lb./ton, agitated 20 minutes. Xanthate 0·36, $CuSO_4$ 1·1, pine oil (Yarmor) 0·12 lb./ton added to cell. No additional reagents were used to clean the concentrates.

Test No. 9—Reagents as in Test No. 8.

Series B

Test No.	Product	Weight, per cent	Au, oz./ton	Per cent of values
1	Concentrate......................................	6·8	0·21	25·3
	Tailing..	93·2	0·0453	74·7
	Tailing screened on 200 mesh—			
	+200..	26·3	0·06	34·9
	−200..	73·7	0·04	65·1
	Average....................................		0·0453	
2	Concentrate......................................	6·6	0·14	17·1
	Tailing..	93·4	0·0478	82·9
	Tailing screened on 200 mesh—			
	+200..	25·8	0·07	37·8
	−200..	74·2	0·04	62·2
	Average....................................		0·0478	
3	Concentrate......................................	5·3	0·14	14·6
	Tailing..	94·7	0·0456	85·4
	Tailing screened on 200 mesh—			
	+200..	28·4	0·06	37·2
	−200..	71·6	0·04	62·8
	Average....................................		0·0456	

Series B—Concluded

Test No.	Product	Weight, per cent	Au, oz./ton	Per cent of values
4	Concentrate	7·2	0·25	38·7
	Tailing	92·8	0·0307	61·3
	Tailing screened on 200 mesh—			
	+200	26·9	0·06	52·4
	−200	73·1	0·02	47·6
	Average		0·0307	
5	Concentrate	4·2	0·24	18·7
	Tailing	95·8	0·0459	81·3
	Tailing screened on 200 mesh—			
	+200	29·1	0·06	38·1
	−200	70·9	0·04	61·9
	Average		0·0459	
6	Concentrate	6·4	0·17	18·7
	Tailing	93·6	0·0508	81·3
	Tailing screened on 200 mesh—			
	+200	34·5	0·08	48·4
	−200	65·5	0·04	51·6
	Average		0·0508	
7	Concentrate	4·1	0·24	16·8
	Tailing	95·9	0·0505	83·2
	Tailing screened on 200 mesh—			
	+200	26·3	0·08	41·6
	−200	73·7	0·04	58·4
	Average		0·0505	
8	Concentrate	4·1	0·18	14·5
	Tailing	95·9	0·0455	85·5
	Tailing screened on 200 mesh—			
	+200	27·3	0·06	36·0
	−200	72·7	0·04	64·0
	Average		0·0455	
9	Concentrate	8·6	0·16	29·5
	Tailing	91·4	0·0361	70·5
	Tailing screened on 200 mesh—			
	+200	20·4	0·06	33·8
	−200	79·6	0·03	66·2
	Average		0·0361	
10	Concentrate	9·1	0·16	29·8
	Tailing	90·9	0·0379	70·2
	Tailing screened on 200 mesh—			
	+200	26·1	0·06	41·4
	−200	73·9	0·03	58·6
	Average		0·0379	
11	Concentrate	15·4	0·08	28·5
	Tailing	84·6	0·0365	71·5
	Tailing screened on 200 mesh—			
	+200	21·5	0·06	35·3
	−200	78·5	0·03	64·7
	Average		0·0365	
12	Concentrate	7·5	0·12	17·7
	Tailing	92·5	0·0451	82·3
	Tailing screened on 200 mesh—			
	+200	25·1	0·06	33·4
	−200	74·9	0·04	66·6
	Average		0·0451	

Reagents—

Test No. 1—Soda ash 3·0 lb./ton, agitated 20 minutes. Coal-tar creosote 0·12, $CuSO_4$ 0·85. xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell,

Test No. 2—Soda ash 3·0 lb./ton, agitated 20 minutes. Coal-tar creosote 0·12 $Al_2(SO_4)_3$ 0·85, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 3—Soda ash 3·0 lb./ton, agitated 20 minutes. Coal-tar creosote 0·12, $FeSO_4$ 0·85, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 4—Soda ash 3·0, $CuSO_4$ 1·7 lb./ton, agitated 20 minutes. Coal-tar creosote 0·12, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 5—Soda ash 3·0, $Al_2(SO_4)_3$ 1·7 lb./ton, agitated 20 minutes. Coal-tar creosote 0·12. xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell,

Test No. 6—Soda ash 3·0, $FeSO_4$ 1·7 lb./ton, agitated 20 minutes. Coal-tar creosote 0·12, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 7—Soda ash 3·0 lb./ton, agitated 20 minutes. Na_2S 0·85, coal-tar creosote 0·12, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 8—Soda ash 3·0 lb./ton, agitated 20 minutes. Na_2SO_4 0·85 lb./ton, coal-tar creosote 0·12, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 9—$CuSO_4$ 1·7 lb./ton, agitated 20 minutes. Soda ash 3·0 lb./ton, agitated 20 minutes. Coal-tar creosote 0·12, xanthate 0·36, $CuSO_4$ 0·85 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 10—$FeSO_4$ 1·7 lb./ton, agitated 20 minutes. Soda ash 3·0 lb./ton, agitated 20 minutes. Coal-tar creosote 0·12, xanthate 0·36, $CuSO_4$ 0·85 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Test No. 11—$FeSO_4$ 1·7 lb./ton, agitated 20 minutes. Soda ash 3·0 lb./ton, agitated 20 minutes. $FeSO_4$ 0·85, coal-tar creosote 0·12, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12, Aerofloat 0·08 lb./ton, added to cell. Voluminous froth.

Test No. 12—$Al_2(SO_4)_3$ 1·7 lb./ton, agitated 20 minutes. Soda ash 3·0 lb./ton, agitated 20 minutes. $FeSO_4$ 0·85, coal-tar creosote 0·12, xanthate 0·36 lb./ton, agitated 5 minutes. Pine oil No. 5, 0·12 lb./ton, added to cell.

Shipment No. 2

Test No.	Product	Weight, per cent	Au, oz./ton	Per cent of values
1	Concentrate	8·5	0·31	61·3
	Tailing	91·5	0·018	38·7
	Tailing screened on 200 mesh—			
	+200	32·6	0·035	63·0
	−200	67·4	0·010	37·0
	Average		0·018	
2	Concentrate	8·8	0·28	58·6
	Tailing	91·2	0·0192	41·4
	Tailing screened on 200 mesh—			
	+200	30·8	0·04	64·1
	−200	69·2	0·01	35·9
	Average		0·0192	
3	Concentrate	4·4	0·52	38·6
	Tailing	95·6	0·0381	61·4
	Flotation tailing was tabled—			
	Table concentrate	3·7	0·12	11·5
	Middling	10·3	0·06	16·3
	Tailing	54·7	0·0417	59·8
	Slimes	31·3	0·015	12·4
	Table tailing screened on 200 mesh—			
	+200	39·2	0·06	56·4
	−200	60·8	0·03	43·6
	Average		0·0417	

Shipment No. 2—Concluded

Test No.	Product	Weight, per cent	Au, oz./ton	Per cent of values
4	Concentrate	9·2	0·26	49·0
	Tailing	90·8	0·0274	51·0
	Tailing screened on 200 mesh—			
	+200	18·5	0·06	40·5
	−200	81·5	0·02	59·5
	Average		0·0274	
5	(Four rougher concentrates were made and combined, then cleaned.)			
	Concentrate (Iron in concentrate 38·09 per cent)	2·0	10·7	41·9
	Middling	2·4	0·03	1·4
	Tailing	95·6	0·0303	56·7
	Tailing screened on 200 mesh—			
	+200	25·9	0·06	51·2
	−200	74·1	0·02	48·8
	Average		0·303	
	Tailing tabled—			
	Table concentrate	4·2	0·09	13·0
	Middling	4·7	0·06	9·9
	Tailing	44·7	0·0388	60·9
	Slimes	46·4	0·01	16·2
	Table tailing screened on 200 mesh—			
	+200	29·4	0·06	45·4
	−200	70·6	0·03	54·6
	Average		0·0388	

Reagents—

Test No. 1—Soda ash 2·5 lb./ton, agitated 20 minutes. $CuSO_4$ 0·85, P.T. & T. Co. No. 1580 0·16, xanthate 0·2 lb./ton, agitated 5 minutes. Floto 0·15 lb./ton, added to cell.

Test No. 2—Soda ash 2·5 lb./ton, agitated 20 minutes. $CuSO_4$ 0·85 lb./ton. Coal-tar creosote 0·16, xanthate 0·2 lb./ton, agitated 5 minutes. Floto 0·15 lb./ton, added to cell.

Test No. 3—Soda ash 3·0 lb./ton, agitated 20 minutes. Na_2S 0·85, coal-tar creosote 0·16, xanthate 0·2 lb./ton, agitated 5 minutes. Floto 0·15 lb./ton, added to cell.

Test No. 4—Soda ash 2·75, pine oil No. 350 0·12 lb./ton, agitated 20 minutes. $CuSO_4$ 1·25, xanthate 0·2 lb./ton, agitated 5 minutes. Floto 0·10 lb./ton, added to cell.

Test No. 5—Soda ash 3·0 lb./ton, agitated 20 minutes, coal-tar creosote 0·12 lb./ton, agitated 5 minutes. Xanthate 0·36, $CuSO_4$ 1·1, pine oil (Yarmor) 0·12 lb./ton added to cell. No additional reagents used for cleaning the rougher concentrates.

SUMMARY

Shipment No. 1, Series A.—The average recovery in seven tests in which the rougher concentrates were not cleaned was 42·5 per cent, the average grade of concentrate was 0·26 ounce per ton and the average ratio of concentration 11·2 : 1. In the two tests where the rougher concentrates were cleaned the recovery was 39·2 per cent with an average grade of 0·785 ounce per ton and a ratio of concentration of 45·4 : 1.

Shipment No. 1, Series B.—The average recovery in twelve tests was 23 per cent with a maximum of 38·7 per cent and a minimum of 14·5 per cent. The average grade of concentrate was 0·174 ounce per ton, and the average ratio, 15·9 : 1.

Shipment No. 2.—The average recovery from three tests, No. 3 omitted, in which the concentrates were not cleaned was 56·3 per cent. The average grade of concentrate was 0·28 ounce per ton, and the average ratio of concentration, 11·4 : 1. In Test No. 5 where the rougher concentrate was cleaned the recovery was 41·9 per cent, the grade of concentrate 1·07 ounces per ton and the ratio of concentration was 50 : 1.

CONCLUSIONS

In Shipment No. 1, Series A, and Shipment No. 2, the recoveries may seem unsatisfactory and the grade of concentrate too low to warrant any further treatment, but it should be noted that the flotation heads assayed 0·05 ounce per ton. Should the cyanide tailing increase in value with the same grinding an increase in recovery and grade of concentrate may be expected. Shipment No. 1, Series A, and Shipment No. 2 were very similar in behaviour and the slightly better results on Shipment No. 2 are attributed to practice rather than a difference in the tailings. The concentrates were bright in colour and the general flotation conditions were good. Pyrite forms the bulk of the concentrates. In Test No. 9, Shipment No. 1, Series A, the concentrate contained 35·43 per cent iron, and in Test No. 5, Shipment No. 2, 38·09 per cent iron. In those tests where tabling followed flotation the table concentrate contained very little free sulphides, but consisted of the coarser sands which contained very finely disseminated sulphide or free milling gold.

The chief features in the tests on Shipment No. 1, Series B, were the poor recoveries, the erratic results, and the white frothy nature of the concentrates which contrasted with the pyritic colour of the concentrates obtained in those tests where washing of the tailings preceded flotation. These conditions are attributed to the depressing influence of cyanide especially in the presence of lime and have been experienced in the flotation of similar tailings both in large and laboratory-scale tests.

Unsuccessful attempts were made to neutralize the effect of the cyanide by the addition of such cyanicides as the sulphates of copper, iron, and aluminium. Where copper sulphate is added with xanthate in the presence of cyanide the xanthate is partly decomposed and free sulphur floats off with the first part of the concentrate. This latter effect was not noticed in the tests on Series A or Shipment No. 2, nor when the copper sulphate is replaced by ferrous sulphate or aluminium sulphate. Apparently the reaction is between the copper sulphate and xanthate in the presence of small amounts of cyanide.

From these tests and past experience in the flotation of cyanide tailings, the most satisfactory condition for flotation is where double filtration is practised with repulping between filtrations, and where water in place of dilute cyanide solution is used on the discharge half of the second filter. This operation leaves the feed to flotation practically free from cyanide and lime in solution.

Report No. 275

CONCENTRATION OF COPPER-GOLD ORE FROM ARCHEAN MINES, CLERICY TOWNSHIP, QUEBEC

C. S. Parsons

Shipment.—A shipment of 113 pounds of ore was received October 5, 1927, from the Archean Mines, Ltd., Clericy, Quebec. The shipment was made by Alderson, MacKay, and Armstrong, consulting engineers, Montreal.

Characteristics and Analysis.—The sample received consisted of disseminated chalcopyrite and pyrite with a small amount of zinc blende. The gangue was chiefly siliceous, but also contained considerable altered rock of a talcy nature.

Copper	0·66 per cent	Gold	0·32 oz./ton
Zinc	0·46 "	Silver	0·26 "

Purpose of Experimental Work.—Concentration tests on this ore were requested, and in view of the fact that occasional high assays are obtained from the vein, it was desirable to determine the nature in which the gold causing this condition occurred; as the presence of tellurides was suspected, tests were also required for tellurides, both by chemical and microscopical methods.

EXPERIMENTAL TESTS

Test No. 1.—Ore −14 mesh, 1,000 grammes ground 30 minutes in ball mill, water 1,000 c.c., soda ash 5 lb./ton; 3 drops Aerofloat added to cells—pulp strongly alkaline.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Concentrate	7·1	8·70	4·12	97·1	86·4
Tailing	92·9	0·02	0·05	2·9	13·6

Test No. 3.—Ore, 1,000 grammes, 1,000 c.c. water, lime 2 lb./ton, ground 30 minutes in ball mill, 3 drops Aerofloat No. 15 added to cells. Obtained small float of copper, then added 0·06 lb./ton copper sulphate; some zinc floated.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Concentrate	3·9	15·72	5·08	92·8	80·8
Tailing	96·1	0·05	0·05	7·2	19·2

Test No. 4.—Ore 1,000 grammes, water 1,000 c.c., soda ash 2 lb./ton, cyanide 0·2 lb./ton, ground 30 minutes in ball mill; 3 drops Aerofloat added to cells. Very little copper floated, added 0·6 lb./ton copper sulphate, obtained good float of copper; and zinc came up.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Concentrate	4·4	14·22	5·68	97·1	86·8
Tailing	95·6	0·02	0·04	2·9	13·2

Test No. 5.—Ore 1,000 grammes, water 1,000 c.c., soda ash 2 lb./ton, cyanide 0·1 lb./ton, ground 30 minutes in ball mill; 3 drops Aerofloat added to cells; *p*H 6·8.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Concentrate	3·0	20·02	6·17	94·0	54·5
Tailing	97·0	0·04	0·16	6·0	45·5

Amalgamation Test on Flotation Tailing—

Amount used—800 grammes 0·16 oz./ton	=	154·24	oz. content
Recovered by amalgamation		126 31	"
Assay of amalgamation tailing 0·03 oz./ton	=	27·93	"
Recovered by amalgamation		81·8	per cent
Gold in flotation tailing, or heads		37·3	"
Recovery in flotation concentrate and amalgamated = 54·5 + 37·3		91·8	"

Test No. 6.—Ore 1,000 grammes, water 1,000 c.c., ground 30 minutes in ball mill, 3 drops Aerofloat added to cells; *p*H 6·7.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Concentrate	7·9	8·06	3·72	94·5	88·8
Tailing	92·1	0·04	0·04	5·5	11·2

Remarks.—It will be observed that under the conditions of Test No. 1, good recoveries of both the gold and copper were obtained, but that the assay of the copper in the concentrate was low. In Test No. 2 the results of which are not reported, lime was used instead of soda ash as in Test No. 1, other conditions being the same. It was found that the amount of lime used caused a large amount of gangue materials of a talcy nature to float. In Test No. 3 the amount of lime used was reduced from 5 pounds per ton to 2 pounds per ton. Very little talcy material floated under this condition. Copper sulphate was also added in order to float the zinc in case the gold should be partly associated with the zinc blende. A high-grade copper concentrate was obtained with a good recovery of both copper and zinc.

In Test No. 4 a combination of soda ash and cyanide was used. The cyanide was to prevent the pyrite from floating. It was found that the addition of such a large amount of cyanide also prevents the copper from floating and it was necessary to reactivate the copper with copper sulphate, which also brought up the zinc. In Test No. 5 the amount of cyanide was reduced and no copper sulphate was necessary. It will be observed that the flotation tailing was high in gold. An amalgamation test was then made on this tailing and it was found that approximately 82 per cent of the gold could be amalgamated. Test No. 6 was run in a neutral pulp resulting in a good recovery of the gold and copper, but a low-grade copper concentrate.

SUMMARY

Tests were made for the presence of tellurides by chemical and microscopical methods both on the ore and on the concentrate; no telluride could be discovered. The fact that the gold remaining in the tailing of Test No. 5 could be amalgamated proves fairly conclusively that the spotty nature of the assays obtained in sampling the vein is due to free gold.

The tests made show that the ore can be satisfactorily concentrated by flotation. By floating in a neutral pulp a low copper content concentrate can be produced carrying approximately $75 in gold per ton, or by using soda ash and cyanide together with copper sulphate a high-grade copper concentrate can be produced running $110 in gold. The cost of addition reagents to produce the concentrate as compared to the method of floating in a neutral pulp would be approximately 20 cents per ton of ore.

It would seem advisable, owing to the presence of free gold, to introduce amalgamation either before or after flotation, depending on the economic considerations in regard to markets for the concentrate. The smelter rates on the different grade of concentrates will also determine the choice between the production of a high-grade copper concentrate or the production of the lower grade product at a low cost for reagents.

Report No. 276

CONCENTRATION OF THE LEAD-ZINC ORE OF THE STIRLING MINE, CAPE BRETON, N.S.

C. S. Parsons

Shipment.—Two small shipments of about 50 pounds each were received October 14, 1927, from the British Metal Corporation (Canada) Ltd., from the Stirling mine, in Cape Breton, N.S.

Characteristics and Analysis.—The samples consisted of a complex mixture of chalcopyrite, galena, sphalerite, and pyrite, with small values in gold and silver finely disseminated and associated with an altered gangue containing magnesium and lime silicate minerals of a talcy nature, and some quartz. Lot No. 1 represented the high-grade ore and Lot No. 2 the low-grade ore. Analysis:—

	Lot No. 1	Lot No. 2
Copper	0·49 per cent	0·50 per cent
Lead	4·02 “	1·16 “
Zinc	16·11 “	7·99 “
Gold	0·06 oz./ton	0·08 oz./ton
Silver	2·67 “	2·14 “
Iron	6·54 per cent	5·43 per cent

Purpose of Tests.—It is necessary to separate the lead and zinc, which are the principal values, and concentrate them into two products, a lead concentrate for the lead smelter and a zinc concentrate for the zinc smelter. As much of the copper, gold, and silver values as possible should be recovered in the lead concentrate as the zinc smelter would not pay for these metals in the zinc concentrate. The samples were submitted for the purpose of running a series of preliminary tests on the ore with the view of following up with further tests on a larger and more representative sample if the development at present being carried out on the ore deposit indicated ore-bodies of commercial extent.

EXPERIMENTAL TESTS

Most of the work was carried out on the low-grade Lot No. 2. One test, however, was made on the high-grade ore.

Lot No. 1—High-grade Ore

1,000 grammes ore −14 mesh ground 45 minutes in ball mill, the discharge approximately 80 per cent −200 mesh.

Reagents—

To ball mill—		*To lead cell—*		*To zinc cell—*	
Soda ash	5·0 lb./ton	Aerofloat	0·2 lb./ton	Copper sulphate	2·0 lb./ton
Cyanide	0·4 “	C.T.C.	0·2 “	Aerofloat	0·15 “

Lead concentrate cleaned with Aerofloat................ 0·1 lb./ton

Results—

Product	Weight, per cent	Assay					Per cent of values			
		Cu, per cent	Pb, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Pb	Zn	Au	Ag
Lead concentrate	11·1	0·71	27·61	11·81	0·28	9·56	64·9	8·05	54·0	38·59
Table concentrate	0·7	0·43	11·97	15·12			18·6	0·69		
Table tailing	5·7	0·69	4·83	19·32	0·06	4·12	5·9	7·36	7·75	10·74
Zinc concentrate	27·1	0·17	1·39	46·52	0·02	4·52	8·2	77·15	12·45	44·49
Tailing	55·4		0·25	1·98	0·02	0·31	2·4	6·75	25·8	6·2

Lot No. 2—Low-grade ore

Test No.	Product	Weight, per cent	Assay					Per cent of values			
			Cu, per cent	Pb, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Pb	Zn	Au	Ag
1	Lead concentrate.....	10·02	0·5	8·75	3·79	......	......	75·7	5·1	......	......
	Zinc concentrate......	17·50	......	1·36	35·43	......	......	20·5	81·8	......	......
	Tailing...............	72·48	......	0·06	1·37	......	......	3·8	13·1	......	......

(Ore 1,000 grammes —14 mesh ground 45 minutes. Discharge from mill approximately 80 per cent —200 mesh.)

2	Lead concentrate.....	13·7	2·45	7·09	7·46	......	......	80·0	12·9	......	......
	Zinc concentrate......	17·0	......	1·06	38·23	......	......	14·9	82·7	......	......
	Tailing...............	69·3	......	0·09	0·50	......	......	5·1	4·4	......	......

(Ore 1,000 grammes —14 mesh ground 60 minutes. Discharge from mill approximately 90 per cent —200 mesh.)

3	Lead concentrate.....	16·2	0·98	3·72	5·43	......	......	52·7	11·3	......	......
	Zinc concentrate......	22·7	......	2·16	28·18	......	......	43·0	82·7	......	......
	Tailing...............	61·1	......	0·08	0·76	......	......	4·3	6·0	......	......

(Ore 1,000 grammes —14 mesh ground 60 minutes in ball mill.)

4	Lead concentrate.....	13·9	0·30	6·89	4·49	......	......	82·7	7·9	......	......
	Zinc concentrate......	21·0	......	0·80	31·91	......	......	14·5	84·9	......	......
	Tailing...............	65·1	......	0·05	0·86	......	......	2·8	7·1	......	......

(Ore 1,000 grammes —14 mesh ground 45 minutes in ball mill.)

6	Talc concentrate......	4·7	0·24	1·56	1·56	0·10	1·82	6·2	0·9	5·9	4·5
	Lead concentrate.....	8·3	......	10·66	6·06	0·16	7·54	74·9	6·6	16·4	32·7
	Zinc concentrate......	15·4	......	1·03	39·26	0·27	6·56	13·4	80·9	57·3	52·7
	Tailing...............	71·6	......	0·09	1·21	0·03	0·27	5·5	11·6	20·4	10·1

(Ore 2,000 grammes —14 mesh ground 45 minutes in ball mill.)

