CANADA
DEPARTMENT OF MINES
HON. W. A. GORDON, MINISTER; CHARLES CAMSELL, DEPUTY MINISTER

MINES BRANCH
JOHN MCLEISH, DIRECTOR

INVESTIGATIONS IN ORE DRESSING AND METALLURGY

(Testing and Research Laboratories)

1932

OTTAWA
J. O. PATENAUDE
PRINTER TO THE KING'S MOST EXCELLENT MAJESTY
1934

No. 736

General view of the mineragraphic laboratory. The microscope table is shown at the left. The photomicrographic unit is shown in the centre foreground; the Harvard polishing machine and the mounting unit in the centre background; a portion of the spectrographic apparatus appears at the right.

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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.

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MINES BRANCH INVESTIGATIONS IN

ORE DRESSING AND METALLURGY, 1932

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm

Chief of Division

The foregoing list of investigations, conducted and reported on during the year 1932, is evidence of the extent to which those engaged in the mining industry of Canada make use of the facilities of the Ore Dressing and Metallurgical laboratories, for the solution of problems in ore treatment. Sixty-four investigations were reported upon, the greatest number for any one year since laboratory facilities were provided for this type of investigative work. Thirty-two, or one-half the total, were on gold ores, showing the activity which took place in this branch of the industry. A number of metallurgical investigations were conducted for the Aeronautical Division of the Department of National Defence and for the steel companies which are not included in this summary report.

Based on the results of the investigations in Ore Dressing and Metallurgy, new milling plants and concentrators have been built, are underway or contemplated, and changes have been made to existing plants to effect economies of operation and increased recoveries. In British Columbia, the milling plant of the Cariboo Gold Quartz Mining Company has commenced operations and the concentrator at the Surf Point mine, Porcher Island, is under construction. In Manitoba, the San Antonio and a small mill at Herb Lake commenced milling operations and a milling plant is being built at the Island Lake properties of Ventures Limited. In Ontario, a milling plant was built on the Ashley property of the Ashley Gold Mining Company and a concentrator was under construction for the treatment of the disseminated copper-nickel ore of the Falconbridge Mine. In Quebec, the first unit of a milling plant was being built at the Beattie mine; changes were made to the Granada mill and also to the Siscoe mill to effect economies and increased recoveries. At Port Hope, Ontario, a treatment plant and refinery was built by the Eldorado Gold Mines, Limited, and commenced operations in January, 1933, for the extraction of radium from the Great Bear Lake pitchblende. The steel plants are extending their operations along new lines, manufacturing new and higher grade products.

The Reports of Investigations are in five parts. The investigations dealing with the milling and concentration of metallic ores were conducted by C. S. Parsons, A. K. Anderson, J. D. Johnston, and W. S. Jenkins and are given in Section II. Those dealing with the dressing of non-metallic minerals were conducted by R. K. Carnochan and R. A. Rogers and are given in Section III. Section IV contains the report on "Methods of Treating Great Bear Lake Pitchblende for the Extraction of Radium" by R. J. Traill, the report of the Radium-Measuring Laboratory and a report on "Precautions for Workers in the Treating of Radium Ores" by W. R. McClelland. Section V contains a summary of the work of the Section of Ferrous Metallurgy and the reports of investigations conducted on iron ores by T. W. Hardy and H. H. Bleakney. The report of the Chemical Laboratory is given by H. C. Mabee, Chief Chemist, in Section VI.

During 1931 there were added to the laboratory equipment for the investigation of problems in ore treatment, modern facilities for the microscopic examination and spectrographic analysis of ores and mill products. The mineragraphic laboratory and its facilities are fully described in Memorandum Series No. 58 of the Mines Branch. The microscopic study of the ores before commencing experimental tests and of the mill products during progress of the test work gives the investigator valuable information as to the mineral constituents, their characteristics, relationships, mode of association, and grain size, which are of great value in conducting the experimental work. A summary of the results of the microscopic examination of the ores under investigation and of the mill products is embodied in the reports of investigations. M. H. Haycock, mineralographer, issued during the year sixty reports giving the results of the examination of 411 polished and 30 thin sections, and 156 spectrographic analyses. In addition to the microscopic and spectrographic work in connexion with the investigations in ore dressing and metallurgy he was engaged in making a fairly complete study of the minerals in the pitchblende and silver ores of the Great Bear Lake area, North West Territories.

The use of the laboratory facilities and the co-operation of the staff were extended to and taken advantage of by consulting engineers and metallurgists from operating companies, who spent considerable time in the laboratories investigating their own particular problems. The benefit of the experience and knowledge of the staff gained in conducting the investigative work was given to consulting engineers and metallurgists engaged on the design, construction and operation of new milling plants, and on operating problems of existing plants.

II

REPORTS OF INVESTIGATIONS: METALLIC ORES SECTION

Report No. 424

THE RECOVERY OF GOLD FROM THE ORE OF THE ASHLEY PROPERTY, MATACHEWAN AREA, ONTARIO

Shipment. Thirteen sacks of ore, weighing 700 pounds, were received November 21, 1931, shipped by the Mining Corporation of Canada, Limited. The shipment was said to be assay rejects representing composites of drift samples of the Ashley property, Matachewan area, Ontario.

Characteristics of the Ore. The shipment consisted of crushed material passing approximately 14 to 20 mesh, apparently siliceous in nature.

Purpose of Experimental Tests. The shipment was made to determine what recovery of the contained gold could be recovered by cyanidation, and also to note its response to other methods of treatment.

Straight cyanidation was found to be applicable giving a recovery of 97·8 per cent from material minus 150 mesh.

Sampling and Analysis. The entire shipment was mixed and passed through a Jones riffle sampler cutting out one-twentieth of the weight. This portion was stage-ground to pass succeeding finer meshes with intervening cuts through the sampler until a representative portion minus 100 mesh was secured for assay which showed the shipment to contain 1·83 ounces gold, and 0·20 ounce silver per ton. The ore did not contain coarse gold as all but 0·10 ounce passed 100 mesh.

Experimental Tests

The investigation included tests by cyanidation, amalgamation and cyanidation, and flotation and cyanidation. The cyanide tests were checked by a continuous run on 150 pounds of the ore.

STRAIGHT CYANIDATION

Test No. 1

Representative portions of the ore were ground to pass 48, 100, 150 and 200 mesh and cyanided for 48 hours, 1 : 3 dilution with a sodium cyanide solution equivalent to 1·0 pound KCN per ton. Lime equal to 9 pounds per ton of ore was added for protective alkalinity.

Results:

Grind	Heads, Au, oz./ton	Cyanide tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton	
				KCN	CaO
− 48 mesh	1·83	0·09	95·1	0·3	4·65
−100 mesh	1·83	0·06	96·7	0·3	5·85
−150 mesh	1·83	0·04	97·8	0·8	7·5
−200 mesh	1·83	0·08	95·1	1·2	8·4

These results indicate that ore ground to pass 150 mesh can be cyanided to yield 97·8 per cent of the contained gold, leaving a tailing of 80 cents.

AMALGAMATION AND CYANIDATION

Test No. 2

A series of tests was run on ore ground to pass 48, 100, 150, and 200 mesh. The samples were ground dry and amalgamated in a 1 : 1 pulp. After removing the mercury, the pulp was filtered and cyanided 1 : 3 dilution without further grinding for 24 hours with a 1·0 pound KCN solution. Six pounds of lime per ton of ore was added.

A second series of tests was made on the amalgamation tailing reground to pass 200 mesh.

Amalgamation Tailing Cyanided

Grind	Heads, Au, oz./ton	Amalgam. tails = Cy.heads, Au, oz./ton	Recovery by Amalgamation, per cent	Cyanide tailing, Au, oz./ton	Recovery by cyanide, per cent	Total recovery, per cent	Reagent consumption, lb./ton	
							KCN	CaO
− 48 mesh	1·83	0·41	77·6	0·19	12·0	89·6	0·6	3·75
−100 mesh	1·83	0·37	79·8	0·10	14·7	94·5	0·6	3·3
−150 mesh	1·83	0·38	79·2	0·18	11·0	90·2	0·3	5·1
−200 mesh	1·83	0·61	66·6	0·08	29·1	95·7	0·9	5·7

Reground Amalgamation Tailing Cyanided

Grind	Heads, Au, oz./ton	Amalgam. tails = Cy.heads, Au, oz./ton	Recovery by Amalgamation, per cent	Cyanide tailing, Au, oz./ton	Recovery by cyanide, per cent	Total recovery, per cent	Reagent consumption, lb./ton	
							KCN	CaO
− 48 mesh	1·83	0·41	77·6	0·08	18·0	95·6	0·3	4·8
−100 mesh	1·83	0·37	79·8	0·06	16·9	96·7	0·6	5·2
−150 mesh	1·83	0·38	79·2	0·07	17·0	96·2	0·9	5·6
−200 mesh	1·83	0·61	66·6	0·09	28·5	95·1	0·9	5·7

The results indicate that there is no metallurgical advantage in amalgamation followed by cyaniding the amalgamation tailing. The recovery made on 100-mesh material by this system is the same as that secured by cyanidation alone, viz.: 96·7 per cent. Cyanidation of the ore ground to pass 150 mesh gives a recovery of 97·8 per cent.

FLOTATION AND CYANIDATION

A portion of the ore was ground, 70 per cent solids, until 78 per cent passed 200 mesh and then floated with 4 pounds soda ash, 0·09 pound Aerofloat No. 25, 0·10 pound potassium ethyl xanthate and 0·06 pound pine oil per ton. After removing the concentrate, the tailing was dewatered and cyanided for 24 hours, 1 : 3 dilution, with a 1·0 pound KCN solution and 4 pounds lime per ton.

Flotation:

Product	Weight, per cent	Assay		Metals, per cent	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Heads (cal.)	100·00	1·44	0·20	100·0	100·0
Flotation concentrate	3·65	35·62	4·06	90·3	75·0
Flotation tailing	96·35	0·15	.....	9·7	

Cyanidation of the flotation tailing left a residue containing 40 cents per ton. This combined with the recovery made by concentration gives a total recovery of 98·7 per cent.

The flotation concentrate was not cyanided. As the raw ore yields to cyanidation, the concentrate should cyanide quite readily. However, as nothing is gained by cyaniding both products of flotation, the proper treatment is straight cyanidation.

CONTINUOUS CYANIDE TEST

Ore ground to pass 20 mesh was fed at the rate of 30 pounds per hour to a small rod mill in closed circuit with a drag classifier. A 1·0 pound per ton KCN solution was fed at the head of the mill to make the rod mill discharge about 70 per cent solids. Lime equal to 4 pounds per ton was mixed with the ore in the bin. The rod mill discharge was diluted with a 1·0 pound per ton KCN solution to give a classifier overflow approximately 30 per cent solids. This was transferred to an agitating tank which when full contained 155 pounds of ore. This tank had a conical bottom through which the pulp circulated through a pump to the top of the tank. A jet of air assisted agitation and ensured sufficient air for cyanidation. Agitation was continued for 40 hours, samples being taken at 18-, 24- and 40-hour periods. The classifier overflow was also sampled to determine the extraction in the grinding circuit. It was found necessary to add additional lime to the pulp during the agitation period to maintain protective alkalinity, 11 pounds per ton being required for the total operation. No additional cyanide was added.

Results:

Heads	Au 1·83 oz./ton
Classifier overflow	Au 0·33 "
Extraction in grinding circuit	82·0 per cent
18-hour tailing	Au 0·30 oz./ton
Extraction	83·6 per cent
24-hour tailing	Au 0·08 oz./ton
Extraction	95·6 per cent
40-hour tailing	Au 0·07 oz./ton
Extraction	96·2 per cent
Ratio of ore to solution in agitation tank	1:1·74
Cyanide consumption	0·5 lb./ton
Lime "	10·3 "

Screen Analysis of 40-hour Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
+100	15·5	0·05	12·1
−100+150	10·9	0·04	6·8
−150+200	13·5	0·03	6·3
−200	60·1	0·08	74·8

These results show that the highest recovery is made on 150-mesh material, checking the small-scale tests, and also showing that minus 200-mesh grinding results in lower recoveries.

SUMMARY AND CONCLUSIONS

The investigation shows that the shipment of ore contains no coarse gold as all but 0·10 ounce passes through 100 mesh.

Straight cyanidation is the method to apply for the recovery of the gold. Grinding should approach minus 150 mesh.

Overgrinding of the ore should be avoided. Two-stage grinding will help to reduce the formation of slimes.

The ore is fast-settling. Tests show that 1·34 square feet settling area to be required per ton per 24 hours to secure a thickener underflow 50 per cent solids from a feed 1 of ore to 1·74 solution and a clear overflow.

The treatment of this ore should not present any metallurgical difficulties as it responds quite readily to simple cyanidation.

Report No. 425

THE CONCENTRATION OF CHROMITE ORE FROM OBONGA LAKE, ONTARIO

Shipment. A shipment of 40 bags of ore was received at the Ore Dressing and Metallurgical Laboratories on June 19, 1931, by freight from Collins, Ont. D. A. Mutch, Haileybury, Ont., for the Consolidated Chromium Corporation, 52 Vanderbilt Avenue, New York City, was the shipper.

The forty bags included two lots of equal size, Lot No. 1 of vein material and Lot No. 2 of zone material.

Characteristics of the Ore. Lot No. 1, vein material, consisted of a chlorite schist carrying some magnetite crystals and black grains scattered throughout the rock which could not be ascertained by microscopic methods, but did not look like typical chromite. Part of Lot Nos. 1 and 2 consisted of extremely altered pyroxenite or peridotite rock almost completely transformed into a talcose rock. Throughout this, grains of magnetite or chromite were scattered in limited amounts. Their average size was not much above one-sixth millimetre.

Sampling and Analysis. The two lots were sampled separately. Of the forty bags all but six were identified.

The lots were crushed to one-half inch, quartered, crushed again, and by succeeding finer grinding and quartering through a Jones riffle sampler, a representative portion minus 100 mesh was secured for analysis. The analyses were as follows:—

Lot No. 1 (20 bags)	4·44 per cent	Cr_2O_3
" 2 (14 ")	1·89	"

Purpose of Experimental Tests. The shipment was made for the purpose of ascertaining the possibilities of economically concentrating the ore to 48 to 50 per cent chromium oxide. The ore on analysis proved to be much lower in chromium than was anticipated.

Summary of Experimental Tests. The results of the small-scale tests are not conclusive and in order to obtain more definite information a larger scale gravity test will be made.

The flotation tests were not successful as very little evidence of a separation was observed. The large amount of talc present in the ore also interfered with the separation as it floated more readily than the chromite, and must be eliminated before the chromite can be floated.

The results of the gravity tests on the small laboratory table, made on a sized feed, are disappointing inasmuch as the grade of concentrate produced was under the requirements of 48 per cent chromium oxide.

Experimental Tests

All tests were made on Lot No. 1, consisting of vein material and assaying 4·44 per cent Cr_2O_3. The tests consisted of gravity separations made on a small laboratory Wilfley table and series of small-batch flotation tests.

The gravity tests on the table were made on a sized feed and an attempt was also made to secure a classified feed for the table, but the large amount of talc present in the ore interfered with the operation.

A representative 6,000-gramme portion of the ore was ground to pass 48 mesh and screened on 65, 100, 150 and 200 mesh. These products were then passed over a Wilfley table giving the following results:—

Table Concentration of −48 mesh and screened

Product	Weight, per cent	Assay, Cr_2O_3, %
− 48+ 65	12·5	6·10
− 65+100	20·9	5·90
−100+150	10·7	5·70
−150+200	12·3	4·30
−200	43·6	3·00

Table Concentration of −48+65-mesh size

Product	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
Heads (cal.)	100·0	5·93	100·0
Table concentrate	6·0	38·70	39·2
Table middling	14·4	15·66	38·0
Table tailing	79·6	1·70	22·8

At this mesh there is no economic recovery of chromium. The concentrate is also lower in grade than that specified.

Table Concentration of −65+100-mesh size

Product	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
Heads (cal.)	100·0	5·30	100·0
Table concentrate	5·0	36·95	34·8
Table middling	11·4	22·00	47·2
Table tailing	83·6	1·90	18·0

Concentration of this size of material does not yield the results required.

Table Concentration of −100+150-mesh size

Product	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
Heads (cal.)	100·0	5·19	100·0
Table concentrate	1·2	37·14	8·6
Table middling	22·2	15·40	65·9
Table tailing	76·6	1·73	25·5

Here again, the grade of concentrate produced is not up to the required limits of 48 to 50 per cent Cr_2O_3.

Table Concentration of −150+200-mesh size

Product	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
Heads (cal.)	100·0	2·87	100·0
Table concentrate	1·7	35·56	15·4
Table middling	7·9	21·92	44·2
Table tailing	90·4	1·75	40·4

Table Concentration of −200-mesh size

Product	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
Heads (cal.)	100·0	2·87	100·0
Table concentrate	0·6	30·70	6·4
Table middling	1·4	25·37	12·4
Table tailing	98·0	2·38	81·2

None of these tests indicates that a concentrate containing 48 to 50 per cent Cr_2O_3 can be obtained. Apparently the chromite is closely associated with other minerals, possibly iron oxides which have approximately the same specific gravity.

No economic recovery is obtained as the tailings are all quite high in chrome oxide.

Report No. 426

THE CONCENTRATION OF CHROMITE FROM LAKE SHEBANDOWAN, ONTARIO

Shipment. One bag containing 120 pounds of material was received on November 3, 1931. This was shipped by freight from Port Arthur, Ontario, by J. G. Cross, 14 High St. It consisted of chrome-bearing rock said to come from the Cross nickel-copper claims in the Lake Shebandowan area.

Characteristics of the Ore. The ore was greenish grey, intermediate or basic schist containing minute crystals of chromite and possibly magnetite.

Purpose of Experimental Tests. The shipment was sent in to determine the commercial possibilities of the deposit. Concentration tests were undertaken with the object of making a commercial grade of chromite concentrate.

Sampling and Analysis. The entire shipment was crushed to pass $\frac{1}{4}$ inch and quartered. One quarter was ground to succeeding finer sizes with intervening cuts through a Jones riffle sampler until a representative sample minus 100 mesh was secured. This was found to contain 16·85 per cent chromium oxide.

Experimental Tests

The investigation consisted of table concentration of the ore sized through different meshes, and concentration of classified products.

TABLE CONCENTRATION OF SIZED FEED

Test No. 1

A representative 3,000-gramme portion of the ore was ground to pass 65 mesh and screened on 100, 150, and 200 mesh. Each size was then passed over a laboratory-size Wilfley table making a concentrate, middling, and tailing.

Results:

Screening of Head Sample

Screen size	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
− 65+100	20·3	24·24	27·5
−100+150	20·2	17·65	19·9
−150	59·5	15·79	52·6

Concentration of −65+100-mesh product

Product	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
Table concentrate	42·0	30·77	53·3
" middling	41·0	23·87	41·4
" tailing	17·0	7·57	5·3

Concentration of −100+150-mesh product

Product	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
Concentrate	18·3	18·66	19·3
Middling	46·5	24·37	64·2
Tailing	35·2	8·26	16·5

This test indicates that chromite is not the heaviest mineral present as the middling has a higher Cr_2O_3 content than the concentrate.

Concentration of —150-mesh product

Product	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
Concentrate	7·3	30·76	14·2
Middling	14·3	29 96	27·1
Tailing	78·4	11·82	58·7

The combined concentrates secured from the above products represent 16·6 per cent of the weight of ore milled and have a content of 27·49 per cent Cr_2O_3. An additional 26·3 per cent of the weight of feed is recovered as middlings with a content of 25·19 per cent Cr_2O_3. The concentrate and middlings together contain 26·07 per cent Cr_2O_3. A total recovery of 64·4 per cent of the chrome oxide is obtained.

TABLE CONCENTRATION OF CLASSIFIED FEED

Test No. 2

A 5,000-gramme sample of the ore, ground to pass 48 mesh, was fed to a pulsating hydraulic classifier fitted with a screen bottom, making four products, a hutch, the material passing through the screen, spigot No. 1, spigot No. 2, and classifier overflow.

All products excepting the hutch were passed over the Wilfley table making a concentrate, a middling, and a tailing.

Results:

Classification

Product	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
No. 1 spigot	29·7	16·84	29·3
No. 2 "	12·5	14·73	10·8
Classifier overflow	36·6	13·17	28·2
Hutch	21·2	25·56	31·7

Concentration of No. 1 Spigot

Product	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
Concentrate	38·8	19·23	44·3
Middling	37·2	19·64	43·4
Tailing	24·0	8·65	12·3

Concentration of No. 2 Spigot

Product	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
Concentrate	13·8	23·16	21·7
Middling	37·3	20·75	52·5
Tailing	48·9	7·76	25·8

Concentration of Classifier Overflow

Product	Weight, per cent	Assay, Cr_2O_3, per cent	Chromite, per cent
Concentrate	4·3	22·70	7·4
Middling	32·9	19·00	47·4
Tailing	62·8	9·47	45·2

The hutch product, being a pure concentrate, was not passed over the table. It had the highest content of Cr_2O_3 of any of the concentrates, viz., 25·56 per cent.

The combined concentrates made in this test had a content of 23·30 per cent Cr_2O_3 and weighed 36·0 per cent of the feed. The middlings assayed 19·83 per cent Cr_2O_3.

The combined concentrates and middlings represented 63·7 per cent of the weight of ore milled, contained 21·79 per cent Cr_2O_3, representing a total recovery of 80·9 per cent of the chromite in the ore.

An endeavour was made to raise the grade of concentrate by magnetic separation. This was not successful as the entire concentrate was magnetic.

Flotation did not make any separation of the chromite from the gangue.

FINE GRINDING AND TABLE CONCENTRATION

Test No. 3

In this test the ore was all ground through 24 mesh and tabled without sizing. The middling was reground to minus 100 mesh and tabled again.

The results are summarized as follows:—

Product	Weight, grms.	Weight, per cent	Analysis						Distribution
			Cr_2O_3	Fe_2O_3	Al_2O_3	SiO_2	MgO	CaO	Cr_2O_3
Heads (feed)	29,056·0	100·0	16·58	.....	.....	.....	.....	.....	100·0
— 24 mesh concentrates	2,043·0	.....	34·9	31·8	14·0	7·2	7·9	.....	14·3
—100 mesh " ..	3,527·0	.....	34·8	30·0	19·0	7·2	6·2	1·2	22·6
—100 mesh " ..	2·347·0	.....	32·4	32·6	13·0	10·2	.....	.....	15·1
—100 mesh tailing No. 1	1,707·0	.....	7·17	.....	.....	.....	.....	.....	2·4
—100 mesh " No. 2	1,710·5	.....	12·8	.....	.....	.....	.....	.....	4·4
—100 mesh middling	1,496·0	.....	16·0	.....	.....	.....	.....	.....	4·8
Slimes	16,225·0	.....	11·2	.....	.....	.....	.....	.....	36·4
Totals	29,056·0	.....	.....	.....	.....	.....	.....	.....	100·0

This test shows a recovery of only 52·0 per cent of the chromite but it is possible that 50 per cent of the chromite in the slime product could be saved by thickening and treatment on slime tables or vanners.

The concentrate obtained from this test was practically pure mineral and represents the highest grade possible to obtain from this sample.

SUMMARY AND CONCLUSIONS

A summary of these tests shows that it is possible to obtain a concentrate containing 30 per cent Cr_2O_3 with a recovery of between 65 and 70 per cent. They also show that in order to obtain this grade of concentrate grinding to minus 100 mesh is required.

The conclusion reached from these tests is that the chromite itself contains a high ratio of iron and aluminium. The ratio of iron to chromium for practical purposes can be taken as one to one. A concentrate containing 30 per cent Cr_2O_3 is, therefore, about as high as it is possible to obtain from this ore.

Report No. 427

EXPERIMENTAL TESTS ON GOLD-BEARING SULPHIDES FROM PLACER DEPOSITS IN THE CARIBOO DISTRICT OF BRITISH COLUMBIA

Shipments. A shipment of six lots of sulphide concentrates, weighing in all 12,820 grammes, was received January 1, 1932. The samples were submitted by A. R. Larkin, President, Cedar Creek Placer Gold Company, Ltd., 541 Georgia Street West, Vancouver, B.C.

Details of the shipment are as follows:—

Drill holes 23-40	3,420	grammes	from 9 cubic feet rocker tailing.
Drill hole 59	2,891	"	mixed.
Drill hole 58	2,689	"	concentrates.
Drill hole 56	3,300	"	from 7 cubic yards rocker tailing.
Drill holes 14-20	270	"	from 12 cubic feet rocker tailing.
Drill hole 57	250	"	from 0·12 cubic yard.
	12,820	"	

Characteristics and Analysis of the Sample. The sulphides are gold-bearing and are said to occur in large placer deposits in the Cariboo District of British Columbia. The samples submitted were obtained from drill cores and would pass through a 20-mesh screen. The cores were washed and the resulting concentrates passed over a small laboratory-size Wilfley table. Free gold was panned from the table concentrate and the sulphides remaining constituted the sample submitted for test purposes. The sulphides are chiefly iron pyrite and pyrrhotite in various stages of oxidation. The gangue is siliceous.

The six lots comprising the shipment were mixed together and a head sample cut by a Jones sampler. Assays showed, gold 0·33 ounce per ton.

Purpose of Experimental Tests. The concentrate, as it occurs, contains some free gold which can be panned out or readily amalgamated and some gold intimately associated with the sulphide particles. The problem is to find an efficient and economical method of recovering the gold in the sulphides. The work carried out consisted of tests by cyanidation and by amalgamation.

Summary of Results. Duplicate tests by straight cyanidation gave an average extraction of 92·4 per cent of the gold. The concentrates were ground dry through 40 mesh and were agitated for 36 hours in 3 : 1 pulp. Consumption of reagents was as follows:—

KCN	1·5 lb./ton
CaO	23·0 "

With the concentrate ground to 40 to 45 per cent through 200 mesh amalgamation for one hour extracted 93·9 per cent of the gold. By grinding 80 to 85 per cent through 200 mesh 97 per cent of the gold was extracted in 38 minutes. The addition of a small amount of lime at the beginning of the grinding operation did not prove beneficial but was perhaps harmful as the extraction was lowered to 95·5 per cent of the gold with all other conditions the same as in the test described immediately above.

Experimental Tests

Amalgamation Tests

Tests Nos. 1, 4, and 5

In this series of tests the concentrate was ground wet in jar mills to varying degrees of fineness in 500-gramme lots, with and without the addition of lime. They were filtered and repulped to 1 : 1 density using some of the water in which they were ground. The pulp was amalgamated in a jar mill using 50 grammes of mercury in each test but varying the time in different tests. The amalgam was separated and found to be in good condition. The amalgamation tailings were filtered and washed and assayed for gold. Screen tests were made to determine the fineness of grinding.

Summary:

Head sample: Au, 0·33 oz./ton.

Test No.	Tailings, oz./ton	Extraction, per cent	Time of grinding, minutes	Time of amalgamation, minutes	Reagents, Hg, grm.	Used, CaO, lb./ton
1	0·02	93·9	10	60	50	nil
4	0·015	95·5	20	38	50	4
5	0·01	97·0	20	38	50	nil

Screen Tests on Tailings:

Test No. 1

Mesh	Weight, grms.	Weight, per cent
+ 48	2·7	1·4
+ 65	5·0	2·5
+100	18·5	9·2
+150	55·7	27·9
+200	32·6	16·3
−200	85·4	42·7
Total	200·0	100·0

Test No. 5

Mesh	Weight, grms.	Weight, per cent
+100	1·4	1·4
+150	4·4	4·4
+200	11·4	11·4
−200	82·8	82·8
Total	100·0	100·0

CYANIDATION

Test No. 2 a and b

The cyanidation test was done in duplicate. The concentrate was ground dry through 40 mesh. Agitation was carried on for 36 hours in 3 : 1 pulp with 25 pounds of lime added per ton of concentrate and a solution of 0·15 per cent KCN. The tailing was filtered and washed and assayed for gold.

Summary:

Head sample: Au, 0·33 oz./ton.

Test No.	Tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
2a	0·02	93·9	1·5	23
2b	0·03	90·9	1·5	23

SUMMARY AND CONCLUSIONS

It is evident from the foregoing results of tests that grinding 80 to 85 per cent through 200 mesh followed by amalgamation gives the highest extraction of the gold, viz., 97 per cent. The addition of lime does not seem to be worth while and coarser grinding lowers the extraction to 93·9 per cent.

Extraction by cyanidation is somewhat lower than that by amalgamation and at the same time the consumption of lime is rather high—23 pounds per ton. The average extraction obtained in duplicate tests using this process is 92·4 per cent of the gold after 36 hours' agitation and a KCN consumption of 1·5 pounds per ton.

It seems obvious that amalgamation is the most practical and economical method of handling these sulphides. Grinding is a simple matter since 20 minutes in a jar mill will reduce the material to 80 to 85 per cent minus 200 mesh. In this state 97 per cent of the gold is extracted by amalgamation. Furthermore, it would be unnecessary to pan out any free gold, which has been done with these samples, but simply grind the table concentrates and amalgamate directly.

The fineness of grinding necessary or desirable is a matter for speculation and observation, but the slightly higher extraction shown in Test No. 5, as compared with Test No. 4, hardly justifies the extra grinding necessary to bring it about.

70362—2½

Report No. 428

EXPERIMENTAL TESTS ON GOLD ORE FROM GOLD RAND MINERAL CLAIM, NEAR SANCA, B.C.

Shipment. A shipment of six sacks of ore, total weight 500 pounds, was received December 16, 1931. The sample was submitted by W. Frampton, Sanca Mines, Ltd., Sanca, B.C., in the name of Canada Smelters, Ltd., Sanca, B.C.

Characteristics and Analysis of the Sample. The sample submitted was highly oxidized material, high in iron, and contained practically no sulphides. The gangue was siliceous.

An average analysis of the sample was as follows:—

Au	0·48 oz./ton	Zn	0·05 per cent
Ag	4·84 "	Cu	0·35 "
Fe	28·0 per cent	Insol	30·05 "
Pb	0·91 "		

Purpose of Experimental Tests. The purpose of investigation was to find how the ore responded to treatment and to form an opinion whether the mill the company is now building will handle the ore satisfactorily. The work included tests by table concentration, flotation, amalgamation and cyanidation.

Summary of Results. A small-scale test was carried out on a small laboratory-size table. Three products were taken off with recoveries of the gold as follows:—

Concentrate	31·5 per cent
Middling	20·7 "
Tailing	47·8 "

A screen analysis showed that 98·7 per cent of the gold in the table tailing was in the minus 200-mesh size.

Flotation tests were made on the ore and showed recoveries in the concentrates of 41·3 per cent and 47·9 per cent of the gold, whereas the tailings ran approximately six dollars per ton in gold.

Amalgamation gave very poor results. Samples of the ore were ground dry through 48 and 100 meshes and amalgamation tests were made on each size. The recoveries of gold were 10·4 and 6·3 per cent respectively.

A series of cyanidation tests was made on samples of the ore ground dry through 48, 100, 150 and 200 meshes. Agitation for 24 hours gave a maximum recovery of 79·2 per cent with a cyanide consumption of 1·5 pounds per ton of ore and a lime consumption ranging from 74·5 pounds per ton of ore on the coarser sizes to 76·5 pounds per ton of ore on the finer. Agitation for 48 hours gave the same maximum recovery with the same cyanide consumption but the lime consumption was higher ranging from 80 pounds per ton of ore on the coarser sizes to 82·5 pounds per ton of ore on the finer.

Two tests were made in which the ore was water-washed before it was treated with cyanide solution. This made no change in the lime consumption but the cyanide consumption rose from 1·5 to 1·8 pounds. per ton of ore treated.

Experimental Tests

TABLE CONCENTRATION

Test No. 9

In this test 500 grammes of ore, ground dry through 100 mesh, was passed over a small laboratory-size table. Three products were taken off, concentrate, middling, and tailing.

Results:

Product	Weight, per cent.	Assay, Au, oz./ton	Recovery, per cent
Head (cal.)	100	0·46	100·0
Concentrate	23·14	0·62	31·5
Middling	26·14	0·36	20·7
Tailing	50·72	0·43	47·8

Screen Analysis of Table Tailing

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Average tailing (cal.)	100·0	0·43	100·0
+200	3·63	0·16	1·3
−200	96·37	0·44	98·7

FLOTATION

Tests Nos. 7 and 8

An attempt was made to concentrate the ore by flotation. In Test No. 7, 1,000 grammes of ore in 500 c.c. of water was ground to 75 to 80 per cent minus 200 mesh in a jar mill with 0·14 pound per ton of Aerofloat No. 25 and 20 pounds per ton soda ash added. The pulp was then transferred to the flotation machine and 0·05 pound per ton potassium xanthate added. The pulp was floated for a few minutes and then 0·12 pound per ton of cresylic acid was added and flotation continued. The concentrate and tailing were filtered, washed and assayed for gold. It was found that the soda ash added was not sufficient to make the pulp alkaline.

In Test No. 8 the same reagents were used as in Test No. 7 but the soda ash was raised to 40 pounds per ton of ore. Flotation was carried on longer than in Test No. 7 and more concentrate was collected but it assayed lower in gold. The pulp was alkaline at the end of the test.

Results Test No. 7:

Product	Weight, per cent	Assay, Au, oz./ton	Recovery, per cent
Head (cal.)	100·0	0·47	100·0
Concentrate	6·04	3·20	41·3
Tailing	93·94	0·29	58·7

Results Test No. 8:

Product	Weight, per cent	Assay, Au, oz./ton	Recovery, per cent
Head (cal.)	100·0	0·48	100·0
Concentrate	13·92	1·66	47·9
Tailing	86·08	0·29	52·1

AMALGAMATION

Tests Nos. 1 and 2

Amalgamation tests were made on two samples of ore ground dry through 48 and 100 meshes. In each case 1,000 grammes of the ore in 1 : 1 pulp was amalgamated for 30 minutes with 100 grammes of mercury. The tailings were filtered, washed, and assayed for gold.

Results of Tests Nos. 1 and 2:

Test No.	Tailing, Au, oz./ton	Recovery, per cent
1	0·43	10·4
2	0·45	6·3

CYANIDATION

Tests Nos. 4, 5, and 6

This series of tests was made on samples of ore ground dry through the following meshes: 48, 100, 150 and 200. In each case 200 grammes of ore was agitated in 3 : 1 pulp with 75 pounds of lime per ton of ore added. The solution strength was 1 pound per ton KCN. In Tests Nos. 4 and 5 the periods of agitation were 24 and 48 hours respectively. In Test No. 6 two samples of ore were water-washed before they were treated with cyanide solution; one of these was agitated for 24 hours and the other for 48 hours.

Results of Test No. 4: 24-hour agitation periods.

Head sample: Au, 0·48 oz./ton.

Mesh	Average tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
−48	0·16	66·6	1·65	76·24
−100	0·12	75·0	1·50	74·33
−150	0·11	77·1	1·50	74·33
−200	0·10	79·2	1·50	76·50

Screen Analysis: Cyanide Tailing —100 mesh

Product	Weight, per cent	Assay, Au, oz./ton	Au, per cent	Average tailing (cal.), Au, oz./ton
+200	32·0	0·17	44·5	0·12
−200	68·0	0·10	55·5	

Results of Test No. 5: 48-hour agitation periods.

Head sample: Au, 0·48 oz./ton.

Mesh	Average tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
			KCN	CaO
− 48	0·14	70·8	1·65	80·0
−100	0·11	77·1	1·50	80·24
−150	0·10	79·2	1·50	80·33
−200	0·10	79·2	1·65	82·40

Screen Analysis: Cyanide Tailing —150 mesh

Product	Weight, per cent	Assay, Au, oz./ton	Au, per cent	Average tailing (cal.), Au, oz./ton
+200	5·6	0·15	8·23	0·10
−200	94·4	0·10	91·77	

Results of Test No. 6: Samples water-washed.

Head sample: Au, 0·48 oz./ton.

Mesh	Average tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton		Period of agitation, hours
			KCN	CaO	
− 48	0·15	68·8	1·8	74·0	24
−150	0·11	77·1	1·8	80·24	48

CONCLUSIONS

The tests indicate that the ore is difficult to treat. Because of its highly oxidized condition and the absence of sulphides it cannot be treated by flotation. The very poor recovery by amalgamation shows the absence of any appreciable quantity of free gold, and since nearly 99 per cent of the gold in the table tailing is in the minus 200-mesh size fine grinding will be necessary to liberate it. The lowest tailing obtained by cyanidation was two dollars per ton in gold and that with a lime consumption of 75 to 80 pounds per ton of ore treated.

Report No. 429

THE RECOVERY OF GOLD IN ORE FROM THE DE SANTIS GOLD MINING COMPANY, LIMITED, TIMMINS, ONT.

Shipment. Three bags of ore, weighing approximately 250 pounds, were received, March 12, 1932, from the De Santis Gold Mining Company, Limited, Box 1299, Timmins, Ontario.

Characteristics of the Ore. The shipment consisted of material crushed to approximately $\frac{1}{2}$ inch, highly mineralized with crystals of iron pyrite in a carbonate gangue. No free gold was observed.

Purpose of Experimental Tests. Tests were required to determine the most suitable method for the recovery of the contained metals.

Sampling and Analysis. The shipment was mixed and crushed to 8 mesh and quartered by passes through a Jones riffle sampler. By further stage-crushing with intervening cuts through the sampler, a representative portion minus 100 mesh was obtained. This showed the shipment to contain 0·43 ounce gold and 0·07 ounce silver per ton.

Experimental Tests

The investigation included tests for the recovery of gold by amalgamation, flotation, and cyanidation. The results indicate that at moderately fine grinding 95 per cent of the gold is recoverable by cyanidation, while 98·8 per cent is recovered when the ore is ground minus 200 mesh.

AMALGAMATION

Two representative portions of the ore were ground dry, the first to pass 48 mesh and the other, 100 mesh. These were amalgamated with mercury with the following results:—

Mesh	Heads, Au oz./ton	Amalgamation tailing	Recovery, per cent
− 48	0·43	0·29	32·6
−100	0·43	0·19	55·8

Amalgamation does not recover sufficient of the gold for this process to be economic.

FLOTATION

A representative portion of the ore was ground 70 per cent solids with 4 pounds soda ash and 0·09 pound Aerofloat No. 25 until 75 per cent passed 200 mesh. After conditioning for 5 minutes with 0·10 pound sodium ethyl xanthate and 0·06 pound pine oil, a concentrate was removed which was cleaned once, yielding a concentrate and a middling.

Results:

Product	Weight, per cent	Assay, Au oz./ton	Gold, per cent
Heads (cal.)	100·0	0·40	100·0
Concentrate	14·3	2·40	85·7
Middling	22·4	0·20	11·2
Tailing	63·3	0·02	3·1
Rougher concentrate	36·7	1·057	96·9

This test shows that 96·9 per cent of the gold is recovered in the rougher concentrate. On cleaning, the grade is raised to 2·40 ounces gold per ton.

CYANIDATION

Sample portions of the ore were ground to various sizes and cyanided with a 1·0 pound KCN solution, 1 : 3 dilution. Sufficient lime to maintain protective alkalinity was added.

Results:

Mesh	Agitation, hours	Head assay, Au oz./ton	Tailing, assay Au oz./ton	Extraction, per cent	Reagent consumption, lb./ton	
					KCN	CaO
− 48	24	0·43	0·07	83·7	0·3	3·9
− 48	48	0·43	0·06	86·1	0·9	6·0
−100	24	0·43	0·027	93·7	0·6	7·9
−100	48	0·43	0·025	94·2	0·9	7·9
−150	24	0·43	0·021	95·1	0·3	3·8
−150	48	0·43	0·015	96·5	0·9	7·8
−200	24	0·43	0·015	96·5	2·1	19·3
−200	48	0·43	0·005	98·8	2·7	14·4

These results show that with increasing fineness of grinding, the recovery increases until at minus 200 mesh the ore, after a period of 48 hours' agitation, yields all but 10 cents of its contained gold.

SUMMARY AND CONCLUSIONS

The ore does not contain sufficient coarse gold to warrant the introduction of amalgamation as only 32·6 per cent can be recovered by this method from ore crushed to pass 48 mesh.

Flotation shows a recovery of 96·9 per cent of the gold in a rougher concentrate 36·7 per cent by weight of ore milled. On cleaning, the grade is raised to 2·40 ounces gold per ton.

Cyanidation of the ore ground minus 150 mesh gives a recovery of 96 per cent. This can be raised to over 98 per cent by still finer grinding.

As the ore does not present any difficulties to treatment by cyanide, flotation concentrates also should respond to similar treatment.

Report No. 430

EXPERIMENTAL TESTS ON SAMPLES OF CONCENTRATES FROM GRANADA GOLD MINES, LTD., ROUYN, QUE.

Shipment. A shipment of two grades of concentrates, labelled No. 1 and No. 2, consisting of 158 pounds of the former and 278 pounds of the latter, was received January 21, 1932. The samples were submitted by W. A. Gamble, Secretary-Treasurer, Granada Gold Mines, Limited, Federal Building, Toronto 2, Ontario.

Characteristics and Analysis of the Samples. Concentrate No. 1 represents amalgamation tailings from the primary blanket concentrate. Screen tests showed it to be 99·5 per cent minus 200 mesh. Concentrate No. 2 represents the secondary blanket concentrate and was much coarser than the primary. Both contained iron pyrite and arsenopyrite in a siliceous gangue.