8	Talc concentrate......	6·8	......	0·61	2·31	0·07	2·67	3·8	2·0	7·5	8·4
	Lead concentrate.....	3·6	......	23·74	6·73	1·04	32·88	75·7	2·9	57·3	54·2
	Zinc concentrate......	7·8	......	1·03	58·42	0·02	4·34	7·3	55·4	2·4	15·6
	Zinc middling.........	7·6	......	1·32	32·27	0·08	3·18	9·2	30·1	9·6	11·2
	Tailing...............	74·2	......	0·06	1·07	0·02	0·31	4·0	9·6	23·2	10·6

(Ore 1,800 grammes —14 mesh ground 45 minutes in ball mill.)

Reagents—

Test No. 1—

To ball mill—		*To lead cell—*		*To zinc cell—*	
Soda ash	5·0 lb./ton	Aerofloat	0·2 lb./ton	Aerofloat	0·2 lb./ton
Cyanide	0·25 "			Copper sulphate	2·0 "

Test No. 2—

Soda ash	5·0 lb./ton	Aerofloat	0·2 lb./ton	Copper sulphate	2·0 lb./ton
Aerofloat	0·5 "			Aerofloat	0·2 "

Test No. 3—

Soda ash	5·0 lb./ton	TT	0·6 lb./ton	Copper sulphate	2·0 lb./ton
Cyanide	0·2 "			Aerofloat	0·2 "

Test No. 4—

Soda ash	2·0 lb./ton	Aerofloat	0·2 lb./ton	Copper sulphate	2·0 lb./ton
Cyanide	0·4 "			Xanthate	0·4 "

Test No. 6—

Soda ash	5·0 lb./ton	Aerofloat	0·2 lb./ton	Copper sulphate	2·0 lb./ton
Cyanide	0·5 "			Aerofloat	0·2 "

Refloated talc from lead concentrate—Agitated 15 minutes with—

Lime	10 lb./ton
Cyanide	1 "
Pine oil	To froth.

Test No. 8—

To ball mill—		*Talc float—*	
Soda ash	5·0 lb./ton	Pine oil	0·1 lb./ton
Cyanide	0·4 "	*To zinc cell—*	
To lead cell—		Copper sulphate	2·0 "
Aerofloat	0·1 "	Xanthate	0·1 "
		Pine oil	0·1 "

SUMMARY OF RESULTS ON LOW-GRADE ORE

It was found on attempting to float the ore that a large amount of gangue of a talcy nature was present and that this talcy gangue floated with the lead leaving the grade of the lead concentrate below that of a marketable product. Tests Nos. 1 to 4 were run varying the alkalinity of the pulp and trying different collecting reagents, all without success. In Test No. 6 the talc and lead were floated together, then the concentrate was treated by the addition of chemicals to depress the lead and an attempt made to float the talc. The results show that this was not successful. The method, however, has promise. In Test No. 8, it was decided to attempt to float the talc before concentrating the lead. The reagents used are given. In floating the talc by itself great care had to be taken not to add an excess of frothing reagent which would cause an increase in losses of lead, zinc, gold, and silver in that product. The results of this test are very encouraging and it is noteworthy that by eliminating the talc in such a manner the recovery of the gold and silver in the lead concentrate is increased to a rather remarkable extent.

In mill operation it would no doubt be profitable to table this talc product to recover part of the lead and precious metal values removed with it during flotation.

Report No. 277

EXPERIMENTAL TESTS ON GOLD ORE FROM RENO MINE, SHEEP CREEK DISTRICT, BRITISH COLUMBIA

J. S. Godard

Shipment.—A shipment of 110 pounds of ore was received November 9, 1927, from E. P. Crawford, Ymir, B.C.

Characteristics of the Ore.—The sample submitted is a high-grade gold ore in which the gold, though not visible, is nearly all free. No mineralization other than that of a greatly oxidized pyrite could be detected. The oxidization has been so great that the iron present resembled limonite. The gangue is a coarse-grained quartz.

Sampling and Analysis.—The entire shipment was crushed and sampled. The analysis of the head sample was as follows:—

Gold	2·51 oz./ton	Arsenic	Trace
Silver	1·28 "	Zinc	"
Copper	0·12 per cent	Lead	0·35 per cent
Iron	5·38 "	Insoluble	88·16 "

EXPERIMENTAL TESTS

Test No. 1—Amalgamation and Tabling the Amalgamation Tailing

Results—

Product	Weight, per cent	Au, oz./ton	Per cent of values
Concentrate	4·3	1·33	32·1
Middling	15·3	0·18	15·5
Tailing	53·4	0·109	32·7
Slimes	27·0	0·13	19·7
Amalgamation tailing from products		0·178	

Screen Test on Table Tailing—

+100 & 150	12·8	0·10	11·8
+200	17·2	0·11	17·4
−200	70·0	0·11	70·8
Average		0·109	

Test No. 2—Amalgamation

Screen Test on Amalgamation Tailing—

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+150	3·1	0·48	9·5
+200	21·3	0·17	23·1
−200	75·6	0·14	67·4
Average		0·157	

Test No. 3—Amalgamation

Amalgamation Tailing Screened on 200 mesh—

Mesh	Weight, per cent	Au, oz./ton	Per cent of values
+200	23·3	0·14	23·3
−200	76·7	0·14	76·7

Summary of Amalgamation Tests Nos. 1, 2, and 3—Head sample, 2·51 oz./ton.

Test No.	Amalgamation tailing Au, oz./ton	Recovery, per cent
1	0·178	92·9
2	0·157	93·7
3	0·140	94·5

Cyanidation Tests

Each of these tests was made on 1,000 grammes of ore. The ore was ground in 1 : 1 pulp, 0·05 per cent KCN, with lime equivalent to 4 pounds per ton. The grinding was done in porcelain pebble mills. The time of grinding was varied in each test.

Screen Test on Cyanide Tailings—

Test No.	Mesh	Weight, per cent	Au, oz./ton	Per cent of values
4	+65	1·0	27·31	16·1
	+100	16·5	2·61	25·5
	+150	32·8	1·34	26·0
	+200	10·5	1·19	7·4
	−200	39·2	1·08	25·0
	Average		1·69	
5	+100	0·9	40·03	21·1
	+150	31·7	2·20	40·9
	+200	10·9	1·58	10·1
	−200	56·5	0·84	27·9
	Average		1·70	
6	+200	33·1	2·25	71·7
	−200	66·9	0·44	28·3
	Average		1·04	
7	+150	3·6	10·01	26·2
	+200	14·7	2·21	23·5
	−200	81·7	0·85	50·3
	Average		1·38	

Summary of Tests Nos. 4, 5, 6, and 7.—Head sample, 2·51 oz./ton.

Test No.	Average tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
4	1·69	32·7	0·10	6·16
5	1·70	32·3	0·05	6·36
6	1·04	58·6	0·12	6·36
7	1·38	45·0	0·10	5·84

Amalgamation and Cyanidation of the Amalgamation Tailings

Screen Tests on Amalgamation Tailings—

Test No.	Mesh	Weight, per cent	Assay, Au, oz./ton	Per cent of values	Average tailing, Au, oz./ton
8	+150	18·3	0·28	24·8	0·207
	+200	27·7	0·21	28·2	
	−200	54·0	0·18	47·0	
9	+150	13·7	0·23	17·4	0·182
	+200	33·8	0·18	33·5	
	−200	52·5	0·17	49·1	
10	+200	29·1	0·21	36·4	0·168
	−200	70·9	0·15	63·6	
11	+200	17·4	0·17	20·4	0·145
	−200	82·6	0·14	79·6	

Screen Tests on Cyanide Tailings—

8	+65	1·1	0·30	3·3	0·099
	+100	34·0	0·14	48·2	
	+150	8·9	0·06	5·4	
	+200	22·7	0·07	16·1	
	−200	33·3	0·08	27·0	
9	+100	16·7	0·04	13·8	0·048
	+150	12·5	0·05	13·0	
	+200	35·1	0·04	28·9	
	−200	35·7	0·06	44·3	
10	+100	9·6	0·16	31·6	0·049
	+150	10·4	0·03	6·4	
	+200	16·9	0·03	10·4	
	−200	63·1	0·04	51·6	
11	+150	7·2	0·06	11·7	0·037
	+200	38·6	0·03	29·0	
	−200	54·2	0·04	59·3	

Summary of Tests Nos. 8, 9, 10, and 11—

1. Amalgamation—Head sample 2·51 oz./ton.

Test No.	Amalgamation tailing, Au, oz/ton	Recovery, per cent
8	0·207	91·7
9	0·182	92·7
10	0·168	93·3
11	0·145	94·3

2. Cyanidation of the amalgamation tailings.

Test No.	Amalgamation tailing, Au, oz./ton	Cyanide tailing, Au, oz./ton	Extraction per cent	Reagent consumption, lb./ton	
				KCN	CaO
8	0·207	0·099	52·2	0·30	5·6
9	0·182	0·048	73·6	0·20	5·5
10	0·168	0·049	70·8	0·20	5·6
11	0·145	0·037	74·4	0·20	5·5

3. Combined recoveries.

Test No.	Recoveries, per cent		
	Amalgamation	Cyanidation	Total
8	91·7	4·3	96·0
9	92·7	5·3	98·0
10	93·3	4·8	98·1
11	94·3	4·3	98·6

CONCLUSIONS

1. Amalgamation gave very satisfactory results on this ore, over 90 per cent of the gold being recovered. In Test No. 1, wherethe amalgamation tailings were tabled, 32 per cent of the gold in the amalgamation tailing was recovered in the table concentrate.

2. The results of the cyanide tests were unsatisfactory from a standpoint of extraction. The tailings were very high, especially on the coarse screen. There is no reason to believe that in mill practice the cyanide tailings would be any higher than those obtained when using amalgamation previous to cyanidation. The free gold would be retained in the circuit until sufficiently fine to be attacked by the cyanide. Former tests on similar ores have shown this to be the case, good extractions having been obtained where the ore was ground in a ball mill in closed circuit with a classifier. The small size of the shipment did not allow of such a test being made. The cyanide tests were therefore only indicative of cyanide and lime consumption. The cyanide consumption is very low, and the consumption of lime is moderate.

Report No. 278

THE CONCENTRATION OF THE SILVER ORE OF THE TORIC MINE, ALICE ARM, BRITISH COLUMBIA

C. S. Parsons

Shipments.—A shipment of 80 pounds was received November 17, 1927, followed by another shipment of 40 pounds received December 19, from the Toric Mines Co., Ltd., Vancouver, B.C. The shipments were from the company's property situated on the east side of the Kitsault river, about 18 miles from tidewater at Alice Arm, B.C. The narrow-gauge Dolly Varden railway is within a mile of the mine.

Character and Analysis of Ore.—The chief value in the ore is silver which occurs in the native state and as argentite, and a small amount of ruby silver. Considerable pyrite is present as well as some galena. Shipment No. 1 was the product from the discharge of the ball mill, ground to pass approximately 35 mesh. A screen analysis of the product is as follows:—

+48	8·1 per cent
−48 +65	14·7 "
−65 +100	26·8 "
−100+150	18·8 "
−150+200	13·5 "
−200	18·1 "

It showed on assay to contain 21·9 ounces of silver per ton. Shipment No. 2 was similar in character to No. 1, but the ore was higher grade. Assay showed it to contain 139·1 ounces of silver per ton.

Purpose of Experimental Tests.—The present milling process which consists of amalgamation and tabling does not give satisfactory recoveries. Tests were desired to determine whether flotation could increase the recovery of the silver values in a high-grade concentrate.

Test No. 1

Ore, 1,000 grammes, ground 10 minutes in ball mill with 0·75 lb./ton of 50-50 mixture of coal tar and coal-tar creosote.

Reagents added to cells: xanthate 0·1 lb./ton and pine oil 4 drops added to cells.

Product	Weight, per cent	Ag, oz./ton	Per cent of values
Concentrate	16·8	98·20	80·9
Tailing	83·2	4·68	19·1

Test No. 2

Ore, 1,000 grammes, ground 10 minutes in ball mill to brighten, with soda ash 2 lb./ton and 0·75 lb./ton coal-tar mixture.

Reagents added to cells: for concentrate No. 1, xanthate 0·1 lb./ton, pine oil 4 drops; for concentrate No. 2, copper sulphate 2 lb., xanthate 0·1 lb./ton, and pine oil 3 drops.

Product	Weight, per cent	Ag, oz./ton	Per cent of values
Concentrate No. 1	13·9	116·7	77·5 } 92·3
Concentrate No. 2	1·6	190·4	14·8
Tailing	84·5	1·9	7·7

Test No. 3

Ore, 1,000 grammes, ground 10 minutes in ball mill to brighten, with soda ash 5 lb. and coal-tar mixture 0·75 lb./ton.

Reagents added to cells: copper sulphate 2 lb., xanthate 0·1 lb./ton, and pine oil 5 drops.

Product	Weight, per cent	Ag, oz./ton	Per cent of values
Concentrate	7·9	256·55	92·3
Tailing	92·1	1·84	7·7
Screen analysis of tailing—			
− 65+100	3·88	17·59	34·1
−100+150	12·10	2·94	18·0
−150+200	8·92	1·88	8·5
−200	15·10	1·06	39·4

Test No. 4

Ore, 1,000 grammes, ground 30 minutes in ball mill with soda ash 5 lb., coal-tar mixture 0·75 lb./ton.

Reagents added to cells: copper sulphate 2 lb., xanthate 0·1 lb./ton, and pine oil 4 drops. Concentrate was cleaned.

Product	Weight, per cent	Ag, oz./ton	Per cent of values
Concentrate	15·6	109·88	83·1
Middling	13·6	7·24	4·8
Tailing	70·8	3·53	12·1
Screen analysis of tailing—			
−100+150	1·7	120·89	52·6
−150+200	6·9	9·38	18·3
−200	91·4	1·12	29·0

Test No. 6

Ore, 2,000 grammes, ground 15 minutes in ball mill with 1,500 c.c. water, soda ash 5 lb., coal-tar mixture 0·75 lb./ton.

Reagents added to cells: copper sulphate 1·5 lb., xanthate 0·05 lb./ton, pine oil 2 drops. Tailing was tabled.

Product	Weight, per cent	Ag, oz./ton	Per cent of values
Flotation concentrate	7·8	249·0	86·4
Table concentrate	3·03	48·8	5·6
Table tailing	89·17	2·38	8·0

Note: In this test the cell was not frothed so hard as in the preceding tests.

Test No. 7—High-grade Ore: (Shipment No. 2)

Ore, 2,000 grammes, which had been crushed to 14 mesh was ground for 35 minutes in ball mill with 1,500 c.c. water, soda ash 5 lb., and coal-tar mixture 0·75 lb./ton.

Reagents added to cells: copper sulphate 1·5, xanthate 0·1 lb./ton, and pine oil 2 drops. Grinding carried to practically all through 65 mesh. Flotation tailing was tabled.

Product	Weight, per cent	Ag, oz./ton	Per cent of values
Flotation concentrate	20·0	377·12	54·3 } 82·5
Table concentrate	10·0	393·00	28·2 }
Table tailing	52·0	41·24	15·4
Table slime	18·0	16·28	2·1
Head from products	....	139·10	

SUMMARY AND CONCLUSIONS

These tests show that more than 85 per cent of the silver remaining in the ball mill discharge after amalgamation can be recovered by flotation. The grade of concentrate produced should exceed 225 ounces to the ton. The screen analysis made on the flotation tailing clearly indicates that this product should be tabled in order to save any coarse native silver which on account of its malleability is rolled out by the action of the balls, and owing to its shape and character, is not saved by flotation.

The tests were made in a combination bubble column type of machine, similar to the Fahrenwald principle. The successful flotation of the ore apparently depends on the introduction of copper sulphate to the flotation circuit. The procedure in the use and addition of the reagents was to add a heavy coal tar dissolved in an acid coal-tar creosote to the grinding mill in order to obtain a heavy filming of the metal and sulphides. Soda ash in just sufficient quantities to keep the pulp alkaline was also added to the mill. This was to deflocculate the slimes in order to increase the effect of the oil. Copper sulphate was added after grinding in order to clean and flocculate the sulphides and native silver. The small amount of xanthate was then added directly to the flotation cell, together with a frothing oil, such as steam-distilled pine oil. It is probably necessary to give the copper sulphate a short interval of time in which to act before flotation is started.

Report No. 279

CONCENTRATION OF A LEAD-SILVER ORE FROM THE CONFEDERATION GROUP, CAMBOURNE, B.C.

C. S. Parsons

Shipment.—A shipment of 179 pounds of ore was received from the Lardeau Mines Exploration, Limited, of Vancouver, B.C., November 18, 1927. The ore was from the Confederation group near the head of Poole creek, at Cambourne, B.C., Claim Alma No. 2, lot 3051 C.G.

Character and Analysis.—The shipment received consisted of a lead-silver ore containing only a trace of zinc and gold. The ore is quite siliceous and a part of the lead is disseminated throughout the gangue in relatively fine particles. Analysis:—

Lead	24·75 per cent	Silver	8·93 oz./ton
Zinc	0·06 "	Gold	trace

Purpose of Experimental Tests.—Concentration tests were run in order to determine what recovery and grade of concentrate could be produced from the samples.

Test No. 1

The ore was crushed to pass 14 mesh and 2,000 grammes taken for a flotation test. This lot was ground in a small ball mill until approximately 90 per cent would pass 65 mesh. The pulp was made alkaline with soda ash; xanthate and pine oil were used to float the lead. The following results were obtained:—

Product	Weight, per cent	Analysis			Per cent of values		
		Pb, per cent	Zn, per cent	Ag, oz./ton	Pb	Zn	Ag
Concentrate	30·6	75·85	0·05	25·7	92·6		89·9
Middling	6·2	14·83	0·21	7·73	3·7		5·5
Tailing	63·2	1·46	0·03	0·64	3·7		4·6

SUMMARY

The ore presents no problem in concentration as shown by the test. An excellent recovery of both the silver and the lead was obtained in a high-grade concentrate. The grade of the concentrate is not surprising, considering that the sample contained over 24 per cent lead.

III

REPORTS OF INVESTIGATIONS: NON-METALLIC LABORATORY

Report No. 280

THE DRY GRINDING OF CALCITE TAILINGS FROM THE KINGDON LEAD MINE, GALETTA, ONTARIO

R. K. Carnochan

Shipment.—Six cans of calcite tailings, shipping weight 700 pounds, were received February 22, 1927, from the Kingdon Mining, Smelting, and Manufacturing Co., Galetta, Ont.

Purpose of Tests.—Grinding tests were desired on the shipment for the production of samples which could be submitted to prospective purchasers. It was also desired that as much silica as possible be eliminated during grinding and a suitable method determined for grinding the calcite tailings on a commercial scale.

Characteristics of the Tailings.—The tailings are from the concentration of the lead ore of the Kingdon mine at Galetta and consist of white crystalline calcite containing a small amount of impurities, mostly selenite, feldspar, quartz, hornblende, augite, and white and brown mica. The tailings assay 14·9 per cent insoluble.

EXPERIMENTAL TESTS

Grinding in Raymond Pulverizer in Circuit with Gayco Air Separator

Part of the shipment was dried and ground in a No. 0000 Raymond pulverizer. This machine is of the beater-blade type. The discharge of the Raymond was fed to a 30-inch Gayco air separator, the oversize being fed back to the Gayco twice to remove fines, and the Gayco fines from the three passes combined. This method of grinding and separating gave a product 99·60 per cent −325 mesh. The product was a little off colour due to the dark impurities in the material and also to iron wearing off the beater blades of the pulverizer. At the end of this run some Gayco oversize remained which was combined with the clean-out of both machines. This product analysed 17·04 per cent insoluble, and the Gayco fines analysed 10·63 per cent insoluble, showing that the calcite grinds faster than the impurities.

Grinding in Pebble Mill with Separation in Gayco Air Separator

The remainder of the shipment was dried and ground in a 2- by 3-foot pebble mill lined with porcelain. The mill was charged with 65 pounds of tailings and 130 pounds of pebbles and run for one hour, then cleaned out and the ground product passed through the 30-inch Gayco separator. The oversize was re-run two to four times to remove fines. In all, five charges were run. All fines were combined, and the oversize products were combined with the clean-up of the separator. The fines were 98·89 per cent −325 mesh and analysed 10·12 per cent insoluble. The oversize analysed 20·50 per cent insoluble. The fines were slightly dark in colour due to impurities present in the tailing, but not so dark as were those produced by grinding in the Raymond pulverizer.

CONCLUSIONS

The tailing could be ground commercially to 99·6 per cent −325 mesh by a high-side Raymond roller mill, a Hardinge mill fitted with a rotary and superfine separator, or by means of a pebble mill followed by a Gayco air separator.

In the last two methods of grinding it might be possible to remove some of the silica from the tailings by means of screening the return to the pebble mill.

The Gayco fines from each of the tests were shipped to the Kingdon Mining, Smelting and Manufacturing Co., so that the company might submit samples to interested people, and obtain opinions of possible uses for the product.

Report No. 281

ELIMINATION OF IRON FROM SAND SAMPLES

R. K. Carnochan

Shipment.—Three samples of sand, total weight 63 pounds, were received from Silico Limited, Montreal, on April 8, 1927. The samples were marked:—

Run of mill	33 pounds
Foundry	15 "
Standard	15 "

Screening and Analysis.—Part of the standard sample was screened on 40 and 65 mesh and the −40+65 used as a fourth sample, designated Standard −40+65. Sampling and analysis showed these samples to run as follows:—

—	SiO_2 per cent	Fe_2O_3 per cent	Al_2O_3 per cent
Run of mill	98·90	0·07	0·32
Foundry	99·11	0·09	0·41
Standard	99·02	0·07	0·38
Standard −40+65	99·37	0·056	0·26

Experimental Tests.—The four samples were run separately over a magnetic pulley. This reduced the iron content slightly by removing a small amount of metallic iron introduced by the grinding machinery used in preparing it. The following table shows the reduction in the iron content obtained in each sample:—

—	Per cent Fe_2O_3 before treatment	Per cent Fe_2O_3 after treatment
Run of mill	0·07	0·06
Foundry	0·09	0·06
Standard	0·07	0·064
Standard −40+65	0·056	0·052

Report No. 282

THE CONCENTRATION OF A GARNET ROCK FROM LANGLADE, QUEBEC

R. K. Carnochan

Shipment.—Five bags of garnet rock, net weight 526 pounds, were received July 9, 1927. The material had been mined near Langlade, Baudin township, Quebec, and sent in by the Langlade Garnet Syndicate, 109 Mountain Hill, Quebec.

Purpose of Tests.—The shippers desired tests on the rock to determine the amount and quality of garnet concentrates that could be produced. They also wanted samples of the finished concentrates to submit to garnet paper manufacturers to determine if the concentrates were suitable for abrasive papers.

Characteristics of the Rock.—The shipment consisted of small crystals of garnet, the largest being one-half inch in diameter, in a gangue of quartz, hornblende, pyroxene, and black mica.

EXPERIMENTAL TESTS

After the selection of a few specimens, the ore was all crushed to about 1-inch in a small jaw crusher, one-sixteenth of the lot was cut out by a Jones riffle sampler and used in Test No. 1.