Screen Test:

Secondary Concentrate

Mesh	Weight, per cent
+ 28	1·3
+ 35	3·4
+ 48	5·8
+ 65	9·5
+100	12·5
+150	11·5
+200	10·6
−200	45·4

The concentrates as received were quite wet, No. 1 containing 18·5 per cent water and No. 2, 10·5 per cent approximately. Samples were taken with a pipe sampler and analysis on the dry basis showed the following:—

Primary Concentrate		Secondary Concentrate	
Au	5·32 oz./ton	Au	2·64 oz./ton
Ag	2·68 "	Ag	0·90 "
Fe	18·36 per cent	Fe	15·80 per cent
As	10·50 "	As	7·90 "
S	12·75 "	S	10·25 "
Cu	0·01 "	Cu	trace
Pb	0·20 "	Pb	0·05 "
Zn	0·15 "	Zn	0·10 "

Purpose of Experimental Tests. The concentrates are being sold to a smelter but on account of the arsenic content the penalties are high. A method of treating them on the property and recovering the gold is desired. The work carried out consisted of amalgamation, cyanidation and flotation tests.

Summary of Results. Amalgamation tests were made on samples from each concentrate to determine if free gold were present. A negligible quantity was found in the primary concentrate, but from the secondary concentrate 59·8 per cent of the gold was recovered after grinding 90 to 95 per cent minus 200 mesh.

Results obtained by cyanidation were entirely satisfactory. Samples of the individual concentrates, as well as a composite sample of both, were treated by this process. In all cases the recoveries were in the neighbourhood of 98 to 99 per cent of the gold.

Flotation was not so successful, only 76·7 per cent of the gold being recovered in the concentrate.

Experimental Tests

AMALGAMATION

Tests Nos. 1 and 2

Amalgamation tests were made on samples of each of the concentrates. In each case 550 grammes of the wet concentrate, with 450 c.c. of water added, was amalgamated with 50 grammes of mercury. The primary concentrate was treated as received and the secondary concentrate was ground to 90 to 95 per cent minus 200 mesh. The tailings were filtered, washed, and assayed for gold.

Summary:

Test No.	Head sample, Au, oz./ton	Tailing, Au, oz./ton	Recovery, per cent
1	5·32	5·22	1·9
2	2·64	1·06	59·8

CYANIDATION

Tests Nos. 3, 4, 6, and 7

This series of tests was made on samples of the individual concentrates and on a sample of a mixture of the two. In Tests Nos. 3 and 4 the secondary concentrate was ground to 95 per cent minus 200 mesh and in Tests Nos. 6 and 7 it was treated as received. The primary concentrate was treated as received in all tests. The composite sample was made up of 58 per cent primary and 42 per cent secondary concentrates (dry weights). The periods of agitation were 24 hours in Test No. 3, 48 hours in Test No. 4, and 30 hours in Tests Nos. 6 and 7. Pulp dilution was approximately 3 : 1 in each test with the initial solution strength 3 pounds per ton KCN and lime added at the rate of 45 pounds per ton of concentrate. The tailings were filtered, washed, and assayed for gold.

In the following summary, Tests Nos. 3 and 4 are subdivided into parts (a) and (b). In each test part (a) refers to primary concentrate and part (b) to secondary concentrate.

Summary Tests Nos. 3a and 3b: 24-hour agitation periods.

Test No.	Head sample, Au, oz./ton	Tailing, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
				KCN	CaO
3a	5·32	0·050	99·06	8·0	41·5
3b	2·64	0·031	98·8	3·8	42·6

Screen Analysis of Cyanidation Tailing, Test No. 3b:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, Au, per cent	Average tailing, Au, oz./ton
+200	5·5	0·045	8·0	0·031
−200	94·5	0·030	92·0	

Summary Tests Nos. 4a and 4b: 48-hour agitation periods.

Sample	Head sample, Au, oz./ton	Tailing, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
				KCN	CaO
4a	5·32	0·03	99·4	10·4	46·6
4b	2·64	0·01	99·6	5·1	47·8

Summary Tests Nos. 6 and 7: 30-hour agitation periods.

Sample	Head sample, Au, oz./ton	Tailing, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
				KCN	CaO
6	*4·19	0·04	99·04	5·3	43·1
7	2·64	0·05	98·1	2·1	43·9

*Assay composite head sample calculated.

FLOTATION

Test No. 5

A flotation test was made on a sample of the secondary concentrate as received. The charge to the flotation machine was as follows:—

Concentrate	1,100	grammes (wet weight)
Na_2CO_3	20	lb./ton
Aerofloat No. 25	0·14	"
Potassium xanthate	0·05	"

The flotation concentrate and tailing were filtered, washed, and assayed for gold.

Summary Test No. 5

Product	Weight, per cent	Assay, Au, oz./ton	Recovery, per cent
Head (cal.)	100·0	2·40	100·0
Concentrate	19·0	9·70	76·7
Tailing	81·0	0·69	23·3

Screen Analysis of Flotation Tailing

Mesh	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Average tailing, Au, oz./ton
+200	46·5	0·96	64·95	0·69
−200	53·5	0·45	35·05	

CONCLUSIONS

The foregoing tests show that the concentrates can be treated best by cyanidation. As a good recovery is made, and consumption of cyanide and lime is not excessive, there is no reason for introducing other operations. While the tonnage to be treated daily is small and its gold content high, it would not be economical to concentrate or roast, as the extra cost would outweigh any saving resulting from the smaller agitation plant necessary or from the reduced fouling of the solution.

Grinding the secondary concentrate improves the extraction as can be seen by comparing the results of Tests Nos. 3b and 7. Extraction can be further improved by lengthening the period of agitation as shown by a comparison of Tests Nos. 3b and 4b. The economical limit to which these two operations may be carried will be governed largely by local conditions.

Report No. 431

CONCENTRATION OF SCHEELITE ORE FROM INDIAN PATH MINES, LUNENBURG, NOVA SCOTIA

Shipment. A shipment of 430 pounds of tungsten ore was received January 28, 1932. The shipment is said to represent the ore from the property of the Indian Path Mines, Ltd., situated near Riverport, Lunenburg County, Nova Scotia. The shipment was made by E. Hart Nicholls, President, Indian Path Mines, Ltd., St. Paul Building, Halifax, Nova Scotia.

Characteristics and Analysis. The tungsten occurs as the mineral scheelite ($CaWO_4$). It is associated with arsenopyrite, pyrite, and pyrrhotite in a gangue consisting mainly of quartz vein matter. Some slate was present in the sample but it is likely from the sides of the vein.

The analysis of the ore is as follows:—

WO_3	3·95 per cent
Gold	0·01 oz./ton

Purpose of Shipment. The market requirements for tungsten ores demand a product containing upwards of 65 per cent WO_3 and not more than 0·5 per cent of each of the following impurities: tin, copper, phosphorus, arsenic, and sulphur—the penalty limit being 0·05 per cent of each.

It was, therefore, required to determine the most suitable method of concentrating the ore to produce the desired grade of concentrate.

Summary of Experimental Tests. It was necessary to grind the ore to 10 mesh in order to free the scheelite.

Two tests were made on the sample. The first test consisted of grinding to 10 mesh and classifying the ground ore into four products in a hydraulic classifier. Each product was then treated separately on a laboratory-size Wilfley table. The results were very encouraging, over 90 per cent of the tungsten was recovered in a rough concentrate assaying 48·7 per cent WO_3. The quantity of ore used in this test was only 25 pounds, and consequently the amount of concentrate produced was too small to conduct further tests for determining whether this low-grade concentrate could be cleaned to a readily marketable grade. A second test was, therefore, made on the remainder of the shipment amounting to about 340 pounds. The ore crushed to 10 mesh was roughed over a large standard Wilfley table, without any previous sizing or classification. The tailing produced by this operation, as would be expected, was not so low as that obtained in the first test; the grade of concentrate, however, was virtually the same.

About 23 pounds of concentrate was obtained from the two tests and this was sufficient to carry on the further tests necessary to determine the grade and purity of the scheelite concentrate which could be expected to be produced from the ore. The concentrate was sized and re-cleaned by concentrating on a small Wilfley table, and a concentrate obtained which ran 64·8 per cent WO_3. Unfortunately, this concentrate contained both sulphur and arsenic in prohibitive quantities. It was, therefore, necessary to carry out a further series of experimental tests, which showed that it would be necessary either to dead-roast the table concentrate, to eliminate the arsenic and sulphur, or to give them a slight roast to make the arsenopyrite and pyrite magnetic, then by passing the roasted product through a high-intensity magnetic separator to extract these minerals. Dead roasting produced a final concentrate analysing WO_3, 60·53 per cent; sulphur, 0·11 per cent; arsenic, 0·05 per cent. Magnetic roasting and magnetic separation gave a final concentrate analysing WO_3, 68·9 per cent; sulphur, 0·11 per cent; arsenic, 0·05 per cent.

Experimental Tests

Test No. 1

Twenty-five pounds of ore was crushed to pass 10 mesh and classified by hydraulic classification into the following products:—

Products	Weight, per cent	WO_3, per cent	Distribution, WO_3, per cent
Feed	100·0	3·95	100·0
Product No. 1	32·4	5·68	46·6
Product No. 2	17·0	2·57	11·0
Product No. 3	15·0	3·34	13·3
Product No. 4	25·5	2·96	19·1
Loss	9·4	4·2	9·9

Each of the above products was concentrated on a small, laboratory, Wilfley table with the following results:—

Product	Weight, per cent	WO_3, per cent	Distribution, WO_3, per cent
Heads	100·0	3·95	100·0
Combined concentrates	6·0	48·7	90·7
Combined tailing	94·0	0·32	9·3

Test No. 2

About 338·5 pounds, the remainder of the shipment, was crushed to 10 mesh and tabled on a large Wilfley table.

Product	Weight, per cent	WO_3, per cent	Distribution, WO_3, per cent
Heads	100·0	3·95	100·0
Concentrate	6·9	49·48	86·8
Tailing	93·1	0·56	13·2

The concentrate analysed—Arsenic, 0·47 per cent; sulphur, 7·44 per cent.

The concentrate was sized as follows:—

Product	Weight, per cent	WO_3, per cent
+48 mesh	53·6	50·7
−48+65 mesh	17·1	42·8
−65 mesh	29·3	53·6
Totals	100·0	49·5

Each of the above sizes was tabled separately with the following results:—

Product	Weight, per cent	WO_3, per cent	Distribution, WO_3, per cent
Heads	100·0	49·5	100·0
Concentrate	70·2	64·8	90·8
Middling	16·2	23·7	7·7
Tailing	13·6	5·5	1·5

In mill practice the middling and probably also the tailing from the re-cleaning of this concentrate would be returned to the head of the mill for re-treatment.

The concentrate in the above test was found still to contain arsenic and sulphur in prohibitive amounts, namely, 0·24 per cent arsenic and 2·08 per cent sulphur.

It was, therefore, necessary to eliminate these two elements. Dead roasting was first tried. The results obtained were as follows: 500 grammes of the concentrate containing 64·8 per cent WO_3 was roasted until all sulphur and arsenic fumes ceased to be given off.

Analysis of Dead-roasted Concentrate

WO_3	60·53 per cent
As	0·05 "
S	0·11 "

A satisfactory elimination of both the arsenic and sulphur was obtained but the grade of the concentrate, namely 60·5 per cent WO_3, is not very satisfactory.

A magnetic roasting was then tried, which consists of driving off only the first atom or part of the sulphur, thereby converting the sulphides into products which can be extracted by magnetic separation.

Five hundred grammes of concentrate was given a slight roast and then passed over a high-intensity magnetic separator. The results are given in the following table:—

Products	Weights		Analyses			Units			Distribution		
	Gms.	Per cent	WO_3, Per cent	S, Per cent	As, Per cent	WO_3, Per cent	S, Per cent	As, Per cent	WO_3, Per cent	S, Per cent	As, Per cent
Magnetic product	131·1	27·3	26·82	7·3	0·65	7·3	2·00	0·20	12·7	96·2	83·3
Non-magnetic product	348·3	72·7	68·90	0·11	0·05	50·1	0·08	0·04	87·3	3·8	16·7
Totals	479·4	100·0	57·4	2·08	0·24	57·4	2·08	0·24	100·0	100·0	100·0

The grade of the above concentrate, namely 68·9 per cent WO_3, is excellent and the elimination of both the arsenic and sulphur can be considered as satisfactory.

Comparing this method of roasting and magnetic separation with that of dead roasting, it will be observed that the elimination of the arsenic and sulphur is the same in each case. The magnetic roast method gives a much higher grade of concentrate due to the iron being eliminated by the magnetic separation.

The magnetic product contains much tungsten. This product in mill practice would be returned to the mill for re-treatment with ore.

SUMMARY AND RECOMMENDATIONS

The only difficulty presented in the treatment of this ore is the elimination of the arsenic and sulphur from the concentrate. The method recommended to accomplish this is a magnetic roast followed by magnetic separation in high-intensity type machines.

The recovery of the tungsten should be over 80 per cent and the grade of concentrate averaging over 65 and close to 70 per cent WO_3.

The accompanying flow-sheet is suggested in principle as the best method of milling the ore; minor changes no doubt will be necessary to suit the initial capacity decided on for the mill.

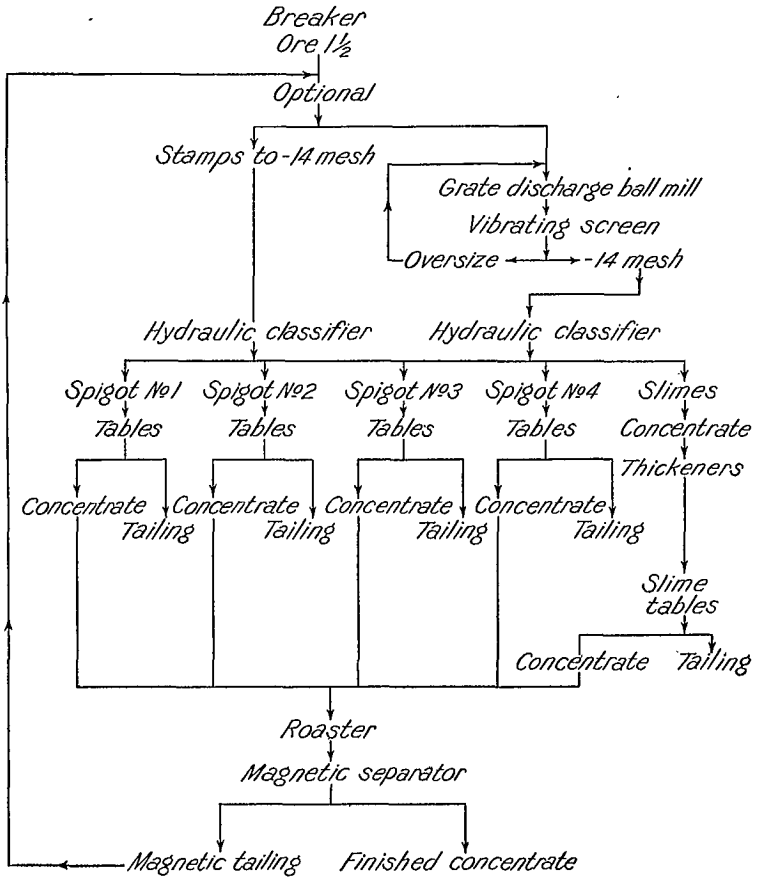

Figure 1. Flow-sheet for treatment of scheelite ore from Indian Path Mines, Ltd., Nova Scotia.

Report No. 432

THE RECOVERY OF GOLD FROM THE ORE OF THE ST. GERMAINE-GALE PROPERTY, DUBUISSON AREA, QUE.

Shipment. A shipment of 3,150 pounds of ore was received on March 18, 1932, from the St. Germaine-Gale property, Dubuisson area, about 2 miles from the Siscoe Gold Mines, Quebec. This sample was sent by W. A. Gamble, Granada Gold Mines, Ltd., 1,109 Federal Building, Toronto 2, Ontario.

Characteristics of the Ore. The shipment consisted of a quartz and carbonate gangue carrying sulphides of iron, principally iron pyrite. No coarse free gold was visible.

Purpose of Experimental Tests. It was required to determine the gold content of the sample and also to establish the most economical method for the recovery of the gold contained. The test work showed that while no free gold coarser than 100 mesh was detected, 81 per cent of the metal is recovered by amalgamation of minus 48-mesh material. Flotation of the tailing after amalgamation recovered an additional 12 per cent. Straight cyanidation at minus 100-mesh grinding gave an extraction of 96·9 per cent of the gold.

Sampling and Analysis. The entire lot was crushed to $\frac{1}{2}$ inch and $\frac{1}{10}$ of the weight cut out by an automatic sampler. This portion was stage-crushed and ground to succeeding finer sizes with cuts through a Jones riffle sampler until a representative portion minus 100 mesh was obtained for assay. The ore assayed 0·326 ounce gold and 0·06 ounce silver per ton.

Experimental Tests

The investigation included amalgamation, flotation, and cyanidation at various degrees of grinding.

AMALGAMATION

Representative portions of the ore were ground dry to pass 48 and 100 mesh, and amalgamated.

Mesh	Head assay, Au, oz./ton	Amalgamation tailing, Au, oz./ton	Recovery, per cent
− 48	0·326	0·06	81·6
−100	0·326	0·055	83·1

Amalgamation does not recover all the gold in the sample as $1.20 is left in the tailing.

AMALGAMATION AND FLOTATION

A representative portion of the ore was ground wet 70 per cent solids and amalgamated.

Mesh	Per cent
− 48+ 65	1·2
− 65+100	11·5
−100+150	18·0
−150+200	5·7
−200	63·6
	100·0

After removing amalgam, the pulp was floated with 4 pounds soda ash, 0·10 pound sodium ethyl xanthate, 0·06 pound Aerofloat No. 25, and 0·06 pound pine oil per ton. The concentrate was cleaned producing a concentrate and middling.

Mesh	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Heads	100·0	0·326	100·0
Amalgamation tailing (cal.)	.....	0·059	82·0*
Flotation concentrate	3·11	0·87	8·3
Flotation middling	3·22	0·40	4·0
Flotation tailing	93·67	0·02	5·7

*Recovered by amalgamation.

These results show that after removing 82 per cent of the gold by amalgamation, an additional 12·3 per cent is recovered in a rougher concentrate which is 6·3 per cent of the weight of ore milled. This concentrate is cleaned to a product worth $17.40 a ton.

CYANIDING

Sample lots of the ore were ground to various degrees of fineness and agitated for 48 hours with a 1·0 pound cyanide solution, 11 pounds of lime was added to each test to provide sufficient protective alkalinity.

Results:

Mesh	Agitation, hours	Assay, Au, oz./ton		Extraction, per cent	Reagent consumption, lb./ton	
		Heads	Cyanide tails		KCN	CaO
− 48	24	0·326	0·021	93·6	0·9	9·2
− 48	48	0·326	0·015	95·4	1·2	9·7
−100	24	0·326	0·015	95·4	1·2	9·7
−100	48	0·326	0·01	96·9	1·5	10·1
−150	24	0·326	0·021	93·6	1·2	9·5
−150	48	0·326	0·01	96·9	1·5	9·9
−200	24	0·326	0·02	93·9	1·2	9·9
−200	48	0·326	0·01	96·9	1·5	10·7

These results show that 95 per cent of the gold is extracted from material ground to pass 48 mesh, with the maximum recovery of 96·9 per cent on minus 100-mesh product.

SUMMARY AND CONCLUSIONS

Approximately 82 per cent of the gold can be recovered by amalgamation from material ground to pass 48 mesh. Flotation of the residue from this operation recovers an additional 12 per cent of the gold.

Cyanidation of the ore ground to the same degree of fineness recovers 95 per cent of the gold, which can be raised to 97 per cent by finer grinding.

Straight flotation unless preceded by amalgamation has a tendency to loss of free gold in the tailing as shown by a test not reported. Erratic results in the assays of the tailing indicate free gold. This gold would be caught by passing the tailing over corduroy-covered inclined plates.

As the raw ore cyanides without difficulty, it is probable that the flotation concentrate would also.

In the event of coarse gold being present in the ore it will prove effective to install amalgamation plates or corduroy blankets ahead of the main treatment plant, whether flotation coupled with cyanidation, or straight cyanidation be used. Which of these two methods to adopt will depend upon a detailed study of the economics of each.

Report No. 433

RECOVERY OF PYRITE FROM COAL WASHERY WASTE OF THE DOMINION COAL COMPANY, SYDNEY, N.S.

Shipment. A shipment of 2 tons of coal washery waste was received March 15, 1932, from the Dominion Coal Company, Sydney, Nova Scotia.

Characteristics and Analysis. The material consisted of coal washery waste and contained a high content of sulphur in the form of pyrite. The greater part of the pyrite was attached to the coal along the cleavage planes of the coal and the bedding of the slate.

The average analysis of the shipment was about 11·0 per cent sulphur.

Purpose of Experimental Work. This shipment was made for the purpose of determining whether a sulphide concentrate could be produced which would be suitable for the production of sulphuric acid by the Freeman process of flash roasting.

Experimental Tests

The following large-scale continuous tests were made. The feed rate was about 400 pounds per hour.

Test No. 1

The material was fed to a small ball mill in closed circuit with a Callow belt screen. The undersize, minus 65-mesh product, was fed to a Wilfley slime table. The results were as follows:—

Product	Weight, per cent	Sulphur, per cent	Insoluble, per cent
Feed to table	100·00	8·10	10·27
Concentrate	15·84	24·84	15·29
Tailing	84·16	5·02	8·99

Recovery of sulphur—48·2 per cent.

Test No. 2

The grinding circuit was the same as in Test No. 1. The minus 65 mesh discharge from the screen was fed to a mechanical flotation cell, where a combined float of coal and pyrite was made. The flotation concentrate was then tabled on a Wilfley slime table. The results follow:—

—	Sulphur, per cent	Insoluble, per cent
Feed to flotation	11·76	10·27
Flotation concentrate	11·11	7·83
Flotation tailing	9·44	31·87
Table concentrate	40·15	7·90
Table tailing	5·39	7·87

It is evident from the results that this test was not successful.

Tests Nos. 5 and 6

Two other tests were run preliminary to Tests Nos. 5 and 6, the same flow-sheet being used in all four tests. The flow-sheet was as follows: The coal was ground to minus 65 mesh as in the previous tests, but instead of endeavouring to float both the coal and sulphide together an attempt was made to float a coal product containing as little sulphur as possible. The tailing from the coal float was then floated in a second machine for the recovery of the pyrite. The coal concentrate was tabled to recover as much as possible of the sulphur floating with it.

Test No. 5

Product	Weight, per cent	Analysis		Units, S	Distribution, S, per cent
		S, per cent	Insoluble, per cent		
Feed to flotation	100·0	11·76	9·80	117·6	100·0
Coal flotation concentrate	26·5	5·3	5·08	14·1	12·0
" " tailing	73·5	14·06	12·36	103·3	87·8
(1) Pyrite flotation concentrate	35·7	20·03	6·37	71·5	60·8
" " tailing	37·8	8·41	27·23	31·8	27·0
(2) Table concentrate	0·02	35·47	5·58	0·01	0·1
Table tailing	26·48	5·28	5·37	14·0	11·9
Total concentrate 1 and 2	35·72	20·0	6·3	71·6	60·9

Test No. 6

Product	Weight, per cent	Analysis		Units, S	Distribution, S, per cent
		S, per cent	Insoluble, per cent		
Feed to flotation	100·0	12·68	11·41	126·8	100·0
Coal flotation concentrate	50·6	8·58	6·68	43·4	34·2
" " tailing	49·4	16·89	21·91	83·4	65·8
(1) Pyrite flotation concentrate	14·03	39·00	4·9	54·7	43·1
" " tailing	35·37	8·11	30·0	28·7	22·6
(2) Table concentrate	3·04	41·1	5·32	12·5	9·9
Table tailing	47·56	6·5	7·74	30·9	24·4
Total concentrate 1 and 2	17·07	39·4	5·0	67·2	53·0

DISCUSSION OF TESTS

The best method of treating this product for the recovery of the pyrite is undoubtedly the method used in Tests Nos. 3, 4, 5 and 6. The best results are represented in Test No. 6, but unfortunately even these are not very good. The low recovery of sulphur, namely only 53 per cent, is attributed to the high dilution of the pulp going to the machines for the flotation of the pyrite. The density of the pulp going to the coal flotation machines was 1 : 2·3, but after taking out 50·6 per cent of the weight of solids in the pulp as a coal float, the pulp discharging from the coal flotation machine and going to the pyrite flotation had a density of only 1 : 5. This pulp should have been thickened before attempting to float the pyrite, but as no thickening apparatus was available this could not be done. It is always difficult to obtain a good recovery by flotation on such a thin pulp, particularly on pyrite. It is quite safe to assume that in a 1 : 2·5 pulp the sulphur in the flotation tailing could be reduced to 2·5 per cent and even lower. This would bring the overall recovery of sulphur up to 65 per cent which is a safe and conservative figure to assume as the minimum recovery on this material.

The recovery by flotation of the coal could, without doubt, also be improved in practice. The amount of material for the tests was not great and, therefore, only a few reagents could be tried. There are also reagents which might be added to keep the pyrite from floating with the coal. In the test work as conducted, owing to difficulty in floating the pyrite in such a dilute pulp, it was not considered advisable to use any of these reagents as their effect would be carried through to the pyrite flotation and make it still more difficult to obtain a low-sulphur tailing.

An analysis was made of the pyrite concentrate.

Ash	59·9 per cent
Carbon	13·4 "
Hydrogen	1·1 "
Sulphur	38·9 "
(a) Total mineral matter	85·3 per cent
Coal by difference	14·7 "
Total	100·0
(b) Total carbon in concentrates uncorrected for C from carbonates	13·4 per cent
Hydrogen	1·1 "
Nitrogen and oxygen	0·5 to 1 "
Estimated ash-free coal	15·0 "

TREATMENT RECOMMENDED

The flow-sheet suggested for the recovery and separation of the pyrite is as follows:—

The material as received can be fed directly to a ball mill in closed circuit with a Dorr classifier. The Dorr classifier overflow should then pass over a Callow belt-type of screen with between 48- and 65-mesh openings. The screen is to take out coarse clean coal which will float over the classifier without being ground to the same size as the pyrite and slate. The fines from the screen then should go to a flotation unit for the flotation of the coal. The coal concentrate should be tabled to recover part of the sulphur floated with it. The tailing from this first float containing the bulk of the sulphur should then go to a Dorr thickener and be thickened to 35 per cent solid before the flotation of the pyrite is attempted. The thickener underflow then goes to the pyrite flotation machines.

Report No. 434

EXPERIMENTAL TESTS ON GOLD ORE FROM McKELLAR-LONGWORTH PROPERTY, SCHREIBER, ONTARIO

Shipment. A shipment of three sacks of ore, total weight 220 pounds, was received April 14, 1932, from R. N. Palmer, Schreiber, Ontario.

Characteristics and Analysis of the Sample. The sample submitted was a straight quartz ore carrying much free gold and a small amount of disseminated pyrite. The sample assayed gold, 2·76 ounces per ton.

Purpose of Experimental Tests. The work was done to find a simple process for recovering the gold from the ore. As not more than ten tons daily will be available for milling, the plant must be simple and compact. This, together with the fact that nearly all the gold is coarse and free, at once eliminates cyanidation as a possible method.

Summary of Results. Tests by flotation alone were not satisfactory as free gold adhered to the walls of the flotation machine and was unaccounted for in the products. Test No. 1, with the ore ground through 48 mesh, gave a tailing assaying gold, 0·55 ounce per ton, and a concentrate assaying gold, 13·45 ounces per ton. The assay of the head sample calculated from these two products was gold, 1·53 ounces per ton.

Test No. 3, with the ore ground through 100 mesh, gave a tailing assaying gold, 0·37 ounce per ton, and a concentrate assaying gold, 5·24 ounces per ton. The assay of the head sample calculated from these two products was gold, 1·47 ounces per ton.

Amalgamation gave better results. With the ore ground through 48 mesh a tailing assaying gold 0·295 ounce per ton was produced, representing a recovery of 89·31 per cent. A similar test with the ore ground through 100 mesh produced a tailing assaying gold 0·185 ounce per ton, representing a recovery of 93·30 per cent. By floating the amalgamation tailing the recoveries were increased to 95·60 per cent and 96·33 per cent respectively.

In the tests in the milling unit a recovery of 92·0 per cent was obtained by amalgamating and blanket concentrating the amalgamation tailing. Flotation of the blanket tailing increased the recovery to 96·0 per cent, leaving a final tailing which assayed gold, 0·06 ounce per ton.

Experimental Tests

FLOTATION

Tests Nos. 1 and 3

In Test No. 1, 1,000 grammes of ore at minus 14 mesh was ground for 15 minutes in a small jar mill. The following reagents were added:—

To ball mill:

Na_2CO_3	3·0 lb./ton
Aerofloat No. 25	0·07 "

To flotation machine:

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

The flotation concentrate and tailing were washed and assayed for gold. The ratio of concentration was 13·7 : 1. A screen test of the flotation tailing showed the following:—

Mesh	Weight, per cent
+ 35	0·1
+ 48	1·2
+ 65	8·4
+100	21·2
+150	17·1
+200	10·8
−200	41·2

Summary Test No. 1:

Product	Weight, Au, oz./ton	Assay, per cent	Recovery, per cent
Concentrate	7·6	13·45	66·8
Tailing	92·4	0·55	33·2
Head (cal.)	100·0	1·53	

In Test No. 3, 1,000 grammes of the ore at minus 14 mesh was ground for 30 minutes in a jar mill. The following reagents were added:—

To ball mill:

Na_2CO_3	5·0 lb./ton
Aerofloat No. 25	0·07 "

To flotation machine:

Sodium ethyl xanthate	0·10 lb./ton
Pine oil	0·10 "

The ratio of concentration was much lower than in Test No. 1, 4·4 : 1, no doubt due to the extra pine oil used. The concentrate and tailing were washed and assayed for gold. A screen test on the flotation tailing showed the following:—

Mesh	Weight, per cent
+ 65	0·3
+100	3·5
+150	17·0
+200	16·4
−200	62·8

Summary Test No. 3:

Product	Weight, per cent	Assay, Au, oz./ton	Recovery, per cent
Concentrate	22·5	5·24	80·4
Tailing	77·5	0·37	19·6
Head (cal.)	100·0	1·47	

AMALGAMATION

Tests Nos. 2 and 4

In these two tests 500-gramme lots of the ore at minus 14 mesh were ground in a jar mill for 15 and 30 minutes, giving minus 48-mesh and minus 100-mesh products respectively. The pulp was filtered and repulped to 1 : 1 density and amalgamated with 50 grammes of mercury for 30 minutes. The tailings were filtered, washed, and assayed for gold.

Head sample: Au, 2·76 oz./ton.

Test No.	Tailing, Au. oz./ton	Recovery, per cent
2	0·295	89·31
4	0·185	93·30

Tests Nos. 5 and 6

In these tests two lots of ore at minus 14 mesh were ground for 15 and 30 minutes and amalgamated with mercury as was done in Tests Nos. 2 and 4. The amalgamation tailings were floated and the products assayed for gold.

Summary Test No. 5:—Head to amalgamation: Au, 2·76 oz./ton.

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Flotation concentrate	6·5	4·36	70·0
Flotation tailing	93·5	0·13	30·0
Head to flotation, or amalgamation tailing (cal.)	100·0	0·405	100·0

Recovery by amalgamation	85·33 per cent
Recovery by flotation	10·27 "
Recovery total	95·60 "

Summary Test No. 6:—Head to amalgamation: Au, 2·76 oz./ton.

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Flotation concentrate	6·96	1·72	53·9
Flotation tailing	93·04	0·11	46·1
Heads to flotation, or amalgamation tailing (cal.)	100·0	0·22	100·0

Recovery by amalgamation	92·03 per cent
Recovery by flotation	4·30 "
Recovery total	96·33 "

Test on Milling Unit

The ore, ground dry through 14 mesh, was fed into a small rod mill in closed circuit with a classifier. The classifier overflow, with most of the free gold removed by mercury in the classifier well, was passed over an amalgamation plate and the amalgamation tailing was passed over a special corduroy blanket. Flotation tests were subsequently made on the blanket tailing. The feed rate was maintained at 12 pounds per hour and the classifier overflow contained approximately 15 per cent solids.

At the end of the run the rod mill and feed-box were cleaned out and the material ground in a jar mill and amalgamated with mercury. The tailing from this operation, rich in sulphide, was weighed and assayed for gold. It was considered as part of the tailing from the major operation and, although not actually united with it, the results for its weight and assay were used to calculate an average tailing for the test and from this the recovery.

Summary of Test on Milling Unit:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Tailing from major operation	99·27	0·169	75·11
Tailing from clean-up	0·73	7·617	24·89
Average tailing	100·00	0·223	100·00

Head sample: Au, 2·77 oz./ton.

Product	Weight, per cent	Assay, Au, oz./ton
Head	100·0	2·77
Average tailing	99·37	0·223
Blanket concentrate	0·63	2·16

Recovery by amalgamation	91·51 per cent
Recovery by blanketing	0·49 "
Recovery total	92·00 "

Flotation tests on the blanket tailing (tailing from mill clean-up not included) which carried about 6 per cent of the gold in the original feed showed a recovery of 67 per cent of this gold. This brings the total recovery up to 96·0 per cent, which checks closely the results obtained on small-scale tests.

CONCLUSIONS

The tests show that the ore is a simple one to treat. Amalgamation alone will recover more than 90 per cent of the gold, and concentration of the amalgamation tailing will bring this figure up to 96 per cent or better. The blanket concentrate, assaying gold 2·16 ounces per ton, after regrinding all through 150 mesh, will yield 60 per cent of its gold to amalgamation. It would, therefore, seem advisable to amalgamate and concentrate the

amalgamation tailing on blankets. The blanket concentrate could be reground in an amalgamation barrel. If desirable the blanket tailing could be floated and the flotation concentrate disposed of to a smelter.

Report No. 435A

FLOTATION OF SISCOE GOLD MINES CLASSIFIER OVERFLOW

Shipment. A shipment of 150 pounds of material, said to be washed and dried classifier overflow, was received April 20, 1932, from Mr. C. O. Stee of the Siscoe Gold Mines, Ltd., Siscoe, Quebec.

Flotation tests were desired in order to determine whether concentration of the gold was possible in a product suitable for cyanidation or shipment to a reduction plant.

On sampling, the shipment was found to contain 0·14 ounce gold and 0·10 ounce silver per ton.

Experimental Tests

Two flotation tests were made, the first on the material as received and the second after washing to remove lime.

A sizing test was made on the shipment which shows the following:—

Mesh	Weight, per cent
+ 48	7·9
− 48+ 65	13·7
− 65+100	18·9
−100+150	12·8
−150+200	6·5
−200	40·2

FLOTATION

Test No. 1

A sample of the material was conditioned for 5 minutes in a 1 : 1 pulp with 5 pounds soda ash per ton; 0·5 pound copper sulphate was then added and after three minutes 0·10 pound sodium xanthate, and 0·05 pound Aerofloat No. 25 were added. Pine oil, 0·05 pound per ton, was required to produce a froth.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Heads (calculated)	100·00	0·138	100·0
Concentrate	1·36	9·07	89·3
Tailing	98·64	0·015	10·7

Ratio of concentration, 73·6 : 1.

Flotation conditions were very poor. The lime left in the material from previous treatment produced a thick gummy froth which would give trouble in dewatering thickeners.

Test No. 2

A sample of the material was water-washed on a filter until most of the alkalinity was removed. This required an excessive amount of water.

A flotation concentrate was then removed following the same procedure as in Test No. 1.

Results:

Product	Weight, per cent	Assay, Au.oz./ton	Gold, per cent
Heads (calculated)	100·00	0·152	100·0
Concentrate	0·97	14·16	90·2
Tailing	99·03	0·015	9·8

Ratio of concentration, 104·7 : 1.

This test gives the same tailing of 30 cents as secured in the previous one but with a higher grade of concentrate.

A screen analysis of the flotation tailing shows:—

Mesh	Weight, per cent	Assay, Au, oz./ton
+ 48	7·1	0·10
− 48+ 65	12·6	0·01
− 65+100	20·9	0·01
−100+150	11·6	0·025
−150+200	9·2	0·015
−200	38·6	0·01

The results show that the gold is freed at minus 48-mesh grinding. The higher values of the plus 150- and 200-mesh portions probably indicate fine gold not caught in the flotation circuit. As a safeguard, the flotation tailing could be passed over corduroy blankets to trap any gold remaining.

CONCLUSIONS

Flotation of the ore which had not been ground with lime and cyanide will yield better results than those indicated in the above tests. Lime and cyanide are both depressants of gold and pyrites. Best results will be secured when the ore has not been in contact with these reagents prior to flotation.

Report No. 435B

FLOTATION CONCENTRATION OF SISCOE GOLD ORE

Shipment. On July 13, 1932, 4,090 pounds of ore was received from the Siscoe Gold Mines, Limited, Siscoe, Quebec. The shipment was said to be run-of-mine ore taken over a period of two weeks from the feed belt leading to the ball mills.

Characteristics of the Ore. The material consisted of white quartz and dark-coloured diorite. Sulphides of iron and copper were present in the quartz.

Purpose of Experimental Tests. The object of this investigation was to study a replica of the grinding and blanket concentration practice now in use at the Siscoe mill. Instead of treatment by cyanidation, the blanket tailing was concentrated by flotation and the main bulk of the feed discarded as a flotation tailing. Concentrates caught in the ball mill trap and on the blankets were amalgamated and the tailings from these operations added to the flotation concentrate. This mixture was then cyanided.

The test work showed that 99·6 per cent of the gold can be concentrated to a ratio of 10 : 1 and that 99·4 per cent of the gold in the concentrate can be extracted by cyanidation.

Sampling and Analysis. The entire lot was crushed to 14 mesh and a sample cut out by an automatic sampler. This portion was stage-crushed to succeeding finer sizes with intervening passes through a Jones riffle sampler until a representative portion minus 100 mesh was secured for assay. All metallics appearing on the screens were calculated in the assay. This showed the shipment to contain 1·15 ounces gold per ton.

In the following tests, the ore was fed at the rate of 100 pounds per hour to a small rod mill. The mill discharged into a trap with a revolving stirrer, thence into an Aikens classifier in closed circuit with the grinding mill. The classifier overflow passed over a table $6\frac{1}{2}$ feet long by $6\frac{1}{2}$ inches wide, covered with special corduroy cloth. After leaving this table, the pulp passed through a conditioning tank to the first cell of a six-cell flotation machine. The rougher concentrate from the last five cells was returned with the feed to the first cell where a finished concentrate was taken off. Reagents were added as indicated in each test.

A sample of the feed was taken from each run.

Mill Run No. 1:

Circuit: Neutral.

Reagents: To conditioning tank, sodium xanthate, 0·10 lb./ton.
To 3rd cell, pine oil, 0·08 lb./ton.

Classifier Overflow: 40 per cent solids.

Grinding:

+ 65	4·2 per cent
− 65+100	11·6 "
−100+150	15·8 "
−150+200	16·3 "
−200	52·1 "
	100·0

Slope of blankets	$3\frac{3}{4}$ inches in 12 inches
Blanket area	4·5 sq. ft.
Blankets changed	4·0-hour intervals.

Assays:	Au, oz./ton
Head sample	1·64
Classifier overflow	0·35
Blanket tailing	0·155
Flotation concentrate	2·84
Flotation tailing	0·005

Recovery in ball mill and trap	78·66 per cent
Recovery on blankets	11·39 "
Recovery in flotation concentrates	9·17 "
Total recovery	99·72 "
Ratio of flotation concentration	19 : 1

Mill Run No. 2:

The conditions were the same as in Run No. 1, with 0·05 pound xanthate and 0·05 pound Aerofloat No. 25 added to the conditioning tank. The classifier overflow was approximately 25 per cent solids which gave too high a velocity over the blankets.

Grinding: Classifier Overflow:	
+ 65	2·0 per cent
− 65+100	11·7 "
−100+150	16·0 "
−150+200	15·6 "
−200	54·7 "
	100·0 "

Assays:	Au, oz./ton
Head sample	1·26
Classifier overflow	0·35
Blanket tailing	0·33
Flotation concentrate	2·56
Flotation tailing	0·005

Recovery in ball mill and trap	72·22 per cent
Recovery on blankets	1·59 "
Recovery by flotation	25·85 "
Total recovery	99·66 "
Ratio of concentration by flotation	7·9:1

Aerofloat gave a voluminous froth and lowered the ratio of concentration.

Mill Run No. 3:

In this run, 0·5 pound soda ash per ton was added to the grinding mill; and 0·10 pound sodium xanthate, and 0·10 pound cresylic acid to the flotation unit.

Grinding: Classifier Overflow:	
+ 65	2·2 per cent
− 65+100	13·2 "
−100+150	12·6 "
−150+200	17·0 "
−200	55·0 "

Assays:	Au, oz./ton
Head sample	1·27
Classifier overflow	0·51
Blanket tailing	0·21
Flotation concentrate	2·08
Flotation tailing	0·005

Recovery in ball mill and trap	59·84 per cent
Recovery on blankets	23·63 "
Recovery by flotation	16·18 "
Total recovery	99·65 "
Ratio of flotation concentrate	10·1:1

Soda ash is of no apparent benefit.

Run No. 4:

The slope of the blanket table was changed from $3\frac{3}{4}$ inches to 2 inches in 12 inches. It was found necessary to maintain not over 25 to 30 per cent

solids over the blankets to keep the sands moving. The blankets were fully loaded and changed at 3-hour intervals. Reagents for flotation were 0·10 pound sodium xanthate and 0·15 pound cresylic acid per ton.

Grinding: Classifier Overflow:	
+ 65	14·7 per cent
− 65+100	17·9 "
−100+150	13·5 "
−150+200	11·9 "
−200	42·0 "

Assays:	Au, oz./ton
Head sample	0·965
Classifier overflow	0·70
Blanket tailing	0·115
Flotation concentrate	10·55
Flotation tailing	0·005

Recovery in ball mill and trap	27·46 per cent
Recovery on blankets	60·62 "
Recovery by flotation	11·40 "
Total recovery	99·48 "
Ratio of concentration by flotation	95·9:1

Run No. 5:

In this run no soda ash was used. The blankets were also omitted, and the classifier overflow passed to the conditioning tank where 0·10 pound xanthate and 0·08 pound pine oil per ton were added.