Test No. 1

One-sixteenth of the lot at 1-inch was crushed in the small jaw crusher and screened after each crushing on an 8-mesh screen. The crushing and screening were continued until all the rock passed through the screen. The −8 mesh was then screened on 10, 14, 20, 40, and 100 mesh, which gave:—

− 8+10	2,060 grammes	=	15·0 per cent
−10+14	1,442 "	=	10·5 "
−14+20	2,110 "	=	15·3 "
−20+40	3,866 "	=	28·1 "
−40+100	2,768 "	=	20·1 "
−100	1,518 "	=	11·0 "

The three coarser sizes were jigged in a small Richards pulsating jig and the three finer tabled on a small Wilfley table. The six concentrates obtained were run separately over an Ullrich magnetic separator. Each concentrate was passed through the machine once, making a garnet concentrate and a tailing. In some of the sizes some iron collected at the feeder, making a separate iron product.

Size	Jig or table concentrate, grammes	Jig or table tailing, grammes	Ullrich concentrate, grammes	Ullrich tailing, grammes	Ullrich iron product, grammes	Per cent garnet Ullrich concentrate
−8+10	589·9	1,460·2	580·6	8·2	....	66
−10+14	406·2	987·2	401·0	4·1	....	76
−14+20	615·0	1,394 3	609·8	7·2	0·3	81
−20+40	1,408·6	2,422·5	1,326·0	81·9	2·1	69
−40+100	854·5	1,861·0	793·3	54·6	0·1	
−100	307·7	907·2	245·6	46·5		

The total garnet concentrate recovered, 3,956·3 grammes, equals 28·7 per cent of the rock treated.

The first Ullrich garnet concentrates, less the samples taken for analysis, were re-run through the Ullrich machine, first using a low current to lift black mica and hornblende, and then using a stronger current to lift the garnet.

Size	Product	Weight, grammes	Per cent garnet
−8+10	Magnetic	101·0	
	Garnet concentrate	419·2	72
	Tailing	8·7	
−10+14	Magnetic	42·5	
	Garnet concentrate	302·0	80
	Tailing	2·0	
−14+20	Magnetic	32·7	
	Garnet concentrate	526·0	86
	Tailing	6·2	
−20+40	Magnetic	19·0	
	Garnet concentrate	1,197·0	78
	Tailing	76·7	
−40+100	Magnetic	10·7	
	Garnet concentrate	732·8	
	Tailing	7·6	
−100	Magnetic	3·5	
	Garnet concentrate	213·4	
	Tailing	3·7	

The second Ullrich garnet concentrates were treated on a Huff electrostatic separator, but it was found that very little impurities were removed and to accomplish even this the concentrates had to be passed through the machine a number of times.

Summary.—Test No. 1 shows that the garnet rock can be concentrated to 80 per cent garnet concentrate by jigging and tabling, and re-concentrating the jig and table concentrates by the use of a magnetic machine operating so as to lift first a magnetic product, and then a garnet product, leaving a tailing. Electrostatic separation was not found to be of any real value in cleaning the garnet concentrates.

Test No. 2

Out of the material that had been crushed to 1-inch size, a thirty-second part was cut and crushed to—20 mesh by means of a small breaker and small rolls. After each pass through the breaker or rolls, the crushed product was screened on the 20-mesh screen. A small amount of black mica remained on the 20-mesh screen after the rest of the garnet rock had been crushed fine enough to pass through. This was kept separately.

+20 Black mica	41 grammes	0·5 per cent
−20+40	3,150 "	40·1 "
−40+100	2,875 "	36·6 "
−100	1,785 "	22·8 "

The three sizes, −20+40, −40+100, and −100, were tabled separately on a small Wilfley table. The resulting three garnet concentrates were treated separately on an Ullrich magnetic separator, making a garnet concentrate and tailing in each case. Some iron collected at the feeder while running each size. This was kept separately.

Size	Table		Ullrich			Per cent garnet Ullrich concentrate
	Concentrate	Tailing	Concentrate	Tailing	Iron	
−20+40	1,237·7	1,890·2	1,135·9	96·5	0·6	74
−40+100	987·9	1,844·0	919·3	56·9	1·0	
−100	281·2	1,174·5	246·2	12·2	3·1	

The Ullrich concentrate produced, 2,301·4 grammes, equals 29·3 per cent of the garnet rock treated.

The Ullrich garnet concentrates, after the removal of a small sample from each, were re-run through the Ullrich, first with a low current to lift a magnetic product, and leave the garnet, and then with a stronger current to lift the garnet and leave a tailing.

Size	Product	Weight, grammes	Per cent garnet
−20+40	Magnetic	40·8	
	Garnet concentrate	966·5	79
	Tailing	21·2	
−40+100	Magnetic	23·3	
	Garnet concentrate	798·7	
	Tailing	10·4	
−100	Magnetic	6·0	
	Garnet concentrate	212·4	
	Tailing	2·5	

The second Ullrich concentrates were treated on the Huff electrostatic separator, but although some impurities were removed, the concentrates had to be passed many times through the separator to obtain this result.

Summary.—Test No. 2 shows that there would be no advantage in crushing garnet rock finer and treating it all on tables. The amount of garnet recovered is slightly increased, from 28·7 to 29·3 per cent, the grade remaining about the same, but to offset this gain in recovery there would be an increased milling cost as table concentration is more expensive than jigging and the finer garnet concentrate produced would not be likely to sell as readily as the coarser concentrate obtained by jigging.

Test No. 3

This test was run to produce enough finished garnet concentrate to submit to garnet paper manufacturers to determine whether the concentrates would be suitable for making abrasive paper. The procedure was the same as in Test No. 1. Seven-eighths of the material crushed to 1-inch was used. After crushing and sizing the weights were:—

− 8 +10	60 pounds	= 13·1 per cent
−10 +14	42 "	= 9·2 "
−14 +20	72 "	= 15·7 "
−20 +40	122 "	= 26·6 "
−40 +100	97 "	= 21·2 "
−100	65 "	= 14·2 "

The table and jig concentrates were run through the Ullrich twice, first lifting out a magnetic product with a low current and then the garnet concentrate with a stronger current. The final garnet concentrates weighed and analysed as follows:—

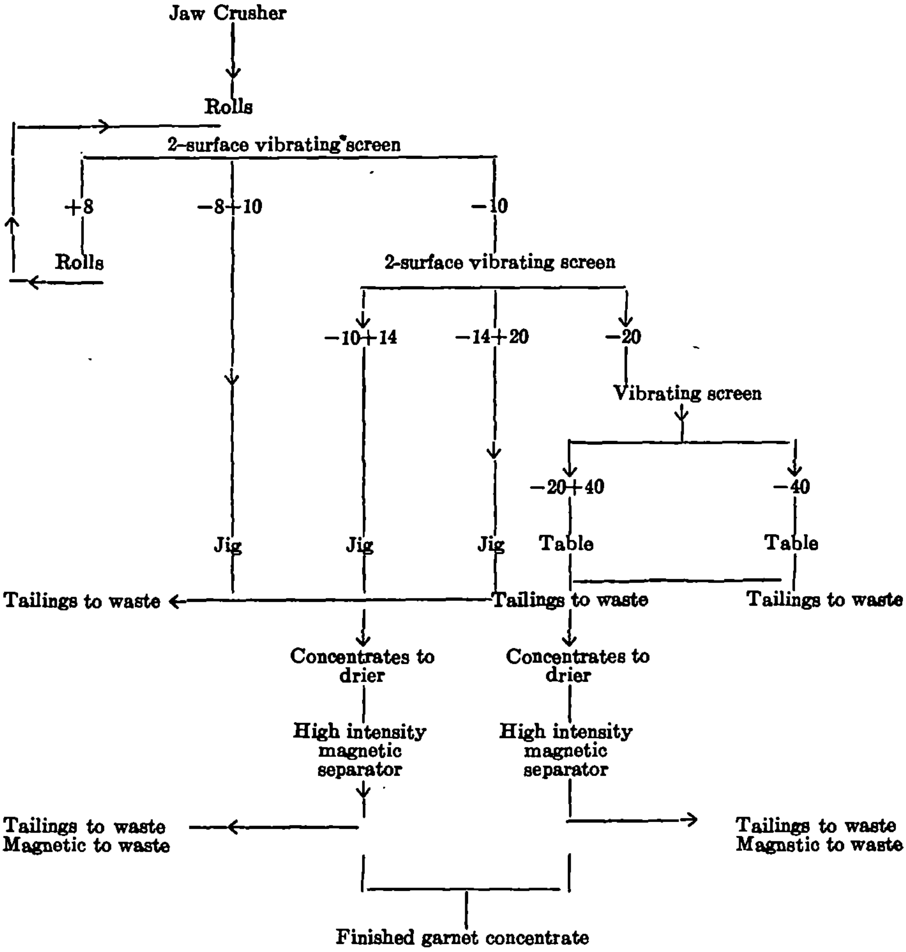
		Garnet
− 8 +10	14·5 pounds	75 per cent
−10 +14	12·0 "	70 "
−14 +20	25·0 "	80 "
−20 +40	44·0 "	79 "
−40 +100	28·0 "	..
−100	3·0 "	..

The total garnet concentrate recovered, 126·5 pounds, equals 27·6 per cent of the garnet rock treated, and contains about 80 per cent garnet.

CONCLUSIONS

1. About 28 per cent of the rock can be recovered as a garnet concentrate, running approximately 80 per cent garnet.
2. To effectively concentrate the rock it should be crushed to −8 mesh.
3. The sizing used for jigging in the final test may be too close, but the rock is difficult to concentrate, and the closer sizing aids in concentrating. Due to the low value of the finer sizes of garnet concentrate and the small amount of these that can be produced by a table, it would seem advisable not to size for tabling as close as was done in the final test.
4. Some type of high intensity magnetic separator must be used to clean the jig and table concentrates.

5. The following flow-sheet is recommended for the concentration of the garnet rock:—



Report No. 283

THE DRYING OF EPSOM SALTS

R. K. Carnochan

Purpose of Tests.—Deposits of Epsom salts occur in British Columbia, but due to the long haul to markets and consequent high freight charges it is not possible to ship to eastern points. Epsom salts contain 51·2 per cent of water of crystallization and if this could be removed cheaply, the anhydrous salt could be shipped at a great reduction in freight and handling charges, and the production of Epsom salts made a profitable venture.

In order to find a suitable method of drying the salts the following test work was carried out.

Experimental Tests.—Epsom salts when heated liquefy in the water of crystallization forming a paste. This phenomenon makes the drying of the salts difficult. After a careful consideration of the different methods of drying materials it was decided that a rotary drier with suitable alterations would be the most likely method of drying the salts.

A small gas-fired rotary drier was fitted up with a screw feeder, and a scraper at the top which extended the full length of the cylinder. The purpose of the screw feeder was to keep the salts moving into the drier when they formed into a paste. The scraper was to remove the paste from the inside of the cylinder at the top and drop it to the bottom, and in this way keep the inside of the drier from becoming caked.

A number of runs were made with this rotary drier, using as feed ordinary commercial Epsom salts purchased locally. The results of these tests were quite satisfactory. As expected the salts became pasty inside the feeder but the screw kept them in motion. Near the feed end of the drier the scraper was very useful in keeping the cylinder clean. As the charge moved along the cylinder it became drier and formed small nodules which did not stick to the inside of the cylinder. The discharge of the drier contained 10 per cent water of crystallization.

CONCLUSIONS

1. For small-scale work the results are very encouraging and indicate that the method used would be suitable for large-scale work.
2. During the test work some experiments were made on the drying of Glauber's salts. These indicated that this salt could be dried in a rotary drier as described, provided with the addition of an arrangement to feed back to the drier from three-quarters to two-thirds of the discharge in order to absorb the water formed when Glauber's salts are heated, and so have a paste to dry and not salts and liquid.

Report No. 284

THE FLOTATION OF GRAPHITE TAILINGS FROM DOMINION MINE, BUCKINGHAM, QUEBEC

R. K. Carnochan

Shipment.—One hundred bags of graphite tailings, shipping weight 10,000 pounds, were received July 18, 1927, from the Dominion mine, Buckingham, Que.

Purpose of Tests.—It was desired to determine the graphite content of the tailing, and whether it could be commercially recovered.

Characteristics and Analysis.—The tailings came from the dump of the Dominion mine and consisted of finely ground material containing graphite in flakes, some of the flakes being fairly large. A small portion of the tailing was taken from each bag and mixed to form a grab sample on which to conduct small-scale tests. This grab sample assayed 3·30 per cent carbon.

Experimental Tests.—Four small-scale flotation tests were made on the grab sample to indicate the most likely lines of treatment. In all these tests only a small proportion of No. 1 flake was recovered, and the concentrates assayed very low in carbon, the highest assay being 67 per cent and the others 59, 53, and 40 per cent.

Two large-scale flotation tests were then made. These were numbered 5 and 6, and the following are the results:—

Test No. 5—

	Per cent carbon
Tailing used in test, 3,135 pounds, assaying	3·29
Concentrates +80 mesh	54·52
Concentrates −80 mesh	21·55

Test No. 6—

Tailing used in test, 3,047 pounds, assaying	2·29
Concentrate +80 mesh	61·89
Concentrate −80 mesh	25·41

These large-scale tests show that the grade of concentrate produced is too low to be of commercial value. The tailing is also too low in graphite content to be of commercial value.

A small-scale test was made and the products screened on 28 mesh. Flake picked from the screened products assayed as follows:—

	Per cent carbon
Flake from concentrate	49·33
Flake from middling	46·50
Flake from tailing	55·31

This shows that some foreign substance is present in what appears to be flake, and accounts for the low grade of concentrates produced.

In order to obtain the graphite content of the entire shipment, the tailings not used in the above tests were weighed and sampled:—

Weight................ 3,782 pounds Assay........... 2·14 per cent carbon

CONCLUSIONS

1. The tailings submitted are too low in graphite content to be of commercial value. The shipment contained only 2·55 per cent carbon.
2. The flake is very impure, and because of this it is impossible to produce a high-grade concentrate.

Report No. 285

THE WET MILLING OF ASBESTOS ROCK FROM DANVILLE, QUEBEC

R. K. Carnochan

Shipment.—A shipment of 60 bags, gross weight 6,777 pounds, was received July 21, 1927. It was submitted by Selective Treatment, Limited, of Montreal, and consisted of three lots of rock from different parts of the Nicolet mine, 8½ miles from Danville, Quebec.

Purpose of Tests.—It was desired to determine how much asbestos fibre could be recovered from each of the three lots by the wet-milling process for asbestos rock.

Arrangements for Experimental Tests.—Arrangements were made by Samuel H. Dolbear, of Selective Treatment, Limited, holders of the wet-process patents, whereby he would supervise the test work. It was the intention of this company to erect a mill on the property, employing the wet-treatment process, should the rock be of high enough grade to warrant it.

Characteristics of the Rock.—The three lots consisted of typical coarse fibre rock of the Quebec district. The rock was medium green in colour and the fibre was white and soft. Very little long fibre was present.

Sampling.—Each lot was crushed separately to about 1½ inch in an 8 by 12-inch jaw crusher, and one-twentieth cut out for a head sample by a Vezin sampler.

Experimental Tests

Each of the lots was ground in a 4½-foot x 13-inch Hardinge mill, as much water as possible being fed with the rock, extra water being added by means of a pipe through the mill discharge. The idea of this excess of water was to float the fibre out of the mill as soon as it was free. A charge of 500 pounds of large steel balls was used so that the rock would be cracked and not ground fine. The mill discharge was screened on a $\frac{1}{32}$-inch round, punched-hole, brass screen mounted over a tank. The fibre deposited on the screen was raked off and put into a tub by hand. The overflow from the tank under the screen went to another tank and then to a large Callow cone. The cone overflow was clear and flowed to waste. The screen oversize was run through a two-compartment James jig to remove sand. The jig fibre discharge was screened on the $\frac{1}{32}$-inch punched screen, the screen oversize being raked to one side and removed by hand, the undersize material going into a tank underneath the screen, the overflow from this tank going to another tank and thence to the Callow cone. The contents of the first tanks of the mill and jig runs were mixed with the jig hutch product and tabled on a large Wilfley table. The contents of the second tanks and mill run were also mixed and tabled. The material in the Callow cone was discarded, although it included a lot of fine fibre. It was not considered worth while to recover this fibre as the amount of long and medium fibres that could be recovered would indicate if the ore were worth milling.

The fibre obtained from the screen and by tabling was put into bags and pressed to remove most of the water, then dried and fluffed up or fiberized in a fiberizer. After fiberizing the table fibre was screened on a 16-mesh vibrating screen.

Summary.—The following tables show the weights of products and results of the tests on the three lots:—

Lot No. 1—

	Pounds
Weight of rock treated	2,159
Oversize fibre from screen	65
Fibre from table	543
Fibre from table, +16 mesh	279
Fibre from table, −16 mesh	264
Average test oversize fibre	0–5·03–5·98–4·99
Average test table fibre, +16	0–0–0·67–15·33

	Per cent	Lb./ton
First box fibre	......	trace
Second "	0·946	18·92
Third "	1·666	33·32
Fourth "	25·548	510·96
	28·160	563·20

Lot No. 2—

	Pounds
Weight of rock treated	1,920
Oversize fibre from screen	124·85
Fibre from table	459
Fibre from table, +16 mesh	122
Fibre from table, −16 mesh	337
Average test oversize fibre	0–0–8·78– 7·22
Average test table fibre, +16	0–0–1·11–14·89

	Per cent	Lb./ton
First box fibre	......	nil
Second "	......	nil
Third "	4·009	80·18
Fourth "	26·399	527·98
	30·408	608·16

Lot No. 3—

	Pounds
Weight of rock treated	2,288
Oversize fibre from screen	158
Fibre from table	426
Fibre from table, +16 mesh	154
Fibre from table, −16 mesh	272
Average test oversize fibre	0–0–9·78– 6·22
Average test table fibre, +16	0–0–0·24–15·76

	Per cent	Lb./ton
First box fibre	......	nil
Second "	......	nil
Third "	4·322	86·44
Fourth "	21·203	424·06
	25·525	510·50

Average of Three Lots—

	Per cent	Lb./ton
First box fibre	......	trace
Second "	0·315	6·30
Third "	3·332	66·64
Fourth "	24·383	487·66
	28·030	560·60

	Pounds 2nd box	Pounds 3rd box	Pounds 4th box
Recovered from one ton of rock	6·30	66·64	487·66
67·2 pounds of 0–1½–9½–5=	6·30	39·90	21·00
Leaving		26·74	466·66
85·57 pounds of 0–0–5–11=		26·74	58·83
Leaving			407·83

Therefore, from one ton of rock there would be recovered 67·2 pounds of 0–1½–9½–5, 85·57 pounds of 0–0–5–11, and 407·83 pounds of fourth box material.

67·2	pounds of 0–1½–9½–5 at $70 per ton	$2·35	
85·57	" 0–0–5–11 at $25 per ton	1·07	
407·83	" fourth box at $10	2·04	
			$5·46

(Approximate market prices, Thetford mines, as of February, 1928.)

This shows that there would be recovered from one ton of rock, fibre having a value of $5.46 at the mine.

CONCLUSIONS

An average recovery of fibre having a value of $5.46. can be made from a ton of the asbestos rock submitted, by the wet process of milling. It is commercially possible to mine and mill asbestos rock of this grade.

Report No. 286

THE ELIMINATION OF IRON FROM SANDSTONE FROM EAST TEMPLETON, QUEBEC

R. K. Carnochan

Shipments.—A number of shipments of sandstone were received at the Ore Testing Laboratories from the Canada Glass Products, Limited, Ottawa. The samples were from their property at East Templeton, Quebec, and ranged in size from small lots of a few pounds, to large lots of 1,000 to 3,500 pounds. These were received during the period from December, 1926, to January, 1928, and had a total weight of 7,956 pounds.

Characteristics of the Sandstone.—When crushed to its natural grain it consists of rounded quartz particles grading in size from 10 mesh down. Some pyrite is present, free and attached to the quartz grains, as well as a small number of rusty particles of weathered pyrite. Examination shows that the pyrite formed after the deposition of the sand, filling some of the voids between the grains. On crushing, some of this pyrite breaks free, and part remains attached to the quartz grains.

Purpose of Tests.—The iron content of the sandstone submitted was 0·15 per cent to 0·30 per cent Fe_2O_3. For the making of clear glass a sand having an iron content of 0·07 per cent or lower is necessary, and it was desired that tests be conducted to determine a method of reducing the iron content of the sandstone to this amount.

EXPERIMENTAL TESTS

Roasting and Magnetic Separation

Small samples of crushed and washed sandstone were roasted in a gas-fired muffle furnace to a dull red heat. After cooling, the product was run through an Ullrich magnetic separator, using as strong a magnetic field as possible. Analyses of the non-magnetic products showed the iron content to be reduced to about 0·06 per cent Fe_2O_3. The roasted samples were a faint pink in colour. The following table gives the results of these tests:—

Test	Head		Non-magnetic product, Fe_2O_3 per cent
	SiO_2 per cent	Fe_2O_3 per cent	
1	99·40	0·29	0·076
2	98·92	0·15	0·078
3	99·20	0·21	0·062
4	98·66	0·20	0·050
5	....	....	0·052

The above tests made on five different samples of the sandstone show that roasting and high intensity magnetic separation will reduce the iron content to the desired limit, but it is doubtful if the method would be economical to use on a material of such low value per ton as glass sands.

Flotation.

This method gave poor results. The pyrite is too coarse for flotation, only a small portion of it floated, and this at a high reagent cost. Flotation is not feasible for the removal of the pyrite.

Tabling.

Small tests made on a laboratory table showed a band of pyrite which was readily removable, indicating that tabling could be used for the removal of any free pyrite in the sands. Examination of the table tailing under the microscope showed it to contain no free pyrite, but to contain some pyrite attached to the sand grains, especially the coarser sand pieces.

Large-scale Tests.—Forty bags of the washed sand, approximate dry weight 3,315 pounds, were run over a quarter-size Plat-O concentrating table. The maximum feed rate to the table was over 2,000 pounds per hour. A clean band of pyrite reported on the table. Considerable sand was cut into this concentrate which was later cleaned up, giving 7·5 pounds of clean pyrite concentrate assaying:—

Iron	32·83 per cent
Sulphur	37·21 "
Gold	trace

A sample was cut out from the tailing and a screen analysis of this sample was as follows:—

Screen size	Weight, per cent	Fe, per cent		Fe_2O_3, per cent	
+ 28	8·47	0·32		0·46	
− 28+ 35	32·33	0·14	0·14	0·20	0·20
− 35+ 48	35·46	0·13		0·19	
− 48+ 65	14·91	0·15		0·21	
− 65+ 80	3·03	0·14		0·20	
− 80+100	3·97	0·16		0·23	
−100	1·83	0·41		0·59	

The washed sands as received assayed 0·23 per cent iron or 0·33 per cent Fe_2O_3. As stated above, a microscopic examination of the table sands showed no free pyrite, but some attached pyrite. This attached pyrite is more with the +28 mesh sands. The −100 mesh sands also assay high in iron due no doubt to a certain amount of washing action, and perhaps to some very fine pyrite. Both the +28 mesh sands and the −100 mesh sands could be readily screened out, giving a product representing 90 per cent of the sands and assaying 0·14 per cent iron or 0·20 per cent Fe_2O_3.