Grinding: Classifier Overflow:	
+ 65	2·7 per cent
− 65+100	15·2 "
−100+150	15·7 "
−150+200	15·5 "
−200	50·9 "

Assays:	Au, oz./ton
Head sample	0·86
Classifier overflow	0·33
Flotation concentrates	33·04
Flotation tailing	0·01

Recovery in ball mill and trap	61·62 per cent
Recovery by flotation	37·23 "
Total recovery	98·85 "
Ratio of flotation concentration	103·2:1

CYANIDATION OF CONCENTRATES

The daily clean-ups of the ball-mill discharge trap together with the accumulated blanket concentrates from the five runs were amalgamated without regrinding. After separating the amalgam and mercury in a hydraulic classifier, the residues were added to the accumulated flotation concentrates. These were thoroughly mixed and sampled showing the product to be cyanided contained 2·31 ounces gold per ton. A larger sample was cut out for regrinding. This contained 2·30 ounces gold per ton.

Representative portions of the ground and unground samples were agitated for 72 hours, 1 : 3 dilution with a 5-pound KCN solution; 10 pounds of lime was added for protective alkalinity.

The following screen tests show the nature of the material:—

Screen size	Unground	Reground
+ 65 mesh	1·3	0·2
− 65+100	6·1	1·2
−100+150	3·3	2·7
−150+200	13·3	12·1
−200	76·0	83·8

Results of Cyanide Tests

Unground concentrate						Reground concentrate					
Agitation time, hours	Heads Au, oz./ton	Tailing, Au, oz./ton	Extraction, %	Reagent consumption, lb./ton		Agitation, time, hours	Head, Au, oz./ton	Tailing, Au, oz./ton	Extraction, %	Reagent consumption, lb./ton	
				KCN	CaO					KCN	CaO
24	2·31	0·09	96·1	6·9	8·2	24	2·30	0·07	97·0	7·8	8·0
48	.....	.....	.....	.....	.....	48	2·30	0·08	96·5	8·1	8·2
72	2·31	0·065	97·2	11·4	8·3	72	2·30	0·015	99·4	9·3	8·2

These results indicate that the gold is readily soluble in cyanide solution. By lengthy contact, the residues can be reduced to 30 cents.

SUMMARY AND CONCLUSIONS

After removing coarse gold and part of the sulphides by trap and blankets no difficulty should be met with in reducing the gold in the residue to 10 cents a ton by flotation. This gives a ratio of concentration of approximately 10 : 1 with a flotation concentrate containing approximately 2·5 ounces gold per ton. By manipulation of the circuit the grade of concentrate can be raised to 10 ounces with a ratio of concentration of 95 : 1.

Good recoveries of 98 per cent can be secured by omitting the blankets before flotation. This raises the grade of the flotation concentrate to 33 ounces gold per ton with a concentration ratio of 103 : 1. The final tailing contains 20 cents per ton.

The ore does not require to be ground finer than 52 to 55 per cent minus 200 mesh. Soda ash is not necessary in the flotation circuit.

The investigation shows that constant results can be secured under a wide variation of milling practice.

The gold is readily soluble in cyanide solution. Any ratio of concentration from 10 : 1 to 100 : 1 can be obtained.

The blankets could be changed from their present position and placed after the flotation circuit. Any gold or sulphides passing the flotation cells would be entrapped here.

Blanket concentrate and trap clean-ups should be ground and amalgamated before adding to the cyanide circuit. A short regrind of the flotation concentrate will yield highest metallurgical recovery. Whether this practice will yield the highest financial return can only be determined by a study of the costs of operating a small regrind mill.

The tests show a difference of $1.00 per ton in the tailing after cyaniding unground and reground concentrates to which has been added the trap clean-up. This amounts to approximately 10 cents per ton of ore milled.

The most practical initial practice would be to aim for a ratio of concentration of 10 : 1, giving a flotation concentrate of about 2·5 ounces gold per ton. Any increase in tonnage milled can then be taken care of by increasing the ratio of concentration without lowering the overall recovery.

Report No. 436

THE RECOVERY OF GOLD FROM TREADWELL-YUKON ORE, BUSSIÈRE PROPERTY, LOUVICOURT TOWNSHIP, QUEBEC

Shipment. An 830-pound shipment of gold-copper ore was received June 9, 1932, from the Treadwell-Yukon Company, Ltd., per J. P. Norrie, Superintendent. The ore was said to have come from the company's Bussière property, Louvicourt Township, Quebec.

Characteristics of the Ore. The gangue is chiefly white quartz, carrying massive sulphides of iron and copper. No appreciable quantity of coarse free gold is visible.

Purpose of Experimental Tests. Standard methods of treatment were made to determine the amenability of the ore to such processes.

Sampling and Analysis. The entire lot was crushed to ¼ inch and quartered. One quarter was then stage-crushed and ground to the finer sizes with intervening passes through a Jones riffle sampler until a representative portion minus 100 mesh was obtained. Analysis showed the ore to contain 2·31 ounces gold, 0·38 ounce silver per ton, and 0·55 per cent copper.

Experimental Tests

The investigation included tests by cyanidation; amalgamation with cyanidation of the amalgamation tailing; flotation; and amalgamation followed by flotation.

It was found that straight cyanidation at 150-mesh grinding gave 99 per cent extraction of the gold. However, the copper caused an excessive consumption of cyanide.

Amalgamation and flotation gave 99·6 per cent recovery.

AMALGAMATION AND CYANIDATION

Test No. 1

A portion of the ore was ground dry to pass 48 mesh and amalgamated. After removing the amalgam, the tailing was agitated with a 3·0-pound KCN solution, 1 : 3 dilution for 48 hours, 10 pounds of lime per ton was added to maintain protective alkalinity.

Grinding:		
− 48+ 65 mesh		8·9 per cent
− 65+100 "		22·8 "
−100+150 "		11·2 "
−150+200 "		6·7 "
−200		50·4 "
		100·0
Results:		
Heads	Au	2·31 oz./ton
Amalgamation tailing	Au	0·47 "
Recovery		79·7 per cent
Cyanide tailing	Au	0·09 oz./ton
Recovery		16·4 per cent
Total recovery		96·1 "

Test No. 2

A similar test was made on a sample ground to pass 100 mesh.

Grinding:		
−100+150 mesh		7·7 per cent
−150+200 "		16·0 "
−200		76·3 "
		100·0
Results:		
Heads	Au	2·31 oz./ton
Amalgamation tailing	Au	0·40 "
Recovery		82·7 per cent
Cyanide tailing	Au	0·02 oz./ton
Recovery		16·4 per cent
Total recovery		99·1 "

CYANIDATION

Test No. 3

Representative portions of the ore ground to various sizes were agitated with a 1-pound KCN solution, 1 : 3 dilution, together with lime to supply protective alkalinity. It was found necessary to make further additions of cyanide during the agitation periods to maintain the strength of the cyanide.

Mesh	Agitation period, hours	Heads, Au, oz./ton	Cyanide tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton	
					KCN	CaO
− 48	24	2·31	0·55	76·2	5·1	16·1
− 48	48	2·31	0·07	97·0	5·7	14·7
−100	24	2·31	0·30	87·0	5·4	15·2
−100	48	2·31	0·02	99·1	6·3	15·5
−150	24	2·31	0·16	93·1	9·9	16·1
−150	48	2·31	0·015	99·3	8·4	16·7
−200	24	2·31	0·13	94·4	9·9	16·1
−200	48	2·31	0·015	99·3	8·4	16·7

These results show that maximum extraction is not obtained within 24 hours. Copper in the ore attacks cyanide causing excessive consumption.

The gold is readily soluble as indicated by the high recoveries on the finer sizes.

AMALGAMATION AND FLOTATION

Test No. 4

A sample of the ore was ground to pass 48 mesh with 66 per cent minus 200 mesh and amalgamated. After removing the amalgam, the tailing was conditioned 10 minutes with 5·0 pounds soda ash per ton, and floated with 0·12 pound Aerofloat No. 25 and 0·10 pound sodium xanthate. The rougher concentrate was cleaned.

Part of the flotation tailing was cyanided and another portion reground to 96 per cent minus 200 mesh and also cyanided.

Results:

Product	Weight, per cent	Assay			Metals, per cent		
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
Heads	100·00	0·55	2·31	0·38	100·0	100·0	100·0
Amalgamation tailing	100·00	.....	0·246	.....	.....	*89·3	
Flotation concentrate	10·36	4·53	2·16	0·72	90·2	9·7	19·6
" middling	2·96	0·85	0·56	0·26	4·8	0·7	
" tailing	86·68	0·03	0·03	.....	5·0	0·3	

*Recovered by amalgamation.

A 48-hour cyanide treatment reduces the gold in unground flotation tailing to 0·01 ounce gold with a cyanide consumption of 1·2 pounds per ton. The reground portion of the flotation tailing after 24 hours' agitation with cyanide solution was reduced to 0·005 ounce. This combination gives a total recovery of 99·8 per cent of the gold.

Other amalgamation and flotation tests using various combinations of reagents gave results similar to the above. Cyanide and lime gave a higher grade copper concentrate with more gold left in the flotation tailing.

One of these tests was made to see if the gold could be dropped out of the copper concentrate and either segregated in an iron pyrite concentrate or left in the tailing.

Test No. 5

The ore was ground to approximately 70 per cent minus 200 mesh and amalgamated. The tailing was then conditioned with soda ash and cyanide to depress iron pyrite and gold. A copper concentrate was removed with the addition of 0·10 pound sodium xanthate and 0·06 pound pine oil. Copper sulphate 1 pound per ton, 0·10 pound amyl xanthate, and 0·06 pound pine oil were then added and an iron pyrite concentrate removed.

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The flotation tailing was then cyanided for 48 hours with 3·0 pounds KCN solution, 1 : 3 dilution, 8 pounds lime per ton being added for protective alkalinity.

Product	Weight, per cent	Assay		Metals
		Cu, per cent	Au, oz./ton	Au, per cent
Heads	100·00	0·55	2·31	100·0
Amalgamation tailing	.....	.....	0·18	*92·1
Copper concentrate	4·15	14·52	2·56	4·6
Iron pyrite concentrate	10·21	0·19	0·32	1·4
Flotation tailing	85·64	.....	0·05	1·9

*Recovered by amalgamation.

Cyanidation of the flotation tailing reduces the gold from $1 to 20 cents, an extraction of 80·3 per cent of the value, a total recovery of 99·6 per cent by the combined processes.

Apparently most of the gold not recovered by amalgamation is associated with the copper minerals.

CONTINUOUS AMALGAMATION AND FLOTATION

Test No. 6

The ore was fed at the rate of 100 pounds per hour to a small rod mill in closed circuit with an Aikens classifier. Five pounds soda ash per ton was fed to the grinding mill. The rod mill discharge passed over a short amalgamating plate before entering the classifier. The classifier overflow was elevated and flowed over a second amalgamating plate before entering a conditioning tank where 0·10 pound Aerofloat No. 25, 0·10 pound sodium ethyl xanthate per ton were added. This tank discharged into a 6-cell flotation machine where a rougher concentrate was taken off. This was cleaned in a separate 3-cell machine. The cleaner tailing was returned to the roughing circuit.

Screen Analysis of Classifier Overflow

+ 65 mesh	0·3 per cent
− 65+100 "	5·2 "
−100+150 "	9·9 "
−150+200 "	13·7 "
−200	70·9 "

Product	Weight, per cent	Assay,		
		Cu, per cent	Au, oz./ton	Ag, oz./ton
Feed	100·00	0·55	2·31	0·38
Amalgamation tailing	.....	.....	0·54	
Final concentrate	12·18	4·24	4·36	1·32
Rougher "	.....	2·22	2·34	0·98
Cleaner tailing	.....	0·18	0·21	
Flotation "	87·82	0·01	0·01	

The final concentrate had the following analysis:—

Au	4·36 oz.	Ag	1·32 oz.
Cu	4·24 per cent	FeO	37·55 per cent
SiO_2	18·36 "	S	28·24 "

Recovery of gold by amalgamation	76·6 per cent
Recovery of gold by flotation	23·0 "
Total gold recovery	99·6 "
Recovery of silver by flotation	42·1 "
Ratio of concentration	8·2:1
Recovery of copper	94·5 per cent

A run similar to the above was made with the soda ash added to the conditioning tank. This was found to improve the results secured by amalgamation. The recovery of copper was lower, due no doubt to insufficient contact with the reagents, or to building up in the circuit.

Product	Weight, per cent	Assay		
		Cu, per cent	Au, oz./ton	Ag, oz./ton
Feed	.....	0·55	2·31	0·38
Amalgamation tailing	.....	.....	0·20	
Final concentrate	8·09	3·88	2·36	1·42
Rougher "	.....	2·00	1·50	1·26
Flotation tailing	91·91	nil	0·01	

The final concentrate had the following analysis:—

Au	2·36 oz.	Ag	1·42 oz.
Cu	3·88 per cent	FeO	41·0 per cent
SiO_2	16·8 "	S	30·5 "

Recovery of gold by amalgamation	91·3 per cent
Recovery of gold by flotation	8·3 "
Total gold recovery	99·6 "
Ratio of concentration	12·37:1
Recovery of copper	60·4 per cent

This recovery of 60·4 per cent of the copper from a test giving a tailing containing no copper is accounted for by the poor flotation condition experienced. The copper floated in the rougher cells but dropped out on cleaning. The lapsed time of the run was not sufficient to allow the circuit to build up enough to affect the tailing assay.

SUMMARY AND CONCLUSIONS

Copper present in the ore attacks the cyanide, resulting in excessive losses. Flotation removes this copper, leaving the ore in a condition where a normal cyanide consumption is noted. However, most of the gold is in the flotation concentrate, leaving little in a tailing to be cyanided.

By amalgamation of minus 48-mesh material 80 per cent of the gold can be recovered. When ground 70 per cent minus 200 mesh, 91 per cent can be recovered, with an additional 8 per cent in the copper concentrate of a grade suitable for shipment to a smelter.

On this grade of ore, recoveries of 99 per cent can be expected.

The process indicated is amalgamation and flotation. Copper in the ore renders the cyanide process impractical, resulting in high reagent costs and low-grade precipitate.

Report No. 437

EXPERIMENTAL TESTS ON A SAMPLE OF GOLD ORE FROM ALEXANDRIA GOLD MINES, LIMITED

Shipment. A shipment of five sacks of ore, net weight 335 pounds, was received March 24, 1932, from G. F. MacDonnell, Alexandria Gold Mines, Limited, Shoal Bay, via Vancouver, B.C.

Characteristics and Analysis of the Sample. The sample submitted contained stringers of pyrite occurring along the cleavage planes of a quartz gangue. An average head sample assay was gold, 0·25 ounce per ton.

Summary of Results. Amalgamation tests were made on four samples of the ore, ground dry through 48, 100, 150, and 200 mesh. No recovery at all was obtained by this method. Four cyanidation tests on similarly prepared samples gave recoveries of 56, 62, 68, and 70 per cent respectively. By flotation it was possible to recover 79 per cent of the gold in a concentrate assaying gold, 2·12 ounces per ton after grinding 55 per cent minus 200 mesh. By grinding 82 per cent minus 200 mesh it was possible to recover 86 per cent of the gold in a concentrate assaying gold, 2·08 ounces per ton.

Experimental Tests

AMALGAMATION

Tests Nos. 1 to 4

In these tests 500-gramme lots of ore, from each of the four lots ground dry through 48, 100, 150, and 200 mesh, were amalgamated for 30 minutes in 1 : 1 pulp. The tailing was filtered, washed, and assayed for gold.

Summary Tests:

Test No.	Tailing, Au, oz./ton	Recovery, per cent
1	0·29	nil
2	0·26	nil
3	0·26	nil
4	0·25	nil

CYANIDATION

Tests Nos. 1 to 4

The ore used in these tests was ground dry through the following sizes: 48, 100, 150, and 200 mesh. In each test 200 grammes of ore was treated with 600 c.c. solution running 3 pounds per ton KCN. The tailing was filtered, washed, and assayed for gold. The period of agitation was 48 hours in each test.

Summary Tests Nos. 1 to 4: Head sample: Au, 0·25 oz./ton.

Test No.	Mesh	Tailing, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
				KCN	CaO
1	− 48	0·11	56·0	0·3	5·5
2	−100	0·095	62·0	0·3	5·5
3	−150	0·08	68·0	0·3	6·2
4	−200	0·075	70·0	0·3	6·2

CYANIDATION

Test No. 5

This test was made on a composite sample made up of 100 grammes each of flotation concentrates Nos. 1 and 2. The pulp was agitated for 48 hours in 3 : 1 dilution with KCN 3 pounds per ton. The tailing was filtered, washed, and assayed for gold. Considerable frothing was noticed during the agitation.

Summary: Head sample: Au, 2·08 oz./ton.

Test No.	Tailing, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
5	1·04	50·0	2·10	7·65

Screen Analysis:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Average tailing, Au, oz./ton
+200	25·22	1·66	40·3	1·04
−200	74·78	0·83	59·7	

FLOTATION

Tests Nos. 1 to 4

In Test No. 1, 2,000 grammes of the ore at minus 14 mesh was ground in 1 : 1 pulp for 30 minutes in a small rod mill.

Reagents to rod mill:

Na_2CO_3	2·0 lb./ton
Aerofloat No. 25	0·07 "

The pulp was then transferred to a flotation machine and the following reagents added:—

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

The concentrate and tailing were filtered, washed, and assayed for gold. A screen test showed the flotation tailing to be 54 per cent minus 200 mesh.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Concentrate	9·46	2·12	78·7
Tailing	90·54	0·06	21·3
Head (cal.)	100·00	0·255	

Test No. 2 was a duplicate of Test No. 1, except that the pulp was ground in the rod mill for 40 minutes instead of 30 minutes and a screen test showed the tailing to be 55·6 per cent minus 200 mesh.

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Concentrate	9·9	2·04	78·9
Tailing	90·1	0·06	21·1
Head (cal.)	100·0	0·256	

In Test No. 3, 2,000 grammes of the ore at minus 14 mesh was ground in 2 : 1 pulp for 30 minutes in an Abbé ball mill.

Reagents to ball mill:

Na_2CO_3	2·0 lb./ton
Aerofloat No. 25	0·07 "

The pulp was then transferred to a flotation machine and the following reagents added:—

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

The concentrate and tailing were filtered, washed and assayed for gold.

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Concentrate	13·1	1·74	87·64
Tailing	86·9	0·037	12·36
Head (cal.)	100·0	0·26	

Screen Analysis:

Mesh	Weight, per cent	Assay, Au, oz./ton	Values, per cent	Average tailing, Au, oz./ton
+200	11·70	0·09	28·45	0·037
−200	88·3	0·03	71·55	

In Test No. 4, 2,000 grammes of the ore at minus 14 mesh was ground for 25 minutes in an Abbé mill with the following reagents added:—

Na_2CO_3	2·0 lb./ton
Aerofloat No. 25	0·07 "

The pulp was then transferred to a flotation machine and the following reagents added:—

Sodium ethyl xanthate	0·2 lb./ton
Pine oil	0·18 "
$CuSO_4$	1·0 "

The concentrate and tailing were filtered, washed, and assayed for gold.

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Concentrate	10·63	2·08	85·93
Tailing	89·37	0·041	14·07
Head (cal.)	100·00	0·257	100·00

Screen Analysis:

Mesh	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Average tailing, Au, oz./ton
+200	17·57	0·09	39·00	0·041
−200	82·43	0·03	61·00	

CONCLUSIONS

The foregoing tests show that neither cyanidation nor amalgamation could be used to treat this ore. While the former will recover 70 per cent of the gold, the latter will not recover any at all. Concentration by flotation and disposal of the concentrate to a smelter would seem the best method of treating this ore.

Report No. 438

TESTS ON GOLD-SILVER-LEAD-ZINC ORE FROM THE ALICE LAKE GROUP OF MINERAL CLAIMS NEAR QUATSINO SOUND, VANCOUVER ISLAND, B.C.

Shipment. A shipment of two sacks of ore, net weight 200 pounds, was received May 2, 1932, from William Clancy, Jeune Landing, B.C.

Characteristics and Analysis of the Ore. The ore consists of galena, zinc-blende, and pyrite, in a gangue of crystalline limestone.

An average analysis of the sample is:—

Gold	0·335 oz./ton
Silver	3·23 "
Copper	0·14 per cent
Lead	5·76 "
Zinc	6·38 "

Purpose of Experimental Tests. Tests were made to determine whether the ore could be treated to recover the metals from it efficiently and economically.

Summary of Results. The greater part of the silver seems to be associated with the galena, whereas the gold is associated with the pyrite and possibly with the galena. Although galena concentrate showed a relatively high gold content it also contained a lot of pyrite which it was impossible to depress while the galena was being floated. It is therefore impossible to say whether any of the gold is associated with the galena.

Five concentration tests were made, all of them by flotation, and in some cases the grade of the product and the recovery varied considerably and in others it remained quite constant. Four copper-lead concentrates were made in which the assays ranged from 21 to 31 per cent lead, and lead recovery ranged from 91 to 94 per cent approximately.

Four zinc concentrates produced at the same time remained fairly constant in grade at 48 to 50 per cent zinc with corresponding recoveries ranging from 69 to 75 per cent approximately.

The gold recovery in the copper-lead concentrates ranged from 35 to 55 per cent and in the pyrite concentrates from 33 to 39 per cent approximately.

The silver recovery in the copper-lead concentrate ranged from 78 to 94 per cent which accounts for the major portion of it.

A bulk copper-iron-lead concentrate made in Test No. 5, while although assaying only 11·5 per cent lead, contained over 98 per cent of the lead in the sample as well as 94 per cent of the silver and 68 per cent of the gold.

Experimental Tests

FLOTATION

Test No. 2

A sample of 2,000 grammes of the ore at minus 14 mesh was ground in a rod mill in 1 : 1 pulp for 25 minutes with the following reagents:—

Na_2CO_3	5·0 lb./ton
NaCN	0·1 "
Thiocarbanilide	0·05 "

The pulp was floated and a copper-lead concentrate taken off using 0·21 pound per ton cresylic acid as a frother.

The zinc was reactivated with $CuSO_4$ and floated with potassium ethyl xanthate and cresylic acid in the following proportions:—

$CuSO_4$	0·5 lb./ton
Potassium ethyl xanthate	0·01 "
Cresylic acid	0·07 "

The zinc concentrate was recleaned in another cell using no additional reagents other than a couple of drops of the frother. All the products were filtered, washed, and assayed for gold, silver, copper, lead, and zinc.

Test No. 3

A sample of 2,000 grammes of the ore at minus 14 mesh was ground in a rod mill for 25 minutes in 2 : 1 pulp with the following reagents:—

Na_2CO_3	2·0	lb./ton
NaCN	0·1	"
Minerec "B"	0·02	"

A copper-lead concentrate was taken off using cresylic acid, 0·07 pound per ton, as a frother.

In this test 0·1 pound per ton sodium Aerofloat was used to float the zinc instead of potassium ethyl xanthate as was used in Test No. 2. The $CuSO_4$ and cresylic acid were kept the same as in Test No. 2. The zinc concentrate was recleaned in another cell giving a clean concentrate and a middling as before.

A pyrite concentrate was next taken off with the following reagents:—

Potassium ethyl xanthate	0·2	lb./ton
Pine oil	0·05	"

What now remained in the flotation machine was the tailing. All five products were filtered, washed and assayed, the pyrite concentrate for gold and zinc only, the other four for gold, silver, copper, lead, and zinc.

Test No. 4

Test No. 4 was a duplicate of Test No. 3 except that twice the quantity of sodium cyanide was used and the Minerec "B" was added to the flotation machine instead of to the rod mill. The same five products were taken off and assayed.

Test No. 5

In this test an attempt was made to concentrate the major part of the precious metals in a bulk concentrate of the lead and iron since the previous tests showed that most of the gold was associated with the lead and iron concentrates.

Two thousand grammes of the ore at minus 14 mesh was ground for 25 minutes in a Denver rod mill in 2 : 1 pulp with the following reagents:—

Na_2CO_3	3·0	lb./ton
Na_2S	1·0	"

A concentrate was taken off using cresylic acid, 0·21 pound per ton, as a frother. This, the only concentrate taken off, and the tailing were filtered, washed, and assayed for gold, silver, copper, lead, and zinc.

Test No. 7

In this test an attempt was made to produce a high-grade lead concentrate and a pyrite concentrate with most of the gold and silver distributed between the two of them.

Two thousand grammes of the ore at minus 14 mesh was ground for 25 minutes in a Denver rod mill in 2 : 1 pulp with the following reagents added:—

Na_2CO_3	3·0 lb./ton
NaCN	0·2 "
Minerec "A"	0·05 "

The copper-lead concentrate was taken off using cresylic acid as a frother and was later recleaned in another cell with a small additional quantity of NaCN added to further depress the pyrite.

The zinc concentrate was taken off with copper sulphate, sodium Aerofloat, and cresylic acid. The zinc concentrate was recleaned without additional reagents.

The pyrite concentrate was next taken off and this time an additional 0·5 pound per ton $CuSO_4$ was added to reactivate the pyrite further. Then it was floated with the following reagents:—

Sodium ethyl xanthate	0·3 lb./ton
Pine oil	0·05 "

The six products were filtered, washed, and assayed for gold, silver, copper, lead, and zinc, as well as for iron and sulphur in the pyrite concentrate.

Test No.	Product	Weight, per cent	Assays							Recovery				
			Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Fe, per cent	S, per cent	Au, per cent	Ag, per cent	Cu, per cent	Pb, per cent	Zn, per cent
2	Head (cal.)	100·00	0·317	3·19	0·17	5·68	6·59							
	Cu-Pb concentrate	19·41	0·58	13·24	0·50	27·45	5·10			35·56	80·61	56·02	93·84	15·02
	Zn concentrate	9·25	0·20	0·86	0·25	0·15	49·20			5·84	2·50	13·35	0·24	69·05
	Zn middling	10·24	0·38	1·44	0·22	1·19	8·64			12·29	4·63	13·00	2·15	13·43
	Tailing	61·10	0·24	0·64	0·05	0·35	0·27			46·31	12·26	17·63	3·77	2·50
3	Head (cal.)	100·00	0·36				6·76							
	Cu-Pb concentrate	23·38	0·80	11·70	0·48	22·76	5·06			52·39				17·49
	Zn concentrate	9·82	0·06	1·00	0·24	0·72	50·54			1·65				73·37
	Pyrite concentrate	31·29	0·42				0·98	42·70	43·80	36·80				4·53
	Zn middling	2·71	0·36	2·10	0·24	2·51	9·43			2·73				3·78
	Tailing	32·80	0·07	0·23	0·04	0·26	0·17			6·43				0·83
4	Head (cal.)	100·00	0·36				6·85							
	Cu-Pb concentrate	25·66	0·76	10·22	0·46	21·23	5·15			54·81				19·29
	Zn concentrate	10·03	0·06	0·42	0·25	0·10	49·90			1·69				73·09
	Pyrite concentrate	28·07	0·42				0·86	43·00	45·60	33·14				3·53
	Zn middling	1·69	0·34	1·36	0·24		3·48			1·62				0·86
	Tailing	34·55	0·09	0·24	0·04	0·21	0·64			8·74				3·23
5	Head (cal.)	100·00	0·32	3·42	0·20	6·03	5·91							
	Cu-Fe-Pb concentrate	51·57	0·42	6·22	0·36	11·50	5·60			68·05	93·90	95·04	98·39	48·83
	Tailing	48·43	0·21	0·43	0·02	0·20	6·25			31·95	6·10	4·96	1·61	51·17
7	Head (cal.)	100·00	0·34	3·07	0·21	5·55	6·90							
	Cu-Pb concentrate	16·12	0·76	14·90	0·72	31·26	4·30			36·33	78·12	54·67	90·81	10·04
	Zn concentrate	10·73	0·14	0·98	0·30	0·75	48·20			4·45	3·42	15·16	1·45	74·95
	Pyrite concentrate	31·01	0·42	0·84	0·09	0·38	0·88	43·20	42·44	38·62	8·47	13·14	2·13	3·96
	Cu-Pb middling	4·97	0·58	3·08	0·36	3·75	6·25			8·55	4·98	8·43	3·36	4·50
	Zn middling	3·95	0·44	1·96	0·21	1·90	4·30			5·15	2·52	3·91	1·35	2·46
	Tailing	33·22	0·07	0·23	0·03	0·15	0·85			6·90	2·49	4·69	0·90	4·09

CYANIDATION

Test No. 6

A sample of the pyrite concentrate produced in Test No. 3 was treated with cyanide solution to see if the gold could be recovered from it.

Two hundred grammes of the concentrate was agitated for 45 hours with 600 c.c. of solution running 2 pounds KCN per ton. The extraction, however, was almost nil.

Results:

Product	Assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
Head	0·42	4·76	1·50	7·2
Tailing	0·40			

CONCLUSIONS

In considering the treatment of this ore four things must be kept in mind:—

1. The silver is associated with the galena.
2. The gold is associated with the pyrite.
3. Zinc is worth little at present.
4. Any hope of profits will depend on recovering the gold and silver, and as the gold cannot be recovered by cyanidation it will have to be treated at a smelter.

It was found possible to make fair grades of lead and zinc concentrates by selective flotation. The lead concentrate, however, contained much pyrite which it was impossible to depress and this no doubt accounts largely for its gold content. The gold recovered in the lead concentrate was as high as 55 per cent and a pyrite concentrate taken off later contained another 30 to 35 per cent of the gold, these two products accounting for the most of it.

When the galena and pyrite were floated together in Test No. 5, the gold recovery in the resulting concentrate was 68 per cent with a ratio of concentration of 1·94 : 1. Comparing this with Tests Nos. 3, 4, and 7 it is found that the aggregate weights of the lead and pyrite concentrates in each test is approximately half the weight of the ore from which they were obtained, but the gold recovery is not so high in the bulk concentrate as the aggregate of the two selective concentrates.

Since the gold cannot be recovered by cyanidation the ore or concentrates will have to be smelted.

Roughly the situation is this: If the ore be concentrated two tons of ore will yield one ton of concentrate worth about $17.00 which will have to pay for mining, milling, freight, and smelter charges.

Report No. 439

EXPERIMENTAL TESTS ON GOLD ORE FROM HERB LAKE, MANITOBA

Shipment. One sack of ore, net weight 100 pounds, was received June 14, 1932, from Charles B. Morgan, The Pas, Manitoba.

Characteristics and Analysis of the Sample. The sample contained some mica and some disseminated pyrite in a quartz gangue. The gold was nearly all in the free state. An average assay of the sample showed the gold content to be 1·30 ounces per ton.

Purpose of Experimental Tests. The purpose of the work done was to find out the most economical way of treating the ore to recover the gold from it.

Summary of Results. The work on this ore was confined to tests by amalgamation in view of the high recovery resulting therefrom. By grinding the ore to 75 per cent minus 200 mesh 97·0 per cent of the gold was recovered, and the recovery was approximately the same when the grinding was reduced to 50 per cent minus 200 mesh.

Experimental Tests

AMALGAMATION

Tests Nos. 1 and 2

In these tests the ore at minus 14 mesh was ground in a ball mill, for 20 minutes in the first test and 15 minutes in the second, then amalgamated with mercury for 30 minutes. The amalgamation tailings were filtered, washed, and assayed for gold.

Results:

Head sample: Au, 1·30 oz./ton.

Test No.	Tailing, Au, oz./ton	Recovery, per cent
1	0·039	97·0
2	0·043	96·7

Screen Analysis:

Test No.	Mesh	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Average tailing, Au, oz./ton
1	+200	25·85	0·05	33·25	
	−200	74·15	0·035	66·75	0·039
2	+200	50·1	0·05	58·9	
	−200	49·9	0·035	41·1	0·043

CONCLUSIONS

The ore is readily amenable to treatment by amalgamation; 96·7 per cent of the gold was recovered by amalgamation on grinding 50 per cent minus 200 mesh.

The following alternative flow-sheets are recommended:—

First: The ore ground in a ball mill, the discharge of the mill passing over a double-deck, vibrating screen equipped with 4-mesh and 14-mesh screens; the oversize from these two screens returned to the ball mill for regrinding; the throughs of the 14-mesh screen passing over amalgamation plates to a classifier in closed circuit with the ball mill; the oversize of the classifier returned to the ball mill for regrinding; the classifier overflow which should be maintained at about 50 per cent minus 200 mesh, passing over corduroy blankets; the blanket concentrate treated in an amalgamation barrel.

The 4-mesh screen will take the wear from the finer 14-mesh screen, and the removal of oversize from the ball mill discharge will prevent scouring of the amalgamation plates.

Second: The ore ground in a ball mill with a 4-mesh screen on the end of the mill to remove chips and any large oversize, the ball mill discharge going to a hydraulic trap to catch the coarse gold, the trap overflowing to the classifier in circuit with the ball mill; the oversize of the classifier returned to the ball mill for regrinding; the classifier overflow which should be maintained at about 50 per cent minus 200-mesh grinding, passing over corduroy blankets. Once a day the trap should be bled, and the blankets removed and washed every four hours. The material from the trap, which will contain the coarse gold, and from the blanket concentrate, which will contain the fine gold, may be treated in an amalgamation barrel for the recovery of the gold.

Report No. 440

THE RECOVERY OF GOLD FROM THE ARSENICAL ORE OF THE CAMERON ISLAND MINE, SHOAL LAKE, LAKE OF THE WOODS, ONTARIO

Shipment. A shipment of 240 pounds of ore was received May 28, 1932, from J. G. Cross, Port Arthur, Ontario. This sample represents the ore from Cameron Island Mine, Shoal Lake, in the Lake of the Woods district, Ontario.

Characteristics and Analyses of Ore. The important features which have a direct bearing on the metallurgical treatment of the ore are summarized briefly as follows:—

The ore minerals are pyrite, arsenopyrite, pyrrhotite, native gold, and chalcopyrite. These are disseminated throughout the gangue which consists of replacement quartz and probably some siderite, although most of it has been largely replaced.

The larger part of the gold apparently is associated with the arsenopyrite and is thought to be in solid solution. The native gold is found in association with the pyrrhotite. The pyrite does not seem to contain gold.

Gold	0·65 oz./ton
Arsenic	3·55 per cent

Purpose of Experimental Tests. The shipment was made in order that the most economical method for the recovery of the gold might be determined. A previous shipment of this ore was received during the year 1927 and considerable experimental work was performed, the results of which were not conclusive.

Experimental Tests

A few tests were made on the ore by direct cyanidation in order to check the results obtained on the previous shipment. The tests showed conclusively that the ore can not be treated by direct cyanidation.

Flotation concentration was then tried and it was found that the ore could be concentrated very easily and a high recovery of the gold obtained in a concentrate containing over 2·5 ounces of gold per ton with a ratio of concentration of 4·78 : 1.

However, it was found that it would be necessary to roast the concentrate before the gold could be extracted by cyanidation. By a properly controlled roast an extraction of 86 per cent of the gold in the concentrate was obtained giving an overall extraction of about 80 per cent.

CYANIDATION OF RAW ORE

Tests Nos. 1 and 2

The ore was ground to 80 per cent minus 200 mesh and agitated for 48 hours in a 0·2 per cent KCN solution at 1 : 3 dilution.

Reagent consumption: KCN..... 3·0 lb./ton
Lime..... 11·5 "
Extraction: 36·9 per cent of the gold.

Test No. 3

The ore was ground to 80 per cent minus 200 mesh and agitated for 48 hours in a 0·2 per cent KCN solution at 1 : 3 dilution. Before cyanidation the ore was given a wash containing 5 per cent caustic soda and agitated 3 hours with this solution before cyanidation.

Reagent consumption: KCN..... 4·2 lb./ton
Lime..... 8·5 "
Caustic soda..... 5.0 "
Extraction: 33·8 per cent of the gold.

FLOTATION

Test No. 1

The ore was ground to 48 mesh with only 49 per cent minus 200 mesh. The reagents used were: soda ash 7 pounds per ton, copper sulphate 0·5 pound per ton, coal-tar creosote 0·2 pound per ton, sodium xanthate 0·3 pound per ton, and pine oil. The soda ash, copper sulphate, and coal-tar creosote were added to the ball mill and ground with the ore.

Product	Weight, per cent	Assays		Recoveries	
		Gold, oz./ton	Arsenic, per cent	Gold, per cent	Arsenic, per cent
Concentrates	20·92	2·90	14·70	93·4	97·5
Tailings	79·08	0·055	0·10	6·6	2·5
Totals	100·00	0·65	3·2	100·0	100·0
Screen Test of Tailing:					
+ 65	2·2				
− 65+100	22·4				
−100+150	13·7	Ratio of concentration, 4·78 : 1.			
−150+200	12·8				
−200	48·9				
Totals	100·0				

Test No. 2

The reagents used were: sulphuric acid 8 pounds per ton, coal-tar creosote 0·2 pound per ton, and Aerofloat No. 25, 0·2 pound per ton.

The grinding was the same as in Test No. 1.

Product	Weight, per cent	Assays		Recoveries,	
		Gold, oz./ton	Arsenic, per cent	Gold, per cent	Arsenic, per cent
Concentrates	21·7	2·58	14·0	86·7	84·3
Tailings	78·3	0·11	0·72	13·3	15·7
Totals	100·0	0·65	3·6	100·0	100·0

CYANIDATION OF THE RAW CONCENTRATES

Test No. 1

The flotation concentrate was ground in cyanide solution to practically all minus 200 mesh and then agitated in 1 : 3 pulp for 48 hours, the solution being maintained at 1·2 per cent KCN.

Reagent consumption: KCN ..... 4·8 lb./ton
Lime ..... 9·5 "
Extraction was only 25·4 per cent of the gold.

ROASTING AND CYANIDATION OF THE CALCINE

A series of samples of flotation concentrate was roasted at different temperatures. Roasting was started at a low temperature and gradually brought up to the final temperature and held there until all fumes had stopped coming off.

The tests indicated that the essential feature in the roasting was to keep the concentrate at a low temperature during the early stages of the roast and until the arsenic was freely coming off. Then the temperature could be raised quite safely to break up the sulphates without danger of the formation of ferrates with the arsenic.

Test A. The flotation concentrate containing 2·36 ounces gold per ton and about 14 per cent arsenic was roasted in the manner described above. The final temperature was 500° C. This low end temperature gave a calcine containing some sulphates. This is indicated by the higher lime and cyanide consumption. The calcine was cooled and reground in cyanide solution and agitated 48 hours in a 0·2 per cent KCN solution at a dilution of 1 : 3.

Amount of concentrate roasted ..... 200 grms.
Amount of calcine obtained ..... 150 "
Loss ..... 50 "

Assay of concentrate ..... 2·36 oz./ton
Assay of tailing ..... 0·44 "

Extraction of gold ..... 86 per cent

Reagent consumption: KCN ..... 3·3 lb./ton
Lime ..... 9·0 "

Test B. The concentrate was roasted in the same manner as in Test A, except that the final temperature was raised to 750° C. and maintained there for 15 minutes to break up all sulphates.

The calcine instead of being cooled and reground in cyanide solution was dumped hot into lime water.

Amount of concentrate roasted	200 grms.
Amount of calcine obtained	150 "
Loss	50 "
Assay of concentrate	2·36 oz./ton
Assay of tailing	0·48 "
Extraction of gold	84·8 per cent
Reagent consumption: KCN	1·2 lb./ton
Lime	4·25 "

Attention is directed to the increase in extraction obtained by regrinding the calcine and also to the lower reagent consumption obtained by finishing the roast at a temperature high enough to break up the sulphates.

SUMMARY AND CONCLUSIONS

The results of previous test work together with the recent work indicate that a higher extraction can be obtained on roasted concentrate than on roasted ore. The recovery of gold obtained by flotation concentration is high and the ore can be concentrated by grinding only to 50 per cent minus 200 mesh. The process indicated for this ore is therefore flotation concentration at 48-mesh grinding with about 50 per cent minus 200 mesh followed by roasting of the concentrate and cyanidation of the reground calcine. The extraction indicated is about 80 per cent of the gold. The arsenic can be saved and might possibly be a source of additional income provided a market for it can be obtained.

Report No. 441

CONCENTRATION OF A COBALT ORE FROM WERNER LAKE, RED LAKE MINING DIVISION, ONTARIO

Shipment. A shipment of 1,600 pounds of cobalt ore was received March 30, 1932, from the Kenora Prospectors and Miners, Ltd., 100 Adelaide Street West, Toronto, Ontario. The shipment was made from their property at Werner Lake, Red Lake Mining Division, claim Nos. K.R.L. 10590 to 10595 inclusive.

Characteristics and Analysis. The shipment on analysis was found to contain:—

Cobalt	10·72 per cent
Copper	0·30 "

Purpose of Experimental Tests. Tests were made for the purpose of determining whether the chalcopyrite could be removed by selective flotation and a high-grade cobalt concentrate obtained which contained a low copper content.

Summary of Experimental Tests. A series of small-scale batch flotation tests indicated that by selective flotation 35 to 50 per cent of the copper could be removed as a low-grade copper concentrate assaying 3·8 per cent copper and 5 to 10 per cent cobalt and represents a loss of 2 to 7 per cent of the cobalt. A large-scale continuous test made on the balance of the shipment with a feed rate of 200 pounds of ore per hour showed that a copper concentrate assaying 5 to 6 per cent copper and about 10 per cent

cobalt could be made. This represented the removal of 56 to 60 per cent of the copper with a loss of about 3 per cent of the cobalt. A cobalt concentrate can be readily made containing 26 per cent cobalt with a recovery of over 90 per cent of the cobalt. The copper content of this cobalt concentrate with the head sample running 0·3 per cent copper would be under 0·3 per cent copper.