CONCLUSIONS

No doubt the tabling of the washed sands will eliminate the free pyrite and reduce the iron content in the finished product materially. It is possible that classification before tabling would assist in the removal of the grains containing the attached pyrite and lower still further the iron content in the finished product. Both these operations are comparatively cheap and should add but a few cents per ton to the operating costs. As shown above, table capacities on this material are quite large.

Report No. 287

REMOVAL OF IMPURITIES IN GYPSUM FROM FALKLAND, B.C.

R. K. Carnochan

Two shipments of gypsum having a total weight of 475 pounds were received on January 5 and August 13, 1927, from the British Columbia Gypsum Company,* a subsidiary of the Manitoba Gypsum Company* with head offices at Winnipeg, Manitoba. The gypsum was from their quarries at Falkland, B.C. Tests were made to determine a suitable method for removing impurities which caused dark spots in the cast gypsum. Magnetic separation after calcining was found useful. Details of the test work will be published in the report "Gypsum in Canada," by L. H. Cole.

Report No. 288

COMPARATIVE GRINDING TESTS WITH QUARTZ PEBBLES FROM CYPRESS HILLS, ALBERTA, AND DANISH FLINT PEBBLES

R. K. Carnochan

A shipment of three bags of pebbles, weight 239 pounds, was submitted by L. H. Cole of the Mineral Resources Division on June 6, 1927. Comparative grinding tests were desired to determine if these pebbles could be used in pebble mills. It was found that although not so hard as Danish or French flint pebbles, they could be used for grinding purposes, and their use locally would probably be economical. Details of the test work are given in Report No. 686, "Silica in Canada," Part II—Western Canada, by L. H. Cole.

Report No. 289

WASHING TESTS ON SILICA SANDS FROM BLACK ISLAND, LAKE WINNIPEG, MANITOBA

R. K. Carnochan

A shipment of 115 bags of sand weighing 18,000 pounds was received August 25, 1926, from L. H. Cole of the Mineral Resources Division. Washing and classification tests were made, and a product suitable for a

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*Now part of the amalgamation known as Canada Gypsum Company.

glass sand was produced. Several smaller samples of sand were also received, three from lake Winnipeg on which washing tests were conducted, and one from Saskatchewan which was treated on a magnetic separator. The details of these tests appear in Report No. 686, "Silica in Canada," Part II—Western Canada, by L. H. Cole.

Report No. 290

GRINDING AND SEPARATION TESTS ON DIATOMITE FROM LAKE MICHEL, CHERTSEY TOWNSHIP, QUEBEC

R. K. Carnochan

Grinding and separation tests were made on a sample of diatomite from lake Michel, Quebec, weight 52 pounds, received March 30, 1927. The shipment was made by V. L. Eardley-Wilmot of the Mineral Resources Division. Details of the tests are published in Report No. 691, "Diatomite" by V. L. Eardley-Wilmot.

Other Non-Metallic Work

A large quantity of sandstone, 7,000 pounds received February 25, 1927, and 4,000 pounds received August 9, 1927, was submitted by the Experimental Farm for grinding to 6 mesh, the finished product to be used by the Horticultural Division in their experimental work.

A quantity of modelling clay, 5,200 pounds, was mixed and ground for the Ottawa Public School Board.

IV

REPORT OF INVESTIGATIONS: HYDROMETALLURGICAL LABORATORY

R. J. Traill

INTRODUCTION

A further series of experiments in the hydrometallurgical treatment of chalcopyrite concentrates was made during the year with quite encouraging results as to extractions of iron, copper, and sulphur. Two different types of ore concentrates were used and the results obtained show that the method of treatment is technically sound. Some difficulty has been encountered in obtaining an iron electrolyte completely free from other metal impurities, especially lead and zinc, and at present there seems little possibility of overcoming this difficulty.

An effort was made to obtain some data regarding the effect of certain conditions on iron deposition, such as the presence of impurities, ferric ions, and acid. Little or no effect has been noted generally on the tests made, but rather than draw any definite conclusions from the results, it is planned to carry out similar tests on a somewhat larger scale.

THE HYDROMETALLURGICAL TREATMENT OF HIGH-GRADE IRON-COPPER SULPHIDE CONCENTRATES

R. J. Traill, W. R. McClelland, J. D. Johnston

Consequent to the encouraging results that were obtained in the preliminary tests on a method of treatment for high iron-copper sulphide concentrates last year and which were reported in the 1926 Report of Investigations, further tests were carried out with the purpose of checking and corroborating the results obtained in 1926.

Two different ore concentrates were used in this work, namely a pyrite-chalcopyrite concentrate from Eustis ore, Eustis, Que., and a pyrrhotite-chalcopyrite concentrate from Anyox, B.C. While the same method of treatment is adaptable to both, a slight variation may be made in the process of treatment for the pyrrhotite-chalcopyrite product. The general method as outlined and discussed in the report of last year consists of first roasting the iron-copper concentrate with an excess of sulphur present, the sulphur being recovered as elemental sulphur, and the roast product leached with ferric chloride, resulting in the solution of the iron and copper as chlorides. The copper is removed from the solution by precipitation with iron—sponge iron or electrolytic iron being the most suitable for this purpose—the cement copper being then subjected to an oxidizing roast, leached with sulphuric acid, and recovered by electrodeposition as electrolytic copper. Iron is recovered from the copper-free solution by electro-deposition as pure iron if the solution is further purified of lead, zinc, or other impurities before electro-deposition; or as impure electrolytic iron containing small quantities of lead and zinc, which may be subjected to a melting process for production of high-grade or alloy steel.

The various steps in the process and the general conditions obtaining are set out briefly in the following paragraphs.

Roasting.—The roasting was carried out in a muffle-type furnace using vitreosil muffles varying in size according to charge. The muffles were fitted with rectangular-shaped metal boxes which served as sulphur condensers and which were kept cold by running water. It would appear from our experience that the condensation space required to give best results, both as to quality of sulphur and efficient roasting, should be at least two to three times as large as that of the roast container.

The charge to the muffles consisted of raw concentrate mixed with the dry leach residue from an equivalent ore charge. This gives the necessary excess sulphur required for the best roasting conditions. The charged muffles with condensers attached, were placed in the furnace cold and the heat applied until the temperature reached around 750 to 800° C., this temperature being maintained for 1½ to 2 hours. The furnace was then allowed to cool, the muffles taken from the furnace, the sulphur collected, and the roasted product leached with ferric chloride.

Leaching.—The leaching apparatus used was similar to that employed in previous tests and described in former reports and consists of a chemical stoneware leaching tank with agitator and containing a heating element made up of nichrome spirals contained in pyrex glass tubing.

The charge to the leaching vessel consists of a weighed quantity of roasted concentrate and the measured proportion of ferric chloride required for the chemical reaction between the iron and copper in the roasted concentrate. To ensure a completely reduced liquor it is better to use a 5 to 10 per cent excess of ore. The leaching liquor consists of ferric-ferrous chloride anolyte as obtained from the iron electro-deposition cell. The temperature at which most efficient leaching is carried out is around 90° to 95° C. The completely reduced liquor is filtered, the filtrate being treated for recovery of copper as the next step. The insoluble from the leach is washed, dried, and mixed with the fresh concentrate and roasted.

Tables I and II show the results of tests on two concentrate products from Eustis ore, carried out according to the procedure described above.

Tables III and IV show results of tests carried out on Anyox ore concentrate, using the same procedure, tests being made in duplicate. Analysis of the Anyox concentrate is as follows:—

Iron	30·94 per cent
Copper	24·40 "
Zinc	1·01 "
Sulphur	32·90 "
Insoluble	1·24 "

Screen Analysis

−200	86·3 per cent
+200	4·7 "
+150	4·5 "
+100	4·5 "

The results in the tables are quite interesting, and check each other so closely that the method of extraction could be considered almost quantitative. On the whole the results may be said to be quite encouraging.

As was observed at the outset, the Anyox ore is a pyrrhotite-chalcopyrite complex and should therefore react fairly satisfactorily with ferric chloride in straight leaching. Pyrite-chalcopyrite treated in this manner shows an iron extraction of around 56 per cent and a copper extraction of around 89 per cent. Table V shows results on the pyrrhotite-chalcopyrite type. Such an extraction, however, is too low to warrant adoption of this procedure as the means of treatment. With an ore of this description, it would probably be better practice to leach direct as above, roast the leach residues recovering the sulphur, and add the roast product obtained to the next leaching charge. Any iron present as pyrites in the leach residue would be converted to the readily soluble monosulphide form in the roasting. In this way quite a saving in roasting costs would be possible and the extractions should be quite as high as with the method of roasting the complete ore feed.

In the two tests that follow a comparison of the two methods of treatment are shown. These tests were carried out on the 56-litre scale, the same amount of ore being used in each test. In addition to the regular extraction data, data relative to copper precipitation and extraction are shown in the respective tables. The tests were made on the Anyox ore which contains little more than a trace of lead and about 1 per cent zinc. The leach liquor in Test No. 4, Series E-1 consisted of ferrous-ferric chloride anolyte obtained from cell operations, adjusted to proper concentration by the addition of ferric chloride. In the comparative test, Test No. 1, Series E-3, it will be observed that the initial leach liquor used consisted of almost pure ferric chloride and the subsequent leach liquors were prepared from the copper-free liquor of the initial leach with ferric chloride added to give the desired leaching strength.

In Test No. 4, Series E-1, the procedure comprised roasting the concentrate in the presence of sulphur, in cycle No. 1 crude sulphur being added and in the subsequent cycles the leach residue of the preceding cycle being combined with raw concentrate and roasted. In cycle No. 2 in this test, due to a miscalculation, it was necessary to add 1,300 grammes of raw concentrate to the leach to completely reduce the liquor. In Test No. 1, Series E-3, the procedure was: leaching the raw concentrate direct, roasting the residue thus obtained and combining this roasted residue with raw concentrate and leaching the mix, the leach residue again being roasted and combined with raw concentrate and leached, and so on.

Tables VI and VII show the complete data obtained on these methods of treatment. Reference will be made later to the copper precipitation.

An examination of the summation shows a slight variance in the metal extractions and per cent reduction in weight, but on the whole the final results are in such close agreement that no decided preference can be made for either method on the basis of percentage extraction. In Series E-1, 24,955 grammes concentrate plus residue (minus the 1,300 grammes added in cycle No. 2) have been roasted, while in Series E-3 the quantity of residue roasted amounted to 10,703 grammes, or approximately two-fifths that of the former. On this type of ore the Series E-3 process would therefore be the more preferable.

TABLE I

Series D-1 ; Test No. 2—Eustis Concentrate

(Analysis : iron 30·44%, copper 20·52%, sulphur 31·62%)

Roasting Tests :—

Roast	Charge concentrate, grms.	Sulphur or residue added grms.	Per cent loss of weight in roast	Weight of roast, grms.	Analysis of roast			Sulphur recovered, grms.
					Fe, per cent	Cu, per cent	S, per cent	
1..........	300	60 (S.)	35·0	234	36·97	26·44	....	59
2..........	250	92 (R.)	36·5	217	37·70	25·73	....	57
3..........	250	81 "	36·2	211	36·44	25·67	....	53
4..........	250	78 "	36·3	209	36·86	25·85	....	44
5..........	250	80 "	36·0	211	36·54	25·31	....	45
6..........	250	85 "	35·2	217	36·97	24·90	....	53
7..........	....	87 "	71·2	25	26·08	8·16	13·29	42

Leaching Tests:—

Leach	Volume leach, litres	Time, hours	Charge, grms.	Per cent dissolved	Weight of leached residue, grms.	Analysis of leached residue		Extraction	
						Fe, per cent	Cu, per cent	Fe, per cent	Cu, per cent
1..........	3·68	8	230	59·1	94	11·20	5·51	87·6	91·4
2..........	3·90	8	215	61·4	83	7·12	3·54	92·7	94·6
3..........	3·50	8	208	61·5	80	6·72	2·97	92·9	95·5
4..........	3·50	8	207	60·3	82	7·12	2·80	92·3	95·7
5..........	3·50	8	209	58·3	87	9·32	2·50	89·3	95·8
6..........	3·60	8	215	58·6	89	7·59	2·48	91·5	95·8

Summary:—

Total concentrate charged........................	1,550 grammes	
Weight final residue..............................	25 "	
Metals and sulphur removed......................	98·4 per cent	
Sulphur in charge................................	490 grammes	
Sulphur recovered...............................	353 "	=72 per cent
Extraction—Iron.................................	98·6 per cent	
Copper..............................	99·3 "	

Note.—Difference in weights between weight of roast and leaching charge, and residue added to roast and leached residue, due to samples taken for analysis.

TABLE II

Series D-1 : Test No. 3—Eustis Concentrate

(Analysis : iron 33·66%, copper 23·10%, sulphur 35·47%)

Roasting Tests :—

Roast	Charge concentrate, grms.	Sulphur or residue added, grms.	Per cent loss of weight in roast	Weight of roast, grms.	Analysis of roast			Sulphur recovered, grms.
					Fe, per cent	Cu, per cent	S, per cent	
1..........	300	60 (S.)	27·5	261	37·07	26·37	....	67
2..........	250	99 (R.)	31·8	238	37·18	26·06	....	81
3..........	250	90 "	31·2	234	37·07	25·70	....	72
4..........	250	90 "	31·2	234	36·97	25·52	....	70
5..........	250	90 "	31·2	234	36·65	25·46	....	74
6..........	250	94 "	30·5	239	36·73	25·52	....	67
7..........	....	93 "	76·3	24	22·02	8·42	13·18	51

Leaching Tests:—

Leach	Volume leach, litres	Time, hours	Charge, grms.	Per cent dissolved	Weight of leached residue, grms.	Analysis of leached residue			Extraction	
						Fe, per cent	Cu, per cent	S, per cent	Fe, per cent	Cu, per cent
1........	4·1	8	257	60·7	101	8·69	4·59	....	90·7	93·1
2........	4·0	8	235	60·8	92	7·54	2·99	....	92·0	95·5
3........	3·6	8	231	60·2	92	6·76	2·79	....	92·7	95·6
4........	3·5	8	232	60·3	92	6·02	2·34	....	93·5	96·3
5........	3·5	8	232	58·6	96	7·28	3·18	....	91·8	94·8
6........	4·0	8	237	59·5	96	5·86	2·29	74·04	94·6	96·3

Summary:—

Total concentrate charged........................	1,550 grammes	
Weight final residue..............................	24 "	
Metals and sulphur removed.......................	98·45 per cent	
Sulphur in charge..................................	609·8 grammes	
Sulphur recovered..................................	482·0 "	=79·04 per cent
Extraction—Iron....................................	98·9 per cent	
Copper..................................	99·1 "	
Gold in final residue..............................	0·14 oz./ton	

NOTE.—*See* note under Table I.

TABLE III

Series E-1: Test No. 1—Anyox Concentrate

(Analysis: iron 30·94%, copper 24·40%, zinc 1·01%, sulphur 32·90%)

Roasting Tests:—

Roast	Charge of concentrate, grms.	Sulphur or residue added, grms.	Per cent loss of weight in roast	Weight of roast, grms.	Sulphur recovered, grms.	Analysis of roast		
						Fe, per cent	Cu, per cent	Zn, per cent
1a.........	400	80 (S.)	22·0	374	39	32·55	24·43	1·16
1b.........	400	80 (S.)	24·5	362	79	33·80	25·48	1·19
2a.........	250	205 (R.)	37·8	283	136	35·31	23·82	....
2b.........	250	152 (R.)	34·5	263	128	34·58	23·36	....
3a.........	300	113 "	26·6	303	69	33·65	23·76	....
3b.........	300	103 "	26·3	297	78	33·39	24·03	....
4a.........	300	130 "	25·1	322	83	32·50	22·58	....
4b.........	300	121 "	26·1	311	99	32·24	22·96	....
5a.........	....	149 "	59·0	61	87	30·36	2·05	0·24
5b.........	....	137 "	63·6	50	85	24·17	2·01	0·16

Leaching Tests:—

Leach	Volume leach, litres	Time, hours	Charge, grms.	Per cent dissolved	Weight of leached residue, grms.	Analysis of leached residue		Extraction	
						Fe, per cent	Cu, per cent	Fe, per cent	Cu, per cent
1a.........	5·5	8	371	57·4	159	10·42	1·87	86·2	96·7
1b.........	5·5	8	359	60·1	143	8·70	1·73	89·7	97·3
2a.........	4·5	8	281	59·0	115	7·29	1·07	91·5	98·1
2b.........	4·5	8	261	59·7	105	6·09	1·19	92·9	97·9
3a.........	4·0	8	301	56·1	132	9·32	1·96	87·7	96·4
3b.........	4·0	8	295	58·3	123	6·09	1·19	92·5	97·9
4a.........	4·25	8	320	52·8	151	12·60	1·07	81·8	98·0
4b.........	4·25	8	309	55·0	139	8·91	0·82	87·5	98·5

Summary:—

	A			B		
Total concentrate charged....	1,250	grammes		1,250	grammes	
Weight final residue..........	61	"		50	"	
Metals and sulphur removed..	95·1	per cent		96·0	per cent	
Sulphur in charge............	491·3	grammes		491·3	grammes	
Sulphur recovered............	414	"	=84·2 per cent	469	"	=95·4 per cent
Extraction—Iron.............	95·3	per cent		96·7	per cent	
Copper...........	99·4	"		99·7	"	
Zinc..............	98·7	"		99·4	"	
Gold in final residue..........	0·24	oz./ton		0·22	oz./ton	

TABLE IV

Series E-1: Test No. 2—Anyox Concentrate

(Analysis: iron 30·94%, copper 24·40%, zinc 1·01%, sulphur 32·90%)

Roasting Tests:—

Roast	Charge of concentrate, grms.	Sulphur or residue added, grms.	Per cent loss of weight, in roast	Weight of roast, grms.	Sulphur recovered, grms.	Analysis of roast			
						Fe, per cent	Cu, per cent	Zn, per cent	S, per cent
1a	300	60 (S.)	24·0	273	74	34·36	26·05	....	....
1b	300	60 (S.)	24·7	271	73	34·36	26·10	....	....
2a	250	101 (R.)	29·9	246	84	33·95	25·04	....	....
2b	250	98 (R.)	30·4	242	86	33·89	25·09	....	....
3a	250	104 “	23·7	270	68	33·01	23·78	....	....
3b	250	96 “	24·3	262	62	33·12	23·88	....	....
4a	250	116 “	26·7	268	81	34·05	22·46	....	....
4b	250	107 “	26·9	261	79	33·53	23·40	....	....
5a	....	108 “	67·6	35	70	21·10	4·93	0·21	14·82
5b	....	105 “	69·5	32	73	18·38	5·06	0·42	12·31

Leaching Tests:—

Leach	Volume of leach, litres.	Time, hours	Charge, grms.	Per cent dissolved	Weight of leached residue, grms.	Analysis of leached residue		Extraction	
						Fe, per cent	Cu, per cent	Fe, per cent	Cu, per cent
1a	4·0	8	271	62·0	103	6·27	3·01	92·9	95·6
1b	4·0	8	269	62·8	100	5·28	2·37	94·2	96·6
2a	4·0	8	244	56·5	106	11·08	4·85	84·9	91·5
2b	4·0	8	240	59·1	98	9·51	3·87	88·5	93·7
3a	3·5	7	268	56·0	118	12·22	1·52	83·7	97·2
3b	3·5	7	260	58·0	109	9·09	1·87	88·5	96·7
4a	3·5	7	266	58·6	110	7·05	1·77	91·4	93·4
4b	3·5	7	259	58·6	107	5·75	1·98	92·9	96·4

Summary:—

	A		B	
Total concentrate charged	1,050	grammes	1,050	grammes
Weight of final residue	35	“	32	“
Sulphur and metals removed	96·6	per cent	96·9	per cent
Sulphur in charge	405·45	grammes	405·45	grammes
Sulphur recovered	377	“ =93 per cent	373	“ =92 per cent
Extraction—Iron	97·7	per cent	98·2	per cent
Copper	99·3	“	99·3	“
Zinc	99·3	“	98·6	“

TABLE V

Series E: Preliminary Test

Test No.	Volume, litres	Time, hours	Charge, grammes	Per cent dissolved	Residue, grammes	Analysis of residue			Extraction	
						Fe, per cent	Cu, per cent	S, per cent	Fe, per cent	Cu, per cent
1	1	7	72	56·6	31·5	19·37	3·32	68·62	72·6	94·1
2	1	7	72	56·6	31·5	19·58	3·80	68·63	72·3	93·2

TABLE VI

Series E-1: Test No. 4—Anyox Concentrate

Cycle	Charge		Weight of roast, grms.	Loss of weight, per cent	Sulphur recovered grms.	Leaching liquor, litres	Leaching liquor		Leaching charge, grms.	Leached residue, grms.	Per cent dissolved	Iron for Cu precipitation, grms.	Precipitated Cu residue, grms.	Analysis of final liquor Fe' grm./lt.
	Raw conc., grms.	Leached residue, grms.					Fe'', grm./lt.	Fe' grm./lt.						
1.......	3,500	700*	3,163	24·6	679	56	64·37	66·27	3,163	1,376	56·5	1,500 sponge Fe	1,584	139·95
2.......	2,185	1,376	2,469	30·7	897	56	61·77	82·34	2,469 1,300 conc.	1,986	47·0	1,350 sponge Fe	1,812	154·07
3.......	3,100	1,986	3,929	22·7	826	56	68·92	50·65	3,929	1,736	55·8	1,000 elec. Fe	859 Cu 313 Fe	147·77
4.......	3,300	1,736	3,635	27·8	1,125	56	67·01	52·35	3,635	1,664	54·2	1,600 sponge Fe	1,619	152·93
5.......	3,500	1,664	3,879	24·8	962	56	67·00	57·60	3,979	1,908	50·8	1,550 sponge Fe	1,644	156·03
Roast of final leached residue		1,908	871	54·3	900									

*Sulphur.