Experimental Tests

Only one small-scale batch test is included in the report. It represents the best results obtained on the selective flotation of the copper.

It should be noted that in all the small-batch tests the cobalt concentrate obtained assayed more than 26 per cent cobalt and as high as 31 per cent.

Batch Test

Ore, 2,000 grammes, was ground in a small rod mill to approximately minus 48 mesh.

Reagents added and ground with ore:

Soda ash	10·0 lb./ton
Cyanide	0·5 "

Concentrate No. 1 was made by adding to the flotation cell:

Cresylic acid	0·08 lb./ton

Concentrate No. 2 was made by adding:

Amyl xanthate	0·005 lb./ton

Concentrate No. 3 was made by adding:

Copper sulphate	0·5 lb./ton
Amyl xanthate	0·2 "
Pine oil	0·01 "

Product	Weight, per cent	Analysis		Recovery	
		Co, per cent	Cu, per cent	Co, per cent	Cu, per cent
Flotation concentrate No. 1	4·35	5·34	3·8	1·5	30·7
" " No. 2	0·95	11·98	2·96	0·8	5·2
" " No. 3	52·6	27·25	0·50	94·6	48·6
Flotation tailing	42·1	1·12	0·20	3·1	15·5
Totals	100·0	15·1	0·54	100·0	100·0

The tailing from the above test was tabled on a small Wilfley table.

Results:

Product	Weight, per cent	Analysis, Co, per cent	Recovery, Co, per cent
Table concentrate	2·0	3·9	0·5
Table tailing	40·1	1·0	2·6
Totals	42·1	1·12	3·1

From the above it will be noted that the concentrates assayed only 3·9 per cent cobalt and represented an additional recovery of only 0·5 per cent.

Large-scale Test

The following flow-sheet was used: The ore was fed to a ball mill operated in closed circuit with an Akins classifier. The classifier overflow went to a 10-cell flotation machine. A copper float was made in cells Nos. 2 and 3 and recleaned in cell No. 1. The cobalt float made on the copper circuit tailing was carried out in cells 4 to 10 inclusive; cells Nos. 4 to 7 inclusive being used to produce final concentrates and the last 3 cells, rougher concentrate which was returned to cell No. 4, the head of the cobalt circuit.

The reagents and feed rate were as follows:—

Feed rate ..... 200 lb./hour

Reagents to ball mill:

Soda ash ..... 10.0 lb./ton
Cyanide ..... 0·07 "

The copper was floated with:

Cresylic acid ..... 0·10 lb./ton

The cobalt was floated with:

Amyl xanthate ..... 0·2–0·3 lb./ton
Copper sulphate ..... 0·5–0·75 "
Pine oil ..... 0·005 "

The ball mill density was 80 per cent solids.
The classifier overflow was 40 per cent solids.

Results:

Product	Weight, per cent	Analysis		Units		Recovery, per cent		Ratio of concentration
		Co %	Cu %	Co	Cu	Co	Cu	
Feed	100·0	10·72	0·30	1,072·00	30·00	100·0	100·0	
Cu concentrate	9·9	11·23	2·8	111·18	27·72	10·36	86·46	1 : 10·10
Co "	32·9	26·42	0·1	869·22	3·29	81·02	10·26	1 : 3·04
Tailing	52·5	1·76	0·02	92·40	1·05	8·62	3·28	1 : 1·90
Feed	100·0	10·50	0·3	1,050·00	30·00	100·0	100·0	
Cu concentrate	3·24	10·08	5·25	32·66	17·01	3·11	56·76	1 : 30·86
Co "	43·58	22·0	0·2	958·76	8·71	91·32	29·06	1 : 2·29
Tailing	53·18	1·1	0·08	58·50	4·25	5·57	14·18	1 : 1·88
Feed	100·0	10·72	0·3	1,072·0	30·00	100·0	100·0	
Cu concentrate	2·97	9·92	6·03	29·46	17·91	2·75	59·70	1 : 33·67
Co "	41·04	23·45	0·24	962·39	9·85	89·78	32·83	1 : 2·44
Tailing	55·99	1·43	0·04	80·07	2·24	7·47	7·47	1 : 1·79

Run No. 1 had trouble with pump and classifier.

CONCLUSIONS

It will be observed that the cobalt concentrate is not so high grade in Run No. 2 as in Run No. 1, or as was obtained in the batch tests. This

was due to over anxiety on the part of the operator to make a low tailing; the result being that the cells were pulled too fast causing considerable gangue to be carried over with the cobalt concentrate and lowering its grade.

There will be no difficulty in making at least a 26 per cent grade of cobalt concentrate on this type of ore.

In regard to the elimination of the copper, Run No. 2 of the large-scale tests represents the best results which can be expected by selective flotation.

Report No. 442

THE RECOVERY OF GOLD FROM ORE OF THE CARIBOO GOLD QUARTZ MINING COMPANY, LIMITED, BARKERVILLE, BRITISH COLUMBIA

Shipment. Four boxes, weighing approximately 400 pounds, were received May 23, 1932, from the Cariboo Gold Quartz Mining Company, Limited, 543 Granville Street, Vancouver, B.C., Fred M. Wells, Managing Director. The material was said to come from the company's property at Jack-of-Clubs Lake, Barkerville, B.C. The consignment consisted of two samples, one of three boxes marked No. 5 Vein, and one box marked No. 2 Vein.

Characteristics of the Ore. The ore consisted of heavy massive iron pyrite in a white quartz gangue. The material from No. 2 vein had considerably more sulphide in it than that from No. 5 vein. No free gold was visible.

Purpose of Experimental Tests. The shipment was made for the purpose of determining the most efficient method to apply for the recovery of the contained gold preparatory to the erection of a 25-ton mill.

The test work showed that 80·8 per cent of the gold can be recovered by amalgamating material ground to pass 48 mesh; 80 per cent of the gold in the tailing from this operation can be recovered by cyanidation, a total recovery of 96·1 per cent; 97·1 per cent of the gold can be recovered by straight cyanidation of minus 48-mesh material and 98·6 per cent when the grinding is minus 200 mesh.

Sampling and Analysis. Both samples were treated separately. They were crushed to $\frac{1}{4}$ inch, mixed and quartered. By stage-crushing to succeeding finer sizes with intervening passes through a Jones riffle sampler, representative portions minus 100 mesh were secured. On assaying, the samples were found to contain:—

Lot A—No. 5 Vein	Au	1·04 oz./ton
Lot B—No. 2 Vein	Au	2·925 "

Experimental Tests

All tests were made on the larger sample, Lot A, No. 5 Vein. These included tests by amalgamation followed by cyanidation, straight cyanidation, and flotation concentration.

AMALGAMATION AND CYANIDATION

Test No. 1

A representative portion of the concentrate was ground to pass 48 mesh and amalgamated. After removing the amalgam, the residue was agitated, 1 : 3 dilution, with KCN solution 2 pounds per ton; 8 pounds of lime was added to maintain protective alkalinity.

Results:

Heads	Au	1·04	oz./ton
Amalgamation tailing	Au	0·20	"
Recovery by amalgamation		80·8	per cent
Cyanide tailing after 24 hours	Au	0·04	oz./ton
Cyanide tailing after 48 hours	Au	0·025	"

A screen analysis of the amalgamation tailing shows:—

Mesh	Weight, per cent	Assay, Au, oz./ton
+ 65	5·5	0·34
− 65+100	14·0	0·28
−100+150	16·2	0·25
−150+200	13·6	0·25
−200	50·7	0·16

The analysis indicates that fine grinding would be necessary to liberate all the gold. There is a uniform decrease in the gold from 65 to 200 mesh. Doubtless much of the gold left after amalgamation is associated with the sulphides.

Test No. 2

A sample of the ore was ground to pass 100 mesh and treated as in Test No. 1.

Results:

Heads	Au	1·04	oz./ton
Amalgamation tailing	Au	0·195	"
Recovery by amalgamation		81·2	per cent
Cyanide tailing after 24 hours	Au	0·03	oz./ton
Cyanide tailing after 48 hours	Au	0·01	"
Total recovery		99·0	per cent

Finer grinding than minus 48 mesh gives slightly higher recovery by amalgamation, and lowers the gold in the residue after cyanidation from 50 cents to 20 cents.

STRAIGHT CYANIDATION

Test No. 3

Representative samples of the ore were ground to pass 48-, 100-, 150- and 200-mesh screens. These were then agitated 1 : 3 dilution with a

sodium cyanide solution equivalent in strength to 2·0 pounds KCN per ton; lime, 8 pounds per ton, was added to supply protective alkalinity.

Results:

Mesh	Agitation period, hours	Heads, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton	
					KCN	CaO
− 48	24	1·04	0·03	97·1	0·6	5·9
− 48	48	1·04	0·025	97·6	0·75	6·3
−100	24	1·04	0·025	97·6	0·6	5·9
−100	48	1·04	0·02	98·1	0·75	6·3
−150	24	1·04	0·02	98·1	0·9	6·2
−150	48	1·04	0·015	98·6	1·05	7·0
−200	24	1·04	0·015	98·6	1·2	7·0
−200	48	1·04	0·015	98·6	1·2	7·0

Screen analysis of minus 48-mesh, 24-hour cyanide tailing:

Mesh	Weight, per cent	Assay Au, oz./ton
+100	19·3	0·07
−100+150	16·1	0 05
−150+200	13·3	0·04
−200	51·3	0·01

The results show that maximum extraction is obtained by fine grinding.

AMALGAMATION AND FLOTATION

Test No. 4

A sample of the ore was ground to approximately 150 mesh and amalgamated. The residue was then floated with 6 pounds soda ash, 0·10 pound sodium ethyl xanthate and 0·15 pound pine oil per ton. This resulted in a heavy iron pyrite concentrate.

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Heads	100·0	1·04	100·0
Amalgamation tailing	.....	0·17	*83·5
Flotation concentrate	26·1	0·60	15·1
Flotation tailing	73·9	0·02	1·4

*Recovered by amalgamation.

This process gives a heavy iron pyrite concentrate, 26·1 per cent of the total weight of ore milled, and containing $12 per ton in gold.

SUMMARY AND CONCLUSIONS

Eighty per cent of the gold can be recovered by amalgamating the ore crushed to pass 48 mesh. The residue contains $4 per ton, of which all but 50 cents can be recovered by cyanidation.

Fine grinding liberates more of the gold, of which 97·1 per cent can be recovered by the above method.

Straight cyanidation of the ore ground to pass 150 mesh results in a recovery of 98·6 per cent.

Flotation results in a bulky, low-grade concentrate which would have to be treated to recover the contained gold.

The most efficient method to apply to this ore is straight cyanidation.

An alternative method, and probably the one most suitable for a small mill operating on such ore, would be amalgamation at minus 65-mesh grinding. This would result in approximately 80 per cent of the gold being recovered. The residue from this operation, containing $3 to $4 per ton, should be impounded until such time when a straight cyanide mill could be installed, and these residues added to the mill feed.

Report No. 443

CYANIDATION OF FLOTATION CONCENTRATES FROM BRALORNE MINES, LTD., BRIDGE RIVER DISTRICT, B.C.

Shipment. Eight bags of material weighing approximately 625 pounds were received May 6, 1932, from Bralorne Mines, Ltd., Bralorne, British Columbia.

The shipment consisted of damp flotation concentrate.

Purpose of Experimental Tests. The shipment was made to determine what recovery of gold could be made by cyanidation. The results of the test work indicate that 90 per cent of the gold can be extracted by this method.

Sampling and Analysis. The entire lot was passed through a 14-mesh screen to break up lumps, thoroughly mixed, and a representative portion of 10 pounds taken out. This was dried and after further reduction in size, a sample was obtained which showed the shipment to contain 9·56 ounces gold, 3·32 ounces silver per ton.

Experimental Tests

The test work included batch cyanide tests with different periods of agitation, a study of the effects of lead acetate and sodium peroxide, and a cycle test to note the effect of the ore on the solutions.

Test No. 1

A series of tests was made on representative portions of the concentrate. These were agitated for 24, 48, and 72 hours, 1 : 3 dilution with a sodium cyanide solution equivalent to 5·0 pounds KCN per ton. Lime was added to maintain protective alkalinity.

Results:

Test No.	Agitation period, hours	Solution strength KCN, lb./ton	Lime, lb./ton	Lead acetate, lb./ton	Assay cyanide tailing		Extraction		Reagent consumption	
					Au, oz./ton	Ag, oz./ton	Au	Ag	KCN	CaO
1.....	48	5·0	15·0	.....	0·96	0·94	90·0	71·7	2·4	11·6
2.....	48	5·0	15·0	1·0	0·98	1·12	89·7	66·3	2·1	11·8
3.....	72	5·0	15·0	.....	0·94	0·89	90·2	73·2	3·0	9·9
4.....	72	5·0	15·0	1·0	0·96	0·94	90·0	71·7	3·0	11·6
5.....	24	5·0	20·0	.....	0·97	0·86	89·9	74·1	2·25	15·5
6.....	24	5·0	10·0	.....	0·97	0·96	89·9	68·1	1·8	7·9

These results indicate that changes in the use of lead acetate, time of agitation, strength of solution, or amounts of lime added do not affect recovery; 90 per cent of the gold apparently is the maximum that can be recovered.

Test No. 2

A sample of the concentrate was agitated for 24 hours, 1 : 3 dilution, with a KCN solution 3·0 pounds per ton. Lime, 10 pounds per ton, was added for protective alkalinity. After 24 hours the pulp was filtered and a fresh KCN solution, 1·0 pound per ton, was added together with 6 pounds lime, and a further 24 hours' agitation given. At the end of this time the pulp was filtered and the tailing assayed. This was found to have a value of 0·95 ounce gold per ton, an extraction of 90 per cent.

CYCLE TEST

Test No. 3

A representative portion of the concentrate was agitated for 24 hours, 1 : 3 dilution, with a KCN solution 3·0 pounds per ton. Lime, 10 pounds per ton, was added to maintain protective alkalinity. After agitation, the material was filtered, washed with water equal to one-fifth the weight of the concentrate. This wash water was added to the pregnant solution which was then precipitated with zinc dust and filtered. The barren solution was made up to 3·0 pounds KCN per ton and used for the next cycle.

A second series was run similar to the above, but with 1·0 pound lead acetate per ton added.

Cycle No.	Without lead acetate				With lead acetate			
	Tailing assay		Extraction		Tailing assay		Extraction	
	Au, oz./ton	Ag, oz./ton	Au	Ag	Au, oz./ton	Ag, oz./ton	Au	Ag
1.....	0·96	0·95	90·0	71·4	0·98	1·07	89·7	67·8
2.....	0·965	1·09	89·9	67·2	0·99	1·02	89·6	69·3
3.....	1·04	1·11	89·1	66·6	1·05	1·10	89·0	66·9
4.....	1·03	1·13	89·2	66·0	1·04	1·09	89·1	67·2
5.....	1·29	1·14	86·5	65·7	1·17	1·13	87·8	66·0
6.....	1·28	1·21	86·6	63·6	1·33	1·16	86·1	65·1
7.....	1·37	1·21	85·6	63·6	1·63	1·29	82·9	61·1

These results indicate that the solution becomes foul after repeated use and loses its efficiency. The use of lead acetate appears to be detrimental.

Test No. 4

A test was made to note the effect of adding sodium peroxide during agitation.

A sample of the concentrate was agitated 1 : 3 dilution with a KCN solution 3·0 pounds per ton; 7 pounds lime per ton was added and 0·5 pound sodium peroxide. After 6 hours' agitation, an additional 0·5 pound peroxide was added. Agitation was continued for 40 hours.

This treatment gave a residue containing 0·97 ounce gold per ton, representing an extraction of 89·9 per cent.

Sodium peroxide does not improve recoveries.

SUMMARY AND CONCLUSIONS

An average extraction of 90 per cent of the gold is indicated. This leaves $18.20 in the residues.

The solution after contact with the concentrate becomes foul, resulting in decreased extraction.

Sodium peroxide or lead salts are not beneficial.

Cyanidation of this material could best be done in batch lots, discarding the solutions at a point where the re-use falls below the economic point.

Disposal of the rich residue also would be a factor. The gold in this might about balance the smelting costs on the raw concentrate; this is a problem of economics best decided by the management of the property.

Report No. 444

RECOVERY OF GOLD FROM THE ORE OF THE BEATTIE GOLD MINES LIMITED, DUPARQUET TOWNSHIP, ABITIBI COUNTY, QUEBEC

Shipments. The following ore samples were received from Ventures Limited, 100 Adelaide Street West, Toronto, Ontario. These samples represent the ore from the Beattie Gold Mines, Limited, located in Duparquet Township, near La Sarre, Quebec.

Shipment No.	Date received	Material, weight
1	June 13, 1931	2 boxes of rejects from diamond drill cores
2	" 16, 1931	600 pounds of ore samples
3	July 7, 1931	15 tons " "
4	Jan. 1, 1932	22 " " "
5	April 12, 1932	3 " " "
6	" 18, 1932	10·5 " " "
7	" 19, 1932	11·5 " " "
8	" 20, 1932	10·0 " " "

Character and Analysis of Ore Shipments. The ore is exceedingly fine-textured and consists of a quartz-carbonate gangue through which pyrite and arsenopyrite are disseminated in varying proportions. The sulphides, pyrite and arsenopyrite, are confined almost exclusively to the quartz of the gangue, but a definite tendency for the arsenopyrite to occur in greater quantity where carbonate is present is to be noted.

The metallic minerals are represented chiefly by pyrite and arsenopyrite, with small quantities of magnetite and ilmenite. The pyrite occurs in grains ranging in size from 0·3 mm. to 0·001 mm., commonly around 0·025 mm. The arsenopyrite is much finer than the pyrite and occurs as small crystals and skeletal forms often in clusters.

No free gold was observed in the sections studied. A quantitative microscopic determination of the grain sizes of the sulphides is shown below. Pyrite and arsenopyrite are considered collectively as "sulphides", and the percentages are by volume. This treatment is consistent throughout the report.

Grain Sizes of the Sulphides in the Ore

Mesh	Sulphides
− 35+ 48	4·9
− 48+ 65	3·8
− 65+100	8·1
−100+150	12·5
−150+200	7·1
−200	63·6

Analyses of the shipments were as follows:—

Shipment No. 1	Gold	0·15 oz./ton		Silver	0·08 oz./ton		
" 2	"	0·26	"	"	0·16	"	
" 3	"	0·18	"	"	0·08	"	
" 4	"	0·23	"				
" 5–8 average	"	0·20	"				

Purpose of Experimental Tests. Experimental tests were run to determine the most economical method of recovering the gold from this ore.

Experimental Tests

The fact that the problem was the recovery of the gold from a large low-grade deposit was kept constantly in mind in all the experimental work.

Progress reports were submitted to the owners giving detailed results of all tests. For this reason and on account of the large number of tests made, only a summary of the results and conclusions which led up to the adoption of flow-sheets used in the large continuous-scale tests will be given in this report.

CYANIDATION

A thorough attempt was made to cyanide the ore direct. The following table gives an idea of the results of this work:—

Grinding	Tailing assay, Au, oz./ton	Recovery by cyanidation, per cent
− 48 mesh	0·07	53·3
− 65 "	0·07	53·3
−100 "	0·06	60·0
−150 "	0·07	53·3
−200 "	0·06	60·0

Reagent consumption was 0·5 to 0·75 pound per ton potassium cyanide and lime normal.

BATCH FLOTATION TESTS

The first tests were made on comparatively coarse grinding and it was found that at only 50 per cent 200-mesh material in the feed that a $15 concentrate could be produced with a recovery of 83 per cent of the gold. The plus 200-mesh tailing averaged about 80 cents while the minus 200-mesh material ran only 20 cents. Finer grinding to 80 per cent and 90 per cent minus 200-mesh gave higher grade concentrates and 90 per cent recovery, but it was found that the minus 200-mesh tailing increased in gold to an average of about 40 cents. Particular attention is drawn to this fact as it may have some bearing on the results obtained in the large-scale tests. These small-batch tests gave a definite indication that there was some middling floating, but also that the middling was difficult to float.

The following table gives a brief summary of typical results from the batch flotation tests:—

Product	Weight, per cent	Assays, oz./ton		Recovery, per cent	
		Au	Ag	Au	Ag
Heads	100·0	0·15	0·08	100·0	100·0
Concentrate	18·4	0·74	0·32	82·7	73·6
Tailing	81·6	0·035	.....	17·3	.....
Screen Analysis of Tailing:					
+ 48	16·5	0·09	.....	.....	.....
− 48+ 65	15·2	0·05	.....	.....	.....
− 65+100	12·5	0·03	.....	.....	.....
−100+150	9·8	0·03	.....	.....	.....
−150+200	5·3	0·03	.....	.....	.....
−200	40·7	0·01	.....	.....	.....
Totals	100·0	0·035	.....	.....	.....
Heads	100·0	0·19	.....	100·0	100·0
Concentrate	16·3	0·92	0·34	88·8	69·3
Tailing	83·7	0·022	.....	11·2	30·7
Screen Analysis of Tailing:					
+100	0·9	0·05	.....	.....	.....
−100+150	4·1	0·04	.....	.....	.....
−150+200	7·0	0·04	.....	.....	.....
−200	88·0	0·02	.....	.....	.....
Totals	100·0	0·022	.....	.....	.....

CONTINUOUS SMALL-SCALE FLOTATION TESTS

Following the batch testing series of preliminary small-scale continuous tests were run at the rate of 100 pounds of ore an hour. A few typical examples of the results obtained under different conditions follow:—

Product	Weight, per cent	Assays, oz./ton		Recovery, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Heads	100·0	0·18	0·08	100·0	100·0	.....
Concentrate	16·08	1·00	0·36	89·0	72·4	6·13 : 1
Tailing	83·92	0·023	.....	10·7	27·6	.....
Screen Analysis of Tailing:						
+200 mesh	18·2	0·04	.....	.....	.....	.....
−200 "	81·8	0·02	.....	.....	.....	.....
Totals	100·0	0·023	.....	.....	.....	.....
Heads	100·0	0·18	0·08	100·0	100·0	.....
Concentrate	10·11	1·54	0·66	86·5	83·4	9·89 : 1
Tailing	89·89	0·027	.....	13·5	16·6	.....
Screen Analysis of Tailing:						
+200 mesh	12·7	0·04	.....	.....	.....	.....
−200 "	87·3	0·025	.....	.....	.....	.....
Totals	100·0	0·027	.....	.....	.....	.....

The two tests given above were chosen from a number and are given to show the effect of raising the grade of the concentrate—that even with finer grinding the tailing is higher if too much middling is allowed to drop out of the concentrate. However, it was also observed that over 80 per cent of the gold in the tailing was in the minus 200-mesh material. A series of tests was therefore carried out in which the tailing from the first flotation was sent to a classifier, and the sands returned for regrinding and a second flotation. The result of one of these six tests is given:—

Product	Weight, per cent	Assay, Au, oz./ton	Au, recovery, per cent	Ratio of concentration
Heads	100·0	0·18	100·0	.....
Concentrates	9·71	1·54	83·1	10·3 : 1
Tailing	90·29	0·034	16·9	.....

The tailing from the above test was pumped to a Dorr classifier and the sand return and slime overflow sampled. The sand return was collected and reground and a batch flotation test made on it.

Product	Weight, per cent	Assay, Au, oz./ton	Au, per cent	Ratio of concentration
Dorr sand	48·3	0·049	69·5	2·07 : 1
Dorr slime	51·7	0·02	30·5	.....
Totals	100·0	0·034	100·0	.....

Screen analysis made of the slime showed 99 per cent minus 325 mesh, and of the sand as follows:—

Mesh	Weight, per cent	Assay, Au, oz./ton
+100	5·0	0·06
−100+150	15·3	0·06
−150+200	29·6	0·05
−200	50·1	0·045
Totals	100·0	0·049

The sand was reground and floated with the following results:—

Product	Weight, per cent	Assay, Au, oz./ton	Au, per cent	Ratio of concentration
Heads	100·0	0·055	100·0	.....
Concentrate	4·5	0·80	65·3	22·2 : 1
Tailing	95·5	0·02	34·7	.....

This meant that by regrinding the classifier sand, representing 48·3 per cent of the tailing, 45·3 per cent of the gold in the first flotation tailing could be recovered increasing the overall recovery to 91·0 per cent or by an additional 7·8 per cent.

CYANIDATION OF THE RAW FLOTATION CONCENTRATE

The cyanide tests made on the flotation concentrate were disappointing. A large number of tests were made under great variety of conditions. A turbo-agitator was tried with no better results. The extraction varied between 60 and 70 per cent. These results were confirmed by other laboratories.

CYANIDATION OF ROASTED CONCENTRATE

Preliminary tests indicated fairly good and uniform extraction. A series of cycle tests was run to determine the tendency of the solution to fouling. The results were fairly uniform and it was found that there was little tendency for the solution to foul, provided the roasted concentrates were given a preliminary water wash. Roasting was carried out both by batch lots and on a 1,000-pound lot in a 6-hearth, gas-fired, Herreshoff furnace, 3 feet in diameter. Results from roasting in the Herreshoff only are given.

The temperatures in the hearths were as follows:—

No. 3	1,223° F.
" 4	1,297° F.
" 5	1,320° F.
" 6	1,320° F.

The dust loss could not be determined owing to the hook-up used to exhaust the fumes. However, it was undoubtedly high. There was no tendency for the material to run on the hearths.

The roasted material assayed:—

Product	Gold, oz./ton	Silver, oz./ton	Fe, per cent	CaO, per cent	Total, S, per cent	Soluble S, per cent	Arsenic, per cent	Insoluble
Before roasting	1·33	0·52	.....	.....	.....	.....	2·5 to 3	.....
After roasting	1·50	0·56	21·75	5·06	2·69	2·56	0·45	52·82

The calcine gave an alkaline reaction with water and did not require the addition of lime to maintain protective alkalinity in any of the following cyanide tests, the details of which follow and are shown in the accompanying tables.

A screen analysis was made of the cyanide tailing and the minus 200-mesh material invariably ran the highest:—

Test No.	Weight, per cent +200 mesh	Assay, Au, oz./ton	Au, per cent	Weight, per cent −200 mesh	Assay, Au, oz./ton	Au, per cent
1a	22·0	0·23	21·3	78·0	0·24	78·7
1b	22·2	0·17	18·0	77·8	0·22	82·0
1c	18·7	0·17	15·1	81·3	0·22	84·9
3a	25·0	0·17	22·1	75·0	0·20	77·9
3d	20·7	0·175	17·9	79·3	0·21	82·1

Cycle Tests Nos. 1 to 7

In each of these tests 500 grammes of calcine, unwashed, was agitated for 48 hours with 1,500 c.c. of solution, KCN, 3 pounds per ton. As each test was completed the solution was measured and made up to the proper strength in KCN and used to treat the next lot of calcine. The tailing was washed and assayed for gold.

Results of this series of tests are given in the following table:—

Head sample: Au, 1·50 oz./ton.

Test No.	Tailing, Au, oz./ton	Recovery, per cent
1	0·215	85·67
2	0·22	85·33
3	0·24	84·00
4	0·22	85·33
5	0·235	84·33
6	0·24	84·00
7	0·24	84·00

Owing to the nature of the tests accurate data on cyanide consumption could not be obtained. No lime was added, the calcine itself containing sufficient for protective alkalinity. The lime built up to approximately 2 pounds per ton of solution, or 6 pounds per ton calcine.

Cycle Tests Nos. 8 to 14

The procedure was the same as in Tests Nos. 1 to 7 except that the calcine was washed before being treated with cyanide solution and, because of the washing, the addition of a small amount of lime was necessary to maintain protective alkalinity.

Head sample: Au, 1·50 oz./ton.

Test No.	Tailing, Au, oz./ton	Recovery, per cent
8	0·21	86·00
9	0·215	85·67
10	0·215	85·67
11	0·215	85·67
12	0·225	85·00
13	0·23	84·67
14	0·225	85·00

Analyses of the final solutions showed the following results:—

	Tests Nos. 1 to 7:		*Tests Nos. 8 to 14:*	
SO_3	2·75	grm./litre	2·51	grm./litre
Fe	0·0060	" "	0·0060	" "
Cu	0·044	" "	0·059	" "
As_2O_3	0·0012	" "	0·0001	" "
CaO	1·71	" "	1·41	" "
One litre reduces	450 c.c. $\frac{N}{10}$ $KMnO_4$		One litre reduces	468 c.c. $\frac{N}{10}$ $KMnO_4$

The results of both cycle tests show no appreciable decrease in extraction due to repeated use of the solution.

Extraction is consistently higher on the washed material in each test and by a margin sufficient to allow this operation to be carried out at a profit.

The concentrate used in the large-scale roasting test was obtained from large-scale pilot plant flotation tests made on Shipment No. 4.

LARGE-SCALE CONTINUOUS FLOTATION TESTS ON SHIPMENT No. 4

About 18 tons of ore was run at an average rate of 500 to 600 pounds per hour. Seven separate runs were made, all of which showed a uniform condition and an identical tailing of 40 cents. The flow-sheet used was similar to that shown for Test No. 4, except that one run was made using but one flotation unit, the classifier overflow from the primary and secondary grinding circuits being united, (see flow-sheet of Test No. 7). The same results were obtained in this case as when the reground material was treated in a separate and additional flotation unit, thus proving that it would not be necessary to increase flotation capacity in order to regrind and re-treat the sands from the first flotation tailing. The principal reason for using the second flotation unit in these tests was to obtain data which could not be obtained if the two circuits were combined.

As the results from all seven runs were uniform, only the details of Test No. 4, which may be regarded as an average, are included in this report. The preliminary grinding was done in a Hardinge ball mill closed circuit with a Hardinge classifier.

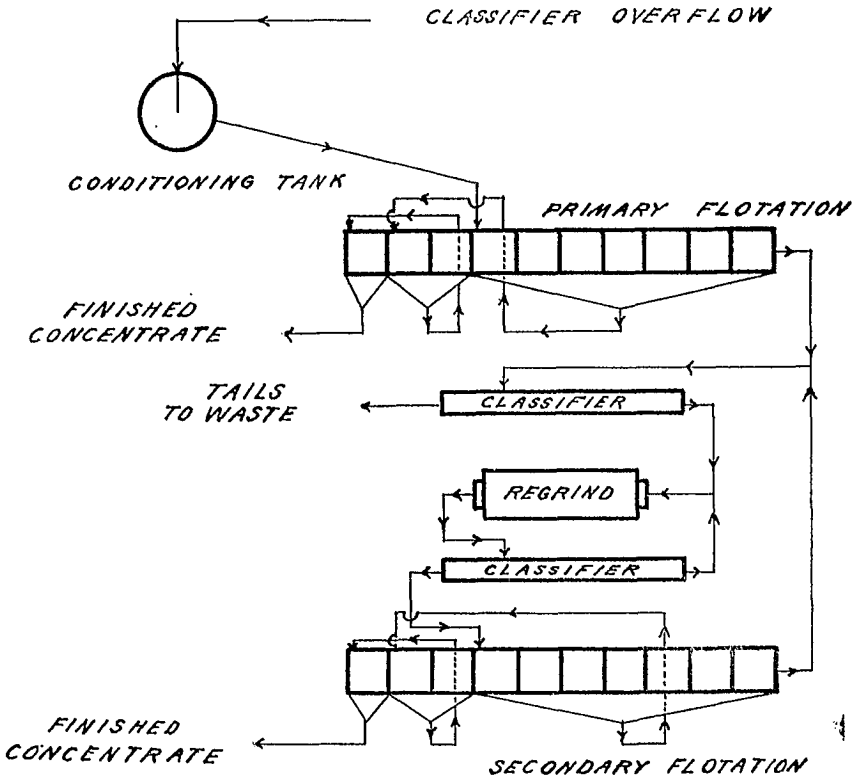

Figure 2. Flow-sheet of Test No. 4, on ore of Beattie mine.

Reagents used were as follows, and are given in pounds per ton:—

To Ball Mill:

Soda ash	0·75
Sodium ethyl xanthate	0·08
Aerofloat No. 25	0·03

To First Flotation Circuit:

Sodium ethyl xanthate	0·04
Cresylic acid	0·04

To Second Flotation Circuit:

Soda ash	0·75
Sodium xanthate	0·08
Aerofloat No. 25	0·03
Cresylic acid	0·015

Product	Weight, 100-ton unit	Assay, Au, oz./ton	Au, per cent	Ratio of concentration
Feed to first flotation circuit	100·0	0·24	100·0	.....
Concentrate—First flotation	10·35	1·98	88·0	9·6 : 1
Tailing—First flotation	89·65	0·035	12·0	.....
Feed to second flotation circuit	.....	0·06	.....	.....
Concentrate of second flotation	2·29	0·46	4·5	14·7 : 1
Final tailing of both circuits (Dorr overflow)	87·36	0·02	7·5	.....
Combined concentrates	12·64	1·665	92·5	7·92 : 1
Feed to Dorr classifier	89·65	0·035	100·0	.....
Dorr sand to regrind	33·61	0·06	64·4	2·67 : 1
Dorr overflow	56·04	0·02	35·6	.....

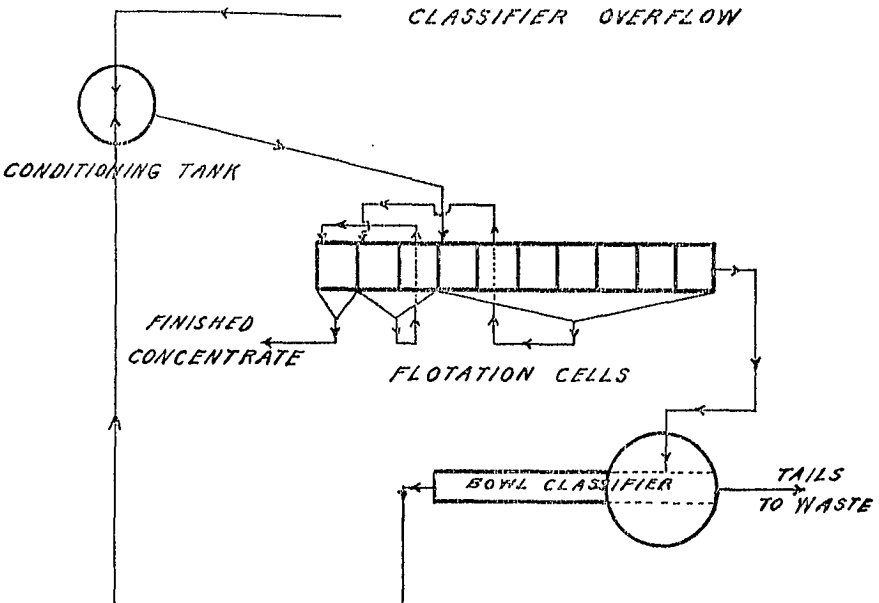

Figure 3. Flow-sheet of Test No. 7, on ore from Beattie mine.

This test indicates that out of every 100 tons treated and ground to the sizes shown in the following screen tests, about 33·6 tons will be returned for regrind averaging $1.20 per ton.

Grinding:

		Per cent
First flotation tailing	+200 mesh	21·0
	−200 "	79·0
Second flotation tailing	+200 mesh	24·0
	−200 "	76·0
Final tailing, Dorr overflow	+200 mesh	0·2
	−200 "	99·8

The results of Test No. 7 are given to show that the regrinding circuit can be handled by the primary flotation circuit without additional cell capacity and without affecting the grade of the primary concentrate.

Results of Test No. 7:

Product	Weight, 100-ton unit	Assay, Au, oz./ton	Au, per cent	Ratio of concentration
Feed	100·00	0·24	100·0	.....
Concentrate	10·65	2·08	92·6	9·4 : 1
Dorr overflow, final tailing	89·35	0·02	7·4	.....

Head sample: Au, 1·50 oz./ton.

Test No.	Treatment preceding cyanidation	Pulp dilution	Solution strength, lb./ton, KCN	CaO added, lb./ton, calcine	Period of agitation, hours	*Assay tailing, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton calcine	
								KCN	CaO
1a	None	3 : 1	3	5	24	0·238	84·13	1·20	none
1b	None	3 : 1	3	5	48	0·209	86·07	1·20	none
1c	None	3 : 1	3	5	72	0·21	86·00	1·50	none
1d	None	3 : 1	3	none	24	0·22	85·33	1·20	none
3a	Water-washed	3·31 : 1	0·98	3·2	72	0·193	87·13	1·26	none
3b	Water-washed and ground	3·58 : 1	0·96	6·9	72	0·18	88·00	1·29	5·50
3c	Ground in KCN solution 87 per cent −200 mesh	2·5 : 1	1	none	72	0·215	85·67	2·00	none
3d	None	3 : 1	5	none	72	0·203	86·47	2·10	none

NOTE.—Average tailing calculated from products of screen analysis.

The quantitative microscopic determinations were carried out on polished sections of the mounted mill products, all results being only approximate.

TYPICAL FLOTATION CONCENTRATE

The results of the microscopic analysis are tabulated below. The amount of free gangue is low, due to the fact that small particles of sulphide commonly adhere to fragments of gangue of much larger size. Only rarely, and in the smallest size, can a fragment of gangue be seen attached to a grain of sulphide of larger size; hence no figures are shown in the middlings column headed "sulphides and gangue".

Typical Flotation Concentrate

Mesh	Grain size, in mm.	Free sulphides	Middlings		Free gangue	Totals	Combined sulphides	Combined gangue
			Sulphides +gangue	Gangue+ sulphides				
+150.....	+0·104	.....	.....	18·4	.....	18·4	.....	18·4
−150..... +200.....	−0·104 +0·074	1·9	.....	8·9	1·9	12·7	.....	8·9
−200..... +325.....	−0·074 +0·044	1·4	.....	7·3	2·6	11·3	0·1	7·2
325.....	0·044	3·4	.....	17·9	4·7	26·0	0·1	17·8
	0·020	2·4	.....	7·0	2·7	12·1	0·8	6·2
	0·010	7·5	.....	4·9	3·5	15·9	1·9	3·0
	0·004	2·7	.....	0·9	.....	3·6	0·7	0·2
		19·3	.....	65·3	15·4	100·0	3·6	61·7

Two tests were made to see if the Dorr sand could be concentrated on tables in order to discard part before regrinding, but the results were not satisfactory.

Classifier oversize (feed to table)	Au	0·06 oz./ton
Table concentrate	Au	0·08 "
Table tailing	Au	0·05 "
Ratio of concentration		3 : 1

Dorr Sand from Test No. 10

A sample of Dorr sand from Test No. 10 was screened to four sizes, the results are:—

Dorr Sand from Test No. 10:

Mesh	Free sulphides	Middlings		Free gangue	Totals, per cent
		Sulphides +gangue	Gangue+ sulphides		
+100	0·05	.....	52·05	47·9	100·0
−100 +150	0·10	.....	37·5	62·4	100·0
−150 +200	0·14	.....	30·76	69·1	100·0
−200	0·17	.....	24·23	75·6	100·0

ELUTRIATION TEST ON FLOTATION TAILING

A sample of the final tailing from the Dorr classifier overflow was separated in a kinetic elutriator and the product assayed. This sample had a gold content of 0·02 ounce per ton, all minus 200 mesh.

The following table gives the results and indicates that 40 cents is about the lowest tailing that can be obtained economically from a $4 head sample:—

Mesh	Weight, per cent	Assay, Au, oz./ton	Au, per cent
−200+250	9·2	0·032	17·3
−250+800	24·6	0·023	33·2
−800	42·2	0·02	49·5
Cal. heads from products	100·0	0·0224	100·0

Elutriation Products

Three fractions from the elutriation of flotation tailing were studied. The following table shows the results of the study of two of these. The third fraction, which was approximately 1,000 mesh, was too fine for determination. However, most of this material is free and the proportion of sulphides to gangue is about the same as for the 300-mesh fraction, viz., sulphides : gangue :: 1·2 : 98·8.

Mesh	Free sulphides	Middlings		Free gangue	Totals, per cent
		Sulphides +gangue	Gangue+ sulphides		
−200 +300	0·3	.....	27·0	72·7	100·0
300	0·4	.....	10·6	86·0	100·0

LARGE-SCALE FLOTATION TESTS MADE ON SHIPMENTS NOS. 5 TO 8

The following tests are given in detail as a different flow-sheet was used in each. They were made under the direction of the Beattie Gold Mines' engineer, Mr. W. G. Hubler, who was present and took charge of the work. The comments given after each test, enclosed in quotation marks, are his. A flow-sheet showing the hook-up used accompanies each test and can be identified by the date. A number of runs were made which are not reported for the reason that more or less trouble was encountered due to mechanical interruptions.

Run No. 3

Date, April 21, 1932.

Feed rate approximately 500 pounds per hour.
Length of run, 16 hours.

Reagents:

—	Na_2CO_3	$CuSO_4$	B_4	Amyl xanthate	Pine oil	X CaCH	H_2SO_4
Ball mill	3·0	0·5	0·10				
Cond. tank				0·15	0·05		
Cell No. 4, primary rougher				0·05	0·05		
Cell No. 1, sec. rougher				0·05	0·03		
Cell No. 5, sec. rougher						0·01	
Cell No. 2, cleaner circuit							0·04

Screen Tests on Products:

Mesh	No. 1 heads	No. 2 heads	No. 3 heads	No. 1 tailing	No. 2 tailing	No. 3 tailing	No. 4 tailing	Rougher concentrate	Cleaner tailing	Final concentrate
+ 48				2·2	0·2	nil				
+ 65	0·1	0·1	0·2	6·2	0·7		0·2	0·3		
+100	1·7	1·5	2·6	13·3	5·3	1·1	1·7	1·1		
+150	5·3	6·8	11·4	10·6	9·0	6·7	7·1	2·8		0·03
+200	6·5	7·0	7·0	6·7	10·6	8·5	6·5	4·2	0·1	0·98
−200	86·5	84·6	78·8	60·9	74·2	83·7	84·5	91·6	99·9	99·8
+325							27·0			
−325							78·0			

Densities:

Tailing Sample No. 1, 43 per cent solids, 10 a.m. to 11.15 a.m., cleaner tail to waste.
Tailing Sample No. 2, 34 per cent solids, 11.15 a.m. to 1.45 p.m., cleaner tail to waste.
Tailing Sample No. 3, 31 per cent solids, 1.45 p.m. to 6.45 p.m., cleaner tail returned to circuit.
Tailing Sample No. 4, 38 per cent-41 per cent solids, 8.30 p.m. to 11.15 p.m., cleaner tail returned to circuit.

"The flotation machines acted as thickeners when no middlings were returned to the circuit; thus heads 34 per cent solids, flotation tailing 56 per cent solids, cleaner tailing 6 per cent solids."

NOTE.—X This symbol represents a private reagent of Mr. Hubler.

Results of Samples:

Samples—10 a.m. to 11.15 a.m.	
Heads	$ 3 40
Rougher concentrate	14 80
Tailing	0 70
Ratio of concentration	5·22 : 1
Recovery	83·3 per cent
Samples—11.15 a.m. to 1.45 p.m.	
Heads	$ 3 40
Rougher concentrate	14 80
Tailing	0 50
Ratio of concentration	4·93 : 1
Recovery	88·3 per cent

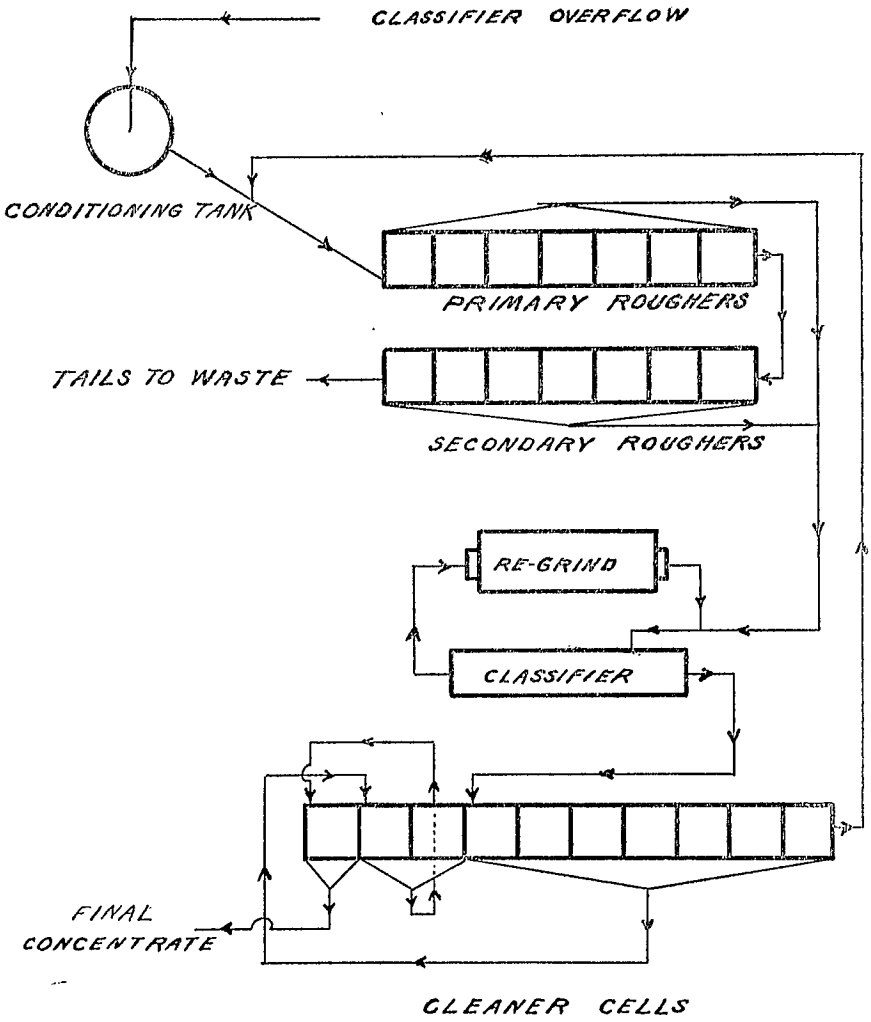

Figure 4. Flow-sheet of Run No. 3, April 21, 1932, on ore from Beattie mine.

This recovery was obtained in a rougher concentrate when grinding 84·6 per cent minus 200 mesh.

No sample was taken of the final concentrate. The circuit as shown in the accompanying flow-sheet was not complete.

Samples—1.45 p.m. to 6.45 p.m.	
Heads	$ 3 60
Rougher concentrate	14 80
Tailing	0 40
Final concentrate	40 80
Ratio of concentration	12·61 : 1
Ratio of rougher concentrate	4·5 : 1
Recovery	89·88 per cent
Grinding	83·7 " −200 mesh

"It is calculated that for each 100 tons milled there would be 27·2 tons returned to regrind circuit. This concentrate is 91·6 per cent −200 mesh which leaves 8·4 per cent of +200 mesh product to regrind."

Samples—6.45 p.m. to 11 p.m.	
Heads	$ 3 70
Rougher concentrate	14 80
Final concentrate	40 80
Final tailing	0 40
Ratio of concentration	12·24 : 1
Recovery	90·1 per cent
Grinding	84·5 " −200 mesh

"It is calculated that there would be 1·92 tons of +200-mesh product to be reground for each 100 tons milled."

Rougher Concentrate. The results of a microscopic determination are as follows:—

Mesh	Grain size, in mm.	Free sulphides	Middlings		Free gangue	Totals, per cent
			Sulphides +gangue	Gangue +sulphides		
−150 +200	−0·104 +0·074	.....	.....	8·5	.....	8·5
−200 +325	−0·074 +0·044	.....	.....	6·6	2·4	9·0
325	0·044	5·2	.....	7·0	9·2	21·4
	0·020	7·9	.....	7·6	12·0	27·5
	0·010	6·5	.....	1·9	12·0	20·4
	0·004	3·3	.....	1·5	8·4	13·2
Totals	.....	22·9	.....	33·1	44·0	100·0

Cleaner Tailing:

Mesh	Grain size, in mm.	Free sulphides	Middlings		Free gangue	Totals, per cent
			Sulphides +gangue	Gangue +sulphides		
−150	−0·104	.....	.....	.....	.....	0·00
+200	+0·074					
−200	−0·074	.....	.....	5·36	2·9	8·26
+325	+0·044					
325	0·044	.....	.....	4·97	3·8	8·77
	0·020	.....	.....	7·27	26·6	33·87
	0·010	0·38	.....	11·09	26·7	38·17
	0·004	0·96	.....	1·91	8·06	10·93
Totals	.....	1·34	.....	30·60	68·06	100·00

Run No. 4

Date, April 22, 1932.

Feed rate approximately 500 pounds per hour.
Ball charge in mill, 2,200 pounds.

Reagents:

—	Na_2CO_3	$CuSO_4$	B_4	Amyl xanthate	Pine oil	CaCH
To ball mill	2·5	0·50	.....	.....	.....	.....
To conditioning tank	.....	.....	.....	0·10	0·10	.....
Cell No. 8, primary rougher	.....	.....	.....	0·05	0·10	.....
Cell No. 1, secondary "	.....	.....	.....	0·01	0·15	.....
Cell No. 5, " "	.....	.....	.....	.....	.....	0·03

NOTE.—"Circuit was contaminated with oil or grease, which accounts for low-grade concentrates".

Screen Tests:

Mesh	Heads	Rougher concentrate	Final tailing	Returned classifier overflow	Returned classifier sand	Cleaner tailing	Primary rougher concentrate
+ 65	0·3	nil	0·2	.....	1·0	.....	0·2
+100	2·6	1·8	1·7	.....	7·9	.....	1·5
+200	16·4	6·3	14·2	nil	32·0	.....	8·7
−200	80·7	91·9	83·9	.....	59·1	100·0	90·4
+325	30·0	.....	24·0	2·0	62·0	nil	20·0
−325	70·0	.....	76·0	98·0	38·0	100·0	80·0

Samples—

Heads	$ 3·70
Final concentrate	No sample
Final tailing	0·40
Cleaner tailing	1·00
Rougher concentrate	4·80
Regrinding classifier overflow	6·40
Regrinding classifier sand	11·20

NOTE—"Samples do not check. Rougher concentrates 4·80 split into sands and slimes report $6.40 and $11.20."

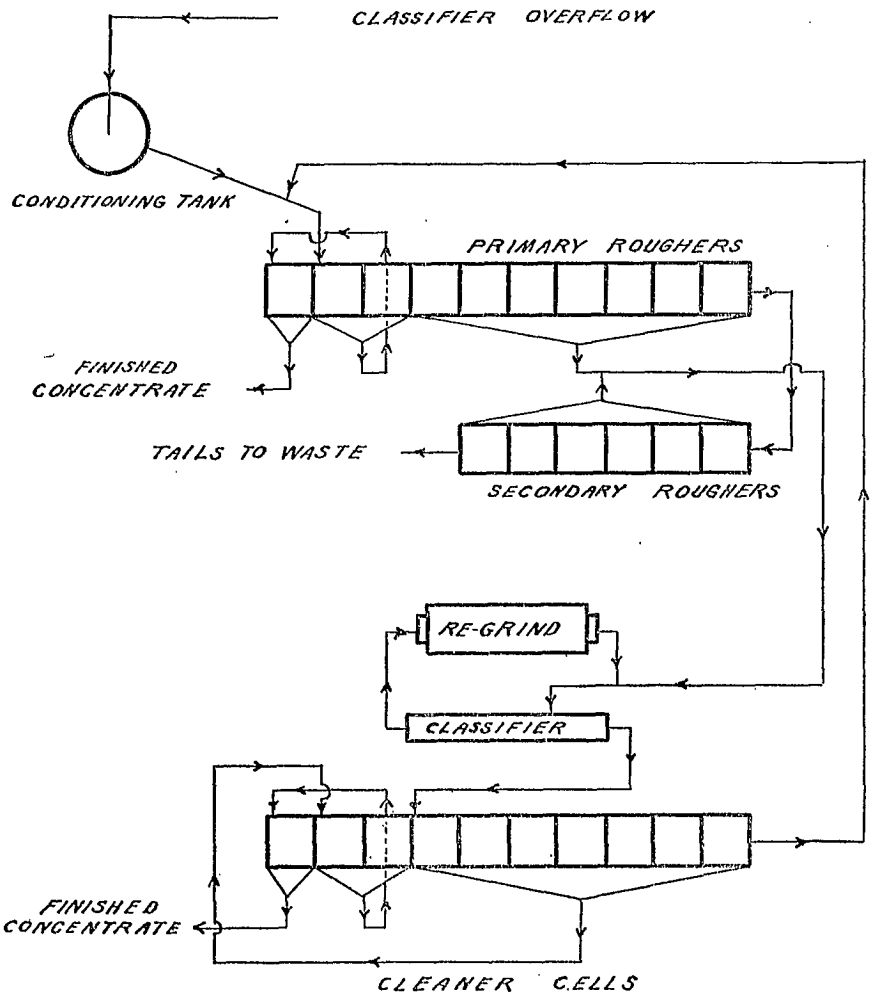

Figure 5. Flow-sheet of Run No. 4, April 22, 1932, on ore from Beattie mine.

Run No. 5

Date, April 25, 1932.

"In this test primary concentrate was made and the cleaner tailing was reground. The amount of cleaner tailing was so small that no attempt was made to operate the regrind circuit."

Feed rate approximately 500 pounds per hour.
Length of run, 16 hours.
Ball charge in mill, 1,800 pounds.

Sample Series No. 1:

Density ball mill 68 per cent solids; feed to cells 36 to 42 per cent.

Reagents:

——	Na_2CO_3	$CuSO_4$	B_4	Amyl xanthate	Pine oil	H_2SO_4	X CaCH
To ball mill	1·71	0·51	0·10				
Conditioning tank				0·05	0·03		
Cell No. 8, primary rougher				0·05	0·044		
Cell No. 1, secondary rougher				0·05	0·016		
Cell No. 5, secondary rougher							0·01
Cleaner cell No. 2						0·05	

X This symbol represents a private reagent of Mr. Hubler.

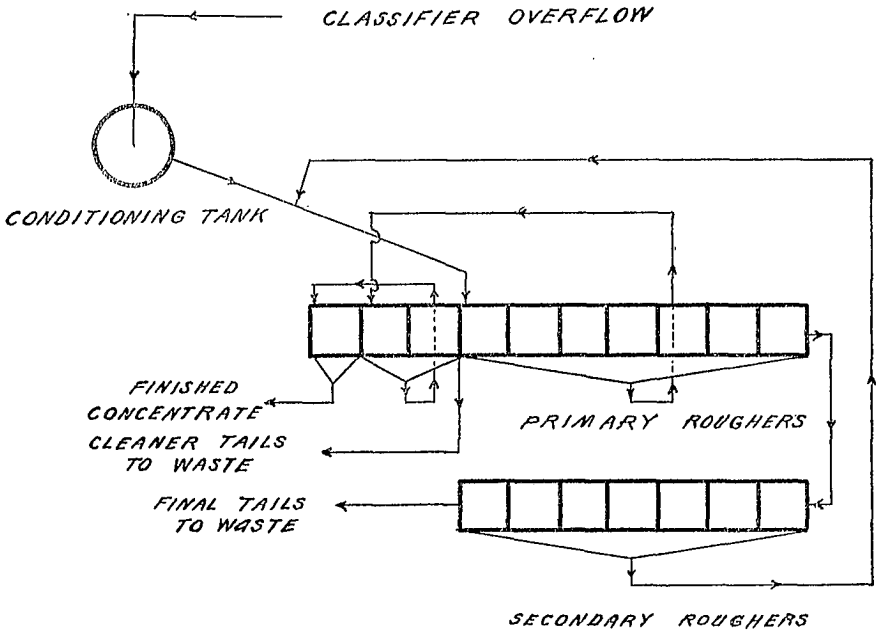

Figure 6. Flow-sheet of Run No. 5, April 25, 1932, on ore from Beattie mine.

Screen Tests:

Mesh	Heads	Final concentrate	Cleaner tailing	Final tailing	Special tailing
+ 65	0·5	.....	0·7	0·5	.....
+100	2·8	1·2	2·6	3·3	0·7
+200	17·2	11·8	6·1	17·2	10·3
−200	79·5	87·0	90·6	79·0	89·0
+325	54·0	24·0	12·0	32·0	24·0
−325	46·0	76·0	88·0	68·0	76·0

Assays:

Heads	$ 3.80
Cleaner tailing	1.90
Concentrate	31.80
Final tailing	0.40

Ratio of concentration 9·23 : 1; recovery 90·6 per cent.

"By calculation, 8 tons of cleaner tailing would be sent to regrind for every 100 tons of ore milled; and as cleaner tailing is 88·2 per cent −200 mesh, there would be $\frac{8 \times 12}{0{\cdot}80} = 1{\cdot}2$ tons of sand to regrind if classifier efficiency is taken at 80 per cent."

Second Series of Samples. These samples were taken using the same flow-sheet, but changing classifier overflow dilution from 42 per cent solids to 29 to 32 per cent solids, and thickening ball mill to 72 per cent solids.

Screen Tests:

Mesh	Heads	Final concentrate	Cleaner tailing	Final tailing
+ 65	.....	.....	.....	.....
+100	0·6	0·9	0·9	1·6
+200	8·9	6·9	5·3	13·0
−200	90·5	92·2	93·8	85·0
+325	20·0	12·0	8·0	26·0
−325	80·0	88·0	92·0	74·0

Assays:

Heads	$ 3 70
Cleaner tailing	2 40
Concentrate	37 80
Final tailing	0 40

Ratio of concentration 11·33 : 1; recovery 90·1 per cent.

"It is calculated that there would be 0·8 ton of +200-mesh sand to regrind for each 100 tons milled."

Run No. 7

Date, April 29, 1932.

Feed rate, 500 pounds per hour.
Length of run, 7 hours.

Reagents:

—	Na_2CO_3	$CuSO_4$	B_4	Xanthate	Pine oil	X CaCH
To ball mill	3·1	0·50	0·10			
Conditioning tank				0·08	0·022	
Cell No. 8, primary rougher				0·05	0·022	
Cell No. 1, secondary "				0·050		
Cell No. 5, " "				0·025		
Cell No. 8, " "						0·03
Cell No. 9, " "					0·022	

Screen Tests:

Mesh	Heads	Concentrate	Tailing	Rougher sand	Concentrate slime
+ 65				0·4	
+100	3·6	3·3	3·15	12·5	
+200	16·5	17·3	18·8	64·0	0·2
−200	79·8	79·3	78·10	23·2	99·8
+325	32·0	30·0	32·0	96·0	2·0
−325	68·0	70·0	68·0	4·0	98·0

Assays:

Heads	$ 3·90
Concentrate	16·80
Tailing	0·40
Special rougher concentrate sand	3·00
Special rougher concentrate slime	1·20

"These two last should be sent to regrind; they were taken from cells 8–9–10–11, secondary roughers."

X. This symbol represents a private reagent of Mr. Hubler.

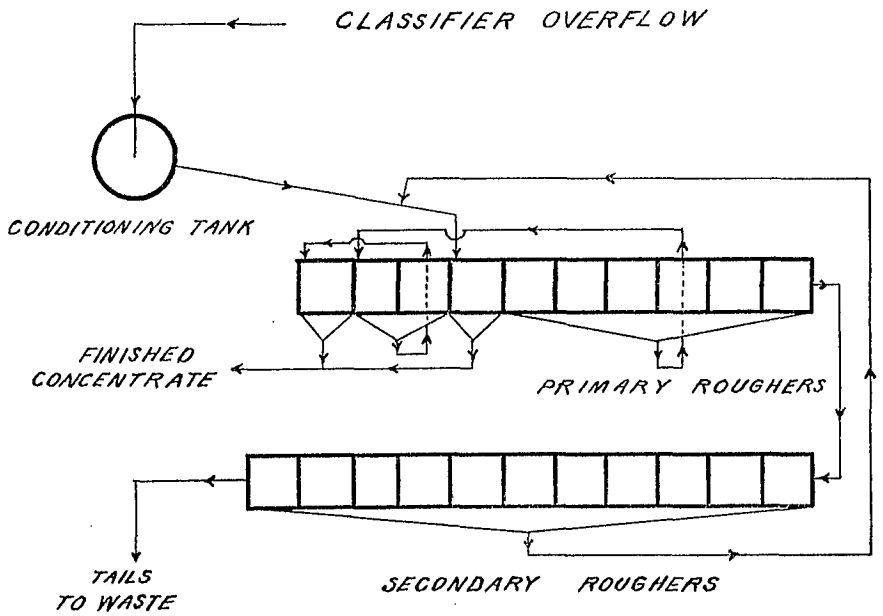

Figure 7. Flow-sheet of Run No. 7, April 29, 1932, on ore from Beattie mine.

Mines Branch Test No. 12

Feed rate, 500 pounds per hour.
Length of run, 8 hours.

Reagents:

—	Na_2CO_3	Xanthate	Pine oil	Cresylic acid	B_4
To ball mill	0·75	.....	.....	.....	0·1
To conditioning tank	.....	0·20	0·05	.....	.....
To cell No. 8	.....	0·10	.....	0·05	.....
To cell No. 11	.....	0·10	0·05	.....	.....
To regrind mill	0·25	0·05	.....	.....	.....

Assays:

Sample No. 1—Heads	0·20	oz./ton
" " 1—Concentrate	1·68	"
" " 1—Primary tailing	0·035	"
" " 1—Regrind classifier sand	0·04	"
" " 1—Final tailing	0·02	"
Sample No. 2—Heads	0·205	oz./ton
" " 2—Concentrate	1·84	"
" " 2—Primary tailing	0·03	"
" " 2—Regrind classifier sand	0·045	"
" " 2—Final tailing	0·02	"
Ratio of concentration	10·65 to 1	
Recovery	91·1 per cent	

"Tonnage of sand to regrind circuit as circulating load on classifier obtained by actual measurement—36·2 tons per 100 tons of ore milled."

Screen Analysis:

Mesh	Heads			Concentrate			Primary tailing		
	Weight, per cent	Assay, oz./ton	Gold distribution	Weight, per cent	Assay, oz./ton	Gold distribution	Weight, per cent	Assay, oz./ton	Gold distribution
+100	6·8	.....	.....	5·0	.....	.....	3·6	.....	.....
+150	10·4	0·13	18·0	11·2	0·52	.....	6·2	0·06	16·7
+200	10·8	.....	.....	8·4	.....	.....	9·0	0·055	14·2
−200	72·0	0·23	82·0	75·4	2·08	.....	81·2	0·03	69·1
Total	100·0	0·20	100·0	100·0	1·69	.....	100·0	0·035	100·0

Mesh	Sands to regrind			Reground sand			Final tailing		
	Weight, per cent	Assay, oz./ton	Gold distribution	Weight, per cent	Assay, oz./ton	Gold distribution	Weight, per cent	Assay, oz./ton	Gold distribution
+100	7·1	0·06	10·0	0·3	.....	.....	.....	.....	.....
+150	17·2	0·05	20·4	3·4	0·06	19·3	.....	.....	.....
+200	29·0	0·045	30·0	11·5	.....	.....	1·2	.....	.....
−200	46·7	0·035	39·5	84·8	0·045	80·7	98·4	.....	.....
Total	100·0	0·042	100·0	100·0	0·047	100·0	100·0	.....	.....

In presenting the results of the experimental work on the Beattie ore no specific recommendations are offered as to the treatment of the ore. In concluding this report, however, certain salient facts from the results of tests are pointed out in an endeavour to show their bearing on possible metallurgical methods for the treatment of the ore.

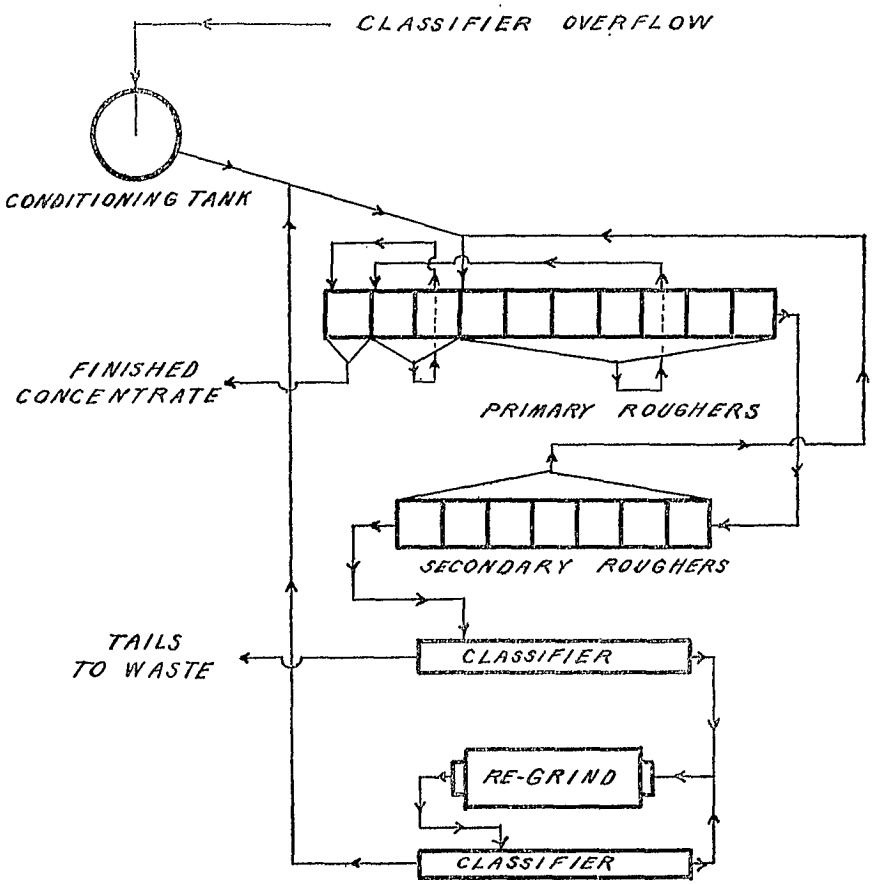

Figure 8. Flow-sheet of Test No. 12, on ore from Beattie mine.

The results of the experimental work can be summarized briefly as follows:—

1. The ore cannot be treated directly by the cyanide process. The average recovery was only about 60 per cent of the gold, ground minus 200 mesh.

2. The ore can be concentrated by flotation.

3. Between 65 and 70 per cent of the gold in the flotation concentrate can be extracted by direct cyanide treatment of the raw concentrate.

4. By roasting the concentrate and cyanide treatment of the calcine 88 to 90 per cent of the gold can be extracted.

5. By concentration, by flotation, roasting the concentrate, and cyaniding of the calcine, a maximum metallurgical extraction of 82 per cent of the gold in the ore can be obtained.

6. There is a slight possibility that other methods of treatment for the flotation concentrate may prove more economical, such as direct smelting. This, however, is doubtful.

Concentration by Flotation. This step in the metallurgy will undoubtedly be necessary. A few observations on the results of this part of the work are therefore in order.

An analysis of the grain size of the sulphide particles by volume shows that 63·6 per cent of the sulphides are finer than 200 mesh. The gold, more by elimination than by determination, has been placed as being contained in these sulphides. There was no difficulty in making a concentrate containing 80 to 85 per cent of the gold even at such comparatively coarse grinding as 50 per cent minus 200 mesh. The difficulty lies in raising the recovery to 90 per cent and maintaining a reasonable ratio of concentration such as 10 to 1, which would seem necessary when the re-treatment of the concentrate is considered. The 63·6 per cent by volume of sulphides finer in grain size than 200 mesh represents by far the largest number of the individual grains in the ore; that is to say, there are probably 100 grains smaller than 200 mesh for every grain that is larger. The greater number of these grains are extremely fine, that is, 600 to 1,000 mesh in size, which accounts for the difficulty of increasing the recovery by flotation without going to extremely fine grinding. The choice of flow-sheet to be adopted for flotation concentration, therefore, undoubtedly lies between two principal methods of treatment, namely those employed in Tests Nos. 2 to 8 inclusive and 11 and 12, which may be called Flow-sheet No. 1, and in the tests made by Mr. W. G. Hubler during April which may be called Flow-sheet No. 2. In the latter tests a middling product was floated which would go to a regrinding section; and in the former the tailing from the primary flotation containing the middling would go to a bowl classifier, and the sand from the bowl would go to a regrinding section.

At this point in the discussion it is necessary to define the term "true middling" as distinct from that of "middling"—loosely used in flotation practice. A *true middling* is a piece of ore containing gangue and sulphide still attached to each other, and it is this definition of middling which is considered in this report, not that part of a product consisting of free sulphides or gangue which is dropped out in a recleaning operation.

There is no doubt that the best grinding balance in Flow-sheet No. 1 was not obtained, as an oversize mill was used for the primary grinding, making it difficult to grind coarse. It was the intention to feed a product of about 70 per cent minus 200 mesh to the primary flotation circuit, instead of that of 80 per cent minus 200 mesh, which was actually obtained; when full advantage could have been taken of the ideal operating conditions for a bowl classifier, as it would be possible to run it at a very high dilution, thereby promoting high efficiency.

The test work shows that with Flow-sheet No. 1 a ratio of concentration exceeding 10 : 1, with a grade of concentrates of about $40, will be assured with a recovery of over 90 per cent, and for every 100 tons of ore milled with a primary grind at 80 per cent minus 200 mesh there would be between 30 to 50 tons of material returned for regrinding, consisting of a mixture of true middling and free gangue with but very little free sulphide. It would not be necessary to use additional cells to handle the reground product.

With the use of "Flow-sheet No. 2 the same tailing is obtained and the possibility of increasing the ratio of concentration above that with Flow-sheet No. 1. The tailing and recovery were obtained by doubling the flotation capacity used in Flow-sheet No. 1 (refer to flow-sheet used on Shipment No. 4)", when 10 cells were used whereas from 14 to 21 cells were used in Flow-sheet No. 2.

Runs Nos. 1 to 7 on ore Shipments Nos. 5 to 8 inclusive, were made using various hook-ups, with the same end in view, that is, of sending a middling to a regrind circuit. This product is not a true middling, it is a mixture of free sulphides, free gangue, and true middling.

The best hook-up required to obtain this product is immaterial as it would not affect the general layout of the mill, but it is important to determine if the whole of this product must be reground or whether it could be sent to a bowl classifier and only the sand reground. Mr. Hubler in all his comments and calculations on the test work on these shipments infers that only the sand need be reground.

This assumption, if correct, makes a very pleasing conclusion inasmuch as it would be necessary to regrind not more than 5 tons of material for every 100 tons of ore milled.

This point is open to discussion for in order to retain the grade of concentrate shown in Mr. Hubler's test work and also the ratio of concentration and the low tailing it would appear that the middling product must be sent to a thickener and the whole reground. The test work was not carried far enough to prove his assumption because the middling products were taken out of the circuit instead of being locked by regrinding and returning for retreatment. This conclusion is based mainly on the result of the microscopic study made on the middling products from the test runs of April 21 and April 26, which showed that 90 per cent of the sulphides in the middling exist as true middling, and that from 30 to 80 per cent of this true middling is finer than 325 mesh. Before a final decision is arrived at as to which flow-sheet should be adopted, a careful study should be made taking into consideration the following factors:—

1. With Flow-sheet No. 1 the concentrate obtained will be much coarser, only 75 per cent minus 200 mesh as compared to 98 per cent minus 200 mesh from Flow-sheet No. 2. This should be considered in connexion with the roasting problem.

2. The extra power consumption required by doubling the flotation capacity in Flow-sheet No. 2 could be balanced against the cost of the extra grinding required in Flow-sheet No. 1.

3. Flow-sheet No. 2 requires an initial grind be made to give a product running 80 per cent minus 200 mesh, and the feed to flotation circuit should be at least 35 per cent solid or denser, otherwise the capacity of the flotation cells will fall off rapidly. The test work showed this in a convincing manner. On such a light ore as the Beattie it will be very difficult and probably almost impossible to obtain efficient classification in a single-stage grinding circuit and at the same time overflow 35 to 40 per cent solids.

4. It is clear a lower cost for grinding can be obtained on the Beattie ore than is obtained with the same degree of grinding at Kirkland Lake. This applies only to Flow-sheet No. 1.

Report No. 445

RECOVERY OF GOLD FROM BLACK SAND CONCENTRATE FROM NORTH BEND, B.C.

Shipment. A shipment of about five pounds of black sand concentrate was received May 1, 1932, from J. A. Bourcier, North Bend, British Columbia.

Character and Analysis of Shipment. This shipment consisted of characteristic black sand concentrate recovered from the placer deposits of the lower Fraser River. The gold was present as thin flat flakes. A little platinum was also observed in the sample. On analysing, the sample was found to contain:—

	Oz./ton
Gold	1·85
Platinum and palladium	0·03

Purpose of Experimental Tests. The tests were made for the purpose of determining a practical method for the extraction of the gold from the concentrate without undue loss.

Summary of Experimental Tests. The experimental work showed that by plate amalgamation 95·3 per cent of the gold could be recovered, and that by flotation of the plate tailing an additional 1·9 per cent of the gold could be recovered in a flotation concentrate which assayed 3·01 ounces per ton in gold, and 1·50 ounces per ton platinum and palladium. The platinum recovery obtained by flotation was 58·4 per cent of the total platinum.

Experimental Tests

Test No. 1

A few ounces of sand were carefully panned in a small gold pan. It was noticed that as the panning operations progressed, small very thin flakes of gold floated on top of the black sand and were lost over the edge of the pan. The panning operation was carried to a point where if carried further practically all the gold would have been lost as float gold.

Test No. 2

About half the sample was fed over a small stationary amalgamation plate, 6 inches wide by 4 feet long. The tailing from the plate was caught and treated in a small flotation machine to recover any fine free gold not caught on the amalgamation plate.

Results:

Amount fed to plate	2,123·3 grammes
Assay, gold	1·84 oz./ton
" platinum and palladium	0·03 "
Tailing from plate	2,123·3 grammes
Assay, gold	0·13 oz./ton
" platinum and palladium	0·03 "
Flotation concentrate, weight	24·8 grammes
Assay, gold	3·01 oz./ton
" platinum and palladium	1·50 "

Ratio of concentration 85·7 : 1. From every ton of sand treated 23·4 pounds of concentrate would be obtained containing 0·0352 fine ounce of gold and 0·016 ounce of platinum having a value of about 75 cents.

Summary:

Flotation tailing, weight	2,098·5 grammes
Assay, gold	0·10 oz./ton

	Per cent
Gold recovered by amalgamation	95·3
Gold recovered by flotation	1·9
Total gold recovered	97·2

CONCLUSIONS AND REMARKS

From the above tests it can be seen that the gold contained in the black sand concentrate can be readily recovered by passing the sand over an amalgamation plate.

Test No. 1, however, indicates that the gold present in these black sands must be very difficult to catch by any of the ordinary methods used in washing gold from placer deposits, and it is judged that over half the gold content of the gravel is lost in making the concentrate that was tested. The main problem would, therefore, seem to be the development of a more efficient method of treating the original gravel.

Report No. 446

MAGNETIC CONCENTRATION OF IRON SANDS FROM BLACK BAY, PORT ARTHUR, ONT.

Shipment. A shipment of 200 pounds of black iron sands was received May 3, 1932, from J. S. Dobie, Esquire, Whalen Building, Port Arthur, Ontario, and came from Mining Locations TB1111-TB1115 inclusive, and the waterlots adjoining, on the easterly shore of Black Bay, near Port Arthur, Ontario.

Characteristics and Analysis. The sample consisted of fine-grained iron sands containing considerable hornblende and ilmenite in addition to magnetite.

The sample on analysis was found to contain:—

	Per cent
Gold and silver	None
Iron	33·62
TiO_2	9·76
SiO_2	16·30

Purpose of Experimental Tests. The purpose of the tests was to determine the grade and character of the concentrate which could be produced by magnetic concentration, and also the number of tons of crude sand required to produce one ton of concentrate.

Experimental Tests

A test was run on about 25 pounds of the sample by first passing the sand in its natural state through a magnetic separator.

Results:

—	Weight, per cent	Analysis, per cent					Ratio of concentration
		Fe	TiO_2	SiO_2	S	P	
Crude sand	100·0	33·6	9·8	16·3	.....	.....	2·09 tons of crude produce 1 ton concentrate.
Magnetic concentrate	47·8	51·3	17·4	4·7	0·12	trace	
Middling	2·0	28·3	.....	.....	.....	.....	
Non-magnetic (tailing)	50·2	17·4	.....	.....	.....	.....	

A further test was made in order to determine if the titanium in the above concentrate could be lowered. The concentrate from the above test was ground to 100 mesh and again passed through a magnetic separator.

Results:

—	Weight, per cent	Analysis					Ratio of concentration
		Fe	TiO_2	V_2O_5	SiO_2	S	
Feed	100·0	51·3	17·4	.....	4·7	0·12	1·17 tons of feed produce 1 ton of concentrate.
Magnetic concentrate	85·4	55·8	16·8	1·06	1·25	0·09	
Non-magnetic	14·6	24·8	.....	.....	.....	.....	

SUMMARY

It will be observed that the grinding of the first concentrate and re-treatment did not reduce the titanium to any appreciable extent.

Attention is directed to the analysis made for vanadium oxide in the reground concentrate which was found to contain 1·06 per cent V_2O_5.

Report No. 447

RECOVERY OF GOLD, SILVER, LEAD, AND ZINC FROM ORE OF THE YANKEE GIRL MINE AT YMIR, B.C.

Shipment. A shipment of 22 sacks of ore, net weight 2,023 pounds, was received August 16, 1932, from E. P. Crawford, Yankee Girl Mine, Ymir, British Columbia.

Characteristics of the Ore. The minerals identified are quartz, pyrite, sphalerite, and galena. The chief gangue mineral is quartz, which varies in colour from milky-white to dark grey. The sulphides are distributed irregularly through this gangue, with local concentrations into comparatively large masses.

Pyrite is common, and in places forms a major portion of the sulphide content. Where observed, it is intimately associated with galena and sphalerite, which form a matrix for numerous small pyrite grains.

Sphalerite locally attains the position of major importance so far as the ore minerals are concerned. Its tendency to form stringers and shoots is to be noted.

Galena is possibly the most abundant metallic mineral in the sample studied. It is intimately associated with the other sulphides and also occurs as minute disseminated grains and irregular stringers in the quartz.

No other metallic minerals were observed, with the exception of one minute grain having the appearance of native gold; positive identification of this grain was impossible.

An average analysis of the ore was as follows:—

Au	0·50 oz./ton
Ag	2·59 "
Pb	2·42 per cent
Zn	4·50 "
Fe	9·37 "
S	10·51 "

Experimental Tests

The sample representing the shipment, on which the tests were made, is the aggregate of individual samples taken from each of the bags.

A series of small-scale tests was made on the ore in order to determine the most satisfactory method of treating it, bearing in mind that the recovery of the gold and silver was essential, the lead next in importance and the zinc of little or no value at the present time, except for the gold and silver that might be associated with it. The work included concentration tests by tabling and flotation, and cyanidation tests on both the ore and the concentration products.

The results of the tests indicate that by cyanidation of the ore the maximum gold recovery possible is from 70 to 75 per cent. By flotation a bulk concentrate of the lead and zinc can be made containing well over 90 per cent of the gold, silver, and lead. The same net result was obtained by taking the lead and zinc concentrates off separately.

Test No. 1

This was a table test made on a sample of the ore roll-crushed to minus 20 mesh. An attempt was made to cut out a lead concentrate, but this was not successful on the small table, although it undoubtedly could be done on a large table.

The ore at minus 20 mesh was fed to a table which yielded a table concentrate, table sand tailing, and table slime tailing. The table slime tailing was passed over blankets and yielded a blanket concentrate and blanket tailing.

Results:

Product	Weight, per cent	Analysis				Recovery, per cent		
		Au, oz./ton	Ag, oz./ton	Pb, per cent	Zn, per cent	Au	Ag	Pb
Table concentrate	32·5	1·16	5·40	5·55	9·10	77·9	67·4	68·3
Table sand tailing	44·7	0·13	0·555	0·30	0·75	12·0	9·6	5·1
Slime heads	22·8	0·21	2·71	3·20	4·90	.....	.....	.....
Blanket concentrate	2·4	0·57	4·05	5·20	5·50	2·9	3·7	4·7
Blanket tailing	20·4	0·175	2·45	2·82	4·35	7·2	19·2	21·8

Screen Test on Table Sand Tailing with Analysis

Mesh	Weight, per cent	Au, oz./ton	Ag, oz./ton	Pb, per cent
− 20+ 28	28·3	0·20	0·525	0·28
− 28+ 35	23·2	0·145	0·455	0·28
− 35+ 48	15·8	0·12	0·40	0·23
− 48+ 65	13·2	0·10	0·35	0·23
− 65+100	7·0	0·09	0·33	0·20
−100	12·5	0·24	1·39	1·54
Totals	100·0	.....	.....	.....

From the above results it will be seen that the combined ratio of concentration is 2·87 : 1, and recoveries in the two concentrates of Au, Ag, and Pb are 80·8, 71·1, and 73·0 per cent respectively.

FLOTATION

Test No. 2

The ore at minus 20 mesh was placed in a flotation machine and floated with the following reagents:—

Na_2CO_3	1·5	lb./ton
Sodium xanthate	0·2	"
Pine oil	0·08	"

Results:

Product	Weight, per cent	Analysis				Recovery, per cent		
		Au, oz./ton	Ag, oz./ton	Pb, per cent	Zn, per cent	Au	Ag	Pb
Concentrate	16·8	1·52	9·34	10·9	12·10	58·0	68·0	68·8
Tailing	83·2	0·215	0·895	1·0	2·60	42·0	32·0	31·2

In this test the ratio of concentration is about 5·96 : 1.

Test No. 3

This is a flotation test followed by tabling of the flotation tailing. The ore at minus 20 mesh was placed in a flotation machine and a lead concentrate taken off with the following reagents:—

Na_2CO_3	3·0 lb./ton
Water-gas tar	0·10 "
Pine oil	0·08 "

A zinc-iron concentrate was next taken off with the following additional reagents:—

$CuSO_4$	0·5 lb./ton
Sodium xanthate	0·2 "

The flotation tailing was tabled and produced three products: table concentrate, table sand tailing, and table slime tailing. The weight of the table slime tailing was obtained by difference and the assays calculated in the same manner.

Results:

Product	Weight, per cent	Analysis				Recovery, per cent		
		Au, oz./ton	Ag, oz./ton	Pb, per cent	Zn, per cent	Au	Ag	Pb
Lead concentrate	4·54	3·68	27·46	28·25	16·30	33·42	48·14	53·00
Zn-Fe "	9·37	0·72	3·92	3·28	14·70	13·50	14·18	12·70
Table "	17·99	0·71	2·60	2·42	3·60	25·54	18·05	18·00
Table sand tailing	40·76	0·16	0·53	0·43	1·23	13·04	8·34	7·25
Table slime tailing	27·34	0·265	1·07	0·80	4·51	14·50	11·29	9·05

The combined ratio of concentration is 3·13 : 1.

Test No. 4

In this test an attempt was made to take off a lead concentrate carrying most of the gold and silver and then a zinc-iron concentrate carrying most of the remaining gold and silver which would be recovered from the concentrate by cyanidation. The ore at minus 20 mesh was ground for 20 minutes in a Denver rod mill and then the lead concentrate floated with the following reagents:—

Na_2CO_3	3·0 lb./ton
NaCN	0·20 "
Minerec "A"	0·05 "
Pine oil	0·08 "

The zinc-iron concentrate was taken off with the following reagents:—

$CuSO_4$	1·0 lb./ton
Sodium ethyl xanthate	0·20 "
Pine oil	0·12 "

Results:

Product	Weight, per cent	Analysis				Recovery, per cent		
		Au, oz./ton	Ag, oz./ton	Pb, per cent	Zn, per cent	Au	Ag	Pb
Lead concentrate	13·03	2·84	15·86	17·3	13·0	73·88	78·87	84·18
Zn-Fe "	17·13	0·58	2·09	1·25	13·0	19·85	13·66	8·00
Tailing	69·84	0·045	0·28	0·30	0·50	6·27	7·47	7·82

The combined ratio of concentration is 3·32 : 1. By cyaniding the Zn-Fe concentrate 63·96 per cent of its gold is recovered, giving an overall gold recovery of 86·6 per cent, assuming that the lead concentrate would be smelted and all its gold recovered.