Summary:—

Total concentrate charged...............	16,885 grammes
Weight of final residue..................	871 "
Sulphur and metals removed............	94·8 per cent
Total charge to roasting.................	24,955 grammes
Sulphur recovered.......................	5,389 "
Total charge to leaching.................	18,475 "
Total sponge iron used...................	6,000 "
Total electrolytic iron used.............	1,600 "
Copper precipitate from sponge iron......	6,659 "
Copper precipitate from electrolytic iron.	859 "
Total leaching liquor....................	280 litres
Average time of leaching................	2½ to 6 hours

	Fe, per cent	Cu, per cent	Zn, per cent	S, per cent
Approximate analysis of roast.....	32·05	23·58		
Approximate analysis of leached residue.......................	12·00	6·00		
Analysis of copper precipitated by sponge iron...........................		45·92		
Analysis of copper precipitated by electrolytic iron.......................		97·55		
Analysis final residue.............	25·97	12·48	0·75	21·14
Extraction........................	95·6	97·3	96·2	97·0
Recoveries...............................		94·5.........		86·1

TABLE VII

Series E-3: Test No. 1—Anyox Concentrate

Cycle	Raw concentrate, grms.	Weight of roast, grms.	Loss of weight, per cent	Sulphur recovered, grms.	Leaching liquor, litres	Leaching liquor		Leaching charge, grms.	Leached residue, grms.	Per cent dissolved	Sponge iron for Cu precipitation, grms.	Precipitated Cu residue, grms.	Analysis of final liquor Fe″, grm./lt.
						Fe‴ grm./lt.	Fe″ grm./lt.						
1	8,000	2,202	43·6	1,244	56	116·93	3·80	8,000	3,905	51·2	2,860	2,830	181·65
2	3,000	1,266	51·0	1,128	56	97·39	50·78	5,202	2,584	50·3	2,475	2,489	196·84
3	3,000	1,153	47·1	974	56	59·16	93·20	4,266	2,180	48·9	1,700	1,848	186·89
4	2,885	1,065	47·6	969	56	61·26	93·20	4,038	2,034	49·0	1,400	1,433	188·98

Summary:—

Total concentrate charged	16,885 grammes
Weight of final residue	1,065 "
Sulphur and metals removed	93·6 per cent
Total charge to roasting	10,703 grammes
Sulphur recovered	4,315 "
Total charge to leaching	21,506 "
Total sponge iron used	8,435 "
Copper precipitate residue	8,600 "
Total leaching liquor	224 litres
Average time of leaching	5 to 6 hours

	Fe, per cent	Cu, per cent	Zn, per cent	S, per cent
Analysis of copper precipitate		42·92		
Analysis of final residue	30·37	11·71	0·20	23·85
Extractions	93·8	96·9	98·7	95·4
Recoveries		89·6	....	77·7

COPPER PRECIPITATION

The leach liquors averaged about 17 grammes of copper per litre. The hot filtered leach liquor was treated immediately with the precipitant, sponge or electrolytic iron, with agitation and with no further heating. In Series E-1, precipitation was complete in two to three hours, cycle No. 3 in which electrolytic iron was used showing little advantage as to time. Filtration, however, in the latter case was much more rapid than in the cycles in which sponge iron was used. The sponge iron used contained 63 per cent metallic iron with about 18 per cent silica, the remainder being principally unreduced oxide. A marked difference and behaviour was noticed between the two copper precipitates. The copper precipitated by electrolytic iron is quite crystalline in appearance and dries readily without any serious oxidation. In the case of the copper precipitated by sponge iron, the precipitate resembles a mud and oxidizes very rapidly during drying as does any unused iron present. It is also much more difficult to filter, the tendency being to clog the filter. The purity of the respective precipitates should also be noted, the sponge-iron copper precipitate assaying 42·92 per cent copper, while the electrolytic-iron copper precipitate assays 97·55 per cent. A higher grade sponge iron would probably compare more favourably with electrolytic iron than the product used in the above tests. In Series E-3 the results obtained were quite similar to those obtained in Series E-1, sponge iron being used as precipitant for the copper.

The copper recoveries reported in the above tables are, it may be noted, based on the copper content of the ore charge used and the net recovery from the ore by precipitation. For the ultimate recovery of the copper from these precipitates, as a marketable product, it is proposed to roast the precipitates in an oxidizing atmosphere, leach with sulphuric acid, and recover the metal as electrolytic copper. Such a scheme should not present any serious difficulty as it is already a well established practice.

Preliminary tests were made along this line on 200-gramme samples of the electrolytic-iron precipitated copper and of the sponge-iron precipitated copper. The composition of these two precipitates differs widely, the former having a content of 97 to 98 per cent copper with very little iron present, and the latter, 43 to 48 per cent copper, 22 to 26 per cent iron, and about 10 per cent other impurities such as silica and carbon.

Roasting of the higher grade product is quite simple, and the resulting oxide leaches readily with around 97 to 98 per cent extraction of copper, giving an electrolyte of 38 to 40 grammes copper per litre. With the lower grade product complications arise in that copper-ferrates are liable to be formed in the roasting, leaching is somewhat slower, iron is readily dissolved making a less satisfactory electrolyte, and copper extractions are somewhat lower. Sufficient work has not been carried out to date on this phase of the process to warrant anything more than the above observations, but it is hoped to go fully into details in the next report. From present knowledge, however, it is felt that the use of electrolytic iron as a copper precipitant is the more attractive proposition of the two. Even from the standpoint of costs of the precipitant it is possible that electrolytic iron might have an advantage. The main cost in its production would be the power cost for electro-deposition. In the reaction it simply replaces copper in the solution and is subsequently wholly recovered again in the iron deposition. It does not introduce possible impurities such as silica, as happens with the use of sponge iron, and on the whole seems much more satisfactory in every way as a precipitant than the sponge iron.

ELIMINATION OF LEAD AND ZINC FROM ELECTROLYTE

The majority of the heavy sulphide ores of the type used in these tests contain varying amounts of lead and zinc. To obtain a pure iron it would be necessary, therefore, to free the iron electrolyte from these impurities. Their presence in the electrolyte, moreover, has a tendency to affect the deposit, the effect varying with the quantity of the metals present, and in our present experience being more noticeable in rotating cathode deposits than in plate deposits. The problem has proved a most difficult one and at present writing has not been quite satisfactorily solved, bearing in mind, of course, the necessary economic aspect of any proposed method.

Various methods have been tried, but results show incomplete separation or elimination of the undesirable metals. The precipitants tried include an iron sulphide, commercial grade and chemically precipitated, calcium sulphide (commercial), sodium sulphide reagent and solution, and hydrogen sulphide gas. While all of these precipitants react with the undesired metals and cause precipitation, the difficulty has been to get consistent results. At times almost complete precipitation is obtained, but when duplication is attempted under the same conditions and procedure, results may vary quite widely. No explanation can be offered to account for this inconsistency in results and the matter must be left to further investigation. It may be said, however, that the presence of hydrogen sulphide in the liquor to be purified seems essential.

With the addition of any of the above-mentioned metal sulphides, an addition of free acid, usually hydrochloric acid, is made to satisfy this condition. By this means it is possible to precipitate 70 per cent and upwards of the lead content of the liquor, with the zinc precipitation varying from a few per cent up to 60 per cent. The lower the zinc and lead content in the liquor the more difficult it becomes to effect their precipitation. Complete elimination of the zinc is not considered possible, but fortunately its presence in an electrolyte is not serious to iron deposition when it is in quantities less than 0·35 gramme per litre.

From present experience, therefore, it would appear that an ore containing these impurities would not prove a satisfactory material for the production of pure iron unless the iron so obtained was further treated by a furnace process for elimination of these impurities. Some very satisfactory iron deposits have, however, been made from electrolytes partially freed from lead and zinc, the resulting deposit having lead present to the extent of 0·04 per cent and zinc 0·01 per cent. The possible effect of these impurities on the physical properties of iron have not, so far, been studied.

ELECTRO-DEPOSITION OF IRON

In the following paragraphs a number of electro-deposition tests are reviewed. These tests comprise for the most part short period runs and give only a fair indication of the results that may be expected. The electrolytes used were in all cases obtained from the treatment of ore, no pure or made electrolytes being used. The tests were carried out in cells of three different sizes, most of the results being obtained on the smallest size cell. With the larger sized cells, the amount of electrolyte required taxes rather heavily the present electrolyte storage capacity and so fewer tests were obtainable, and for a similar reason it was found not possible to run longer period tests. Observations relative to the results are set out in the following paragraphs with the data obtained.

LARGER SCALE DEPOSITION TEST

Test No. 11-P

The cell consisted of an asbestos-ebony box having two cathodes and one graphite anode. The anode was supported in an asbestos bag which constituted the anolyte compartment. The electrolyte, heated in a feed tank, was fed in at the bottom of the cell and overflowed through an opening at the top of one end. The electrolyte was a purified ferrous chloride leach liquor containing 161 grammes of ferrous iron per litre, having a *p*H value of 3·7. Monel plates having a total deposition area of 1·39 square feet served as cathodes. These were coated with a thin film of kerosene previous to their immersion in the cell. An increase of acidity in the electrolyte was noted from the overflow during the run. The deposits were somewhat pitted, silver-grey in colour, and very smooth in parts, and their total weight was 207 grammes. One deposit was stripped as a plate. The data from the test follow:—

Time of run	7 hours
Average temperature in cell	77·4° C.
Average voltage	1·93 volts
Current density	21·7 amperes per square foot
Total ampere hours	212
Current efficiency	93·9 per cent
[†]Anode efficiency	94·1 "
Rate of flow of electrolyte	80–85 litres per hour

[†]The anode efficiency is calculated from the total ferric iron generated in the anolyte.

The following four tests were carried out in a concrete cell impregnated with sulphur. The anode compartment consisted of an asbestos bag supported in a bakelite grid. Two cathodes of sheet steel fitted closely against the grid frame. The cathodes were 14 by 23 inches and had an effective deposition area of 4·6 square feet. Hot electrolyte was fed to the cell through an opening in the bottom and overflowed through an opening at the opposite top.

Test No. 13-P

The electrolyte was a ferrous chloride leach liquor having a concentration of ferrous iron of 181 grammes per litre and a *p*H of 4·0. Before immersion in the cell the cathodes were given a thin coating of gasoline. The deposits, although badly pitted, were free from trees and easily stripped from the plates. No evolution of chlorine was detected during the run.

Time of run	6 hours
Average temperature in cell	72·7° C.
Voltage	1·51 volts
Current density	10·9 amperes per square foot
Weight of deposit	278 grammes
Ampere hours	300
Current efficiency	89·1 per cent
Average rate of flow of electrolyte	48·6 litres per hour

Test No. 14-P

The electrolyte flow and current density were reduced slightly. The deposit was not quite so porous as the previous deposit. One plate was fairly smooth, the other very scaly. Both deposits were easily stripped from the cathodes.

Concentration of electrolyte	183 grms. per litre ferrous iron 1·6 grms. per litre ferric iron
*p*H	3·3
Time of run	6 hours
Average temperature in cell	67·6° C.
Average voltage	1·45 volts
Current density	8·65 amperes per square foot
Weight of deposit	205 grammes
Ampere hours	239·25
Current efficiency	80·7 per cent
Average rate of flow of electrolyte	30·6 litres per hour

The low current efficiency may be due to the lower temperature in the cell and the presence of ferric iron.

Test No. 15

The electrolyte used in this test was the overflow from the previous tests reduced by means of sponge iron. A low current density and reduced rate of flow were continued. Considerable oxidation of the electrolyte overflow was observed as the test proceeded. The deposit was pitted, but not so badly as in Test No. 13-P. In certain areas the deposit was fairly smooth, in others slightly nodular. The deposits were easily stripped from the cathodes.

Concentration of electrolyte	179 grms. per litre ferrous iron
*p*H	4·0
Time of run	12 hours
Average temperature in cell	72° C.
Average voltage	1·66 volts
Current density	10·25 amperes per square foot
Weight of deposit	510 grammes
Ampere hours	566
Current efficiency	86·7 per cent
Average rate of flow of electrolyte	32 litres per hour

Test No. 16-P

To reduce the ferric iron formed in the electrolyte during the running of the cell, a U-shaped, cast iron pipe was partly filled with coarse electrolytic iron and connected in the overflow circuit. Water containing 2 c.c. hydrochloric acid per litre was fed slowly to the overflow tank to make up for evaporation losses and to maintain the *p*H value at 3·4. The overflow electrolyte was pumped to a suction filter from which it flowed through the heater feed tank to the cell, making a continuous flow. There was no hydrolysis during the run and the anode bag was entirely free from precipitated oxychloride. The deposits were silver-grey in colour with slight cracking at top of plates. They were slightly pitted, and fairly easily removed from the cathodes. The cathodes were given a thin coating of gasoline previous to the run as in earlier tests.

Concentration of electrolyte at start	176 grms. per litre ferrous iron
*p*H	3·2
Time of run	5½ hours
Average temperature in cell	71·3° C.
Average voltage	1·91 volts
Current density	10·15 amperes per square foot
Weight of deposit	230 grammes
Ampere hours	256·5
Current efficiency	86·2 per cent
Average rate of flow of electrolyte	33 litres per hour

The current efficiency obtained in the above tests appears low in comparison with former tests and the only explanation that can be suggested is that of poor contacts, resulting in a loss of power. Anode efficiencies were not obtained on account of the difficulty or impossibility of recovering all of the anode liquor from the cell.

SMALL-SCALE DEPOSITION TESTS

A number of tests were carried out in small cells for the purpose of studying the effect of different conditions of the electrolyte on the nature of the deposit. The tests were run in glass battery jars, the inside dimensions of which were 3¾ by 3¾ by 8 inches high. They were divided into two compartments by an asbestos cloth diaphragm. The anodes were made from ½-inch graphite plates and the cathodes from ⅛-inch steel sheets. The cathodes had an effective deposition surface of 0·158 square foot. The cells were partly immersed in an electrically heated copper water bath. Two tests were usually carried out simultaneously, the cells being connected in parallel. Feed and overflow were maintained at the required volume through glass and rubber tubing. The tests were made in pure electrolyte and electrolytes containing respectively lead, ferric, and acid. Removal of lead was effected by the calcium sulphide or iron sulphide treatment mentioned.

The results of the tests are summarized in Table VIII. The electrolyte in all cases was a copper-free ferrous chloride liquor from leaches of copper concentrates. *Tank* signifies liquor containing lead; *purified* signifies electrolyte approximately lead free. The cathodes in Tests Nos. 24-P to 37-P inclusively were heated and dipped in kerosene previous to putting in the cell. In Tests Nos. 38-P to 41-P the plates were dipped in hot ferrous chloride solution as a preliminary treatment. From results, the latter treatment was no more satisfactory than the former. The presence of small amounts of lead in the electrolyte, which gave decidedly bad results in depositions on rotating mandrels had little or no effect on the plate depositions. The presence of small amounts of ferric ion in the electrolyte increased the tendency of the catholyte to hydrolyse, but its effect on the deposition was negligible, inasmuch as good deposits were obtained. In Tests Nos. 39-P and 41-P hydrochloric acid was added to the electrolyte in sufficient quantity to bring the *p*H value around 1·0 to 2·0. Excellent deposits were obtained and in the case of Test No. 39-P the current efficiency was high. Hydrolysis was entirely absent in the presence of such an acid electrolyte.

Hydrolysis of the hot electrolyte is probably due to the presence of ferric ion. An increase in the hydrogen ion concentration will reduce or prevent this tendency when ferric ion is present. An absolutely pure ferrous chloride electrolyte had little or no tendency to hydrolyse with a *p*H value of 3·6, while an electrolyte containing ferric ion hydrolysed with a *p*H value of 3·4. The evolution of free chlorine was probably due to the high current density employed and the small size of the cells. In brief, the above tests gave the following results:—

Cathode coatings of either kerosene or hot ferrous chloride showed no difference in the nature of deposit. Small amounts of lead in the electrolyte had no apparent effect on the nature of the deposit. The presence of small amounts of ferric ion in the electrolyte, while increasing the tendency for hydrolysis, in no way affected the deposit. From an acid electrolyte good deposits were obtained with little or no decrease in the current efficiency. It is to be noted that these tests were carried out in small cells. Whether similar results would be obtained in cells of three or four times this size is at present undetermined, but it is the intention to check these results in larger cells at an early date.

TABLE VIII
Small-scale Deposition Tests

Test No.	Electrolyte	Analysis electrolyte, grms./lt.				Time, hours	Temperature in cell, ° C.	Voltage	Current density, amp./sq. ft.	Weight of deposit, grms.	Current efficiency, per cent	Rate of flow of electrolyte, lt./hr.	Anode efficiency, per cent	Nature of deposit	Observations
		Fe''	Fe'''	Pb	*p*H										
24P	Purified	182	tr.	nil	3·6	7·0	82·5	3·07	56·0	63	97·7	....	100	Good	Slight hydrolysis of electrolyte.
25P	Tank	187	tr.	0·04	3·6	6·0	79·4	3·08	52·7	51	97·7	1·05	104	Good	....
26P	Purified	182	tr.	nil	3·6	6·0	80·3	3·08	52·7	50	95·7	....	99	Good	....
29P	Tank	187	tr.	0·04	3·6	7·0	78·2	3·18	53·8	57	92·1	0·728	....	Fair	Slight hydrolysis.
30P	Purified	173	nil	nil	3·6+	7·0	80·8	3·18	53·8	61	98·6	0·828	....	Fair	No hydrolysis, chlorination.
31P	Tank	187	tr.	0·04	3·9	7·0	81·8	3·19	53·9	60	96·8	0·786	97·8	Fair, slight treeing.	Hydrolysis and chlorination.
32P	Purified	173	nil	nil	3·6	7·0	82·4	3·19	53·9	60	96·8	0·800	108·0	Slightly corrugated and treed.	Slight hydrolysis after 4th hour. Chlorination.
33P	Purified	173	tr.	nil	3·7	6·5	85·6	3·23	53·4	51	89·5	0·807	80·8	Good, few small nodules.	Hydrolysis.
34P	Purified	173	tr.	nil	3·0	6·5	85·7	3·23	53·4	56	98·3	0·793	95·1	Good, few small nodules.	No hydrolysis.
35P	Purified	173	tr.	nil	3·7	6·5	85·3	3·24	54·6	53	90·7	1·046	76·0	Very good Smooth dark grey.	Hydrolysis and chlorination.
36P	Purified	173	tr.	nil	3·0	6·5	82·7	3·24	54·6	53	90·7	0·977	92·2	Very good Smooth light grey.	Hydrolysis.
37P	Purified	171	2·0	nil	3·4	6·0	83·1	3·45	59·4	55	93·8	0·833	83·5	Fair to good, slight nodules.	Chlorination. Hydrolysis.
38P	Tank	173	tr.	0·04	1–2	6·5	86·0	2·84	54·2	56	96·7	0·723	122·6	Excellent.	No hydrolysis.
39P	Tank	166	2·1	0·04	3·4	6·5	84·4	2·84	54·2	51	88·0	0·771	102·0	Fair to good.	Hydrolysis from start.
40P	Tank	173	tr.	0·04	3·8+	6·5	85·1	2·68	54·0	55	95·3	0·915	92·5	Rough, otherwise fair.	Hydrolysis.
41P	Tank	170	tr.	0·04	1–2	6·5	84·7	2·68	54·0	49	85·0	0·854	104·0	Good to very good.	No hydrolysis.
44P	Tank	161	nil	0·04	3·6	7·0	86·7	1·95	45·8	82	93·8	1·335	89·3	Good	No hydrolysis. Two anodes and asbestos paper diaphragm.

REPORT ON THE TREATMENT OF MIXED (BULK) CONCENTRATES FROM BASE-METAL SULPHIDE ORES

W. E. Harris[1]

Following a co-operative arrangement between the Base Metal Extraction Company, Ltd., of London, England, and 564 Booth Street, Ottawa, and the Mines Branch, Department of Mines, Ottawa, a laboratory and demonstration plant were built, equipped, and operated by the company for experimental tests and demonstrations of a hydrometallurgical process, developed in England, for the treatment of mixed concentrates obtained from the bulk flotation of base-metal, sulphide ores. The equipment was such that a maximum of 120 pounds cathode zinc per day could be produced if required.

Bulk concentrates were made by the Mines Branch from shipments of copper-zinc ores of the Amulet and Abana mines, western Quebec; of the copper-zinc ore of the Sherritt-Gordon mine, Cold Lake, northwestern Manitoba; of the zinc-lead-copper ores of the Stirling mine, Cape Breton, and the Stralak mine, Sudbury district, Ontario; of the lead-zinc ore of the Tetreault mine, Notre-Dame des Anges, Quebec; and base-metal, sulphide ores from several other localities.

The concentrates were treated in a muffle furnace and in a six-hearth, mechanically rabbled, Herreshoff furnace under conditions to give the maximum solubility of the zinc and copper content for subsequent leaching and deposition tests. It was found, however, that the control of temperatures in these small furnaces was very difficult and undoubtedly accounted for the erratic results obtained as seen in the percentages of zinc and copper rendered soluble.

Another factor must be contended with, as it is well known that "ferrates" are formed when sulphide zinc ores are roasted, and experience has shown that this tendency to produce ferrates is more pronounced when copper is present.

The process developed and patented by the Base Metal Extraction Co., Ltd., consists of roasting the concentrates in an atmosphere of steam whereby the formation of the ferrates is reduced to a minimum. The calcines are leached (batch system) with hot sulphuric acid, and maximum extractions of zinc and copper with one acid attack are obtained; the silica and iron are precipitated in the leaching vat, and by agitating and boiling a granular precipitate of the silica is obtained which acts as a filter bed and admits of easy filtration. After filtration the zinc-copper solution is electrolysed, using insoluble anodes, and as much of the copper as possible is obtained as cathodes. The acid formed is neutralized with zinc oxide (recovered from the residue) and the remaining copper is precipitated with zinc dust. The solution is filtered from the precipitate which is added to

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[1] Electro-metallurgist, Base Metals Extraction Co., Ltd.

the leach residue for further treatment. The solution is purified from nickel, cobalt, cadmium, and the last traces of copper by the use of mercury and zinc dust in a hot solution, this being a specially developed and patented method of purification of zinc solution whereby all the impurities likely to interfere in electrolysis are removed in one operation. The solution is then passed on for electrolysis for the recovery of the zinc. Electrolysis is carried out in the usual type of cell—no neutral solution is added to the circuit. All residues and dross are added together and treated in a rotary furnace in such a manner that the zinc and lead remaining in the residues are volatilized off and collected as oxides leaving a residue which contains the remaining copper and the precious metals; these can be smelted in the usual manner.

The features of the above process are as follows:

(a) Roasting in an atmosphere of steam.

(b) Maximum extractions of zinc and copper in one attack of acid, which means a great saving in plant and labour.

(c) The great bugbear "silica", together with the iron, is precipitated in the leaching tank, constituting a part of the leaching process and by the methods used is precipitated in a form that assists filtration.

(d) From 60 to 70 per cent of the copper in solution is recovered as cathode copper from the zinc-copper solution at a very low cost.

(e) The solution, after this preliminary electrolysis and after neutralization of the acid formed, is then purified, by means of mercury, from all the metals that cause corrosion during electrolysis or lower the purity of the metal by inclusion in it.

(f) The electrolysis of the purified solution is simple and efficient.

(g) The re-treatment of the residues not only gives an enhanced recovery of the zinc (and lead) but allows of the recovery of the precious metals which hitherto has been one of the great problems in the treatment of zinc-copper complex ores.

The following table gives a brief summary of some of the results obtained:

Ore or concentrate	Analysis				Calcined ore or concentrate				Per cent solubility		Leach residue				Recovery in solution			
					Zn		Cu				Zn		Cu		Zn		Cu	
	Zn	Cu	Pb	Fe	Total	Soluble	Total	Soluble	Zn	Cu	Total	Soluble	Total	Soluble	Per cent of total	Per cent of soluble	Per cent of total	Per cent of soluble
	%	%	%	%	%	%	%	%			%	%	%	%				
Tetreault—Notre-Dame des Anges zinc conc ...	48·92	0·58	1·18	12·5	54·6	49·81	....	....	92·0	......	18·05	6·85	....	....	87·00	94·6	....	....
[1]Tetreault mixed zinc and lead concentrates.	40·58	0·97	4·87	11·15	43·02	42·41	....	....	98·5	......	....	....	....	....	....	....	....	....
[2]Abana straight zinc ore.	31·2	0·2	......	16·93	35·0	31·5	0·24	....	90·0	......	7·5	3·2	....	....	87·6	94·0	....	....
[3]Abana high-grade zinc ore (picked sample)...	......	......	......	......	50·4	49·05	0·25	....	97·4	......	7·08	4·94	....	....	94·27	95·89	....	....
[4]Amulet mixed zinc and copper concentrates...	32·3	6·26	......	15·81	32·92	29·76	6·36	5·86	90·4	92·4	14·79	4·42	3·81	0·7	85·02	95·04	80·03	96·08
[4]Amulet zinc—Copper concentrate...	39·15	6·75	......	15·30	44·43	40·18	6·45	5·75	90·7	89·2	13·66	5·02	3·42	0·7	87·00	93·86	74·0	94·0
[5]Stirling ore mixed concentrate..............	37·88	1·1	9·3	7·47	34·8	33·0	0·9	0·8	94·8	89·0	5·36	2·5	0·4	Trace	92·75	96·6	80·65	....
[6]Stralak, bulk concentrate.................	48·0	2·05	5·72	9·95	46·34	43·12	2·35	1·18	93·00	......	12·69	7·05	1·86	1·47	88·0	95·0	....	....

[1] No leach on this roast.

[2] An excellent result on low-grade ore.

[3] A good example of high efficiency of roasting and leaching of a straight zinc ore.

[4] In both these tests the difficulty of temperature control coupled with the presence of an increasing quantity of copper have the tendency to pull down the per cent solubility of the significant metals, but of course these are recovered in the subsequent treatment of residue.

[5] This concentrate had 4·54 per cent of MgO and 3·07 per cent CaO of which 96·35 per cent of the MgO went into solution. This ore could be treated by volatilizing the zinc and lead off first when the MgO would not interfere. Arsenic and antimony are also present in this ore.

[6] This very complex concentrate presents no difficulties and with the re-treatment of residues yields over 95 per cent of all the significant metals.

In the foregoing table the last column "Recovery in Solution" gives the percentage recovery of the metals in solution. Assuming that 90 per cent of the zinc and copper in the residues is recovered the total recovery would be around 95 per cent.

The following are the details of an electrolytic run for zinc on solution made from Amulet concentrates after the copper had been precipitated:

Head tank solution averaged	Zinc	100·0 grms. per litre.
	Acid	20·5 "
Outflowing solution averaged	Zinc	31·0 grms. per litre.
	Acid	124·0 "

Duration of run	54½ hours.
Average amperage	310·0 (Current density about 30 amperes per square foot).
Average voltage	3·86
Current efficiency	91·6 per cent=Kw. hours per ton= 3,150

The resulting zinc assayed (50-gramme assay lots)—

Iron	0·0059 per cent
Lead	0·0066 "
Copper	0·0018 "
Zinc (by difference)	99·9857 "

V

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

H. C. Mabee, Chief Chemist

The increased demand of the mining public for service relative to efficient methods of treatment and utilization of Canadian metallic ores and non-metallic minerals was so great that the present staff and facilities of the chemical laboratory of the Division were taxed to the limit to take care of the analytical and assay work resulting from the experimental test operations in the other laboratories. This condition, and the lack of accommodation and facilities, restricted the work which had been planned on special chemical and metallurgical problems. The research staff of the National Research Council, for whom accommodation had been provided during the past two years, found it necessary to secure other accommodation to carry on their work.

A total of 1,684 samples of ores and minerals, and the products of their concentration, were examined, and reports issued thereon. A large proportion of these represented complex mixtures of lead, zinc, copper, iron, and precious metal values, and therefore involved a large amount of analytical work, including in all nearly 7,000 chemical determinations.

The following is a list of ores and associated minerals in connexion with which more or less exhaustive analytical work was carried on during the year:—

Calcite	25 samples	Iron	3 samples
China clay	1 "	Titaniferous iron	5 "
Cyanite rock	9 "	Lead ores	19 "
Epsom salts	6 "	Lead-zinc-silver	254 "
Gold ores and tailings	748 "	Mercury	14 "
Gold-copper	143 "	Molybdenite	1 "
Gold-copper-arsenic	23 "	Platinum sands	16 "
Garnet	26 "	Silica sands	56 "
Graphite	31 "	Silver ores	29 "
Gypsum	13 "	Zinc-copper-gold-silver	233 "
Identification and valuation	29 "		

A HYDROMETALLURGICAL PROCESS FOR THE TREATMENT OF STIBNITE ORE AND THE RECOVERY OF METALLIC ANTIMONY

This investigation was undertaken at the request of a representative of the Natural Resources Branch of the Canadian National Railways, and was desired for the purpose of developing a process for the treatment and utilization of the antimony ore deposit of Lake George, N.B., and one that would utilize the salt from the deposits at Malagash, N.S., in its operation.

Prof. S. J. Demorest had experimented on a process for the electrolytic production of antimony direct from stibnite ores by leaching with a solution of sodium hydroxide (a product from the electrolysis of salt), or sodium sulphide (Jour. Amer. Inst. Metals XI-XII, 1917-18, pp. 83-87). The process was found to be successful and antimony of high purity was obtained with current efficiencies around 76 per cent. It was therefore decided to carry out a similar investigation along the lines suggested in his paper using a stibnite concentrate of the following analysis, obtained by flotation from Lake George ore:—

Antimony	57·70 per cent
Arsenic	0·89 "

Stibnite reacts with caustic soda solution according to the following equation:—

$$Sb_2S_3 + 2NaOH = NaSbS_2 + NaSbSO + H_2O$$

This reaction proceeds rapidly in hot solutions, the electrolytic reaction, however, is somewhat complex. Metallic antimony and hydrogen deposit at the cathode while sodium thiosulphate is formed at the anode; an additional deposit of a crystalline salt found to contain antimony was also noted on the iron anode.

The experimental work in connexion with the investigation was conducted in the electrochemical laboratory of the Division by W. R. McClelland.

The leaching solution used in all cases was an 8 per cent solution of sodium hydroxide. Deposition was carried out in an ebony-asbestos cell 8 inches deep, 6 inches wide, and 7 inches long, with one cathode and one anode. The cathode in the first three tests was of sheet iron perforated with $\frac{1}{16}$-inch holes. Eventually a smooth cathode was used. Anodes of iron, aluminium, and lead were tried. The latter proved to be the most satisfactory. Aluminium was quite unsuitable as it is attacked vigorously by the free NaOH in the electrolyte. The cathode had an effective plating surface on one side of 35 square inches (7 by 5 inches).

Test No. 1

In this test a continuous operation of leaching and deposition was carried out. The stibnite concentrate was leached hot with caustic solution and agitated in enamel pails, filtered, fed to the cell at the top, siphoned off from the bottom of the cell and returned to the leaching system. The test was run for 2 hours during which time the temperature in the cell dropped from 55° to 37° C. The average voltage was 1·98 with a current of 1·81 amperes. The current density was 7·4 amperes per square foot (calculated on one side of the cathode only). As the solution has a very high throwing power and antimony is deposited on the far side of the cathode as well as on the side facing the anode, this figure is high, the actual current density being undeterminable. It would be from one-half to three-quarters of the current density as calculated for one side only. The deposit cracked and peeled. Considerable material, which on analysis showed the presence of antimony and probably contained some free sulphur, was thrown down in the electrolyte.

Test No. 2

Owing to the unsatisfactory results obtained by continuous operation it was decided to carry out the leaching and electrolysis in two distinct operations. A lot of 480 grammes of concentrate was leached with 5 litres of hot, 8 per cent sodium hydroxide solution for 20 minutes and then filtered.

Weight of residue	179 grammes
Per cent of concentrate dissolved	62·7 "
Antimony in residue	41·56 per cent
Antimony extraction from concentrate	73·0 "

It was observed that a precipitate settled out of the leached liquor after standing. To determine the loss of antimony thus settling out, analyses were made of the liquor immediately after leaching and after standing overnight with the following results:—

Antimony in liquor after leaching	42·53 grammes per litre
" " standing	40·66 " "
" settled out	1·87 " "

This may be due to hydrolysis or to the presence of excess NaOH in the liquor.

Electrolysis: The liquor from the above leach was heated and circulated through the cell. The data from the run follow:—

Time	1¾ hours
Average temperature	55·2 ° C.
" voltage	2·54 volts
" current	2·14 amperes.
Weight of deposit	4·12 grammes
Current efficiency	77·4 per cent

The deposit was silver-grey in colour but cracked and was flaky. No sediment precipitated from the electrolyte. The same crystalline deposit was observed on the anode.

Test No. 3

In this test the electrolyte from Test No. 2 was diluted with 2 litres of 8 per cent NaOH solution. The hot electrolyte was fed in at the bottom of the cell and a siphon overflow at the top maintained a balanced circulation through the cell. The increase in free NaOH gave a lower voltage but the deposit cracked and peeled badly. Excess NaOH in the electrolyte undoubtedly has an adverse effect on the deposition of the antimony. The data of the run follow:—

Time	1¾ hours
Average temperature	72·1 ° C.
" voltage	1·90 volts
" current	2·15 amperes.
Weight of deposit	2·9 grammes
Current efficiency	52·2 per cent

Test No. 4

Leaching: Two 4-litre leaches of stibnite concentrate were made, 600 grammes of concentrate in each. The leaching liquor was 8 per cent NaOH solution. Leaching was carried out for 1 hour at 90° C.

Total weight of residue	505 grammes
Per cent of concentrate dissolved	57·91 "
Analysis of residue Sb	43·67 per cent
Extraction of antimony	68·2 "

Electrolysis: The concentration of the electrolyte was 51·23 grammes of antimony per litre. It was decided to carry out this test with the electrolyte at room temperature. A lead anode was used in place of iron. The antimony concentration was higher than previous tests and the current density was increased. A smooth cathode was used. The temperature of the cell gradually increased during the run from 22·5 to 25° C. The data of this test follow:—

Time of run	2½ hours
Average temperature	23·6 °C.
" voltage	2·0 volts
" current	4·387 amperes.
Weight of deposit	16·23 grammes
Current efficiency	99·0 per cent
Analysis of deposit: Antimony	99·19 per cent (by difference)
Arsenic	0·01 "
Sulphur	0·8 "

The deposit was smooth and adherent. There was a slight cracking at the upper portion of the deposit. The electrolyte remained clear throughout the run.

Test No. 5

This test was carried out using the electrolyte as in the previous test, under similar conditions, except that the electrolyte was heated and circulated hot during the run. The data are as follows:—

Concentration of electrolyte	52·2 grm./lt. antimony
Time of run	1½ hours
Average temperature	61·2 ° C.
" voltage	2·60 volts
" current	4·19 amperes.
Weight of deposit	8·27 grammes
Current efficiency	88·0 per cent

The deposit cracked and flaked and was loosely adherent.

Addition Reagents

No addition reagents were tried out in any of the above tests. It is possible that in some cases the cracking might have been overcome by suitable addition reagents. From work carried out on antimony plating baths, various agents have been suggested and found successful. The most successful bath is the fluoride.[1] In this bath resorcinol, a 0·025 per cent solution twice daily, gave the best deposit. Peptone, gum arabic, and aloin also gave good deposits. Size has also been suggested. Whether these addition reagents would be satisfactory in an alkaline electrolyte remains to be investigated.

CONCLUSIONS

The results of these preliminary tests show that stibnite is readily soluble in hot solutions of sodium hydroxide. Better deposits are obtained from a cold electrolyte than from a hot. High concentrations of antimony with low concentration of free NaOH in the electrolyte appear necessary

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[1] Mathers and Means: Tests on Antimony Plating Baths. Trans. Amer. Elec. Soc., vol. 31.

VI

THE CONCENTRATION OF THE ORES OF WESTERN QUEBEC*

J. S. Godard

The mining developments in western Quebec have shown the ores of this district to be more complex in character than was at first expected. When the initial discoveries were made it was thought that the ores would be similar in character to those of Kirkland Lake and Porcupine areas. Developments have since shown that although the ores along the southern margin of the belt are somewhat similar, those within the belt, constituting by far the greater proportion, are vastly different, being complex mixtures of base-metal sulphides carrying values in the precious metals. These mixtures of base-metal sulphides of copper, zinc, and iron, present several problems to the metallurgist, especially in their concentration into marketable products. For the purpose of this paper the following classification may be made of the ores of the district:—

Group I.—Milling Ores:

(a) Ores in which arsenopyrite is the predominating sulphide. In this class of ore the gold occurs in fractures in the quartz and is associated to a lesser extent with the arsenopyrite and pyrite. Examples of this class of ore are the Granada, south of Pelletier lake, and the O'Brien and the Thompson in Cadillac township.

(b) Ores in which iron pyrite is the predominating sulphide and gold is the valuable constituent. This may also be classed as a concentrating ore, as in Group III *(b)*.

Group II.—Direct Smelting Ores:

(a) Heavy-sulphide ores of copper and iron, in which the copper and gold content is above the point where it is more economical to smelt the ores direct after roasting than to concentrate the copper and gold values into high-grade concentrate before roasting and smelting. The economic point between a direct-smelting and a concentrating ore will depend on the amount of contained gold values and whether these values are recovered in the copper concentrate. It will be somewhere about 4 per cent copper. A large part of the Waite-Montgomery ore-body and a number of the Horne ore-bodies are in this class.

(b) Highly siliceous ores carrying values in copper and gold required for fluxing purposes in the copper converters. A number of the Horne ore-bodies, the gangue of which is a rhyolite with a high percentage of free silica, and the Chadbourne ore, are of this class. This class of ore may also be considered as a concentrating ore provided the gold values

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are recovered in the chalcopyrite. The tonnage of this class of ore to be smelted or concentrated will depend on the smelter requirements for flux, and whether it is more economical to sell such an ore as flux or to concentrate it.

Group III.—Concentrating Ores:

(a) The disseminated chalcopyrite-rhyolite ore-bodies not required for fluxing purposes.

(b) Disseminated ores in which iron pyrite is the predominating sulphide and in which gold is the chief value. This type might also be classed as a milling ore, Group I *(b)*, but owing to their proximity to the smelter it may be more economical to concentrate the values into a pyritic gold concentrate. Examples of this class of ore are the Francœur and Arntfield low-grade ore-bodies.

(c) Disseminated and heavy-sulphide ores of zinc containing varying proportions of iron sulphides such as the zinc ore-bodies of the Horne, Aldermac, and Abana mines.

(d) Disseminated ores containing small amounts of chalcopyrite, sphalerite, and pyrite, but whose chief value is gold. Archean in Clericy township is an example of this type.

(e) Heavy-sulphide ores of copper and iron with pyrrhotite predominating, too low in copper and gold content to be of direct smelting grade, such as the massive pyrrhotite ore-bodies "F" and "H" of the Horne mine, and portions of the heavy-sulphide, direct-smelting ore-bodies which will probably grade off into concentrating ores of this class. This ore presents a concentrating problem in the collection of the gold values into the copper concentrate.

(f) Heavy-sulphide ores of copper, zinc, and iron of low copper, zinc, and precious-metal values, in which the predominating sulphide is pyrite. The outstanding example of this class is the large ore-bodies of the Aldermac mine. This class of ore presents a concentrating problem in the separation of the values into two or possibly three products.

(g) The heavy-sulphide and disseminated-sulphide ores of copper, zinc, and iron, with pyrrhotite predominating and containing some values in gold and silver, such as certain portions of the Waite-Montgomery ore-body, the Amulet ore-bodies, and the copper-zinc ore-body of the Abana mine. This class of ore presents several concentration problems due to unfavourable metal ratio of copper to zinc, the association of the precious metal values, and in some cases the oxidized condition of the ore.

CONCENTRATING ORES

Group III

Much experimental work has already been done on these complex ores and the chief problems encountered in their concentration are as follows:—

1. The concentration of the precious-metal values with the copper concentrates where the greatest monetary return for these metals will be received. In some cases gold and silver are the chief values and copper

is of secondary importance. In all cases the recovery of the precious-metal values in the copper concentrate is of the greatest importance to the net value of the ore mined. Whether they can be made to report with or in the copper concentrate depends on their association with the various constituents of the ore.

2. The concentration of the copper-zinc ores into marketable copper and zinc products with high recoveries of both minerals. This has been found to depend on the metal ratios, namely, the proportion of copper, zinc, and iron sulphides in the ores, and on the oxidized condition of such ores.

The solution of the above problems will depend on very careful control of operating conditions, such as grinding, pulp densities, conditioning of the pulp, proper reagents and amounts of reagents, regularity of feed, and many other factors which affect the flotation of minerals, especially the selective flotation of admixtures of the types presented.

Shipments, ranging from one hundred pounds to carload lots, of a number of the concentrating ores of this district have been made to the Ore Testing Laboratories of the Department of Mines at Ottawa for experimental purposes, and a brief summary of the results obtained and the difficulties encountered in their concentration are given in the following pages:—

(a) Assay of head sample: gold, 0·04 ounce per ton; silver, 0·22 ounce per ton; copper, 4·60 per cent; iron, 19·83 per cent; insoluble, 50·64 per cent.

A number of small-scale flotation tests were made for the purpose of ascertaining how far concentration could be carried consistent with good gold recoveries. The results from three of these tests are shown in the following table:—

Test No.	Product	Weight, per cent	Assay		Per cent of values	
			Cu, per cent	Au, oz./ton	Cu	Au
1	Concentrate	35·3	12·14	0·11	95·0	81·2
	Middling	8·5	1·39	0·04	2·6	7·1
	Tailing	56·2	0·19	0·01	2·4	11·7
2	Concentrate	17·9	23·18	0·15	92·3	71·2
	Middling	9·8	1·33	0·05	2·9	9·0
	Tailing	72·3	0·30	0·015	4·8	18·9
3	Concentrate	14·5	28·72	0·17	89·9	42·9
	Middling	8·2	3·00	0·07	5·3	9·9
	Tailing	77·3	0·29	0·035	4·8	47·2

Should this ore be treated as a concentrating ore, the selective action between the chalcopyrite and the pyrite cannot be carried too far—if so, it will be done at the expense of the gold values. A copper concentrate containing about 12 to 15 per cent copper and containing a large amount of pyrite appears to be as far as concentration may be carried. This will give a recovery of 95 per cent of the copper and 80 per cent of the gold. The 20 per cent loss of gold would amount to about 20 cents per ton of ore of this grade.

(b) Assay of head sample: gold, 0·37 ounce per ton.

Should this ore be considered as a concentrating ore no difficulty should be experienced in its treatment. Concentration by flotation recovered 87 per cent of the gold in a concentrate assaying 2·40 ounces per ton with a concentration ratio of 1 : 8·3. Flotation followed by tabling recovered 91 per cent of the gold in the form of a combined flotation and table concentrate assaying 2·30 ounces per ton with a concentration ratio of 1 : 8. Seventy per cent of the gold in this ore can be recovered by amalgamation. It is possibly amenable to cyanidation, or concentration followed by cyanidation of the concentrate. It, therefore, can be considered either as a milling or a concentrating ore, depending on which is the more economical practice.

(c) Assay of head sample: zinc, 13·0 per cent; iron, 3·28 per cent; insoluble, 71·95 per cent; copper, nil.

Flotation yielded satisfactory results. No difficulty was experienced in making a marketable grade of zinc concentrate assaying 49 per cent zinc with a recovery of 92 per cent, and a concentration ratio of 1 : 4·2.

(d) Assay of head sample: gold, 0·32 ounce per ton; silver, 0·26 ounce per ton; copper, 0·66 per cent; zinc, 0·46 per cent. The gangue is chiefly siliceous, but contains considerable altered rock of a talcy nature.

Concentration gave very satisfactory results. A concentrate assaying 14 per cent copper and 5·7 ounces gold per ton was made with a concentration ratio of 1 : 23, and recoveries amounting to 97 per cent of the copper and 87 per cent of the gold. This is about the maximum concentration that can be made with satisfactory gold recoveries. When the concentration ratio was increased to 1 : 33, the copper concentrate assayed 20 per cent copper and 6·2 ounces gold per ton. This gave a satisfactory copper recovery of 94 per cent, but the gold recovery fell to 55 per cent. The gold loss in the tailing was largely as free gold, as 81 per cent of it was recovered by amalgamation, making a total recovery of 91·5 per cent of the gold. The ore does not present any concentrating problem; either flotation or a combination of flotation and amalgamation, depending on the grade of concentrate desired, gives satisfactory results.

(e) Assay of head sample: gold, 0·25 ounce per ton; silver, 0·28 ounce per ton; copper, 1·55 per cent; zinc, 0·28 per cent; iron, 50·39 per cent; sulphur, 39·86 per cent; insoluble, 2·93 per cent.

A number of flotation tests were made to determine whether the gold values could be recovered in the copper concentrate. The results of these tests are shown in the following table:—

Test No.	Product	Weight, per cent	Assay		Per cent of values	
			Cu, per cent	Au, oz./ton	Cu	Au
1	Concentrate	58·2	2·50	0·48	96·2	95·5
	Tailing	41·8	0·14	0·03	3·8	4·5
2	Concentrate	21·5	6·30	0·74	87·3	62·8
	Tailing	78·5	0·25	0·12	12·7	37·2
3	Concentrate	8·7	13·98	1·10	76·6	39·6
	Tailing	91·3	0·38	0·15	23·4	60·4

It will be noted from these tests that as the grade of the copper in the flotation concentrate is increased a higher gold loss occurs in the tailing. How far the concentration may be carried consistent with good gold recoveries is a matter of more exhaustive tests under actual milling conditions.

The results recorded above are those from small-scale flotation tests. It is reported that this ore has been concentrated to 14 per cent copper with recoveries of 85 per cent of the copper and 70 per cent of the gold, by grinding in a ball mill in closed circuit with a classifier, thus retaining in the grinding circuit any free gold until sufficiently fine for flotation.

A magnetic separation test was made, separating the pyrrhotite from the non-magnetic portion of the ore, for the purpose of ascertaining the association of the gold values. The results are tabulated below:—

Product	Weight, per cent	Assay			Per cent of values		
		Cu, per cent	Au, oz./ton	Fe, per cent	Cu	Au	Fe
Magnetic............	45·4	0·62	0·07	57·73	19·1	12·0	51·4
Non-magnetic........	54·6	2·19	0·43	45·32	80·9	88·0	48·6

Examining the iron content of these two products, it will be seen that the iron of the magnetic portion corresponds very closely with that of pyrrhotite, while that of the non-magnetic product corresponds closely with the composition of pyrite when due consideration is given to the amount of chalcopyrite present. It is therefore evident that the gold is not associated with the pyrrhotite. The magnetic portion, weighing 45 per cent of the total, contained but 12 per cent of the gold. As the separation was made at −80 mesh, some unfreed chalcopyrite and pyrite, and some that was mechanically entrained, reported with the pyrrhotite. This could easily account for the 12 per cent of the gold that was found in the magnetic product.

(f) Assay of head sample: copper, 7·36 per cent; zinc, 6·53 per cent; iron, 35·48 per cent; gold, 0·03 ounce per ton; silver, 2·05 ounces per ton; insoluble, 18·6 per cent.