Test No. 5

This was a duplicate of Test No. 4 except for a slight difference in flotation reagents. In Test No. 5 water-gas tar was used in place of Minerec "A".

Results:

Product	Weight, per cent	Analysis				Recovery, per cent		
		Au, oz./ton	Ag, oz./ton	Pb, per cent	Zn, per cent	Au	Ag	Pb
Lead concentrate	9·43	2·89	18·84	21·65	14·0	59·64	71·53	81·47
Zn-Fe "	21·16	0·74	2·39	1·21	12·70	34·28	20·36	10·22
Tailing	69·41	0·04	0·29	0·30	0·35	6·08	8·11	8·31

Combined ratio of concentration is 3·27 : 1, and overall gold recovery is 84·75 per cent when Zn-Fe concentrate is treated by cyanidation as in Test No. 4.

Test No. 6

This test is the same as No. 5 except that the ore was ground in the rod mill for 10 minutes only and the zinc and iron were taken off as two separate concentrates.

Results:

Product	Weight, per cent	Analysis				Recovery, per cent		
		Au, oz./ton	Ag, oz./ton	Pb, per cent	Zn, per cent	Au	Ag	Pb
Lead concentrate	7·22	3·47	22·21	25·30	15·35	48·82	60·80	73·55
Zn "	3·61	1·22	4·46	2·14	46·60	8·58	6·11	3·11
Fe "	19·66	0·90	3·06	1·71	4·60	34·48	22·81	13·54
Tailing	69·51	0·06	0·39	0·35	0·35	8·13	10·28	9·80

The combined ratio of concentration is 3·28 : 1, and the overall gold recovery by smelting the lead concentrate and cyaniding the iron concentrate only is 70·5 per cent.

CYANIDATION

Tests Nos. 7, 8, and 9

Three cyanidation tests were made on samples of the ore crushed dry through 48, 65, and 100 mesh respectively. Screen analyses of the cyanide tailings showed them to be 44·8, 56·5, and 65·3 per cent minus 200 mesh. Agitation was carried on for 48 hours in 3 : 1 dilution with KCN at 2 pounds per ton of solution.

Results:

Head sample: Au 0·50 oz./ton.

Test No.	Tailing assay, Au, oz./ton	Recovery, per cent	Reagent consumption lb./ton	
			KCN	CaO
7	0·163	67·4	1·8	10·2
8	0·144	71·2	2·4	10·7
9	0·151	69·8	2·4	10·7

Tests Nos. 10, 11, and 12

Cyanidation tests were made on the zinc-iron concentrates produced in Tests Nos. 4 and 5 and on the iron concentrates produced in Test No. 6. No grinding was done on any of these products. Agitation was carried on for 48 hours in 3 : 1 dilution with KCN at 2 pounds per ton of solution.

Results:

Test No.	Head assay, Au, oz./ton	Tailing assay, Au, oz./ton	Recovery, per cent	Reagent consumption lb./ton	
				KCN	CaO
10	0·58	0·209	63·96	4·2	18·5
11	0·74	0·198	73·24	4·2	18·3
12	0·90	0·334	62·89	3·3	18·7

Test No. 13

The ore was ground dry to 87·9 per cent minus 200 mesh.

The pulp was agitated for 48 hours in 3 : 1 dilution with KCN at 2 pounds per ton of solution. Lime was added at the rate of 15 pounds per ton of ore.

Results:

Product	Assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
Head	0·50	62·6	2·7	12·0
Tailing	0·187	.....	.....	.....

Screen Analysis Cyanide Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton	Metals, per cent	Average tailing
+200	12·1	0·205	13·2	0·187
−200	87·9	0·185	86·8	.....

FLOTATION

Test No. 14

This is a flotation test the object of which was to make a lead concentrate, carrying most of the gold for smelting and a zinc-iron concentrate carrying as much as possible of the remaining gold for cyanidation. (See Test No. 16.)

The ore at minus 20 mesh was ground for 20 minutes in a Denver rod mill with the following reagents added:—

Na_2CO_3	3·0	lb./ton
NaCN	0·2	"
Minerec "A"	0·05	"

A lead concentrate was taken off using three drops of pine oil as a frother. The zinc-iron concentrate was then taken off with the following reagents:—

$CuSO_4$	1·0	lb./ton
Sodium ethyl xanthate	0·2	"
Pine oil	0·12	"

Results:

Product	Weight, per cent	Analysis			Recovery, per cent		
		Au, oz./ton	Pb, per cent	Zn, per cent	Au	Pb	Zn
Lead concentrate	18·8	2·10	11·90	14·25	80·7	84·8	62·1
Zn-Fe "	17·2	0·40	1·10	8·50	14·1	7·2	33·9
Tailing	64·0	0·04	0·33	0·27	5·2	8·0	4·0

The combined ratio of concentration is 2·78 : 1 and cyanidation of the zinc-iron concentrate effects an overall recovery of 90·8 per cent of the gold.

Test No. 15

This is a flotation test in which an attempt was made to make a bulk concentrate carrying as much as possible of the gold. The concentrate was taken off in two stages, the lead first and then the zinc-iron, both being caught in the same container and thoroughly mixed. The reagent combination was the same as that used in Test No. 14. This was done to see how the mixed or bulk concentrate would respond to cyanidation, the results of which are given in Tests Nos. 17 and 18.

Results:

Product	Weight, per cent	Analysis			Recovery, per cent		
		Au, oz./ton	Pb, per cent	Zn, per cent	Au	Pb	Zn
Concentrate	40·8	1·12	5·80	9·65	96·9	93·5	96·2
Tailing	59·2	0·025	0·28	0·26	3·1	6·5	3·8

The ratio of concentration is 2·45 : 1 and the results of cyanidation are rather disappointing, the overall recoveries with and without grinding being 62·2 and 58·9 per cent respectively.

CYANIDATION OF CONCENTRATE

Test No. 16

This is a cyanidation test on the zinc-iron concentrate produced in Test No. 14 after it had been ground to 72·1 per cent minus 200 mesh. Agitation was carried on for 48 hours in 3 : 1 dilution with KCN 2 pounds per ton of solution and lime added at the rate of 20 pounds per ton of concentrate.

Results:

Product	Assay, Au, oz./ton	Recovery, per cent	Reagent consumption, lb./ton	
			KCN	CaO
Head	0·40	71·5	4·05	14·7
Tailing	0·114	.....	.....	.....

Screen Analysis:

Mesh	Weight, per cent	Assay, Au, oz./ton	Metals, per cent	Average tailing, Au, oz./ton
+200	27·9	0·19	46·4	0·114
−200	72·1	0·085	53·6	.....

Tests Nos. 17 and 18

These are cyanidation tests on the bulk concentrate produced in Test No. 15. Test No. 17 was done on the concentrate as it came from the flotation machine and Test No. 18 was done after further grinding as shown in the screen analyses.

Agitation was carried on for 48 hours in 3 : 1 dilution with KCN at 2 pounds per ton of solution and lime added at the rate of 20 pounds per ton of concentrate.

Results:

Head sample: Au, 1·13 oz./ton.

Test No.	Tailing assay, Au, oz./ton	Recovery, per cent	Reagent consumption, lb./ton	
			KCN	CaO
17	0·443	60·8	4·8	13·8
18	0·404	64·2	4·8	13·8

Screen Analyses:

Test No.	Mesh	Weight, per cent	Assay, Au, oz/ton	Metals, per cent	Average tailing, Au, oz./ton
17	+200	28·8	0·60	39·0	0·443
	−200	71·2	0·38	61·0	
18	+200	15·6	0·48	18·5	0·404
	−200	84·4	0·39	81·5	

Test No. 19

It was necessary to crush a new lot of ore, the original head sample cut from the shipment having become exhausted. For this purpose one of the sacks of ore was crushed to minus 20 mesh and used for further test work. The new sample was assayed for gold only, the value being Au 0·47 ounce per ton.

In this test an attempt was made to cyanide the ore and then float the cyanide tailing to produce a marketable lead concentrate carrying most of the remaining gold. This, however, was a failure and a bulk concentrate of the lead and zinc had to be made.

The ore at minus 20 mesh was ground in water in a rod mill for 20 minutes. It was agitated for 48 hours at 2·5 : 1 dilution in cyanide solution running, KCN 2 pounds per ton, with lime added at the rate of 14 pounds per ton of ore. The cyanide tailing was filtered and washed, then transferred to a flotation machine and the following reagents added:—

Na_2CO_3	3·0	lb./ton
NaCN	0·2	"
Minerec "A"	0·05	"

After 15 minutes' contact, pine oil was added at the rate of 0·1 pound per ton to float the lead. However, the lead showed little or no tendency to float, having evidently been too completely depressed by the lime during agitation. It was, therefore, necessary to add the following reagents:—

$CuSO_4$	1·0 lb./ton
Sodium ethyl xanthate	0·1 "
Pine oil	0·15 "

and make a bulk concentrate of the lead and zinc.

Results:

Product	Weight, per cent	Analysis				Recovery by flotation of cyanide tailing			
		Au, oz./ton	Ag, oz./ton	Pb, per cent	Zn, per cent	Au	Ag	Pb	Zn
Flotation concentrate	24·7	0·42	4·58	7·74	13·66	79·7	76·6	71·7	85·3
Flotation tailing	75·3	0·035	0·46	1·00	0·77	20·3	23·4	28·3	14·7
Cyanide tailing or head to flotation (cal.)	100·0	0·13	1·48	2·66	3·95	.....	.....	.....	.....

	Au, per cent	Ag, per cent
Recovery by cyanidation	72·3	42·9
Recovery in flotation concentrate	22·1	43·7
Recovery, overall	94·4	86·6

Reagents consumed:	lb./ton
KCN	2·5
CaO	10·3

Test No. 20

In this test an attempt was made to grind the ore and float a lead concentrate in cyanide solution and then agitate the flotation tailing in this same solution to extract the remaining gold.

The ore at minus 20 mesh was crushed in a ball mill, the charge being as follows:—

Ore	1,000 grammes
Solution	500 c.c. 3·6 lb./ton KCN
Lime	14 lb./ton

This charge was ground for 20 minutes. Then it was made up to 5,000 grammes with water and floated. The following reagents were added:—

Cresylic acid	0·20 lb./ton
Potassium amyl xanthate	0·032 "

and a lead concentrate was taken off. The tailing was transferred to a cyanide agitator and with the dilution reduced to 2·5 : 1 was agitated for 43 hours.

Results:

Product	Weight, per cent	Analysis				Distribution of metals contained in flotation concentrate and cyanide tailing, per cent			
		Au, oz./ton	Ag, oz./ton	Pb, per cent	Zn, per cent	Au	Ag	Pb	Zn
Flotation concentrate....	6·5	0·68	12·04	21·83	5·32	32·1	53·8	56·3	9·4
Cyanide tailing from flotation tailing.....	93·5	0·10	0·72	1·18	3·56	67·9	46·2	43·7	90·6

	Au, per cent	Ag, per cent
Total recovery by cyanidation.....	70·7	44·2
Total recovery in flotation concentrate.....	9·4	30·2
Total recovery in tailing.....	19·9	25·6

Reagents consumed:	lb./ton
KCN.....	6·8
CaO.....	13·8

Test No. 21

This test was the same as that of Test No. 20, the difference being in the time of grinding and the amount of reagents used.

Charge to ball mill:

Ore.....	1,000 grammes
Solution.....	500 c.c. 2 lb./ton KCN
Lime.....	14 lb./ton ore

The charge was ground for 15 minutes, then it was made up to 5,000 c.c. with water and a lead concentrate taken off with the following reagents:—

Cresylic acid.....	0·14 lb./ton
Potassium amyl xanthate.....	0·032 "

The flotation tailing was agitated in cyanide solution for 50 hours at 2·5 : 1 dilution.

Results:

Product	Weight, per cent	Analysis				Distribution of metals contained in flotation concentrate and cyanide tailing, per cent			
		Au, oz./ton	Ag, oz./ton	Pb, per cent	Zn, per cent	Au	Ag	Pb	Zn
Flotation concentrate....	5·7	1·0	13·42	23·74	4·62	33·5	46·6	52·5	7·0
Cyanide tailing from flotation tailing.....	94·3	0·12	0·93	1·30	3·70	66·5	53·4	47·5	93·0

	Au, per cent	Ag, per cent
Total recovery by cyanidation.....	63·8	36·6
Total recovery in flotation concentrate.....	12·1	29·5
Total recovery in tailing.....	24·1	33·9

Reagents consumed:	lb./ton
KCN.....	4·6
CaO.....	10·6

CONCLUSIONS

The results of the tests suggest possible methods for its treatment. It is difficult to say which one would be best suited to the problem as freight rates and smelter charges will be important factors in any of them. However, from a purely metallurgical viewpoint the following suggestions are offered:—

The low gold recovery resulting from cyanidation of the ore renders this process unsatisfactory by itself, but in combination with flotation, as in Tests Nos. 19, 20, and 21, it may hold possibilities.

A bulk flotation concentrate, containing well over 90 per cent of the total gold, responds rather poorly to cyanidation, but might be sold profitably to a smelter either before or after cyanidation.

The best method seems to be the one followed in Tests Nos. 4, 5, and 14 in which a lead concentrate and a zinc-iron concentrate were taken off, the former for sale to a smelter and the latter for cyanidation. The overall recoveries of the gold in these tests were 86·6, 84·75, and 90·8 per cent respectively. Screen tests made on the products and microphotographs of the products from Test No. 5 indicate that in order to obtain the recovery mentioned, the ore should be ground to at least minus 65 mesh with approximately 70 per cent minus 200 mesh. The cyanide tailing from the zinc-iron concentrate, being of no value at present, may be thrown away or impounded and kept in the hope of higher prices in the future. If the price of zinc rises provision might be made in the flotation plant for making a lead, a zinc, and an iron concentrate, the last-mentioned being the only one to be treated by cyanidation. The grades of the various concentrates to be made will have to be decided by the operators in the light of their knowledge of controlling economic factors.

Report No. 448

TESTS ON GOLD ORE FROM MINING CLAIMS K-3645, ISLAND 102P, LAKE OF THE WOODS DISTRICT, ONT.

Shipment. A shipment of 120 pounds of gold ore was received on September 15, 1932. The shipment consisted of three lots representing samples taken from three pits designated Nos. 1, 2, and 3. The shipment was from Mr. W. J. Dickson, 270 Fort Street, Winnipeg, Manitoba, from Mining Claim K-3645, Island 102P, Lake of the Woods.

Characteristics of the Ore. Two polished sections were made of selected samples of the ore and studied under the microscope. This examination showed the ore to contain pyrite, pyrrhotite, and, less abundant, sphalerite. Chalcopyrite was rare and gold occurs in considerable quantity. The metallic minerals occur in stringers in a gangue composed of iron-stained grey quartz with small patches of carbonate.

Purpose of the Experimental Tests. The shipment was made for the purpose of determining the most efficient method of recovering the contained gold.

Sampling and Analyses. The ore was crushed through a 14-mesh screen. A representative sample from each pit was obtained by riffling

through a Jones riffler and the laboratory sample was crushed minus 100 mesh and assayed for gold and silver. The average assays were as follows:—

Pit	Au, oz./ton	Ag, oz./ton
1	0·65	0·11
2	0·25	0·08
3	1·55	0·13
Composite head sample	1·25	0·11

Experimental Tests

The ore consisting of all the three samples was carefully mixed and a composite head sample was obtained by riffling. The assay of the head sample is given above. The experimental tests conducted were as follows:—

1. Amalgamation and cyanidation at 48 mesh.
2. Amalgamation and cyanidation at 100 mesh.
3. Straight cyanidation.
4. Flotation.
5. Amalgamation and flotation.
6. Amalgamation, grinding, and flotation.
7. Concentration on blankets, flotation of blanket tailing.
8. Concentration on blankets, flotation of the blanket tailing.
9. Amalgamation on an amalgamated copper pan.

SUMMARY

In *Test No. 1*, the minus 48-mesh ore gives a recovery of 89·7 per cent by amalgamation, and cyanidation of the amalgam tailing gives a total recovery of 98·4 per cent on the 24-hour test.

In *Test No. 2*, the minus 100-mesh ore gives a recovery of 87·6 per cent by amalgamation, and a total recovery by cyaniding the tailing of 98·0 per cent on the 24-hour test.

In *Test No. 3*, straight cyanidation of the raw ore gives 98·0, 98·4, 99·2, 98·8 per cent recovery on the minus 48-, 100-, 150-, 200-mesh ores respectively for the 24-hour test.

In *Test No. 4*, straight flotation gives only 54·1 per cent recovery when grinding to 95·42 per cent minus 200 mesh.

In *Tests Nos. 5 and 6*, amalgamation followed by flotation gives 89·6 and 90 per cent by amalgamation with an additional recovery of 8·82 per cent, and 8·4 per cent on a flotation concentrate when grinding 40·79 per cent minus 200 mesh in Test No. 5, and 86·31 per cent minus 200 mesh in Test No. 6.

In *Tests Nos. 7 and 8* the ore was crushed to approximately 76·9 per cent minus 200 mesh and concentrated on a special corduroy blanket. The tailing from the blanket was concentrated again by flotation. The blanket concentrate, of which there would be between $1\frac{1}{2}$ to 2 tons from every 100 tons of ore treated, was amalgamated by barrel amalgamation and about 99 per cent of the gold contained in it was extracted by the mercury. The percentage of total gold caught by the blankets was 86·72

per cent and 95·22 per cent respectively. In Test No. 7 flotation of the blanket tailing recovered an additional 11·74 per cent of the gold giving a total recovery by amalgamation and flotation of 97·8 per cent, and in Test No. 8 flotation of the blanket tailing recovered an additional 4·2 per cent of the gold, giving a total recovery by amalgamation and flotation of 98·67 per cent.

Test No. 9 was a check test on the methods used to determine the amount of gold which could be recovered by amalgamation. The ore was ground to approximately 77 per cent minus 200 mesh and amalgamated in a copper pan. The recovery was very high, namely, 97·6 per cent of the total gold.

CONCLUSIONS

Most of the gold is present as free gold, and can be recovered either by amalgamation or by blanket concentration followed by barrel amalgamation of the blanket concentrates. The recovery by straight amalgamation was as high as 90 per cent, and as high as 94·47 per cent by amalgamation of the blanket concentrate. Amalgamation is recommended as the first step to be used for milling this ore and a choice between the two methods used is largely a matter of personal preference.

The concentration of the amalgamation or blanket tailing by flotation is recommended to give an additional extraction and the test work indicates that by amalgamation and flotation a total recovery of over 95 per cent can be obtained.

If barrel amalgamation of a blanket concentrate is decided on, the flotation concentrate could also be treated in the barrel and additional gold recovered as bullion at the property.

In order to obtain the recovery mentioned the ore should be ground to at least 65 per cent minus 200 mesh and all minus 48 mesh.

Attention is directed to the possibility of the sample submitted having been taken from the surface that it may contain more fine gold than will later be found in the ore when mined at depth.

AMALGAMATION AND CYANIDATION

Test No. 1

The ore was ground to pass minus 48 mesh and amalgamated. After removing the amalgam the residue was treated by cyanidation in a pulp containing three parts of solution to one of ore. The strength of the sodium cyanide solution used was equivalent to 1·0 pound KCN per ton of solution, and 8 pounds of lime per ton of ore was added for a protective alkalinity.

Results:

Head assay	Au	1·25	oz./ton
Amalgamation tailing assay	Au	0·129	"
Recovery by amalgamation		89·68	per cent
Cyanide tailing after 24-hour agitation	Au	0·02	oz./ton
" " 48-hour "	Au	0·025	"
Total recovery after 24-hour agitation		98·4	per cent
" " 48-hour "		98·0	"

Reagent consumption, lb./ton:

	KCN	CaO
24-hour period	0·40	5·68
48-hour "	1·63	6·25

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Screen Analyses of the Amalgamation Tailing

Mesh	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
+ 65	10·54	0·175	14·25
− 65+100	12·57	0·11	10·68
−100+150	8·86	0·09	6·15
−150+200	7·48	0·06	3·47
−200	60·55	0·14	65·45

Test No. 2

A representative sample of the ore was ground to pass a 100-mesh screen and treated as in Test No. 1.

Results:

Head assay	Au	1·25	oz./ton
Amalgamation tailing assay	Au	0·155	oz./ton
Recovery by amalgamation		87·60	per cent
Cyanide tailing after 24-hour agitation	Au	0·025	oz./ton
" " 48-hour "	Au	0·01	"
Total recovery after 24-hour "		98·0	per cent
" " 48-hour "		99·2	"

Reagent consumption, lb./ton:

	KCN	CaO
24-hour period	0·39	5·78
48-hour "	1·47	6·95

A screen test on the amalgamation tailing shows the ore was ground to the following size:—

Mesh	Weight, per cent
+150	8·02
−150+200	14·98
−200	77·00

STRAIGHT CYANIDATION

Test No. 3

Samples of the ore were crushed to pass 48, 100, 150, and 200 mesh. The ore was treated by cyanidation in a pulp containing three parts of solution to one part of ore. The strength of the sodium cyanide solution used was equivalent to 1·0 pound KCN per ton of solution, and 8 pounds of lime per ton of ore was added to supply a protective alkalinity.

Results:

Heads: Au, 1·25 oz./ton.

Mesh	Agitation period, hours	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton	
				KCN	CaO
− 48	24	0·025	98·0	0·15	5·11
− 48	48	0·02	98·4	1·35	5·90
−100	24	0·02	98·4	0·30	5·26
−100	48	0·02	98·4	1·35	6·05
−150	24	0·01	99·2	0·30	5·20
−150	48	0·015	98·8	1·35	5·96
−200	24	0·015	98·8	0·30	5·41
−200	48	0·015	98·8	1·35	6·20

FLOTATION

Test No. 4

A representative sample of the minus 14-mesh ore was ground in a ball mill with soda ash, 6 pounds per ton, and Aerofloat No. 25, 0·07 pound per ton. The pulp was floated with sodium ethyl xanthate, 0·10 pound per ton, and pine oil, 0·05 pound per ton.

Results:

Products	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Ratio of concentration
Heads	100·0	1·25	100·0	
Flotation concentrate	1·04	64·96	54·1	96·15 : 1
Flotation tailing	98·96	0·35	27·7	

Loss of gold, 18·2 per cent, due to free gold.
Total recovery is 54·1 per cent.

A screen test on the flotation tailing shows:—

Mesh	Weight, per cent
+100	0·02
−100+150	0·82
−150+200	3·74
−200	95·42

Free gold is lost in this test, flotation alone is not adaptable to an ore containing so much free gold.

AMALGAMATION AND FLOTATION

Test No. 5

A sample of the ore was ground to pass a 48-mesh screen and amalgamated. The amalgam was separated and the residue was floated with soda ash, 6 pounds per ton, Aerofloat No. 25, 0·07 pound per ton, sodium ethyl xanthate, 0·10 pound per ton, and pine oil, 0·05 pound per ton.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Ratio of concentration
Heads	100·0	1·25	100·0	.....
Amalgamation tailing	.....	0·13	89·60*	.....
Flotation concentrate	1·19	9·28	8·82	84·1 : 1
Flotation tailing	98·81	0·02	1·58	.....

	Per cent
*Recovery by amalgamation	89·60
Recovery by flotation	8·82
Total recovery	98·42

A screen test on the flotation tailing shows:—

Mesh	Weight, per cent
+ 65	9·61
− 65+100	21·06
−100+150	17·69
−150+200	9·95
−200	40·79

AMALGAMATION, GRINDING, AND FLOTATION

Test No. 6

A sample of the ore was ground to pass a 48-mesh screen and amalgamated. The residue was ground in a ball mill with soda ash, 6 pounds per ton, and Aerofloat No. 25, 0·07 pound per ton.

The pulp was floated with sodium ethyl xanthate, 0·10 pound per ton, and pine oil, 0·05 pound per ton.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Ratio of concentration
Heads	100·0	1·25	100·0	.....
Amalgamation tailing	.....	0·12	90·40*	.....
Flotation concentrate	1·0	10·52	8·40	100 : 1
Flotation tailing	99·0	0·015	1·20	.....

	Per cent
*Recovery by amalgamation	90·40
Recovery by flotation	8·40
Total recovery	98·80

A screen test on the flotation tailing shows the ore was ground to the following size:—

Mesh	Weight, per cent
+100	0·62
−100+150	3·72
−150+200	9·35
−200	86·31

BLANKET CONCENTRATION AND FLOTATION

Test No. 7

A sample of the ore, minus 14 mesh, was ground in a ball mill, dilution 2 : 1 and passed over a standard corduroy blanket. The blanket concentrate was amalgamated. The blanket tailing was floated with soda

ash, 6 pounds per ton, Aerofloat No. 25, 0·07 pound per ton, sodium ethyl xanthate, 0·10 pound per ton, and pine oil, 0·05 pound per ton. The ore was ground to about 76·9 per cent minus 200 mesh.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Recovery, per cent
Heads	100·0	1·24	100·0
Blanket concentrate	2·95	36·53	86·72
Blanket tailing	97·05	0·17	13·28
Amalgamation of Blanket Concentrate			
Heads	.....	36·53	.....
Tailing	.....	0·29	.....
	.....		86·03
Flotation of Blanket Tailing			
Blanket tailing	97·05	0·17	.....
Concentrate	1·79	8·1	11·74
Tailing	95·26	0·02	.....

Summary

	Per cent
Recovery by amalgamation of the blanket concentrate	86·03
Recovery by flotation of the blanket tailing	11·74
Total recovery	97·77

Test No. 8

A sample of the ore, minus 14 mesh, was treated as in Test No. 7.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Recovery, per cent
Heads	100·0	1·24	100·00
Blanket concentrate	1·21	97·64	95·22
Blanket tailing	98·79	0·06	4·78
Amalgamation of Blanket Concentrate			
Heads	.....	97·64	.....
Tailing	.....	0·77	.....
	.....		94·47
Flotation of Blanket Tailing			
Blanket tailing	98·79	0·06	.....
Concentrate	3·01	1·73	4·20
Tailing	95·78	0·0075	.....

Summary

	Per cent
Recovery by amalgamation of blanket concentrate	94·47
Recovery by flotation of blanket tailing	4·20
Total recovery	98·67

AMALGAMATION COPPER PAN

Test No. 9

A sample of the minus 14-mesh ore was ground for 20 minutes in a ball mill, dilution 2 : 1, and amalgamated in an amalgamated copper pan.

Heads	1·25 oz./ton
Tailing	0·03 "
Recovery	97·6 per cent

The ore for this test was ground to approximately 77 per cent minus 200 mesh.

Report No. 449

TESTS ON GOLD-SILVER-LEAD-ZINC ORE FROM BEY MINES, LIMITED, NORTHBROOK, ONTARIO

Shipment. One sack of ore, net weight 168 pounds, was received September 8, 1932. The sample was submitted personally by Colin A. Campbell, Manager, Bey Mines, Limited, Northbrook, Ontario.

*Characteristics of the Ore.**—The gangue is chiefly quartz which locally contains small amounts of impure carbonate.

The metallic minerals observed in the sections are sphalerite, galena, tetrahedrite, chalcopyrite, marcasite, pyrite, and arsenopyrite. Sphalerite, galena, chalcopyrite and marcasite are abundant, and tetrahedrite is common but forms only a small percentage of the metallic mineral aggregate. Pyrite is rather uncommon in the sections studied, and arsenopyrite is rare.

Native gold was not observed.

An average analysis of the ore was as follows:—

Au	0·35 oz./ton
Ag	24·96 "
Cu	0·58 per cent
Pb	6·65 "
Zn	6·60 "

EXPERIMENTAL TESTS

The test work was limited to concentration by flotation with cyanidation tests on some of the products.

The results of the tests show that the best gold recovery can be obtained by floating off a copper-lead concentrate, recleaning it and uniting the cleaner tailing with the flotation tailing for cyanidation. This concentrate will contain from 50 to 75 per cent of the total gold, 80 to 85 per cent of the total silver, and upwards of 90 per cent of the copper and lead.

*From report of Mineragraphic Laboratory by Maurice Haycock.

By cyanidation, 93 per cent of the gold contained in the above tailing can be extracted, giving an overall gold recovery of from 96 to 98 per cent. It is also possible to make good grade zinc concentrates, but unfortunately not possible to make a tailing low enough in gold to throw away.

Test No. 1

The ore at minus 14 mesh was ground in a Denver rod mill for 25 minutes.

Charge to rod mill:

Ore	2,000	grms.
Water	1,000	c.c.
Na_2CO_3	20	lb./ton
NaCN	0·2	"
Minerec "A"	0·05	"

Copper-lead float:

Pine oil	0·05 lb./ton

Zinc float:

$CuSO_4$	0·5	lb./ton
Sodium Aerofloat	0·1	"
Pine oil	0·05	"

The zinc concentrate was recleaned in another cell giving a clean concentrate and cleaner tailing.

Results:

Product	Weight, per cent	Analyses					Recoveries, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Cu-Pb concentrate	14·7	1·75	149·0	3·78	45·36	11·00	73·2	83·9	91·6	90·7	23·1
Zn concentrate	9·2	0·07	8·28	0·20	0·50	54·60	1·8	2·9	3·0	0·6	71·6
Zn middling	1·5	0·38	18·22	0·18	3·78	7·50	1·6	1·0	0·4	0·8	1·6
Tailing	74·6	0·11	4·26	0·04	0·78	0·35	23·4	12·2	5·0	7·9	3·7

Test No. 2

This test is a duplicate of Test No. 1, except that cresylic acid was used as a frother instead of pine oil. The ore at minus 14 mesh was ground for 25 minutes in a Denver rod mill.

Copper-lead float:

Cresylic acid	0·21 lb./ton

Zinc float:

$CuSO_4$	0·5	lb./ton
Sodium Aerofloat	0·1	"
Cresylic acid	0·14	"

The zinc concentrate was recleaned using cresylic acid, 0·07 pound per ton, giving a clean concentrate and a middling.

Results:

Product	Weight, per cent	Analyses					Recoveries, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Cu-Pb concentrate	16·0	1·40	142·9	3·70	42·90	10·50	76·7	86·0	93·3	99·0	23·6
Zn concentrate...	5·6	0·56	7·44	0·17	0·50	55·80	10·7	1·6	1·5	0·4	43·8
Zn middling.....	0·9	2·36	21·98	0·23	2·06	8·40	7·3	0·7	0·3	0·3	1·1
Tailing.....	77·5	0·02	4·0	0·04	0·30	2·90	5·3	11·7	4·9	0·3	31·5

The products of this test, when figured back to a head sample, give a low gold assay. This raises a doubt as to the grade of the particular lot of ore this test was made on as no subsequent test produced a tailing so low in gold.

A polished section was made of the zinc middlings from this test in the hope of obtaining some information as to the occurrence of the gold. This section, in spite of the concentrated condition of the gold, failed to cut any grain of the metal.*

Although the copper-lead concentrate produced in this test is of good grade, the remaining gold is not concentrated in any one of the three remaining products.

Test No. 3

In this test a copper-lead concentrate was taken off and the tailing cyanided. The ore at minus 14 mesh was ground for 35 minutes.

Charge to rod mill:

Ore	2,000 grms.
Water	1,000 c.c.
Na_2CO_3	20·0 lb./ton
NaCN	0·3 "
Thiocarbanilide	0·1 "

Copper-lead float:

Pine oil	0·05 lb./ton

Results:

Product	Weight, per cent	Analyses					Recoveries, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Cu-Pb concentrate	16·9	1·40	131·94	3·80	39·51	11·90	66·3	86·9	90·6	97·1	28·1
Tailing.....	83·1	0·145	4·05	0·08	0·24	6·20	33·7	13·1	9·4	2·9	71·9

This is a very satisfactory test as 93·1 per cent of the gold in the tailing is recoverable by cyanidation, giving an overall recovery of the gold of 97·7 per cent. The copper and lead recoveries in the concentrate are also high.

* From report of Mineragraphic Laboratory by Maurice Haycock.

Test No. 5

In this test an attempt was made to take off a good grade of lead concentrate and a rougher zinc concentrate containing as much as possible of the remaining gold leaving a low-grade tailing to be discarded. The rougher zinc concentrate would then be recleaned with the idea of allowing the gold to drop out into the cleaner tailing and give a high-grade zinc concentrate low in gold. This, however, was not a success as too much of the gold remained with the zinc and in the flotation tailing. The ore at minus 14 mesh was ground for 25 minutes in a rod mill.

Charge to rod mill:

Ore	2,000 grms.
Water	1,000 c.c.
Na_2CO_3	20·0 lb./ton
NaCN	0·2 "
Minerec "A"	0·05 "

Copper-lead float:

Pine oil	0·05 lb./ton

Zinc float:

$CuSO_4$	0·5 lb./ton
Sodium Aerofloat	0·1 "
Pine oil	0·05 "

Zinc concentrate recleaned without additional reagents.

Results:

Product	Weight, per cent	Analyses					Recoveries, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Cu-Pb concentrate	16·8	1·68	131·42	3·41	40·35	13·80	78·7	84·8	93·4	94·7	33·2
Zn concentrate	8·3	0·50	11·80	0·24	2·01	50·80	11·6	3·8	3·3	2·3	60·5
Zn middling	3·5	0·18	10·50	0·17	1·46	5·0	1·8	1·4	1·0	0·7	2·5
Tailing	71·4	0·04	3·65	0·02	0·23	0·37	7·9	10·0	2·3	2·3	3·8

Screen Test of Flotation Tailing, Test No. 5

Mesh	Weight, per cent
+ 65	1·7
− 65+100	14·4
−100+150	25·8
−150+200	10·8
−200	47·3

Test No. 6

In this test a copper-lead concentrate was made and recleaned so as to improve the content of lead and copper and to drop out as much as

possible of the gold and zinc. The cleaner tailing was united with the flotation tailing and this was treated by cyanidation for the recovery of the gold content.

The ore at minus 14 mesh was ground for 35 minutes in a Denver rod mill.

Charge to rod mill:

Ore	2,000 grms.
Water	1,000 c.c.
Na_2CO_3	20·0 lb./ton
NaCN	0·3 "
Thiocarbanilide	0·1 "

Copper-lead float:

Cresylic acid	0·14 lb./ton

Concentrate recleaned without additional reagents.

Results:

Product	Weight, per cent	Analyses					Recoveries, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Cu-Pb concentrate	12·5	1·48	142·14	4·34	55·30	6·90	49·0	80·9	93·9	94·3	12·2
Tailing	87·5	0·22	4·79	0·04	0·48	7·10	51·0	19·1	6·1	5·7	87·8

Test No. 7

In this test the object was to collect as much as possible of the gold in a low grade zinc concentrate. A copper-lead concentrate was taken off and recleaned. Then a zinc concentrate was taken off and combined with the cleaner tailing from the copper-lead concentrate. The final flotation tailing was to be discarded.

The ore at minus 14 mesh was ground for 35 minutes in a Denver rod mill.

Charge to rod mill:

Ore	2,000 grms.
Water	1,000 c.c.
Na_2CO_3	20·0 lb./ton
NaCN	0·3 "
Thiocarbanilide	0·1 "

Copper-lead float:

Cresylic acid	0·14 lb./ton

Concentrate recleaned:

Cresylic acid	0·07 lb./ton

Zinc float:

$CuSO_4$	0·5 lb./ton
Sodium ethyl xanthate	0·15 "
Pine oil	0·10 "

Results:

Product	Weight, per cent	Analyses					Recoveries, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Cu-Pb concentrate	15·5	1·50	139·82	3·73	45·35	13·95	69·2	85·7	94·5	96·3	29·6
Zn concentrate...	20·8	0·13	8·44	0·10	0·85	24·10	8·0	6·9	3·4	2·4	68·5
Tailing.....	63·7	0·12	2·92	0·02	0·15	0·22	22·8	7·4	2·1	1·3	1·9

Screen Analysis Flotation Tailing, Test No. 7

Mesh	Weight, per cent
+ 65	0·0
− 65+100	2·9
−100+150	21·4
−150+200	20·0
−200	55·7

This test was a failure owing to the amount of gold left in the tailing. By cyanidation 61·5 per cent of the gold in the zinc concentrate was recovered giving an overall gold recovery of 74·1 per cent.

Test No. 10

In this test the same object was sought as in Test No. 7. The products were manipulated in the same way but the reagent combination and grinding period were different.

The ore at minus 14 mesh was ground for 25 minutes in a Denver rod mill.

Charge to rod mill:

Ore	2,000 grms.
Water	1,000 c.c.
Na_2CO_3	20·0 lb./ton
Minerec "A"	0·05 "
NaCN	0·3 "

Copper-lead float:

Cresylic acid	0·14 lb./ton

Concentrate recleaned without additional reagents.

Zinc float:

$CuSO_4$	0·5 lb./ton
Sodium ethyl xanthate	0·1 "
Cresylic acid	0·14 "

Copper-lead cleaner tailing was combined with zinc concentrate.

Results:

Product	Weight, per cent	Analyses					Recoveries, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Cu-Pb concentrate	12·4	1·40	169·36	4·25	53·5	8·20	44·9	80·2	87·2	90·3	14·5
Zn concentrate...	26·4	0·32	14·02	0·20	2·0	22·30	21·9	14·1	8·7	7·2	83·8
Tailing.....	61·2	0·21	2·43	0·04	0·3	0·2	33·2	5·7	4·1	2·5	1·7

This test was also a failure because the tailing contains too much gold to be discarded.

CYANIDATION

Test No. 4

This was a cyanidation test on the flotation tailing produced in Test No. 3. A sample of the tailing was agitated for 48 hours in solution containing 2 pounds per ton in KCN, with lime added at the rate of 10 pounds per ton of tailing. Pulp dilution was 3 : 1.

Results:

Product	Assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
Head.....	0·145	93·1	2·7	9·0
Tailing.....	0·010	.....	.....	.....

This, combined with the recovery in the flotation concentrate, gives an overall gold recovery of 97·7 per cent.

Test No. 8

The flotation tailing produced in Test No. 6 was agitated for 48 hours in a solution containing 2 pounds per ton in KCN. Pulp dilution was 3 : 1 and lime was added at the rate of 10 pounds per ton of tailing.

Results:

Product	Assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
Head.....	0·220	93·6	3·2	9·2
Tailing.....	0·015	.....	.....	.....

This, combined with the recovery in the flotation concentrate, gives an overall gold recovery of 96·5 per cent and corresponds closely with Test No. 4.

Test No. 9

The zinc concentrate produced in Test No. 7 was agitated for 72 hours, 3 : 1 dilution, in a solution containing 2 pounds KCN per ton. Lime was added at the rate of 16 pounds per ton of concentrate.

Results:

Product	Assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
Head	0·13	61·5	7·8	14·5
Tailing	0·05	.....	.....	.....

This material responds rather poorly to cyanidation and the total gold recovery is only 74·1 per cent combining the recovery by cyanidation with that in the copper-lead concentrate.

CONCLUSIONS

This ore contains too much copper to be treated by cyanidation in its raw state.

The methods followed in Tests Nos. 3 and 6 seem to be the most satisfactory, particularly No. 6, as the copper-lead concentrate has been greatly improved in grade by cleaning.

The erratic tails produced in Tests Nos. 1, 2, 5, 7, and 10 seem to show evidence of free gold in the ore, although microscopic examinations have failed to confirm this.

The high soda-ash consumption necessary for flotation will be somewhat of a handicap in treating this ore, but this possibly may be reduced when freshly mined ore is being milled.

Report No. 450

THE CONCENTRATION OF COPPER AND GOLD IN GREENE-STABELL ORE, DUBUISSON TOWNSHIP, QUEBEC

Shipment. A shipment of copper-gold ore contained in two boxes, weighing 155 pounds, was received on September 21, 1932, from the Greene-Stabell mine, Dubuisson Township, Quebec.

Characteristics of the Ore. The ore consists of stringers of pyrite, chalcopyrite, and minor sphalerite, with small amounts of gold and calaverite, associated with ferruginous dolomite. Gold, usually associated with calaverite, occurs also through the quartz gangue. Pyrrhotite is rare, and is disseminated in the quartz. The gold and calaverite are extremely fine, and appreciable amounts are present in grains far below 325 mesh in size.

Purpose of Experimental Tests. The object of the investigation was to determine the method most suited for the recovery of the values contained in the ore. As the ore contains appreciable amounts of copper, concentration would seem the process to apply. The tests showed that fine grinding is essential, 94 per cent of the copper and 82 per cent of the gold can be recovered by flotation when grinding is 85 to 90 per cent minus 200 mesh. The ratio of concentration is 16 : 1.

Sampling and Analysis. The entire shipment was crushed to 14 mesh and quartered. By further crushing and grinding to succeeding finer sizes with cuts through a riffle sampler, a representative portion minus 100 mesh was obtained for analysis. This showed the shipment to contain: copper, 0·48 per cent; gold, 0·60 ounce per ton, and silver, 2·32 ounces per ton.

Experimental Tests

Tests were made to note the amount of gold recovered by amalgamation, amalgamation followed by flotation and flotation followed by blanket concentration. Flotation tests were made with various reagent combinations to determine the most suitable conditions for high recovery.

AMALGAMATION

Test No. 1

A sample of the ore ground dry to pass 48 mesh was amalgamated in a 1 : 1 pulp, the amalgam separated and the tailing assayed.