Selective flotation tests were made for the purpose of separating the copper from the zinc, and very favourable results were obtained. The copper concentrate assayed 24 per cent copper, 0·08 ounce gold per ton, 8·0 ounces silver per ton, and 3 per cent zinc. The concentration ratio was 1 : 3·8 and recoveries of 93 per cent of the copper, 75 per cent of the gold, and 76 per cent of the silver were obtained.

The zinc concentrate assayed 49 per cent zinc, making a recovery of 75 per cent of this metal with a concentration ratio of 1 : 11.

This ore does not present any serious concentration problem. The separation of the chalcopyrite from the zinc blende was very good, and the reporting of a large percentage of gold and silver with the copper concentrate is favourable. The ore as received was freshly mined and free from oxidation, and this, coupled with a favourable flotation ratio of copper to zinc, are the contributing factors to the results obtained.

(g) Assay of head sample: copper, 3·85 per cent; zinc, 9·25 per cent; gold, 0·24 ounce per ton; silver, 3·28 ounces per ton. Acidity equivalent to 9 pounds per ton H_2SO_4.

A number of large-scale selective flotation tests were made. The flow-sheet for the various tests was, with a few minor changes, as follows:—

Ore was crushed dry to 4-mesh and fed to a 4½-foot by 13-inch Hardinge ball mill operated in closed circuit with a simplex Dorr drag-classifier. The oversize from the classifier was returned to the ball mill for regrinding and the overflow went to a Greenawalt 8-cell mechanically-agitated flotation machine. The final copper concentrate was taken from the first three cells, and the middling from the last five cells was returned to the ball mill for regrinding. The tailing from the copper flotation was pumped into a launder leading to a flat-bottom Callow pneumatic flotation unit which was used for the flotation of the zinc. The Callow unit consisted of three cells, a rougher, a cleaner, and a recleaner. The middling from the cleaner and the recleaner was returned to the launder leading to the zinc cells. The results from three of these tests are given in the table following:—

Test No.	Product	Assay			
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton
1	Head sample	4·35	9·90	0·24	3·46
	Copper concentrate	19·78	8·71	0·61	12·16
	" tailing	0·35	7·79	0·04	1·16
	Zinc concentrate	0·93	42·77	0·04	0·75
	" tailing	0·28	1·27	0·04	1·16
	Recovery, per cent	92·3	74·1	89·2	73·5
2	Head sample	3·85	9·25	0·24	3·28
	Copper concentrate	21·73	9·03	0·89	12·83
	" tailing	0·40	8·98	0·05	1·50
	Zinc concentrate	1·22	52·54	0·05	1·39
	" tailing	0·33	2·53	0·05	1·41
	Recovery, per cent	90·2	64·4	84·0	61·6
3	Head sample	3·85	9·25	0·24	3·28
	Copper concentrate	24·16	8·53	0·89	13·72
	" tailing	0·49	9·62	0·07	1·47
	Zinc concentrate	1·36	51·11	0·10	1·84
	" tailing	0·37	1·46	0·06	1·29
	Recovery, per cent	88·2	75·3	76·9	61·8

The flotation reagents used were varied in kind and quantity, as was also the place they were introduced to the circuit. The following reagents gave in a general way the most satisfactory results:—

For the Copper. Soda-ash, 12 lb./ton, was added dry to the ball mill.
Aerofloat No. 15 added to the head of the copper cells.

As a depressant for the zinc during the flotation of the copper, sodium cyanide, 0·30 lb./ton, and zinc sulphate, 0·75 lb./ton, were added to the ball-mill discharge.

For the Zinc. Copper sulphate, 2·0 lb./ton, was added to the discharge of the copper cells;
Xanthate, 0·30 lb./ton, was added to the launder leading to the zinc cells; and
Steam-distilled pine oil, 0·05 lb./ton, was added to the head of the zinc cells.

A close check was kept on the pulp densities throughout the testing.

The following percentage of solids in the pulp at various points in the circuit gave the most satisfactory results:—

Ball-mill discharge	56 per cent
Classifier overflow	38 "
Discharge copper cells	29 "

Grinding to about 12 per cent +200 mesh was necessary for good results. A typical screen analysis was as follows:—

+ 65	0·10 per cent
+100	0·84 "
+150	3·10 "
+200	7·80 "
−200	88·16 "

A few difficulties were experienced. These were to be expected in an ore of this type, as the copper-zinc separation is a new problem and nowhere has it been attempted other than in the experimental or pilot stages.

The difficulties in the separation of the chalcopyrite from the sphalerite were greatly increased by the presence of a large quantity of soluble sulphates. These had a disturbing effect on the flotation and were largely responsible for the high zinc content in the copper concentrate. The problem of the separation of these two sulphides is made more difficult by the rather unfavourable metal ratio of copper to zinc, which in this ore is 1 : 2·4. The gold recoveries in the copper concentrate were somewhat erratic. It was found that better recoveries of the gold were obtained when Aerofloat was used for the flotation of the copper instead of xanthate, and that the gold recovery dropped when the alkalinity of the pulp was increased. Some difficulty was at first experienced in making a satisfactory grade of zinc concentrate. In a soda-ash pulp there was a tendency for the iron sulphides to float with the zinc. This condition was overcome by the addition of lime water to the tailings from the copper cells and agitating the pulp for 15 minutes in a contact tank previous to the flotation of the zinc. This had the effect of depressing a large amount of iron sulphides that otherwise appeared in the zinc concentrate.

From the classification of the ores given above one might possibly be impressed with the number of different types and might be of the opinion that they would present rather difficult problems. There will, of course, be minor problems involved in the concentration of each, due largely to variations in type and grade from different parts of the mine. Careful manipulation and plant control are essential in the concentration of all ores by flotation. In fact this is as important, if not more so, than in cyanide practice on gold ores. Of the various types listed, those which will present the most difficulties in their concentration are the heavy-sulphide pyrrhotite ores carrying appreciable values in the precious metals, and the copper-zinc ores, also carrying precious-metal values.

In the former group the difficulty lies in recovering the gold values with the copper concentrate when a satisfactory ratio of concentration is obtained. The pyrrhotite, which has been proved to contain little or no gold values in association with it, may be selectively dropped into the tailing with but small loss of gold. This alone is, however, insufficient, as the ratio of concentration would then be only 1 : 2. Should the gold values be associated with the pyrite and not freed on fine grinding, any selective action between the chalcopyrite and the pyrite would result in a loss of gold. On the other hand, if the gold is freed on fine grinding, which seems likely, at least to some extent, it becomes a matter of securing flotation conditions suitable for its collection with the chalcopyrite. Should flotation be ineffective as a means of recovering a fair amount of the gold, table or blanket concentration of the flotation middling may be an effective means of increasing their recovery by an appreciable amount.

In the second class, a number of problems arise. The first of these is the separation of the copper from the zinc. This is a delicate separation, especially when there is a high zinc and low copper content in the ore, and it will require close control of conditions throughout the concentrator if the copper concentrate is to be kept below the penalty limit in zinc. Just where this limit will be will depend on the quantity of this type of concentrate the smelter will have to treat. The copper-zinc separation is more difficult to make than the lead-zinc separation, because lead is more easily floated than copper, and reagents that are added for depressing the zinc must be added in smaller quantities; otherwise they will also have a depressing effect on the copper, with a resultant loss in the copper recovery. As the greatest return on this type of ore will be from the copper concentrate, it might prove more economical to pay a smelter penalty on the zinc content rather than incur a loss of copper in the endeavour to make a sharper separation of the two minerals. Unfavourable metal ratios is one of the big problems confronting custom concentrators. It is much easier of solution in a concentrator built specially to handle this type of ore, where provision can be made for it.

The second problem is in the recovery of the precious metals with the copper concentrate. In testing this ore, it was found that the recoveries of gold and silver with the copper were somewhat erratic. This may be partly due to the running time of the tests, which averaged from six to seven hours, as well as to other factors, including the oxidized condition of the ore. Under conditions of continuous operations, where the copper middling is returned to the grinding circuit or separately treated in a regrinding unit, it will be interesting to see where the precious-metal values will report.

Gold is worth more in a copper concentrate than in the zinc concentrate, as it is more easily recovered in subsequent smelting operations, and is paid for at a higher rate. Smelters as a rule do not pay for any gold under $1.00 in the zinc concentrates; consequently any gold under this amount may be considered a loss as far as the concentrator is concerned. Hence to realize the greatest return from the precious metals it is essential that they be collected with the chalcopyrite.

The third problem is to be found in the oxidized condition of some of the ores, which has a decided bearing on good flotation results. The results given above on this type of ore are remarkably good when its oxidized condition is considered. The ore represents perhaps the extreme conditions of oxidization that could be expected from the ores of the district. In this particular case it may be necessary to treat this oxidized ore separately and thus prevent interference with the flotation of the unoxidized ore. Even in the mining of the unoxidized ore-bodies, co-operation will be required between the mine and the concentrator to see that the ore is freshly broken and not allowed to remain in the stopes for any length of time, in order to avoid difficulties due to oxidization.

Now that the Noranda custom smelter is producing blister copper from its first 500-ton unit—and the second 500-ton unit will be ready shortly—and a 500-ton custom concentrator is in course of erection, the necessary facilities are at hand for the mines of western Quebec to dispose of their ores and copper concentrates. This should result in the bringing of many properties to the production stage, with the erection of concentrators on those properties whose ores are too low-grade, or which for other economic reasons cannot afford to sell to the custom concentrator.

The problems which will be encountered in the concentration of the ores of this district are no greater than have been met and successfully solved in other mining districts. Much experimental work has already been done, indicating that the ores can be satisfactorily concentrated. This should be followed up with pilot-plant operation on the more difficult ores, and should assist materially in the solution of the problems when actual mill operations are undertaken.

CUSTOM CONCENTRATORS*

C. S. Parsons and A. K. Anderson

INTRODUCTION

Custom milling-plants have been operated successfully for a great many years on this continent. One of the outstanding examples in the United States is the Golden Cycle mill at Colorado Springs, which mills the gold-telluride ores from the Cripple Creek district. In eastern Canada, the Northern Custom Concentrators, Limited, at Cobalt, successfully treated the cobalt-silver ores of that area for a number of years.

Flotation plants have entered the custom field in the past few years. The building of flotation concentrators to treat ores on a custom basis indicates the advance which has been made in flotation. In fact, three years ago it was considered almost impossible to treat successfully a mixture of miscellaneous ores by flotation. The methods used at that time were considered so delicate that any change, even in a single ore, would usually result in decreasing the effectiveness of the separation. To-day, we find four large custom flotation plants in the United States and one in Canada for treating complex lead-zinc ores from many different localities.

The advent of these mills was brought about by the serious problem confronting the lead-smelting plants due to increasing amounts of zinc in the ores presented for treatment. It was necessary to penalize the mines shipping high-zinc ores. The penalty is justified, because high zinc in the lead blast-furnace charge not only decreases the tonnage of the furnace but increases the amount of flux necessary, as well as increasing the metal loss in the slag. It has been estimated that in the Salt Lake valley alone, 75,000,000 pounds of zinc were lost in the slag-dumps from the operations of three large lead blast-furnace plants. It has also been estimated that $750,000 worth of additional flux was required to slag this zinc. It was to avoid at least a part of this waste and to help maintain an ore supply that custom flotation concentrators were built by the smelters.

This policy of the smelters has been amply justified by the results, not only in the conservation of metals and the economy effected in smelting, but it has had the effect of increasing ore shipments. It is noteworthy that most of the ores received come from mines which had previously been inactive or shut down. The operation of the mill affords valuable data to the small-mine operator, who is able to see the separation of his ore demonstrated, and is able to obtain from the smelter custom mill all the information necessary for the erection of his own plant at the mine. An encouraging feature of this is that in both Canada and the United States a large number of the mines which were previously shipping to the custom mill at the smelter are now contemplating, or have erected, mills of their own.

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It is not going too far to suggest that this history of the complex silver-lead-zinc ore deposits of western Canada and the United States might well repeat itself in the complex base-metal ore deposits of eastern Canada. These ores are difficult to concentrate, and the problem of the copper smelter handling zinc-copper ores will, owing to comparatively high costs for flux and fuel, be similar to that of the lead smelter in the west. At present the small mines are facing much the same problem that confronted the silver-lead-zinc mines of the west, inasmuch as the separations of these ores have not been demonstrated as yet by any operating mill.

PRINCIPAL CUSTOM MILLS

The five large custom flotation plants are the International mill built by the International Smelting and Refining Company at Tooele, Utah, which has a daily capacity of 1,000 tons; the 700-ton Midvale mill built by the United States Smelting and Refining and Mining Company at Midvale, Utah; the Timber Butte mill built by the Timber Butte Milling Company, Butte, Montana; the Anaconda mill at Anaconda near Butte, Montana; and in Canada, the mill at Trail in British Columbia, built and operated by the Consolidated Mining and Smelting Company.

METHODS OF HANDLING THE ORES AT THE MILL

In order to maintain as much uniformity as possible in the operating conditions, the mills have a system of bins, similar to those used at smelters, where the ores are bedded and mixed. However, it is necessary in many cases to segregate some ores for separate treatment if a peculiar condition, such as partial oxidation or interfering gangue, is present. Separate bins are provided for such ores, and one section of the mill is operated separately for their treatment.

The general layout of all custom flotation mills consists of one, or two self-cleaning receiving bins of 100 to 150 tons capacity. As ore is paid for as received, no storage ahead of the coarse crushing and sampling is practised. Bin storage is provided after sampling. The bin system may consist of five 150-ton bins for storage of custom ore after sampling. These are linked up with the mill-feed bins by a conveyer onto which the ore from each bin can be fed, and the exact amount from each bin controlled by fine gate operation. By drawing a certain amount of ore from each bin, a synthetic ore is made up on this conveyer and carried to the different mill storage bins at the head of each section of the mill. In flotation plants it is the general practice to divide the mill into 500-ton sections. In custom mills, however, it would seem advisable to have the sections of 250 tons capacity. A double bin system should be at the head of each section so that, if it is necessary to change the ore mixture, one bin can be filled while the other is being drawn. The bins are generally 250 tons capacity, and each ball-mill unit is generally given 500 tons of bin storage. Not only must the ore be bedded in these bins according to its assay, but, most important of all, it must be segregated with respect to its concentrating properties.

METHODS EMPLOYED IN THE PURCHASE OF ORES

The buying of ores for a custom flotation mill is not like purchasing ore for smelting. In buying ores on a smelter schedule, the only thing taken into consideration is the chemical analysis of the slag-forming constituents and the assay value based entirely on this analysis. On the other hand, in purchasing ore for a custom mill, the vital feature is its amenability to flotation. Two ores may have the same content of metal values, but have entirely different flotation properties. The basis on which all new ore is purchased must, therefore, rest entirely on the results obtained in the testing laboratory. The laboratory tests form the basis of the schedules on which the ore is purchased, with some slight deviations according to the interpretation of the results in terms of practice, which in turn is gained by experience. The importance of the testing laboratory and the staff which must be employed is therefore evident. With a prospective shipper, the first thing that must be done is to obtain from 50 to 100 pounds of ore which is representative of his shipping grade. If the test results are satisfactory, a contract is made for a small amount of ore. If the subsequent shipments are similar to the sample submitted, the contract is renewed for a larger tonnage.

It can readily be seen that the success of such a system depends a great deal on the closest co-operation between the ore-purchasing department and the concentrating department. The laboratory testing comes under the ore purchasing department, but the concentrating department should, in all cases, pass on the amenability of any new ore so as to put the responsibility in the proper hands.

A typical schedule for purchasing silver-lead-zinc ores in the Rocky Mountain district is as follows:—

Payment for Metals

SILVER.—Pay for 70 per cent of the silver content at the E. & M. J. average New York quotation for the week next preceding the date of sampling the last car of the lot. No payment for silver when the assay is less than 1·0 oz. per ton.

GOLD.—Pay for 65 per cent of the gold content at $20.00 per oz. No payment for gold when the assay is less than 0·01 oz. per ton.

ZINC.—Deduct 0·2 unit (4 pounds) of zinc for each 1·0 per cent of iron contained in the ore (minimum deduction 1·0 unit) and pay 70 per cent of the remainder of the zinc content at one-half the E. & M. J. average St. Louis quotation for the week next preceding the date of sampling the last car of the lot.

LEAD.—(Determined by wet assay). Deduct 35 per cent of the lead content with a minimum deduction of 0·5 unit (10 pounds) and pay for the remaining lead content at 2·5 cents less than the average of the daily published quotations of the American Smelting & Refining Company for common de-silverized domestic lead for the week next preceding the date of sampling the last car of the lot.

Treatment Charge

The base treatment charge to be $2.50 per ton f.o.b. cars at switching point. Switching charges paid by mill.

Sampling

The seller has the privilege of having the ore sampled at a custom sampler at his expense and such results shall be final; otherwise it will be sampled by the buyer at his expense in the usual manner employed for the sampling of other ores at the mill, and such results shall be final. The seller has the privilege of being represented at the mill during the sampling, and may have an assay pulp and compare assays by mutual agreement.

This schedule will be subject to termination at any time by either party.

This schedule is more generous than could be allowed by a mill operating in eastern Canada, because of the long freight hauls for the marketing of the concentrates to the lead and zinc smelters. Under the above schedule, an ore running 4 per cent lead, 15 ounces per ton silver, and 7 per cent zinc, would net the shipper $9.90 per ton, omitting freight charges.

Take grade of ore: 5 per cent lead, 25 per cent zinc, 10 ounces silver. This ore cannot be sold to the lead smelter, but on a lead-zinc schedule it will net the producer, omitting freight charges to the concentrator, $15.90 per ton. Take grade of ore: 20 per cent lead, 20 per cent zinc, 15 ounces silver; on lead schedule to lead smelter, this ore nets shipper $15.75, but on a lead-zinc schedule it will net him, omitting freight, $22.90 per ton.

The average grade of ore treated by one of the mills in the Salt Lake valley over a period of six months was: lead, 14·5 per cent; zinc, 20 per cent.

From a mine in British Columbia, the following ore was shipped to the Trail custom mill: lead, 3·12 per cent; zinc, 4·60 per cent; gold, 0·35 ounce per ton; silver, 2·48 ounces per ton. Freight rate to Trail, $1.50. Treatment charge, $3.00 per ton. The net return was $4.72 per ton. This would not leave much profit after deducting mining costs.

In starting a custom concentrator in a new district many difficulties will at first be encountered. The first difficulty experienced will be to obtain representative samples of the classes of ore to be treated. Good co-operation with the mines is necessary to overcome this difficulty. Then comes the laboratory problem of working out reagent combinations. It will be found impracticable at first to purchase ores on a schedule basis. As the concentrator pays for the ore as received on an assay value, the schedule must be based on the returns which will be received on the concentrates when shipped to the smelter. For this reason the behaviour of different ores and ore mixtures together with different reagent combinations must first be worked out and the relations between laboratory results and practice determined and correlated. A few test shipments of concentrates should also be made to the smelter. The shipper in the meantime must meet the concentrator half way by agreeing to temporarily accept payment on the basis of actual recovery made by the concentrator *pro rata* with other shippers, not on the assay value of his own ore, the concentrator charging for sampling and treatment. This arrangement answers two purposes: a check on the grade and the flotation properties of the ore as represented by the original sample is obtained, and opportunity is given the mill to try out the ores under operating conditions. In this way a check on the previous test work done in the laboratory is obtained. The mine owner also is benefited by being able to determine quite accurately what he may expect from his mine as to future production. After the necessary information has been obtained a schedule can be made out. The schedule should be subject to termination at any time by either party.

It must be remembered that the difference between the gross value of the marketable constituents of an ore as determined by assay, and the actual return to the miner, when shipping to a custom mill, is governed by three factors:—

1. Concentration treatment charges and the percentage of metals recovered by the concentrator in the form of concentrates suitable to be smelted.
2. Smelting charges, in which are included unavoidable metallurgical losses, this being paid for by the custom concentrator on behalf of the miner.
3. Marketing charges, for which the smelter charges the concentrator, which in turn includes them in the charges to the miner for concentrating his ore.

It will be of interest to take a hypothetical case of custom mill located at a copper smelter and purchasing copper-zinc ores and straight copper ores. Take, for example, an ore of the following grade:—

Copper, 4·35 per cent; zinc, 9 per cent; gold, 0·24 ounce per ton;
silver, 3·46 ounces per ton; iron, 18 per cent.

Previous test work made in the mill testing-laboratory shows that they can expect to make

1. A copper concentrate assaying: copper, 20 per cent; zinc, 8·1 per cent; gold, 0·62 ounce; silver, 12·9 ounces—recovering 92 per cent copper, 51 per cent gold, and 74·5 per cent silver; and
2. A zinc concentrate assaying: zinc, 50 per cent; copper, 0·7 per cent; gold, 0·04 ounce; silver, 0·5 ounce—recovering 78 per cent of the zinc.

Ratio of concentration: 5 : 1 for the copper
7·4 : 1 for the zinc.

The concentrator has to sell these two concentrates to the smelter, the copper concentrate to a copper smelter and the zinc concentrate to a zinc smelter. The gold and silver in the zinc concentrate is too low to be paid for by the smelter.

The smelter return received by the concentrator on the

Copper concentrate is roughly $9.64, omitting freight; and on the
Zinc " " $2.34, including freight to Europe.

The following milling schedule is made up, based on this smelter return:—

COPPER.—Pay for 85 per cent at 6·3 cents off the market.
ZINC.— Pay for 70 per cent after deducting 0·05 units for each per cent of iron in the ore, at the following rates:
Zinc at 6 cents per pound, pay for at 1·6 cents.
" 5·5 " " " 1·3 "
" 5·0 " " " 1·0 "
GOLD.— Pay for 50 per cent at $19.50 per ounce.
SILVER.— " 50 per cent at market.
Treatment charge, $3.00 per ton.

On this ore the return to mine would be:—

Copper	87 lb. × 85 per cent at 13−6·3=6·7 cents	$4.95
Zinc	(180 lb. −18 lb.) × 70 per cent at 1·38 cents	1.47
Gold	0·24 ounce × 50 per cent × $19.50	2.34
Silver	3·46 ounces × 50 per cent × 56 cents	0.96
		$9.72
Treatment charge		3.00
Return to mine, omitting freight		$6.72

This seems a large margin of profit for the concentrator, but the concentrating of such a type of ore is a delicate operation at best, as the grade of concentrates and recovery made from day to day will vary, and the smelter returns on lower grade concentrates are much less than on higher grade products.

It will be observed that the examples of ores sent to the custom mills in the Rocky Mountain section were all fairly high-grade ores. It is also quite evident that this is necessary if a profitable return is to be expected. However, if any of these ores were figured on a straight lead schedule, it would be found that the profit would be much less and that in some cases in high-zinc ores, the lead smelter would not even accept them. The same thing will be found in the case of the examples given of copper ores and copper-zinc ores. Take an ore of the following grade:—

Copper, 3·12 per cent; zinc, 4·63 per cent; gold, 0·02 ounce; silver, 0·72 ounce; iron, 20 per cent.

Applying the above milling schedule to this ore we have:—

Copper	62 lb. × 85 per cent at 6·7 cents	$3.55
Zinc	92−20 lb. = 72 lb. × 70 per cent × 1·3 cents	0.65
Gold	0·02 × 50 × $19.50	0.19
		$4.39
	Treatment charge	3.00
	Net return, omitting freight	$1.39

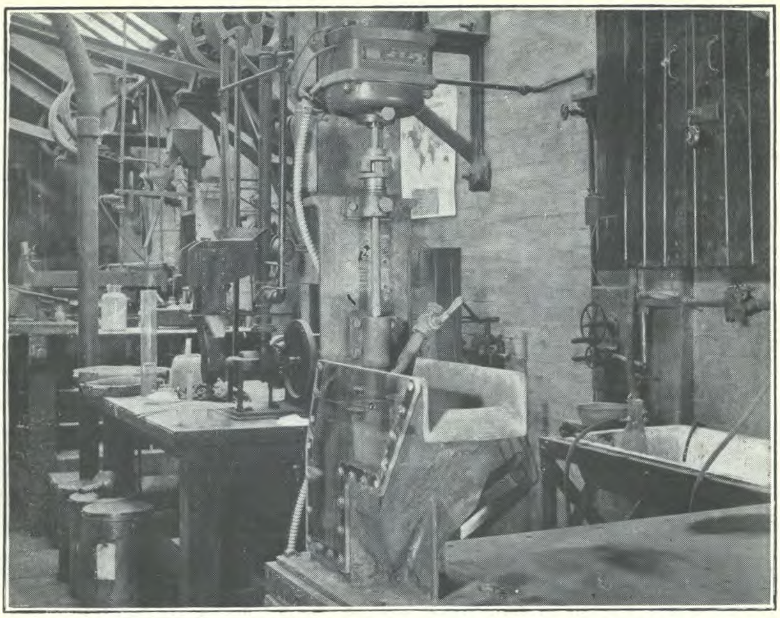
It can be readily seen that it would not pay to ship this grade of ore to a custom mill.

Take the case of a copper ore of grade: copper, 2·5 per cent; gold, 0·10 ounce; silver, 1·5 ounces:—

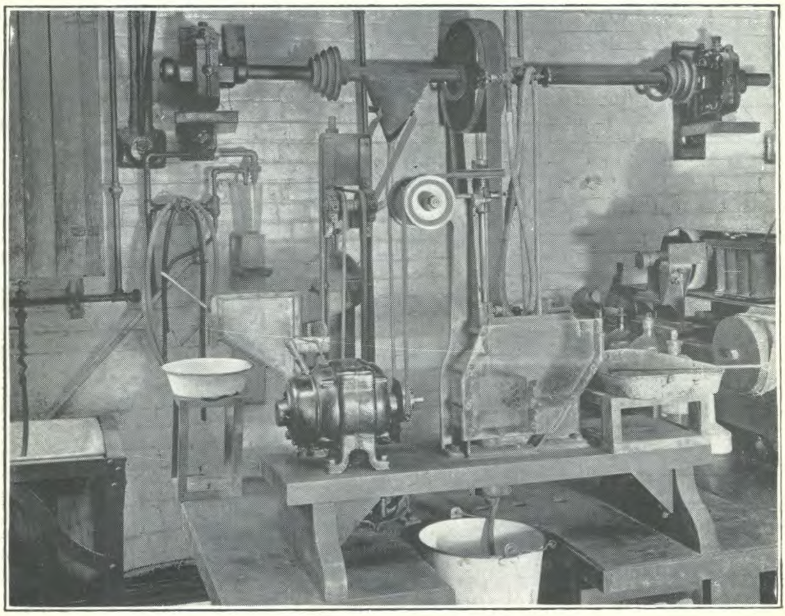
Copper	30 lb. × 85 per cent at 6·7 cents	$1.70
Gold	0·10 ounce × 50 per cent at $19.50	0.97
Silver	1·50 ounce × 50 per cent at 56 cents	0.42
		$3.09
	Treatment in Canada	2.50
	Return, omitting freight	$0.59

It can again readily be seen that in order to profitably operate an ore of this grade it would be necessary to mill a large tonnage and that the mill would have to be located at the mine.

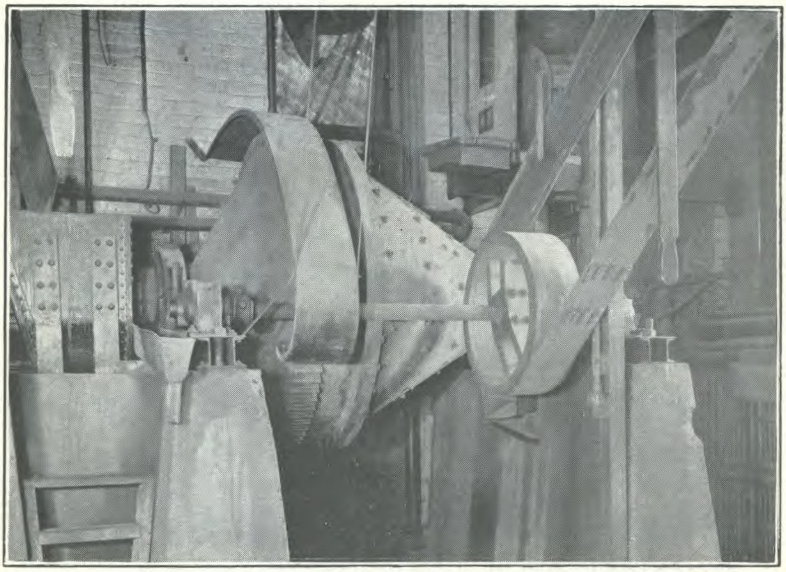
With the increasing discoveries of complex sulphide ore-bodies in northern and eastern Canada being opened up, the opportunities for custom concentrators become apparent. The establishing of such mills will mark one step forward, and one which is being stressed more and more in every branch of commercial endeavour—the conservation of our natural resources. Metals which by earlier metallurgical methods would have been lost, or recovered at such cost that the ore could not be mined at a profit, will then be available in such form that they can be placed on the market with a profit to all concerned.



A. Denver Sub-A (Fahrenwald) laboratory flotation machine used for making preliminary batch flotation tests on 2000-gramme samples of ore. Richards laboratory jig and laboratory Wilfley table in the background.



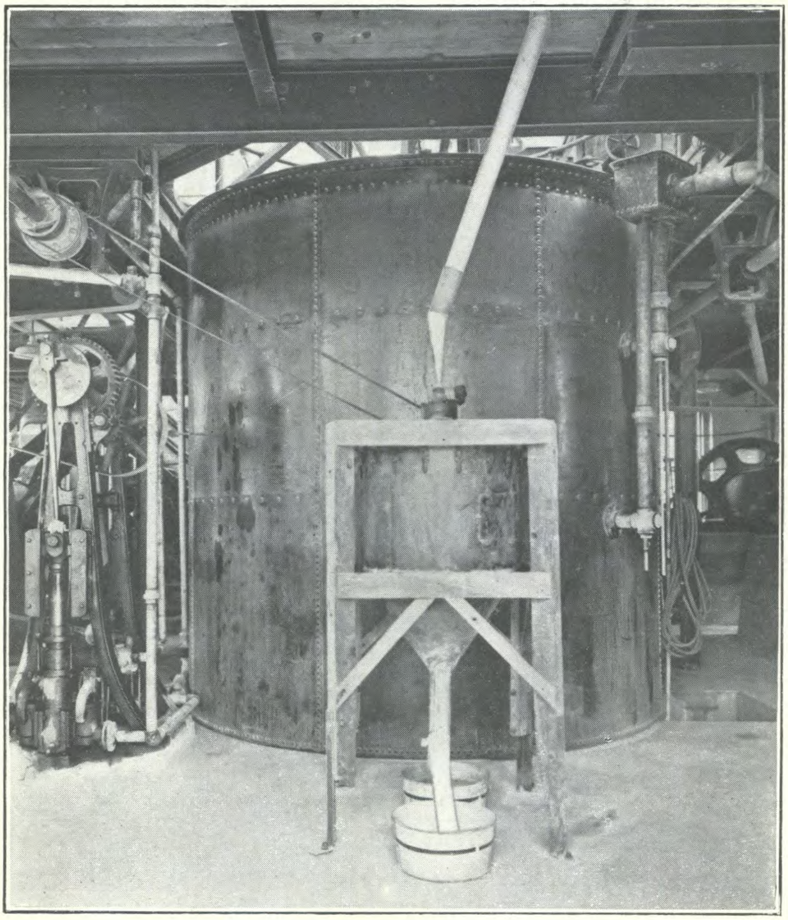
B. Janney and Ruth laboratory flotation machines used for making preliminary batch flotation tests on 1000-gramme samples of ore.



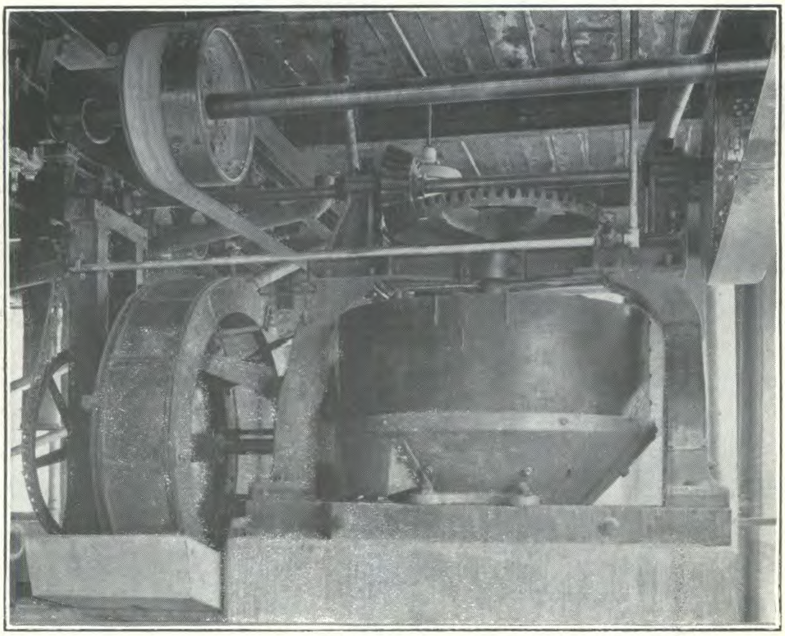
A. Hardinge ball mill (4′ 6″ x 13″). This grinding mill is used in conjunction with a Dorr simplex classifier (15″ x 11′ 8″) having a 3-foot bowl attachment for very fine grinding. The mill has a capacity of about 1,000 pounds per hour and is used for the large-scale or tonnage check tests.

B. Marcy rod mill (3′ x 6′). This grinding mill is used for a similar purpose as the Hardinge and for comparative tests with the Hardinge on the preparation of ore for concentration by gravity means or flotation.

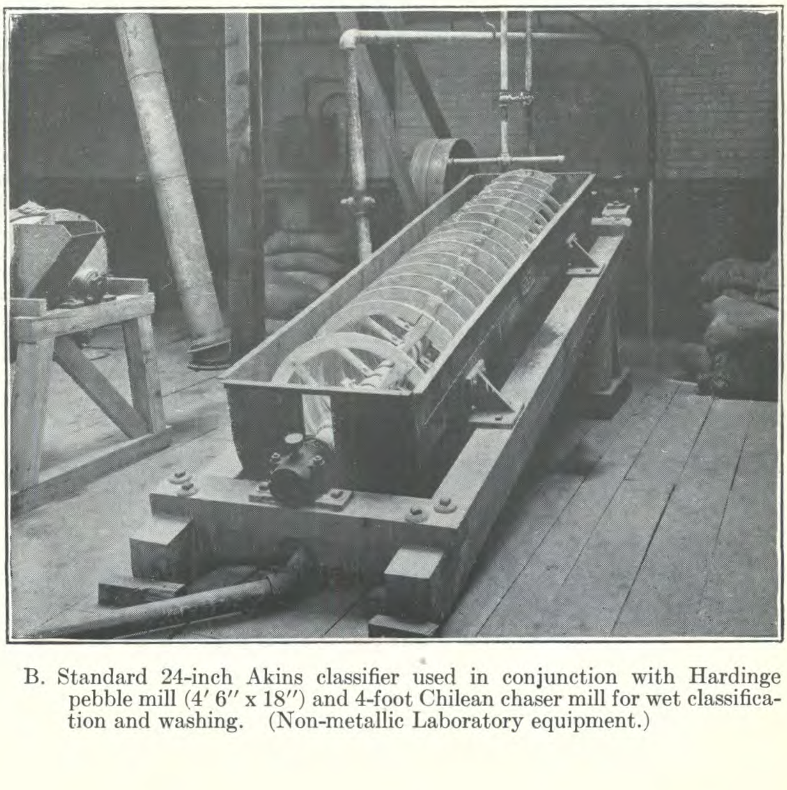
PLATE III



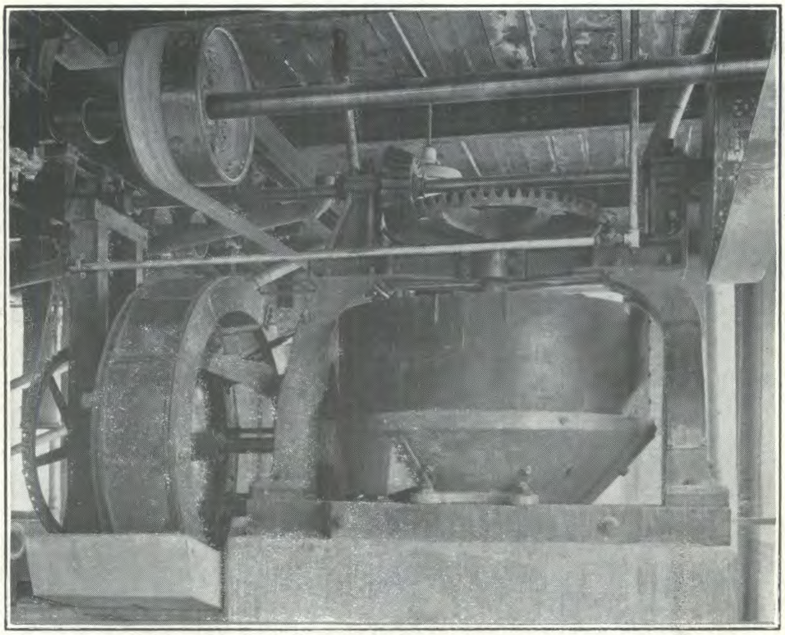
Two-tray (12′ x 12′), Dorr thickener used for thickening water-floated products from bowl classifier. Two-plunger, 4-inch slip pump on the left delivers thickened product to the filter press. In the foreground is a 30-inch Gayco air separator. (Non-metallic Laboratory equipment.)



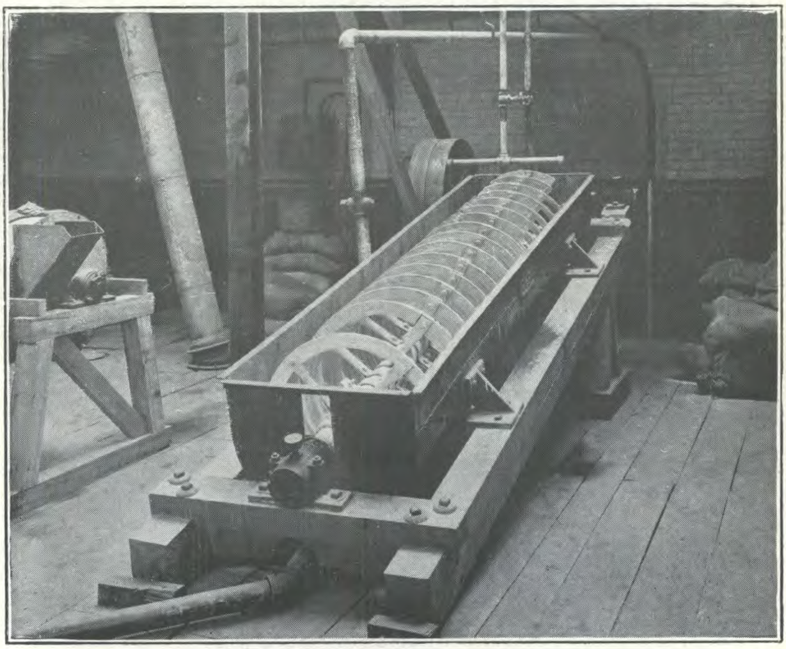
A. Four-foot Chilean chaser mill used for the grinding of silica sands, sandstone, etc. (Non-metallic Laboratory equipment.)



B. Standard 24-inch Akins classifier used in conjunction with Hardinge pebble mill (4′ 6″ x 18″) and 4-foot Chilean chaser mill for wet classification and washing. (Non-metallic Laboratory equipment.)

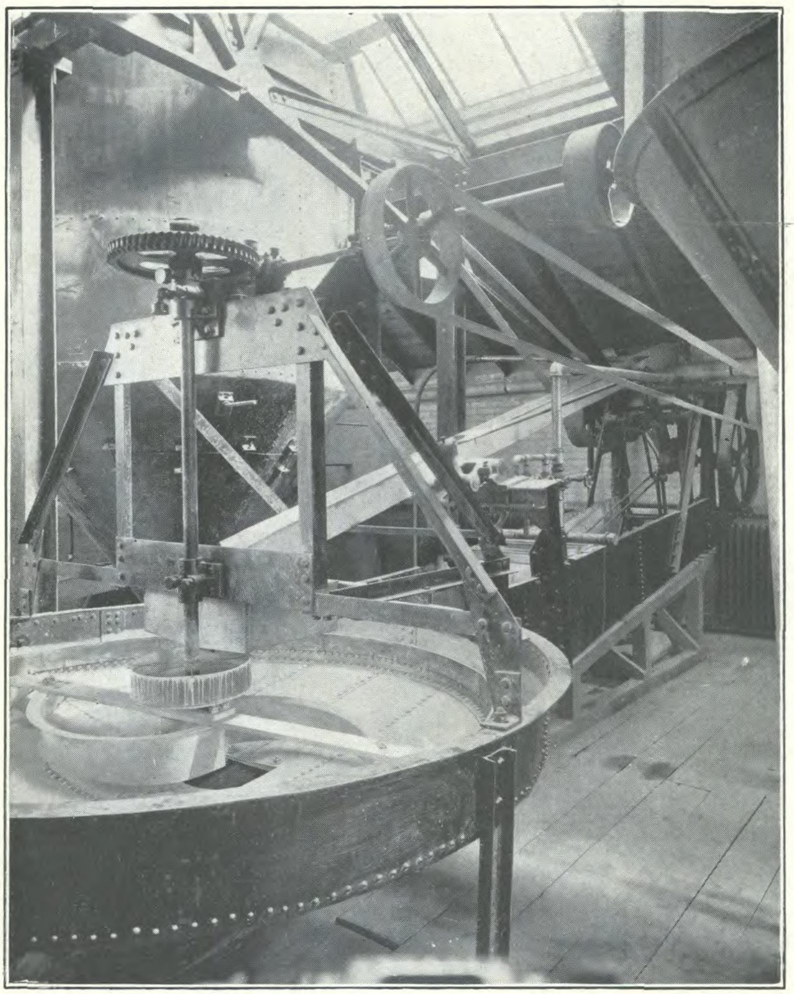


A. Four-foot Chilean chaser mill used for the grinding of silica sands, sandstone, etc. (Non-metallic Laboratory equipment.)

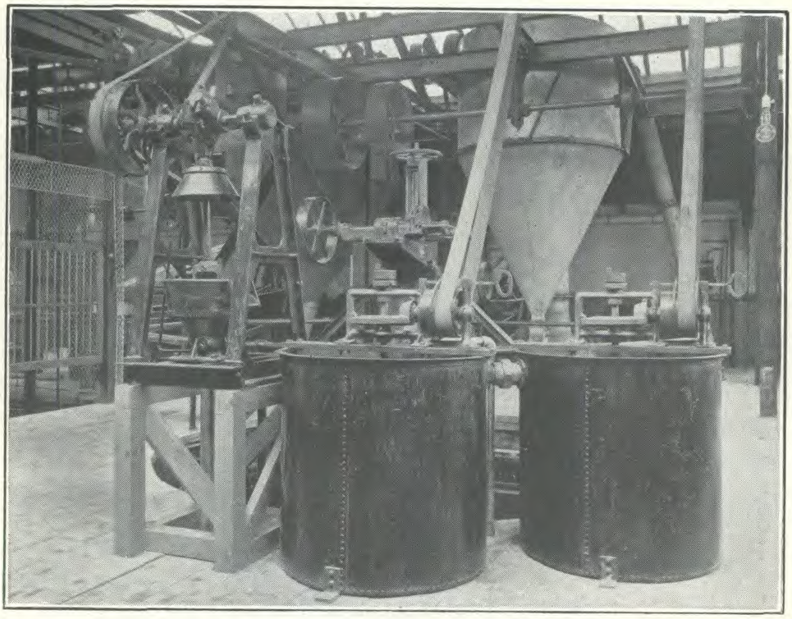


B. Standard 24-inch Akins classifier used in conjunction with Hardinge pebble mill (4′ 6″ x 18″) and 4-foot Chilean chaser mill for wet classification and washing. (Non-metallic Laboratory equipment.)

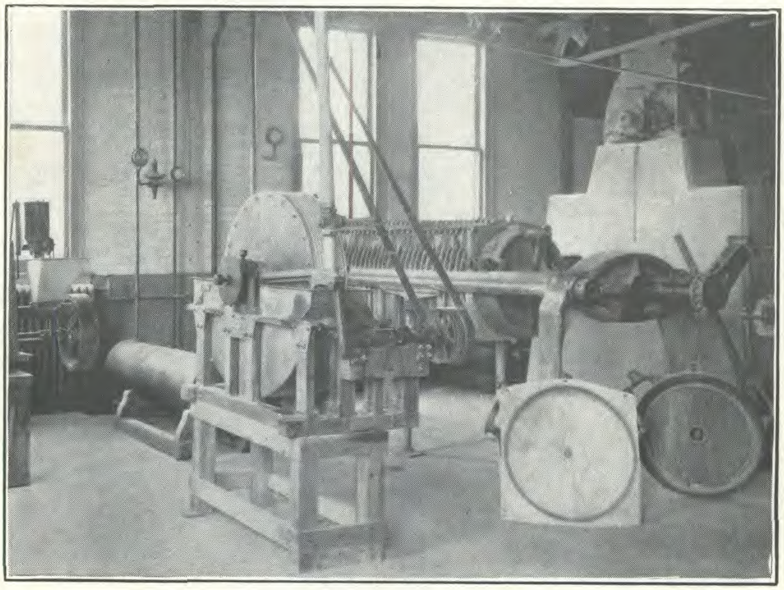
PLATE V



Dorr bowl classifier (2′ 3″ x 16′ 4″) with 7-foot bowl; 8-foot Sturtevant air separator in right background, 8-foot Gayco air separator in left background. The bowl classifier is used to obtain a fine water-floated product and the air separators are used to obtain a fine air-floated product. (Non-metallic Laboratory equipment.)



A. Dorrco pump. Two mechanical mixing tanks (3′ x 3′) in front of and above Dorr thickener. In the background are three air separators. (Non-metallic Laboratory equipment.)



B. Laboratory Oliver filter (3′ x 6″) and 36-chamber, 28-inch diameter, Patterson filter press used for filtering water-floated products from Dorr thickener. (Non-metallic Laboratory equipment).

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