Results:

Heads	Au	0·60 oz./ton
Amalgamation tailing	Au	0·29 "
Recovery		51·7 per cent

Screen Analysis

Mesh	Weight, per cent	Assay, Au, oz./ton
+ 65	6·4	0·44
− 65+100	13·9	0·43
−100+150	12·5	0·38
−150+200	10·1	0·35
−200	57·1	0·21

These results show the necessity of fine grinding to liberate the gold.

Test No. 2

A similar test was made on material ground to pass 100 mesh.

Results:

Heads	Au	0·60 oz./ton
Amalgamation tailing	Au	0·25 "
Recovery		58·3 per cent

This again indicates the necessity of fine grinding as 51·7 per cent of the gold is recovered from minus 48-mesh material and 58·3 per cent from minus 100 mesh.

AMALGAMATION AND FLOTATION

Test No. 3

A sample of the ore was ground wet in a neutral circuit to pass 95 per cent through 200 mesh and amalgamated.

After removing amalgam, the pulp was conditioned with 0·06 pound Aerofloat No. 25, 0·06 pound sodium ethyl xanthate, and 0·06 pound pine oil per ton. A flotation concentrate was then removed.

Results:

Product	Weight, per cent	Assay		Metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·0	0·53	0·60	100·0	100·0
Amalgamation tailing	.....	.....	0·24	.....	60·0*
Flotation concentrate	5·58	7·99	2·61	84·0	24·3
Flotation tailing	94·42	0·09	0·10	16·0	15·7

*Recovered by amalgamation.

The results show that 60 per cent of the gold is recoverable by amalgamation from material ground 95 per cent minus 200 mesh. An additional 24·3 per cent is concentrated in a product containing 2·61 ounces gold per ton. Each 100 tons of ore milled yields 5·58 tons of this concentrate. A total recovery of 84·3 per cent by the two methods is obtained.

FLOTATION

Test No. 4—Neutral Circuit

The ore was ground 95 per cent minus 200 mesh in a ball mill together with 0·06 pound Aerofloat No. 25, and then conditioned in the flotation machine with 0·06 pound sodium xanthate. No soda ash was used in the circuit. A concentrate was then removed after the addition of 0·05 pound pine oil.

Results:

Product	Weight, per cent	Assay		Metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·00	0·51	0·58	100·0	100·0
Concentrate	6·28	7·15	7·20	87·3	77·5
Tailing	93·72	0·07	0·14	12·7	22·5

Ratio of concentration, 15·9 : 1.

Test No. 5—Alkaline Circuit

In this test, 3·0 pound soda ash per ton was added to the grinding mill. In all other respects the test is similar to Test No. 4.

Product	Weight, per cent	Assay		Metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·00	0·48	0·59	100·0	100·0
Concentrate	6·31	7·16	7·66	94·1	82·4
Tailing	93·69	0·03	0·11	5·9	17·6

The results indicate that soda ash is beneficial for recovery of both copper and gold. The ratio of concentration is the same as in the neutral circuit. The indicated recoveries are 94·1 per cent of the copper and 82·4 per cent of the gold.

Test No. 6

In this test, additional reagents were added to the circuit to note the effect of a tougher froth; 0·12 pound coal-tar creosote was added to the grinding mill. Other reagents and conditions were the same as in Test No. 5.

Product	Weight, per cent	Assay		Metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·00	0·51	0·60	100·0	100·0
Concentrate	6·56	7·34	7·44	94·5	81·3
Tailing	93·44	0·03	0·12	5·5	18·7

Analysis of flotation concentrate: Au 7·44 oz./ton; Ag 4·06 oz./ton; Cu 7·34 per cent; Fe 30·9 per cent; S 28·7 per cent; SiO_2 17·46 per cent.

The additional reagents used in this test do not improve the results secured in Test No. 5.

Test No. 7

The effect of sodium sulphide was next investigated, 1·0 pound per ton of this reagent being added to a sample of ore treated as in Test No. 6.

Product	Weight, per cent	Assay		Metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·00	0·51	0·57	100·0	100·0
Concentrate	6·58	7·16	6·92	92·7	80·2
Tailing	93·42	0·04	0·12	7·3	19·8

Analysis of flotation concentrate: Au 6·92 oz./ton; Ag 3·30 oz./ton; Cu 7·16 per cent; Fe 30·1 per cent; S 28·4 per cent; SiO_2 18·9 per cent.

Sodium sulphide does not increase recoveries.

Test No. 8

A sample of the ore was floated as in Test No. 5 and the flotation tailing passed over a corduroy blanket set at a slope of 2 inches in 1 foot.

Product	Weight, per cent	Assay		Metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·00	0·47	0·57	100·0	100·0
Flotation concentrate	6·08	7·33	7·40	93·9	79·3
Flotation tails (cal.)	.....	0·03	0·12	.....	.....
Blanket concentrate	3·51	0·05	0·78	0·4	4·8
Blanket tailing	90·41	0·03	0·10	5·7	15·9

Analysis of flotation concentrate: Cu 7·33 per cent; Fe 36·9 per cent; S 29·6 per cent; SiO_2 15·0 per cent; Au 7·40 oz./ton; Ag 3·40 oz./ton.

The introduction of blankets after flotation results in an additional recovery of 4·8 per cent of the gold. A ratio of concentration by blanketing of 28·5 : 1 on the original feed is recorded. This concentrate has a value of $15.60 per ton.

The flotation results secured in this test are not equal to those of a parallel test, Test No. 5, due no doubt to manipulation of the test machine.

COMPARATIVE GRINDING

Three tests were made to note the results obtained from samples of the ore ground to different degrees of fineness.

Test No. 9

A sample of the ore was ground 70 per cent solids with soda ash 3·0 pounds, Aerofloat No. 25, 0·06 pound, coal-tar creosote 0·12 pound for 10 minutes in a porcelain mill containing iron balls. After removing the balls, the pulp was transferred to a flotation machine and conditioned for 5 minutes with 0·06 pound sodium ethyl xanthate and floated with 0·06 pound pine oil per ton.

Results:

Mesh	Weight, per cent
+ 65	20·0
− 65+100	17·0
−100+150	11·4
−150+200	8·5
−200	43·1

Product	Weight, per cent	Assay		Metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·00	0·52	0·60	100·0	100·0
Concentrate	6·51	7·08	5·92	89·1	64·2
Tailing	93·49	0·06	0·23	10·9	35·8

Test No. 10

This test is the same as Test No. 9 with the time of grinding increased to 20 minutes.

Results:

Mesh	Weight, per cent
+ 65	0·5
− 65+100	2·4
−100+150	9·3
−150+200	13·2
−200	74·6

Product	Weight, per cent	Assay		Metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·00	0·51	0·57	100·0	100·0
Concentrate	5·94	7·78	6·42	90·8	67·2
Tailing	94·06	0·05	0·20	9·2	33·0

Test No. 11

A test similar to the two preceding ones was made with the grinding time increased to 30 minutes. The concentrate secured from this test was cleaned once to note the effect of this operation.

Results:

Mesh	Weight, per cent	Assay, Au, oz./ton
+150	4·2	0·26
−150+200	9·3	0·22
−200	86·5	0·095

Product	Weight, per cent	Assay		Metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·00	0·50	0·57	100·0	100·0
Concentrate	4·81	9·43	9·00	91·2	76·2
Cleaner tailing	2·05	0·75	1·58	3·1	5·7
Flotation tailing	93·14	0·03	0·11	5·7	18·1

Analysis of flotation concentrate: Au 9·00 oz.; Ag 4·38 oz.; Cu 9·43 per cent; Fe 31·5 per cent; S 37·0 per cent; SiO_2 7·52 per cent.

The screen analysis of the flotation tailing shows the necessity of fine grinding for a good recovery of gold; 81·9 per cent of the metal is recovered in the rougher concentrate. When this is cleaned, the cleaner tailing contains 1·58 ounces gold per ton.

Test No. 12

A sample was ground and floated as in Test No. 11, and the flotation tailing passed over a corduroy blanket. The flotation concentrate was not cleaned.

Results:

Product	Weight, per cent	Assay		Metals, per cent	
		Cu, per cent	Au, oz./ton	Au	Cu
Heads (cal.)	100·00	0·49	0·60	100·0	100·0
Flotation concentrate	6·47	7·09	7·08	94·5	76·4
" tails (cal.)	.....	0·03	0·14	.....	.....
Blanket concentrate	4·69	nil	0·74	.....	5·7
" tails	88·84	0·03	0·12	5·5	18·1

The blanket concentrate contains $14.80 per ton and represents 5·7 per cent of the gold in the ore.

A screen analysis of the blanket tailing shows:

Mesh	Weight, per cent	Au, oz./ton
+150	5·2	0·24
−150+200	9·3	0·225
−200	85·5	0·10

The need for fine grinding is again indicated.

SUMMARY AND CONCLUSIONS

1. Fine grinding is essential for highest recoveries. This is shown by a comparison of tests made under similar conditions with varying grinds.

2. An alkaline circuit yields higher recoveries than a neutral one.

3. Amalgamation followed by flotation and flotation followed by blanket concentration give the same recovery.

4. A simple reagent combination such as Aerofloat, sodium xanthate, and pine oil will yield standard results. Further work at the property may establish a more suitable combination.

5. Cleaning the rougher concentrate has a tendency to drop out the gold. If the cleaner tailing is returned to the circuit there is the probability of gold building up in the circuit, eventually increasing tailing losses. A rougher concentrate carrying 7 per cent copper and 6 to 8 ounces in gold per ton is the average product that may be expected.

6. The grade of concentrate yielding the highest profit can best be determined by the operators. Freight, smelter charges, tailing losses, and other costs can only be established at the mill.

The following comparison of concentrate analyses shows the grade of product that can be made.

Test No.	Concentrate analyses					
	Au, oz./ton	Ag, oz./ton	Cu, per cent	Fe, per cent	S, per cent	SiO_2 per cent
6	7·44	4·06	7·34	30·9	28·7	17·5
7	6·92	3·30	7·16	30·1	28·4	18·9
8	7·40	3·40	7·33	36·9	29·6	15·0
11	9·00	4·38	9·43	31·5	37·0	7·5

Test No. 11, in which the rougher concentrate was cleaned, shows the effect of this operation. There is a decided drop in the silica content and an increase in the gold and copper together with a proportional increase in sulphur.

The recommended method of treatment is amalgamation of the ball mill discharge, followed by flotation producing a rougher concentrate of shipping grade and recovering 84 per cent of the gold and 94 per cent of the copper.

An alternative method is flotation followed by blanket concentration. This blanket concentrate will doubtless be less bulky and higher grade than those shown in the tests. This product may be either returned to the grinding mill, or if sufficiently high in free gold, treated in an amalgam barrel.

The mill tailing contains approximately $2.00 in gold per ton. Further work will be necessary to determine if this can be recovered by cyanidation.

Report No. 451

THE RECOVERY OF GOLD FROM ORE AND MILL TAILING OF THE HORSESHOE MINES, LIMITED, KENORA, ONT.

Shipment. A shipment weighing 3,930 pounds consisting of 16 bags of ore and 14 bags of mill tailings was received August 17, 1932, from the Horseshoe Mines, Limited, Kenora, Ontario.

Characteristics of the Ore. The gangue consists of grey to white quartz with locally admixed greenish-grey material, probably chloritic, through which a considerable amount of fine carbonate is disseminated.

Pyrite is the one metallic mineral present in appreciable amount. It is disseminated throughout the gangue as irregular grains and well formed cubes.

Chalcopyrite occurs as exceedingly minute grains within the pyrite and pyrrhotite occurs more rarely in the same manner.

Extremely fine magnetite remnants were observed in the gangue, the original grains being at present almost completely altered.

Gold was not positively identified, but one minute yellow grain in the gangue reacted to KCN; in repolishing the specimen this grain was removed so further tests were impossible.

Grain size of the pyrite.

Mesh	Per cent
−325	0·2
+325−200	0·9
+200−150	1·6
+150−100	3·2
+100− 65	6·6
+ 65− 48	13·5
+ 48− 28	74·0
Total	100·0

It will be seen from the above that relatively coarse grinding should free most of the pyrite.

Experimental Tests

On sampling both lots by standard methods, the 16-bag lot of mine ore was found to contain 0·575 ounce gold per ton. The tailing contained 0·12 ounce gold per ton.

A series of tests was made to establish the most suitable method for the recovery of the contained values. The investigation included amalgamation, flotation and cyanidation, separately and in combination. The results indicate that 63 per cent of the gold can be recovered by amalgamation of minus 48-mesh material. Cyanidation of the tails from this operation raises the recovery to 92 per cent. Straight cyanidation at minus 200 mesh gives an extraction of 94·8 per cent.

The detailed tests follow.

Mine Ore

AMALGAMATION AND CYANIDATION

Test No. 1

Samples of the ore were ground dry to pass 48 and 100 mesh and amalgamated. The tailings from this treatment were agitated for 48 hours, 1: 3 dilution, with a 1·0 pound KCN per ton solution. Lime 8 pounds per ton was added to maintain protective alkalinity.

Mesh	Amalgamation			Cyanidation			Total recovery
	Head, Au, oz./ton	Tailing, Au, oz./ton	Recovery, per cent	Agitation, hours	Tailing, Au, oz./ton	Recovery, per cent	
48	0·575	0·21	63·5	24	0·045	28·7	92·2
48	.....	0·21	63·5	48	0·05	27·8	91·3
100	.....	0·18	68·7	24	0·04	24·4	93·1
100	.....	0·18	68·7	48	0·05	22·6	91·3

A screen analysis of the amalgamation tailing from the 48-mesh test shows:

Mesh	Weight, per cent	Assay, Au, oz./ton
− 48+ 65	3·97	0·34
− 65+100	17·64	0·26
−100+150	16·56	0·24
−150+200	14·52	0·20
−200	47·31	0·17

This test shows that over 60 per cent of the gold is liberated at minus 48-mesh grinding, and a large amount of the gold not recovered by amalgamation is readily soluble in cyanide solution.

AMALGAMATION AND FLOTATION

Test No. 2

A sample of the ore ground to pass 48 mesh was amalgamated and after separating amalgam floated with 6·0 pounds soda ash, 0·07 pound Aerofloat No. 25, 0·10 pound sodium ethyl xanthate, and 0·05 pound pine oil per ton.

Results:

Product	Weight, per cent	Gold, oz./ton	Gold, per cent
Heads	100·00	0·575	100·0
Amalgamation tailing	.....	0·23	.....
Flotation concentrate	9·85	1·78	30·6
" tailing	90·15	0·06	9·4
Recovery by amalgamation	.....	.....	60·0

Ratio of concentration, 10·1 : 1.

This process gives a total recovery of 90·6 per cent of the gold, 60 per cent of which is recovered as amalgam and the remaining 30·6 per cent in a pyrite concentrate assaying 1·78 ounces gold per ton.

Test No. 3

This test is the same as the preceding one except that, after amalgamation, the tailing was reground to 90 per cent through 200 mesh.

Results:

Product	Weight, per cent	Gold, oz./ton	Gold, per cent
Heads	100·00	0·575	100·0
Amalgamation tailing	.....	0·24	.....
Flotation concentrate	8·53	2·46	36·2
" tailing	91·47	0·035	5·5
Recovery by amalgamation	.....	.....	58·3

Ratio of concentration, 11·7 : 1.

Regrinding the amalgamation tailing before flotation gives a total recovery of 94·5 per cent of the gold. An increase in the ratio of concentration, 11·7:1 as against 10·1:1, and a higher grade concentrate are noted.

AMALGAMATION, FLOTATION, AND CYANIDATION

Test No. 4

A sample of the ore was treated as in Test No. 3. The flotation concentrate was cyanided for 72 hours, 1:3 dilution with KCN 5·0 pound per ton solution. Lime, 10 pounds per ton of ore, was added for protective alkalinity.

Results:

Product	Weight, per cent	Gold, oz./ton	Gold, per cent
Heads	100·00	0·575	100·0
Amalgamation tailing	.....	0·22	.....
Flotation concentrate	7·81	2·36	.....
" " cyanided	.....	0·26	3·6
" tailing	92·19	0·035	5·7
Recovery by amalgamation	.....	.....	61·7
" cyanide	.....	.....	29·0

Ratio of concentration, 12·8 : 1.

In this test, 61·7 per cent of the gold was recovered by amalgamation and 32·6 per cent in the flotation concentrate; 89 per cent of the gold in this concentrate was extracted by cyanidation, a total recovery by the combined processes of 90·7 per cent. A cyanide consumption of 7·8 pounds per ton of concentrate was noted.

FLOTATION

Test No. 5

The ore was ground in a ball mill together with 6·0 pounds soda ash and 0·07 pound Aerofloat No. 25 per ton until 92 per cent passed 200 mesh. The pulp was then floated with 0·10 pound sodium ethyl xanthate and 0·05 pound pine oil.

Results:

Product	Weight, per cent	Gold, oz./ton	Gold, per cent
Heads	100·0	0·52	100·0
Concentrate	9·1	5·04	87·9
Tailing	90·9	0·07	12·1

Ratio of concentration, 11·0 : 1.

Highest recoveries are not secured by straight flotation.

CYANIDATION

Test No. 6

Samples of the ore were ground dry to pass 48, 100, 150, and 200 mesh and cyanided for 48 hours, 1:3 dilution, with a 1·0 pound per ton KCN solution; 8 pounds lime per ton ore was added to each lot.

Results:

Mesh	Agitation period, hours	Assay, Gold, oz./ton		Extraction, per cent	Reagent consumption, lb./ton	
		Heads	Tailing		KCN	CaO
− 48	24	0·575	0·065	88·7	0·21	4·0
− 48	48	0·575	0·055	90·4	1·02	5·2
−100	24	0·575	0·04	93·0	0·66	4·3
−100	48	0·575	0·045	92·2	1·11	5·4
−150	24	0·575	0·035	93·9	0·87	5·0
−150	48	0·575	0·045	92·2	1·23	6·0
−200	24	0·575	0·03	94·8	0·69	4·9
−200	48	0·575	0·035	93·9	1·26	6·3

These results indicate that extremely fine grinding is not necessary to secure high extractions, 90 per cent of the gold is recovered from minus 48-mesh material, and 94·8 per cent from minus 200-mesh.

Test No. 7

A series of tests was made to note if fouling of solutions took place. A 7-cycle cyanide test in which the cyanide solution from one lot of ore agitated for 24 hours was used on the following batch of fresh ore showed no falling off in recoveries.

MILL RUN No. 1

The ore ground minus 20 mesh was fed at the rate of 109 pounds per hour to a 12 inch by 24 inch rod mill in closed circuit with a classifier. The mill discharge passed over a short amalgamating plate before entering the classifier. The classifier overflow was pumped over a second amalgamating plate. The tailing from this flowed to a conditioning tank and thence to the first cell of a 6-cell flotation machine where a finished concentrate was taken off. The rougher concentrate from cells 2 to 6 was returned to head of the unit.

Soda ash, 3 pounds, and sodium ethyl xanthate, 0·10 pound per ton, were added to the conditioning tank, pine oil 0·06 pound was added to the cells.

Classifier overflow:

Mesh—	Weight, per cent
+ 48	0·2
− 48+ 65	1·3
− 65+100	3·3
−100+150	9·6
−150+200	13·1
−200	72·5

Product—	Au, oz./ton	
Feed	0·51	
Mill discharge	0·56	
Classifier feed	0·37	
" overflow	0·28	
Amalgamation plate tailing	0·175	
Flotation concentrate	4·22	Ag, 4·38 oz./ton
" tailing	0·08	

Results:

Recovered on short amalgamation plate	37·3	per cent
Held in mill and classifier	7·8	"
Recovered on second amalgamation plate	20·6	"
Recovered by flotation	18·9	"
Loss in flotation tailing	15·4	"
	100·0	

Ratio of concentration, 43·6 : 1.

This test was conducted under poor milling conditions. Coarse grinding was attempted with the result that the mill discharge had to be diluted to move the pulp over the short amalgamating plate. This resulted in a dilute classifier overflow, 17 to 20 per cent solids, making poor flotation conditions.

Cyanide tests on the amalgamating plate tailing containing 0·175 ounce gold per ton, gave a tailing of 0·02 ounce gold. Cyaniding the reground product yielded a tailing of 0·015 ounce or an extraction of 91·4 per cent, 30·4 per cent of the values in the feed to the mill.

MILL RUN No. 2

For this run the weight of rods in the mill was increased and the feed rate cut to 94 pounds per hour. Other conditions were the same as for the preceding run.

Classifier overflow:

Mesh—	Weight, per cent
+ 65	0·2
− 65+100	2·4
−100+150	10·1
−150+200	19·8
−200	67·5
	100·0

Product—	Au, oz./ton
Feed	0·54
Mill discharge	0·605
Classifier feed	0·535
" overflow	0·30
Amalgamation plate tailing	0·19
Flotation concentrate	2·14
" tailing	0·04

Results:

Recovered on short amalgamation plate	13·0	per cent
Held in classifier and mill	31·4	"
Recovered on second amalgamation plate	20·4	"
Recovered by flotation	28·3	"
Loss in flotation tailing	6·9	"
	100·0	"

Ratio of concentration, 14 : 1.

The grinding in this run produced less coarse material and less minus 200 mesh. Flotation conditions were improved, resulting in a tailing of 0·04 ounce gold per ton. More gold remained in the classifier and mill.

Cyanidation of the reground amalgamation plate tailing reduced the values to 0·03 ounce gold, being an extraction of 24·1 per cent of the gold in the mill feed. Cyaniding the unground product left a residue of 0·06 ounce.

Both these mill runs show that gold is trapped in the classifier and grinding mill. To get conclusive results from tests of this kind it would be necessary to run continuously for some weeks until this trapped gold began to spill over the classifier overflow.

CYANIDATION OF FLOTATION CONCENTRATE

The concentrates produced by Mill Runs No. 1 and No. 2 were combined. A representative portion was agitated for 72 hours, 1 : 3 dilution, with a 5·0 pound per ton KCN solution. Lime was added to maintain protective alkalinity at the rate of 7 pounds per ton ore.

Results:

Heads	Au	2·44	oz./ton
Tailing	Au	0·30	"
Recovery		87·7	per cent
Reagent consumption	KCN	4·7	lb./ton
	CaO	6·7	"

Cyanidation of the concentrate reground to pass 93 per cent through 200 mesh left a residue of 0·26 ounce—an extraction of 89·3 per cent.

Assuming that in Mill Run No. 2 all the gold held in the mill and classifier would be recovered either on the amalgamating plates, or in flotation concentrate, the recoveries made by this process supplemented by cyanidation of the flotation concentrate would be 90·1 per cent.

Mill Tailing

Tests similar to those made on the mine ore were undertaken on the shipment of tailings. Sampling and assaying showed the lot to carry 0·12 ounce gold per ton.

A screen analysis of the material follows:

Mesh	Weight, per cent	Assay, Au, oz./ton
+ 35	6·1	0·14
− 35+ 48	18·2	0·10
− 48+ 65	28·5	0·10
− 65+100	12·5	0·09
−100	34·7	0·135

AMALGAMATION

Test No. 1

Tests by amalgamation did not lower the value of the product. Little, if any, free gold is present.

CYANIDATION

Test No. 2

Samples of the tailing as received were agitated 48 hours 1 : 2·5 dilution with a KCN solution 1·0 pound and lime 7 pounds per ton of ore.

These tests reduced the gold content to 0·03 ounce, an extraction of 75 per cent.

Test No. 3

Grinding in cyanide solution and agitating 1 : 2·5 dilution with a 1·0 pound KCN solution gave the following results:

Heads: Au, 0·12 oz./ton.

Per cent −200 mesh	Agitation period, hours	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption	
				KCN	CaO
67·5	24	0·02	83·3	0·10	5·6
62·5	48	0·025	79·2	0·10	6·6
86·9	24	0·02	83·3	0·2	5·8
85·3	48	0·015	87·5	0·2	6·7

By grinding to 85 per cent minus 200 mesh and agitating for 48 hours a residue containing 30 cents gold per ton is obtained.

FLOTATION

Test No. 4

A sample of the tailing was ground 58 per cent minus 200 mesh together with 6 pounds soda ash and 0·07 pound Aerofloat No. 25 per ton. A concentrate was removed following the addition of 0·10 pound sodium ethyl xanthate and 0·05 pound pine oil.

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Heads	100·00	0·12	100·0
Concentrate	2·34	3·10	59·8
Tailing	97·66	0·05	40·2

Ratio of concentration, 42·8 : 1.

MILL RUN

The tailing was fed at the rate of 138 pounds per hour to the mill and classifier used in previous runs together with 6 pounds soda ash per ton. The classifier overflow passed to the conditioning tank and thence to the flotation machine; 0·24 pound xanthate was added to the conditioning tank and 0·05 pound cresylic acid used to produce a froth.

Classifier Overflow:

Mesh—	Weight, per cent
+150	12·5
−150+200	21·4
−200	66·1

Results:

Heads	Au, 0·125	oz./ton
Flotation concentrate	Au, 2·25	"
Flotation tailing	Au, 0·03	"
Recovery	77·1	per cent
Ratio of concentration	23·4 : 1	

CYANIDATION OF FLOTATION CONCENTRATE

Regrinding the concentrate to 90 per cent minus 200 mesh and agitating for 72 hours, 1 : 2·5 dilution, with a 5-pound KCN solution gave an extraction of 88·3 per cent. Cyanide consumption was 6·6 pounds per ton.

Flotation followed by cyanidation of the concentrate does not yield as high a recovery as straight cyanidation.

SUMMARY AND CONCLUSIONS

1. Amalgamation recovers between 60 and 68 per cent of the gold when the ore is ground 48 to 100 mesh. This coupled with cyanidation of the amalgamation tailing results in an overall recovery of 92 to 93 per cent of the values.

2. Amalgamation followed by flotation of the reground tailing, as shown in Test No. 3, gives a total recovery of 94·5 per cent.

3. Amalgamation, flotation, and cyanidation of the concentrate gives 90·7 per cent recovery—Test No. 4.

4. Straight cyanidation of the ore ground to pass 48 mesh gives an extraction of 90·4 per cent; when ground minus 200 mesh, 94·8 per cent —Test No. 6.

5. Cyanidation of the reground mill tailing gives an extraction of 87·5 per cent.

6. Flotation as made in the mill run gives 77·1 per cent recovery.

7. An extraction of 88 per cent of the gold in the flotation concentrate by cyanide is possible.

Straight cyanidation of minus 200-mesh ore yields the highest recovery. However, as there is considerable free gold in the ore, amalgamation followed by regrinding and cyaniding is preferable. Recoveries should be equal to those from straight cyanidation.

Flotation produces a considerable weight of concentrate, a ratio of concentration of from 11:1 to 20:1 is indicated. This product must be either cyanided or shipped.

The mill tailing, apart from the fact that little free gold is present, responds in the same manner as the ore. This material may therefore be added to the tailing from the amalgamating plates entering the cyanide circuit.

Amalgamation followed by cyanidation apparently is the most suitable method. Corduroy blankets may be introduced in place of amalgamating plates. This will necessitate the operation of a clean-up barrel. The residues from this may be passed into the cyanide plant. This concentrate will be of considerable bulk, due to the appreciable amount of sulphides in the ore.

Grinding mills having low discharges should be installed, as much gold will be held in the mill and classifier. When starting the plant, much concern will be occasioned if the classifier return and well are not examined to determine if the gold not recovered as bullion is not held in the grinding circuit.

Report No. 452

TESTS ON GOLD ORE FROM THE HIGHLAND ENTERPRISE COMPANY, BARREN LAKE, MANITOBA

Shipment. A consignment of 5 bags of ore, weighing 520 pounds, was received August 26, 1932. The shipment, which was forwarded from Ingolf, Ontario, by A. A. Fraser, 303 Huron and Erie Building, Portage Avenue, Winnipeg, Manitoba, was said to come from the company's holdings on the west side of Barren Lake, southeastern Manitoba.

Characteristics of the Ore. The ore as received consisted of surface material containing much oxidized matter. Some vein filling of white quartz was present and schisted country rock.

Samples of the ore were selected and two sections polished and studied under the microscope with a view to determining the minerals present and their grain size.

The gangue is composed essentially of glassy grey quartz. The metallic minerals are disseminated rather irregularly throughout this gangue. Besides native gold, only pyrite and arsenopyrite were observed. The gold was rare in the sections and where seen occupied a position in the pyrite. No grains of gold larger than 200 mesh were observed.

This microscopic examination was made on the quartz vein matter and not on the oxidized schisted portion of the shipment.

Grain Analyses. The following table shows the grain sizes of the sulphides for the various meshes, with percentages by volume for each size.

Mesh	Pyrite, per cent	Arseno-pyrite, per cent	Total, per cent
− 14+ 20	45·9	.....	45·9
− 20+ 28	8·7	17·1	25·8
− 28+ 35	7·3	3·6	10·9
− 35+ 48	3·2	3·2	6·4
− 48+ 65	2·9	2·9	5·8
− 65+100	1·8	2·0	3·8
−100+150	0·5	0·4	0·9
−150+200	0·3	0·2	0·5
−200	0·02	0·02	0·04
Totals	70·62	29·42	100·04

Purpose of Experimental Tests. The shipment was made to determine the most efficient method to apply for the recovery of gold.

Experimental Tests

After sampling the entire lot by standard methods, analysis showed the shipment to contain 0·68 ounce gold, 0·16 ounce silver per ton, and 0·17 per cent arsenic.

Tests showed that 98·5 per cent was recovered by amalgamation followed by cyanidation of the residues; 82 per cent was recovered by amalgamation alone. Flotation following amalgamation resulted in 93 per cent of the gold being recovered as amalgam and flotation concentrate.

The investigation included amalgamation, flotation and cyanidation separately and in combination.

AMALGAMATION AND CYANIDATION

Test No. 1

The ore ground dry minus 48 mesh was amalgamated and the residue agitated 1 : 3 dilution with a 1·0 pound KCN solution. Lime equal to 20 pounds per ton was added for protective alkalinity.

Results:

Heads.....Au, 0·68 oz./ton
Amalgamation tailing.....Au, 0·12 "
Recovery..... 82·2 per cent

Cyanidation of amalgamation tailing

Heads: 0·12 oz./ton.

Mesh	Agitation time, hours	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton	
				KCN	CaO
−48	24	0·025	79·2	1·20	17·3
−48	48	0·12	nil	2·39	16·4

This test shows an overall recovery in 24 hours by the combined processes of 96·3 per cent of the gold in the feed. A short-time agitation period is indicated as reprecipitation is noted.

Screen analysis of amalgamation tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
+ 65	2·8	0·16	3·7
− 65+100	23·6	0·17	33·2
−100+150	12·4	0·15	15·4
−150+200	11·1	0·12	10·5
−200	50·1	0·09	37·2

This screen analysis indicates that the gold not recovered by amalgamation is not freed or else is in combination with the sulphides. Finer grinding is indicated.

Test No. 2

A test on material ground to pass 100 mesh was made similar to the preceding one.

Results:

Heads.....Au, 0·68 oz./ton
Amalgamation tailing.....Au, 0·13 "
Recovery..... 81·3 per cent

Cyanidation of amalgamation tailing

Heads: Au, 0·13 oz./ton.

Agitation time, hours	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton	
			KCN	CaO
24	0·01	92·3	1·65	17·6
48	0·075	42·3	3·36	16·7

A total recovery of 98·6 per cent of the gold is obtained by the combined processes. A falling-off in cyanide extraction after 24 hours is again noticed.

STRAIGHT CYANIDATION

Test No. 3

Representative portions of the ore were crushed to pass 48, 100, 150, and 200 mesh and agitated 1 : 3 dilution with a KCN solution 1·0 pound per ton; 20 pounds of lime per ton was added to each test.

Results:

Heads: Au, 0·68 oz./ton.

Mesh	Agitation period, hours	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton	
				KCN	CaO
− 48	24	0·04	94·1	0·76	16·8
− 48	48	0·10	85·3	2·22	17·5
−100	24	0·01	98·5	0·90	17·4
−100	48	0·03	95·6	2·34	18·1
−150	24	0·01	98·5	0·76	17·7
−150	48	0·03	95·6	2·64	19·2
−200	24	0·005	99·3	1·20	18·2
−200	48	0·02	97·1	2·8	20·5

A decrease in extraction after 24 hours' agitation and an increase in cyanide consumption are again noted. The gold yields to cyanidation very readily.

AMALGAMATION AND FLOTATION

Test No. 4

A sample of the ore ground to pass 48 mesh was amalgamated and after removing amalgam, floated with 10·0 pounds soda ash, 0·07 pound Aerofloat No. 25, 0·10 pound sodium ethyl xanthate, and 0·05 pound pine oil per ton.

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Heads	100·00	0·68	100·0
Amalgamation tailing	.....	0·17	.....
Recovery by amalgamation	.....	.....	75·5
Flotation concentrate	4·26	2·90	18·2
" tailing	95·74	0·045	6·3

A total of 93·7 per cent of the gold is recovered in amalgam and in the flotation concentrate. The concentrate contained 2·90 ounces gold, 2·00 ounces silver per ton, and 3·4 per cent arsenic.

Test No. 5

A test similar to the preceding one was made, grinding the tailing after amalgamation until 92 per cent passed 200 mesh.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Heads	100·00	0·68	100·0
Amalgamation tailing	.....	0·15	.....
Recovery by amalgamation	.....	.....	77·9
Flotation concentrate	3·93	3·22	19·2
" tailing	96·07	0·02	2·9

Ratio of concentration, 25·4 : 1.

The flotation concentrate contains 3·22 ounces gold, 2·42 ounces silver per ton, and 2·65 per cent arsenic; 97·1 per cent of the gold is recovered by the two processes.

Regrinding the amalgamation tailing increases the recovery by flotation.

AMALGAMATION, FLOTATION AND CYANIDATION

Test No. 6

A sample of the ore was ground to pass 48 mesh and amalgamated. After removing amalgam, a flotation concentrate was taken off as in Test No. 5. This concentrate was cyanided 1 : 3 dilution with a KCN solution 5·0 pounds per ton for 72 hours; 20 pounds lime per ton of concentrate was added for protective alkalinity.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Heads	100·00	0·68	.....
Amalgamation tailing	.....	0·19	.....
Amalgam	.....	.....	72·1
Flotation concentrate	4·76	3·76	.....
" " cyanided	.....	0·12	0·8
Cyanide extraction	.....	.....	25·1
Flotation tailing	95·24	.....	2·0

These results show that 72·1 per cent of the gold is recovered by amalgamation. Flotation of the residues from this treatment produces 4·76 tons of concentrate from each 100 tons of ore milled; 96·8 per cent of the gold in this concentrate is extracted by cyanide, or 25·1 per cent of the values in the mill feed. A reagent consumption of 24·9 pounds KCN and 23 pounds lime per ton of concentrate is noted.

SUMMARY AND CONCLUSIONS

1. Amalgamation of minus 48-mesh material recovers 82 per cent of the gold. Cyanidation of the residues raises the recovery to 96 per cent.

2. Grinding to minus 100 mesh followed by amalgamation and cyanidation gives a recovery of 98·6 per cent.

3. Straight cyanidation at minus 48 mesh gives an extraction of 94·1 per cent, while minus 200-mesh material yields 99·3 per cent of its values.

4. Flotation alone is not efficient. Amalgamation and flotation recover 97 per cent of the gold. A concentrate containing 3·22 ounces gold and 2·42 ounces silver is obtained with a ratio of concentration of 25 : 1.

5. Amalgamation, flotation, and cyanidation of the concentrate give 97 per cent recovery.

Much of the gold is free and easily recovered by amalgamation. This, therefore, should be recovered before the ore is subjected to additional treatment.

The amalgamation tailing should be reground and cyanided to secure highest recoveries. This combination should give recoveries equal to those of straight cyanidation, 94 to 99 per cent, depending on the fineness of grinding.

If a minimum of equipment is to be installed, amalgamation followed by flotation and custom treatment of the concentrate is indicated as 80 per cent of the gold can be recovered by amalgamation and 97 per cent when the flotation concentrate is cyanided.

In all cases where the ore was cyanided, there were indications of precipitation of gold from solution after 24 hours' agitation together with a high consumption of lime and cyanide. The material tested was from the surface of the ore-body. Before any decision is made on the installation of a cyanide plant, fresh clean ore taken from below the oxidized zone should be tested to determine its true reactions to cyanidation. Less free gold also may be found below the surface.

Report No. 453

RECOVERY OF GOLD FROM BLACK SAND CONCENTRATE FROM ROCK CREEK PLACER DEPOSITS, BRITISH COLUMBIA

Shipment. A small sample of black sand, weighing 715 grammes, was received November 12, 1932, from R. D. Kerr, Midway, B.C.

Characteristics of Sample. The sample consisted of black sand concentrate containing gold. A microscopic examination showed that the gold was fine, but quite granular and rough.

EXPERIMENTAL TESTS

It is impossible to get check results on black sand concentrates by the ordinary method of assaying. In this case, the small amount of material recovered made it necessary to combine the method of recovering the gold with the determination of the amount of gold in the sample.

The sample was found to contain 3·92 ounces of gold per ton, the equivalent of about $78.40.

The recovering of the gold from this black sand concentrate is relatively simple. The method recommended is barrel amalgamation. The sample consisting of 715 grammes of concentrate was placed in a small amalgam barrel and half its weight of water added together with mercury, equivalent to 10 per cent of the weight of the charge of black sand. The barrel was allowed to revolve for 1 hour after which the mercury was separated by washing the charge over a Long Tom sluice box constructed with a pocket at the head of the box to collect the mercury. In practice the riffles in the Long Tom catch the fine globules of broken-up mercury and any gold not amalgamated. The material caught in the riffles is panned to collect the mercury, which is best accomplished by the addition of fresh mercury and a little cyanide of soda or lye.

Summary of Results:

Amount of sample used	715 grms.
Amount of gold extracted by the mercury	96·08 milligrams
Amount of gold remaining in black sand	trace
Recovery by barrel amalgamation	99·9 per cent

Assay of sample calculated from the gold in the products is

$$\frac{96{\cdot}08 \times 29{\cdot}166}{715} = 3{\cdot}92 \text{ oz./ton}$$

Barrel amalgamation is comparatively a simple process and one not requiring any elaborate or expensive equipment. The barrel consists of a small-batch ball mill with a door for charging and cleaning out. A charge of black sand, depending on the size of the barrel, from 100 pounds to a ton, is placed in the barrel with water and mercury and rotated for about one hour, or longer if found necessary, and then dumped into a tray which overflows into a small Long Tom or sluice box. A hose is turned into the tray and the sand washed away from the mercury. The mercury is then drawn off through a plug hole in the bottom of the tray and squeezed through a chamois skin to separate the amalgam. The amalgam is then retorted.

A home-made barrel can be constructed from an empty gasoline drum placed on rolls which can be driven with a 3 h.p. gasoline engine or a small water-wheel. A small barrel for the individual miner can be made from a 5-gallon oil drum or wooden wine cask; paddles can be attached to the cask and by fixing trunnions to each end it can be made to revolve by the current of water in the sluice box. Another way the individual miner can amalgamate his black sand is to purchase a small grinding muller used in an assay office for grinding small samples of rock, or he can buy a cast iron mortar and pestle and grind the black sand with mercury and water. In each case he would pan his amalgamated sand to recover the mercury.

Report No. 454

THE RECOVERY OF GOLD FROM THE ORE OF VENTURES LIMITED, ISLAND LAKE PROPERTY, MANITOBA

Shipment. Five lots of ore, total weight 3,200 pounds, were received between April 19 and September 6, 1932, from the properties of Ven-

tures Limited, Island Lake, Manitoba. All but a small part of these shipments came fron the "Jack of Diamonds" claims. A small 5-bag lot was from the "Jack of Hearts" claims, some 10 miles distant.

Characteristics of the Ore. The ore consisted of schisted siliceous material together with white quartz heavily mineralized with sulphides of iron, lead, zinc, and copper.

The metallic minerals, in their order of deposition, are pyrite; galena, freibergite; chalcopyrite, gold, and sphalerite. Thus the silver values may be expected with galena, the gold values with an association of chalcopyrite and sphalerite and within the latter mineral.

The spectrographic analyses indicated, first, no metals are to be expected in the pyrite; second, the silver occurs mainly in freibergite usually within the galena; third, that probably most or all of the gold occurs in the native state in the sphalerite; and fourth, small amounts of cadmium are present in the freibergite and sphalerite.

Experimental Tests

R. C. Mott, metallurgist for Ventures Limited, personally conducted much of the test work assisted by the staff.

The investigation showed that the ore from the Jack of Hearts property contained 2 per cent copper and did not respond to the same treatment as that from the Jack of Diamonds. This report deals solely with the ore from the latter.

It was found that 60 per cent of the gold was freed at minus 65-mesh grinding and 84 per cent at minus 200-mesh. The gold can be amalgamated in the laboratory but large-scale tests show that in continuous operation the mercury has a tendency to sicken.

Removal of the coarse gold by traps or blankets leaves a residue that can be cyanided with a total recovery of over 90 per cent. When the coarse gold is not removed, cyanidation fails to show an economic recovery.

Blanket concentration recovers 60 to 80 per cent of the gold in a bulky concentrate with a ratio of concentration of over 18 : 1. Barrel amalgamation of this concentrate recovers over 90 per cent of the contained gold.

Each lot was sampled and analysed. Toward the close of the investigation all Jack of Diamonds ore was mixed and used for larger scale continuous runs. This combined lot is called No. 6. The analyses of the various shipments are as follows:—

Lot No.	Description	Weight, pound	Assay							
			Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Fe, per cent	As, per cent	Insol., per cent
1	Jack of Diamonds.....	660	1·43	2·97	0·18	3·35	2·75	3·55	.....	.....
2	" "	420	1·40	3·96	0·28	3·63	2·95	.....	.....	.....
3A	" "	820	1·72	4·50	0·28	1·74	2·14	.....	0·25	.....
3B	Jack of Hearts.....	.....	2·47	10·04	2·00	1·64	0·57	.....	0·06	.....
4	Jack of Diamonds.....	970	0·68	1·86	0·07	0·90	1·50	3·50	.....	86·9
5	" "	340	0·46	1·42	0·09	0·47	1·80	.....	.....	.....
6	" "	.....	1·05	2·23	0·16	1·71	1·54	2·89	.....	85·8

Preliminary small-scale tests were made on Lots Nos. 1 and 2. The results secured indicated a line of procedure which was checked and augmented during the main investigation made on Lots Nos. 3A, 4, 5, and 6.

AMALGAMATION AND FLOTATION

Test No. 1A

In this test, two 1000-gramme samples of the ore were ground 60 per cent solids in a jar mill with mercury. Amalgam and excess mercury were separated from the ground pulp and the tailing treated by flotation.

Reagents to cells—Soda ash	3·0	lb./ton
Aerofloat No. 25	0·15	"
Cresylic acid	0·07	"

Results:

Product	Weight, per cent	Assay				Gold, per cent
		Au, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	
Amalgam						79·4
Flotation concentrate	8·35	2·04	1·61	12·52	11·85	16·3
" tailing	91·65	0·05			0·60	4·2

Ratio of concentration, 12·0 : 1.

Test No. 1B

In this test, flotation was carried out in a high-lime circuit, an attempt being made to depress gold from the flotation concentrate.

Reagents to cells—Lime	10·0	lb./ton
Aerofloat No. 25	0·15	"
Cresylic acid	0·07	"

Product	Weight, per cent	Assay				Gold, per cent
		Au, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	
Amalgam						75·7
Flotation concentrate	9·3	1·365	1·10	7·44	6·05	12·2
" tailing	90·7	0·14	0·15	0·38	1·10	12·1

Flotation in a high-lime circuit results in 4·1 per cent less gold reporting the flotation concentrate than in that of the preceding test.

AMALGAMATION AND CYANIDATION

Test No. 2

In this test, 1000-gramme samples of the ore were amalgamated 1 : 1 dilution in a jar mill. After removing mercury and amalgam, the residues were cyanided 1 : 3 dilution with 2 pounds KCN per ton of solution, 8 pounds lime per ton of ore was added for protective alkalinity.

Screen Analysis of Amalgamation Tailing

Mesh	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
+ 65	10·5	1·36	19·5
− 65+100	27·2	0·85	31·5
−100+150	11·8	0·75	12·5
−150+200	10·8	0·71	10·4
−200	39·7	0·48	26·4

Results:

—	Amalgamation			Agitation, hours	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton		Gold, per cent		
	Au, oz./ton		Recovery, per cent				KCN	CaO	Amalgamation	Cyanidation	Total
	Heads	Tailing									
A	1·72	0·73	57·6	24	0·17	76·7	3·0	5·9	57·6	32·5	90·1
B	1·72	0·73	57·6	48	0·07	90·4	3·0	6·1	57·6	38·3	95·9

Cyanidation of the amalgamation tailing recovers 90 per cent of the values in the product.

STRAIGHT CYANIDATION

Test No. 3

Samples of the ore ground to pass different meshes were agitated 1 : 3 dilution with a KCN solution of 2 pounds per ton; 8 pounds lime per ton of ore was added to each test.

Mesh	Agitation, hours	Au, oz./ton		Extraction, per cent	Reagent consumption, lb./ton	
		Heads	Tails		KCN	CaO
− 48	24	1·72	1·63	5·2	.....	5·8
− 48	48	1·72	1·47	14·5	.....	6·0
−100	24	1·72	1·28	25·6	2·4	6·8
−100	48	1·72	1·15	33·1	2·4	6·8
−150	24	1·72	0·92	46·5	2·6	6·9
−200	24	1·72	0·81	52·9	2·7	7·1
−200	48	1·72	0·65	62·2	2·7	7·4

Two additional tests were made grinding the ore to 95 per cent minus 200 mesh, 1 : 1 dilution, with a KCN solution 4 pounds per ton, and 8 pounds lime per ton of ore. The pulp was then diluted to 1 : 2·5 and the strength of solution raised to 5 pounds KCN per ton.

Agitation, hours	Au, oz./ton		Extraction, per cent
	Heads	Tails	
48	1·72	0·61	64·5
72	1·72	0·38	77·9

Straight cyanidation gives poor recoveries.

BLANKET CONCENTRATION

Test No. 4

In this test, the ore was ground in a jar mill loaded with iron balls until 70 per cent passed 200 mesh. After thickening, the pulp was fed over a corduroy blanket set at 3 inches in 12 inches slope.

This test was made on Lot 3A.

Product	Weight, per cent	Assay		Metals, per cent	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Heads (cal.)	100·0	1·69	3·71	100·0	100·0
Blanket concentrate	5·5	23·74	13·78	77·1	20·5
" tailing	94·5	0·41	3·12	22·9	79·5

Ratio of concentration, 18·2 : 1.

Test No. 5

This test is a continuation of blanket concentration coupled with cyanidation of the blanket tailing.

A sample of the ore was ground as in the preceding test until 60 per cent passed 200 mesh and then passed over blankets set at 3 inches in 12 inches slope.

The blanket tailing was de-watered and agitated 24 hours, 1 : 2 dilution with a KCN solution of 2·5 pounds per ton. Five pounds lime per ton of ore was added for protective alkalinity.

This test was made on Lot No. 6.

Product	Weight, per cent	Assay, oz./ton		Metals, per cent	Gold extraction, per cent
		Au	Ag		
Blanket concentrate	5·56	12·58	10·20	66·8	66·8
" tailing (cal.)	.....	.....	.....	33·2	.....
Extracted by cyanide	.....	.....	.....	.....	24·2
Cyanide tailing	94·44	0·10	.....	.....	91·0

Ratio of concentration, 17·8 : 1.

A test similar to the preceding one was made with the slope of the blankets set at 2 inches per foot instead of 3 inches.

Product	Weight, per cent	Assay, oz./ton		Metals, per cent	Gold extraction, per cent
		Au	Ag		
Blanket concentrate	8·4	9·86	9·24	79·0	79·0
" tailing (cal.)	.....	0·24	.....	21·0	.....
Extracted by cyanide	.....	.....	.....	.....	12·4
Cyanide tailing	91·6	0·09	.....	.....	.....
					91·4

Ratio of concentration, 11·9 : 1.

When the slope of the blankets is decreased, more concentrate is retained accompanied by an increase in gold recovery. The over-all recovery is, however, the same as in the preceding test.

Removing the bulk of the coarser particles of gold allows cyanidation to make a fair extraction of the remainder.

Test No. 6

In this test, a sample of the ore ground 60 per cent minus 200 mesh was passed over blankets set at a slope of 3 inches in 12 inches. The concentrate secured was ground 1 : 1 dilution in a porcelain mill containing a light ball load together with 2 pounds of lime per ton of ore and a small quantity of mercury.

Product	Weight, per cent	Assay, Au, oz./ton	Metals, per cent	Recovery, per cent
Blanket concentrate	9·9	7·06	70·0	.....
Blanket concentrate amalgamation	.....	.....	.....	67·2
Blanket concentrate amalgamation tailing	.....	0·30	.....	.....
Blanket tailing	90·1	0·35	30·0	.....

A recovery of 96 per cent of the gold in the blanket concentrate was made by grinding with mercury.

TRAPS

Test No. 7

A sample of the ore ground wet to pass 52 per cent through 200 mesh was treated in a hydraulic classifier to determine what recoveries could be expected in gold traps. The classifier tailing was then cyanided.

Product	Weight, per cent	Assay, oz./ton		Metals, per cent	Gold extraction, per cent
		Au	Ag		
Heads (cal.)	100·00	1·05	2·23	100·0	.....
Classifier concentrate No. 1	1·60	25·10	11·01	38·1	.....
Classifier concentrate No. 2	1·92	12·24	15·55	22·7	.....
Classifier concentrate total	3·52	18·05	.....	60·8	60·8
Classifier tailing cyanide heads	.....	0·429	.....	39·2	.....
Cyanide tailing	96·48	0·10	.....	9·2	.....
Cyanide extraction	.....	.....	.....	.....	30·0
					90·8

These results indicate that 60 per cent of the gold can be recovered in traps, usually placed at the discharge end of the grinding mill.

AMALGAMATION IN GRINDING CIRCUIT

Test No. 1

Ten per cent by weight of mercury was added to samples of the ore while grinding 66 per cent solids in a rod mill. The amalgam and excess mercury were removed and the residue assayed.

Mesh				Assay, Au, oz./ton		Recovery, per cent
+100	+150	+200	−200	Heads	Tailing	
8·1	21·6	17·6	52·7	1·05	0·245	76·7
3·4	20·0	17·8	58·8	1·05	0·28	73·4
1·3	17·3	19·0	62·2	1·05	0·20	81·0

Lime was added to the second test at the rate of 1·0 pound per ton ore.

LARGE-SCALE MILL RUNS

The following tests were made on Lot No. 6—a combination of all Jack of Diamonds shipments, the head sample of which assayed, Au, 1·05 oz./ton; Ag, 2·23 oz./ton; Cu, 0·16 per cent; Pb, 1·71 per cent; Zn, 1·54 per cent; Fe, 2·89 per cent; insoluble, 85·78 per cent. These larger scale continuous runs were made to check the results obtained in the laboratory.

Mill Run No. 1

Ore crushed to minus 20 mesh, fed at the rate of 110 pounds per hour to a 12-inch by 24-inch rod mill in closed circuit with a classifier. Dry lime, 1 pound per ton, was mixed with the feed. Before entering the classifier, the rod mill discharge passed over a short amalgamating plate. The classifier overflow also passed over a second amalgamating plate before entering a conditioning tank where 0·1 pound per ton sodium ethyl xanthate and 0·08 pound cresylic acid were added. Contact was approximately 15 minutes. The tank discharged into the first cell of a 6-cell flotation machine where a finished concentrate was taken off. The rougher concentrate from the last five cells was returned to the head of the machine.

Large samples of the amalgamation plate tails, flotation concentrate, and flotation tails were taken on which cyanide tests were made.

Samples for screen tests were taken during the run. The feed to the secondary amalgamation and flotation averaged 63·8 per cent minus 200 mesh.

Screen Analyses:

Mesh	Rod mill discharge		Classifier overflow	
	Weight, per cent	Cumulative	Weight, per cent	Cumulative
+100	7·3	.....	2·8	.....
−100+150	18·2	25·5	14·6	17·4
−150+200	21·8	47·3	18·8	36·2
−200	52·7	100·0	63·8	100·0

Results:

Product	Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent
Feed	1·10	3·07	.....	.....	.....
Rod mill discharge	1·49	.....	.....	.....	.....
Primary amalgamation tailing	1·215	.....	.....	.....	.....
Classifier overflow	0·485	.....	.....	.....	.....
Secondary amalgamation tailing	0·445	.....	.....	.....	.....
Flotation concentrate	8·96	56·32	3·88	36·42	20·80
" tailing	0·13	0·42	0·05	0·23	0·88

Distribution of Gold. From the above table, it will be seen that a large percentage of the total gold was retained in the grinding circuit. The classifier overflow assayed 0·485 ounce per ton; thus 0·615 ounce per ton, or 55·9 per cent of the total gold in the ore remained in the circuit. However, 0·275 ounce per ton, or 25·0 per cent of the total was recovered on the short plate between the mill discharge and classifier.

On completion of the three mill runs, sands remaining in the mill and classifier were cleaned up, sampled, assayed and tested in the laboratory for recovery of gold. In a run continuous for some weeks, this gold would unload from the circuit.

—	Weight, pound	Total gold in feed, per cent
Feed	440·0	100·0
Mill and classifier clean-up	.....	30·9
Recovered on primary plate	.....	25·0
" "secondary plate	.....	3·6
" in flotation concentrate	15·4	28·6
Values in flotation tailing	.....	11·9

Laboratory Testing of Mill Products. The following table summarizes the laboratory results obtained when treating various mill products for final extraction of gold.

Product	Treatment	Au, oz./ton		Extraction, per cent
		Heads	Tailing	
Classifier overflow	24-hr. cyanidation	0·485	0·195	59·8
Amalgam tailing	24-hr. cyanidation	0·445	0·16	64·0
Flotation concentrate	42-hr. cyanidation	8·96	4·12	54·4
" "	Regrind +48-hr. cyanidation	8·96	2·40	73·2
" tailing	24-hr. cyanidation	0·13	0·02	84·6
*Mill clean-up	Amalgam +24-hr. cyanidation	.....	.....	80·0

*The mill and classifier were cleaned up at the end of Mill Run No. 3. The 80·0 per cent extraction figure obtained in Mill Run No. 3 has been applied to the gold retained in the grinding circuit in Mill Runs Nos. 1 and 2.

Calculated Recoveries. Combining results in the mill test and by further treatment of products in the laboratory, the following table outlines the gold recovery to be expected in a continuous mill operation when conditions similar to those in Mill Run No. 1 are maintained.

Product	Treatment				Recovery, per cent
	Amalgamation		Cyanidation		
	Gold in product, per cent	Gold in ore, per cent	Gold in product, per cent	Gold in ore, per cent	
Amalgam from first plate	.....	25·0	.....	.....	.....
" " second plate	.....	3·6	.....	.....	.....
" " mill clean-up	69·2	21·4	.....	.....	.....
Amalgam total	.....	.....	.....	.....	50·0
Flotation concentrate	.....	.....	73·2	20·9	.....
" tailing	.....	.....	84·6	10·1	.....
Mill clean-up amalgamation tails	.....	.....	80·0	7·6	.....
Extraction by cyanidation	.....	.....	.....	.....	38·6

An additional 7·7 per cent of the total gold in the ore remained in the cyanided flotation concentrate. This residue, amounting to $3\frac{1}{2}$ pounds per 100 pounds of ore treated, assayed; Au, 2·40 oz./ton; Cu, 3·88 per cent; Pb, 36·4 per cent; Zn, 20·8 per cent. Further extraction would necessitate very fine grinding and prolonged agitation coupled with a high consumption of cyanide.

Mill Run No. 2

In this test, the same flow-sheet was used as in the preceding run. An attempt was made to raise the recovery of gold by amalgamation by frequent dressing of the plates. The amount of lime added to the ore was increased with a view to keeping the plates from staining rapidly and also to depress gold from the flotation concentrate, a very small amount of which was removed.

Grinding. The feed rate was reduced to 105 pounds per hour with the result that a finer grind was obtained as shown in the following table.

Mesh	Weight, per cent
+100	2·1
−100+150	10·7
−150+200	15·5
−200	71·7

Reagents:

Lime	5·0	lb./ton to rod mill
Sodium ethyl xanthate	0·05	"
Cresylic acid	0·04	"

Results:

Product	Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent
Feed	0·98				
Rod mill discharge	1·285				
Primary amalgamation tailing	0·80				
Classifier overflow	0·41				
Secondary amalgamation tailing	0·41				
Flotation concentrate	8·08	42·26	2·22	25·85	21·80
" tailing	0·32	1·59		0·96	1·12

Distribution of Gold. In Mill Run No. 2, 0·57 ounce per ton or 58·1 per cent of the total remained in the grinding circuit; 49·5 per cent was recovered on the primary amalgamation plate and 8·6 per cent remained in the mill and classifier.

Product	Weight, pounds	Total gold in feed, per cent
Feed	420·0	100·0
Mill and classifier clean-up		8·6
Recovered on primary plate		49·5
" " secondary plate		
" in flotation concentrate	5·45	10·7
Values in flotation tailing		31·2

Flotation concentrate carried 10·7 per cent of the total gold as against 28·6 per cent in the preceding run.

Laboratory Testing of Mill Products. Laboratory cyanide tests were carried out on products from the mill run.

Product tested	Treatment	Au, oz./ton		Extraction, per cent
		Heads	Tailing	
Classifier overflow	24-hr. cyanidation	0·41	0·18	56·1
Amalgamation tailing	24-hr. "	0·41	0·19	53·6
Flotation tailing	24-hr. "	0·32	0·02	93·7

Recoveries:

Product	Treatment				Recovery, per cent
	Amalgamation		Cyanidation		
	Gold in product, per cent	Gold in ore, per cent	Gold in product, per cent	Gold in ore, per cent	
Amalgamation from first plate		49·5			
" " second plate					
" " mill clean-up	69·2	5·95			
Amalgamation total					55·45
Flotation concentrate			73·2	7·8	
" tailing			93·7	29·2	
Mill clean-up amalgamation tailing			80·0	2·1	
Extraction by cyanide					39·10
					94·55

The residue remaining after cyanidation of the flotation concentrate contained 2·9 per cent of the total gold in the ore.

A consumption of 32·0 pounds KCN per ton of concentrate treated was recorded.

Mill Run No. 3

In this run, the short amalgamation plate in the grinding circuit was replaced by a mechanically agitated gold trap and the secondary plates by a corduroy blanket set at a slope of $2\frac{1}{2}$ inches in 12 inches. During the run, the pulp was fed to the blanket at 15 per cent solids. Laboratory cyanide tests were made on the mill products.

Product	Assay, Au, oz./ton
Feed	0·94
Rod mill discharge	1·43
Trap overflow	1·17
Classifier overflow	0·44
Blanket tailing	0·35
" concentrate	17·00
Trap cleanings	18·35
Mill clean-up	9·025

Feed to the gold trap and overflow differed by 0·16 ounce. Recovery at this point would be increased by feeding the trap a more dilute pulp.

—	Weight, lb.	Total gold in feed, per cent
Feed	420·0	100·0
Mill and classifier clean-up	.....	25·6
Gold trap	5·94	27·6
Blanket concentrate	2·23	9·6
" tailing	.....	37·2

It is seen that 0·50 ounce of gold per ton, or 53·2 per cent of the total gold was retained in the grinding circuit.

Laboratory Testing of Products. As in Runs Nos. 1 and 2, products were tested in the laboratory with the results shown below.

Product tested	Treatment	Au, oz./ton		Extraction, per cent
		Heads	Tailing	
Blanket feed	24-hr. cyanidation	0·44	0·04	90·9
" tailing	24-hr. "	0·35	0·035	90·0
Mill clean-up	Amalgamation +24-hr. cyanidation	9·025	1·80	80·0

Recoveries:

Product	Total gold in product, per cent	Recovery of gold in product, per cent	Recovery of total gold, per cent
Trap cleanings	27·6	88·6	33·0
Blanket concentrate	9·6		
Mill clean-up	25·6	80·0	20·5
Blanket tailing	37·2	90·0	33·4
			86·9

SUMMARY AND CONCLUSIONS

1. Straight cyanidation of the ore is not applicable; 5 per cent extraction of the gold at minus 48-mesh grind and 78 per cent at minus 200-mesh may be expected.

2. Mill Run No. 2 shows the highest recorded recoveries. Amalgamation recovers 55·45 per cent of the gold. Cyanidation of the plate tailing accounts for an additional 23·9 per cent, a total by the two processes of 79·4 per cent. However, when a small amount of flotation concentrate, ratio of concentration 77 : 1, is removed from the plate tailing a marked improvement is noted. Prolonged cyanidation of the flotation concentrate and a short treatment of the tailing coupled with prior amalgamation, raises the overall recovery to 94·55 per cent.

3. Much of the gold can be recovered by a system of traps. In continuous operation, free gold can be expected to gradually unload from the grinding circuit, eventually to be caught by amalgamating plates or blankets.

Blanket concentration accounts for an additional 9·6 per cent, leaving 37·2 per cent in the tailing. Cyanidation of this product extracts 90 per cent of the contained values. Barrel amalgamation of the blanket concentrate would recover approximately 90 per cent of its gold.

It would be well in the proposed mill to make provision for amalgamation of either the ball mill discharge or classifier oversize. In Mill Run No. 2, 49·5 per cent of the gold was recovered by amalgamating the rod mill discharge.

This ore, because of its location and complexity, presents a difficult metallurgical problem. Shipment of concentrate to a smelter cannot be considered owing to the bulk of material and excessive freight charges. This makes it necessary to recover the gold as bullion at the property. Straight cyanidation is not applicable due to low extractions, 55 per cent of the gold is free and under ideal conditions can be recovered by amalgamation. However, some mineral in the ore, possibly an antimony compound, stains the plates, making it necessary to give the mercury and amalgam constant attention. The removal of a small amount of concentrate makes it feasible to cyanide the remainder of the ore after the coarser particles of gold have been removed.

The suggested initial procedure for a pilot mill is a trap or amalgamating plate between the mill and classifier, or in the classifier return.

To care for gold spilling over the classifier overflow when the grinding circuit is loaded, secondary amalgamating plates or blankets should be installed over which the pulp may flow before entering a flotation unit, where a minimum of concentrate should be taken off. Flotation tailing should then be cyanided for approximately 24 hours. If blankets are used, the concentrate obtained here can be barrel amalgamated, together with any periodic cleanings from the gold trap or classifier. The residues from this may be cyanided with the reground flotation concentrate. It will be well to keep the solutions of this circuit apart from those of the main cyanide plant treating the flotation tailing, as fouling will soon appear.

Ore from the Jack of Hearts property must be treated in a separate section of the mill. If mixed with Jack of Diamonds ore serious complications will arise.

Report No. 455

EXPERIMENTAL TESTS ON THREE SAMPLES OF GOLD ORE FROM CEDAR ISLAND, SHOAL LAKE, LAKE OF THE WOODS DISTRICT

Shipment. Three shipments of the ore were received, one on June 13, 1932, one on August 8, 1932, and one on October 13, 1932. These were designated as Shipments No. 2, No. 3, and No. 4 respectively. The samples were submitted by Geo. H. Ince, Secretary-Treasurer, Kenora Prospectors and Miners, Limited, 100 Adelaide Street West, Toronto, Ontario.

Characteristics and Analyses of the Samples. The three samples were similar in outward appearance except that considerably more free gold could be seen in the third shipment than in the second or fourth.

In all cases the mineralization seemed to be chiefly pyrite and some chalcopyrite in a siliceous gangue. Average head sample assays were as follows:—

	Au, oz./ton	Ag, oz./ton
Shipment No. 2	1·435	0·35
" " 3	5·07	0·38
" " 4	3·41	0·62

Experimental Tests

All three shipments were sampled and treated separately. Small-scale tests only were made on Shipments Nos. 2 and 3, and larger scale mill runs only on Shipment No. 4.

By amalgamation recoveries of 68·6, 76·3, 74·9, and 69·3 per cent respectively were obtained on samples of Shipment No. 2, ground to −48, −100, −150 and −200 mesh, and 48-hour cyanidation tests on the same four samples gave recoveries of 96·0, 96·9, 97·9, and 98·3 per cent respectively.

On samples of Shipment No. 3, ground to the same sizes, recoveries of 67·9, 81·3, 72·4, and 75·4 per cent were obtained by amalgamation; and by cyanidation for 24 hours the corresponding recoveries were 95·9, 98·2, 98·6, and 98·7 per cent. By agitating for 48 hours a slight increase in extraction was noted, the recoveries then being 96·5, 98·4, 99·0, and 98·9 per cent respectively.

Flotation tests were made on samples of amalgamation tailings from Shipments Nos. 2 and 3. High-grade concentrates were produced in both

cases, but the flotation tailings were high enough to warrant blanket concentration or some other treatment if amalgamation and flotation should be adopted as steps in the process.

On Shipment No. 4 three flow-sheets were used, namely:—

(1) Traps and blankets.
(2) Traps and flotation.
(3) Amalgamation plates and flotation.

Average gold recoveries resulting from these were 67·1, 88·0, and 93·9 per cent respectively.

Details of the tests follow:—

Shipment No. 2

AMALGAMATION

Tests Nos. 1 to 4

In these tests 500 grammes of the ore at the following sizes, −48, −100, −150, and −200 mesh, was amalgamated for 30 minutes with 50 grammes of mercury. The tailings were filtered, washed, and assayed for gold.

Results:

Head sample: Au, 1·435 oz./ton.

Test No.	Mesh	Tailing, Au, oz./ton	Recovery, per cent
1	− 48	0·45	68·64
2	−100	0·34	76·31
3	−150	0·36	74·91
4	−200	0·44	69·34

CYANIDATION

Tests Nos. 1 to 4

In these tests 200 grammes of ore from each of the four lots used for the amalgamation tests described above was agitated for 48 hours in 600 c.c. of solution running 2 pounds KCN per ton with lime added at the rate of 10 pounds per ton ore. The tailings were filtered, washed, and assayed for gold.

Results:

Head sample: Au, 1·435 oz./ton.

Test No.	Mesh	Tailing, Au, oz./ton	Recovery,	Reagents consumed, lb./ton	
				KCN	CaO
1	− 48	0·058	95·96	1·20	7·50
2	−100	0·045	96·86	1·50	7·00
3	−150	0·03	97·71	1·50	7·33
4	−200	0·025	98·26	1·80	8·67

Screen Analysis:

Mesh	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Average tailing
+100	18·76	0·12	38·87	0·058
+200	27·58	0·07	33·34	.....
−200	53·66	0·03	27·79	.....

AMALGAMATION, FLOTATION, AND CYANIDATION

Test No. 5

In this test 2,000 grammes of the ore at minus 150 mesh was amalgamated for 30 minutes with 200 grammes of mercury in 1 : 1 pulp. The amalgamation tailing was transferred to a flotation machine and the following reagents added:—

	Lb./ton
Soda ash	2·0
Aerofloat No. 25	0·07
Sodium ethyl xanthate	0·1
Pine oil	0·05

The flotation concentrate assayed 1·75 oz./ton in gold and the tailing 0·07 oz./ton in gold. By cyanidation 96·3 per cent of the gold in flotation concentrate was extracted, leaving a cyanidation tailing assaying Au, 0·065 oz./ton.

Shipment No. 3

AMALGAMATION

Tests Nos. 1 to 4

In this series, as on Shipment No. 2, amalgamation tests were made on four samples of the ore ground to −48, −100, −150, and −200 mesh. In each case 500 grammes of the ore was amalgamated for 30 minutes in 1 : 1 pulp with 50 grammes mercury. The tailings were filtered, washed, and assayed for gold.

Results:

Head sample: Au, 5·07 oz./ton.

Test No.	Mesh	Tailing, Au, oz./ton	Recovery
1	− 48	1·63	67·85
2	−100	0·95	81·26
3	−150	1·40	72·39
4	−200	1·25	75·35

CYANIDATION

Tests Nos. 1 to 4

In this series cyanidation tests were made on four lots of the ore ground to −48, −100, −150, and −200 mesh. Agitation was continued for 24 hours in 3 : 1 dilution with KCN 1·95 pound per ton to start with. The tailings were filtered, washed, and assayed for gold.

Results:

Head sample: Au, 5·07 oz./ton.

Test No.	Mesh	Tailing, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
				KCN	CaO
1	− 48	0·21	95·85	1·05	6·0
2	−100	0·09	98·22	1·05	6·3
3	−150	0·07	98·62	1·05	7·3
4	−200	0·065	98·72	1·50	7·5

Tests Nos. 5 to 8

This series of tests was the same as Tests Nos. 1 to 4 except that agitation was carried on for 48 hours instead of 24 hours.

Results:

Head sample: Au, 5·07 oz./ton.

Test No.	Mesh	Tailing, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
				KCN	CaO
5	− 48	0·18	96·45	1·05	6·5
6	−100	0·08	98·42	1·05	6·8
7	−150	0·05	99·01	1·35	7·7
8	−200	0·055	98·92	2·10	8·0

AMALGAMATION, FLOTATION, AND CYANIDATION

Test No. 9

In this test 2,000 grammes of the ore was amalgamated for 30 minutes in 1 : 1 pulp with 200 grammes of mercury. The amalgamation tailing was floated with the following reagents:—

	Lb./ton
Soda ash	3·0
Aerofloat No. 25	0·07
Sodium ethyl xanthate	0·1
Pine oil	0·05

The flotation concentrate, weighing 255·3 grammes, assayed Au, 4·08 oz./ton and the tailing assayed Au, 0·28 oz./ton. By cyanidation 94·7 per cent of the gold in the flotation concentrate was extracted, leaving a tailing assaying Au, 0·215 oz./ton.

Mill Runs

SHIPMENT No. 4.

These larger scale tests were carried out to check results obtained in batch laboratory testing. Five mill runs were completed with feed rates from 110 to 150 pounds per hour.

The grinding equipment consisted of one Denver rod mill in closed circuit with an Aikens classifier. This unit grinds 100 pounds per hour of the average gold ore from minus 12 mesh to 65 to 70 per cent minus 200 mesh.

In Run No. 1 Cedar Island ore was found to be more readily reduced than average ore. With a rod charge of 270 pounds in the mill and the feed set at 105 pounds per hour, 92 per cent minus 200 mesh was obtained. The charge of rods was reduced and the feed to the mill increased until in Run No. 5, with a rod load of 220 pounds and a feed rate of 150 pounds an hour, 73·3 per cent minus 200-mesh material was produced.

In these tests, three flow-sheets were used.

1. Traps and blankets.
2. Traps and flotation.
3. Amalgamation plates and flotation.

Small-scale tests on Shipments Nos. 2 and 3 proved conclusively that excellent extraction could be expected when cyaniding either the raw ore or a concentrate, hence cyanide tests were not carried out on Shipment No. 4.

Run No. 1

Flow-sheet: Traps and blankets.
Date of test: October 24.
Time of test: 8.15 a.m.-2.30 p.m.
Sampling period: 11.30 a.m.-2.30 p.m.
Feed rate: 105 pounds per hour.
Blanket slope: 2½ inches per foot.

Grinding: Screen analysis of classifier overflow showed.

Mesh	Weight, per cent
+100	0·2
−100+150	2·0
−150+200	5·6
−200	92·2

Assay of Products:

	Au, oz./ton
Feed	2·63
Mill discharge	2·93
Trap overflow	1·68
Classifier overflow	0·85
Blanket concentrate	37·72
Blanket tailing	0·775

Summary of Results:

Product	Total gold, per cent	Recovered by amalgamation,	Total gold recovered, per cent
Mill and classifier clean-up	20·2	92·2	18·6
Trap cleanings	47·5	94·3	44·8
Blanket concentrate	2·8	76·0	2·1
Tailing	29·5	.....	.....
Totals	100·0	.....	65·5

Run No. 2

Flow-sheet: Traps and blankets.
Date of test: October 25.
Time of test: 8.30 a.m.-2 p.m.
Sampling period: 10 a.m.-2 p.m.
Feed rate: 130 pounds per hour.
Blanket slope: 3½ inches per foot.

Screen Analysis of Classifier Overflow

Mesh	Weight, per cent
+100	1·5
−100+150	4·3
−150+200	7·3
−200	86·9

Assay of Products:

	Au, oz./ton
Feed	3·03
Mill discharge	3·11
Trap overflow	2·47
Classifier overflow	1·07
Blanket concentrate	48·46
" tailing	0·745

Results:

Product	Total gold, per cent	Recovered by amalgamation, per cent	Total gold recovered, per cent
Mill and classifier clean-up	43·5	92·2	40·1
Trap cleanings	21·1	94·3	19·9
Blanket concentrate	10·8	81·5	8·8
Tailing	24·6	.....	.....
Totals	100·0	.....	68·8

Run No. 3

Flow-sheet: Traps and flotation.
Date of test: October 26.
Time of test: 10.15 a.m. — 2.30 p.m.
Sampling period: 12.30 p.m. — 2.30 p.m.
Feed rate: 127 pounds per hour.

Screen Analysis of Classifier Overflow

Mesh	Weight, per cent
+100	1·6
−100+150	8·0
−150+200	90·4
−200	.....

Reagents:

	Lb./ton
Soda ash	1·0
Sodium xanthate	0·05
Cresylic acid	0·03

Assay of Products:

	Au, oz./ton
Feed	2·63
Mill discharge	2·74
Trap overflow	2·28
Classifier overflow	1·03
Rougher concentrate	39·80
Clean concentrate	61·58
Flotation tailing	0·20
†Amalgamated flotation concentrate	41·03

Summary:

Product	Total gold, per cent	Gold recovered from products by amalgamation, per cent	Total gold recovered in amalgam, per cent	Gold remaining in concentrate, per cent	Total recovery, per cent
Mill and classifier clean-up	43·3	92·2	39·9	.....	.....
Trap cleanings	17·5	94·3	16·5	.....	.....
Flotation concentrate	31·6	33·2	9·9	.....	.....
" tailing	7·6	.....	.....	.....	.....
Totals	100·0	.....	66·3	†21·7	88·0

Run No. 4

Flow-sheet: Amalgamation and flotation.
Date of test: November 7.
Time of test: 8.15 a.m. — 3 p.m.
Sampling period: 12 noon — 3 p.m.
Feed rate: 140 pounds per hour.

Screen Analysis:

Mesh	Weight, per cent
+100	0·5
−100+150	7·0
−150+200	11·9
−200	80·6

Reagents:

	Lb./ton
Soda ash	5·0
Sodium xanthate	0·10
Cresylic acid	0·07

Assay of Products:

	Au, oz./ton
Feed	2·95
Mill discharge	3·61
Plate overflow	2·34
Classifier overflow	1·02
Flotation concentrate	19·23
Flotation tailing	0·14
Amalgamated flotation concentrate	13·04

Summary:

Product	Total gold, per cent	Recovery by amalgamation, per cent	Total recovered in amalgam, per cent	Total recovery, per cent
Mill and classifier clean-up	22·4	92·2	20·6	.....
Amalgam	43·0	.....	43·0	.....
Flotation concentrate	29·8	32·2	9·6	.....
Tailing	4·8	.....	.....	.....
Totals	100·0	.....	73·2	93·4

Run No. 5

Flow-sheet: Amalgamation and flotation.
Date of test: November 8.
Time of test: 9 a.m.-3 p.m.
Sampling period: 10.30 a.m. — 3 p.m.
Feed rate: 150 pounds per hour.

Grinding:

Screen Analysis

Mesh	Weight, per cent
+100	3·3
−100+150	11·4
−150+200	12·0
−200	73·3

Reagents:

	Lb./ton
Soda ash	3·0
Sodium xanthate	0·075
Cresylic acid	0·07

Assay of Products:

	Au, oz./ton
Feed	3·94
Mill discharge	4·33
Plate overflow	2·85
Classifier overflow	1·29
Flotation concentrate	36·97
Flotation tailing	0·13
Mill and classifier clean-up	24·95

Summary:

Product	Total gold, per cent	Recovery by amalgamation, per cent	Total recovered in amalgam, per cent	Total gold remaining in concentrate, per cent	Total recovery, per cent
Mill and classifier clean-up	29·7	92·2	27·4	.....	.....
Amalgam	37·6	.....	37·6	.....	.....
Flotation concentrate	29·4	32·7*	9·5	.....	.....
Tailing	3·3	.....	.....	.....	.....
Totals	100·0	.....	74·5	19·9	94·4

*Average of recovery obtained in Runs Nos. 3 and 4.

In the tests reported, amalgamation recoveries of 65·5, 68·8, 66·3, 73·2, and 74·5 per cent were made—an average of 69·7 per cent.

High ratios of concentration were obtained by flotation of the amalgamation tails in Runs Nos. 3, 4, and 5. Flotation results from these runs are summarized below:—

Run	Au, oz./ton			Recovery, per cent		Ratio of concentration
	Amalgamation, tailing, flotation, feed	Flotation concentrate	Flotation tailing	Au in feed	Au in ore	
No. 3	1·03	61·58	0·20	80·6	31·6	*72·4 : 1
No. 4	1·02	19·23	0·14	86·2	29·8	21·8 : 1
No. 5	1·29	36·97	0·13	89·9	29·4	31·9 : 1

*Concentrate was cleaned in Run No. 3, with resulting higher ratio and lower recovery than in Runs Nos. 4 and 5.

In Run No. 1, 47·5 per cent of the gold was recovered in cleanings from a small trap. Recovery from the trap dropped to 21·1 and 17·5 in Runs Nos. 2 and 3, largely due to poor operation. Results of Run No. 1 can be duplicated or improved with a properly designed and operated trap.

SUMMARY AND CONCLUSIONS

Testing Cedar Island gold ore has established the following points:—

1. The ore is easy to grind. In the grinding plant increased capacity of 30 to 40 per cent, over average figures, might be expected.

2. A properly designed and operated gold trap will recover 40 to 50 per cent of the gold in a small bulk of concentrate. Barrel amalgamation of this product will recover 94 per cent of the contained gold.

3. Amalgamation will recover 65 to 70 per cent of the gold. Concentration of the amalgamation tails and barrel amalgamation of the concentrate will raise the total amalgamation recovery approximately 10 per cent. In Mill Runs Nos. 3, 4, and 5 this practice increased the amalgamation recovery by 9·7 per cent.

4. Flotation alone yields only fair recoveries and ratios of concentration. By this method 83·0 per cent of the gold was recovered in a concentrate that amounted to 3·1 per cent of the weight of the ore. (Ratio 32·4 : 1.

5. High recoveries have been obtained by combined treatments, traps and flotation, or amalgamation plates and flotation. Traps ahead of flotation offer the advantages of cheap installation and operation.

6. The ore responds readily to cyanide treatment. Extractions as high as 96 per cent were obtained in 24-hour treatment.

To obtain maximum extractions the ore should be cyanided. However, if the operation does not justify a cyanide installation, amalgamation and flotation will yield good recoveries. Seventy to seventy-five per cent of the gold will be recovered in bullion and the concentrate can be shipped or stored for treatment at a later date.

A simple flow-sheet would be to grind the ore in a ball mill and let it discharge onto a 4-mesh screen to take out the large oversize. The screen undersize would go to a hydraulic gold trap where the coarse gold would be caught. The overflow from the trap would go to a classifier in closed circuit with the ball mill. The classifier oversize would be returned to the ball mill for regrinding and the overflow would go to flotation. If desirable the flotation tailing could be passed over blankets and its gold content thus reduced.

The trap cleanings, flotation concentrate, and blanket concentrate would be treated in an amalgamation barrel and the amalgamation tailings cyanided, shipped, or impounded for future treatment.

Report No. 456

CONCENTRATION OF SURF POINT, PORCHER ISLAND, GOLD ORE

Shipments. Two shipments of ore from the Surf Point mine, Porcher Island, British Columbia, were received at the Ore Dressing and Metallurgical Laboratories. The first arrived August 8, 1932, and consisted of one sack weighing approximately 100 pounds, consigned by J. R. Williams, Vancouver, for the N. A. Timmins Corporation, Montreal, Que. The second, weighing 1,450 pounds, was shipped by freight from Prince Rupert, B.C., by R. E. Legg for the same corporation. This lot was received October 25, 1932.

Characteristics of the Ore. The ore consists of rather coarse-textured pyrite, grading locally to a fine-grained texture. The quartz gangue varies from a milky variety to a grey glassy type, and contains small amounts of a soft transparent mineral, possibly sericite.

A spectrographic analysis showed a strong indication of gold, a trace of silver, no copper and no tellurium in the pyrite. This evidence in addition to the apparent absence of any free gold or other mineral which might contain the metal would indicate rather conclusively that the gold is present in the pyrite in solid solution. The silver apparently accompanies the gold in the same relations.

Purpose of Experimental Tests. The object of the investigation was to determine the ratio of concentration and grade of product that could be obtained by flotation.

Experimental Tests

After sampling by standard methods, the two shipments were assayed and found to contain:—

Shipment No. 1	1·64 oz. gold/ton	0·86 oz. silver/ton
" No. 2	1·135 "	0·51 "

Tests were made on Shipment No. 1, but following advice from the shippers further work was delayed pending the receipt of the second.

The investigation showed that no free gold was present. Amalgamation failed to recover any of the values, 66 per cent can be extracted by cyanidation. Flotation recovered 97·6 per cent of the gold in a concentrate assaying 6·78 ounces gold per ton with a ratio of concentration 5·4 : 1.

All tests reported in this investigation were made on Shipment No. 2 and consist of flotation with various reagents and grindings.

Test No. 1

A sample of the ore was ground 66 per cent solids in a rod mill with 6 pounds soda ash, 0·12 pound Barrett No. 4, 0·2 pound sodium ethyl xanthate; 0·1 pound pine oil per ton was added to the flotation machine.

Screen Analysis:

Mesh	Weight, per cent
+ 48	0·02
− 48+ 65	2·30
− 65+100	14·87
−100+150	21·05
−150+200	12·93
−200	48·83
	100·00

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Heads	100·00	1·17	100·00
Flotation concentrate	16·32	7·00	97·85
" tailing	83·68	0·03	2·15

Test No. 2

In this test a coarser grind was made. The sample was ground 66 per cent solids with 10 pounds soda ash, and 0·04 pound Tarol No. 1. After transferring the pulp to a flotation machine, a concentrate was removed by the addition of 0·10 pound pine oil per ton.

Screen Analysis:

Mesh	Weight, per cent
+ 48	5·61
− 48+ 65	18·22
− 65+100	18·30
−100+150	14·64
−150+200	8·80
−200	34·43

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Heads	100·00	1·18	100·00
Flotation concentrate	16·86	6·78	97·2
" tailing	83·14	0·04	2·8

Test No. 3

In this test, grinding was a little coarser than in Test No. 2, 31·2 per cent passing 200 mesh. One pound per ton of a mixture, 10 per cent tar and 90 per cent Barrett No. 4, was added to the mill. Pine oil at the rate of 0·09 pound per ton was added to the flotation machine.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Heads	100·00	1·18	100·00
Flotation concentrate	21·20	5·42	97·33
" tailing	78·80	0·04	2·67

Test No. 4

A sample of the ore was ground 66 per cent solids with 0·2 pound per ton sodium xanthate until 34 per cent passed 200 mesh and floated with 0·04 pound Tarol No. 1, and 0·06 pound pine oil per ton.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Heads	100·00	1·27	100·00
Concentrate	20·60	5·96	96·87
Tailing	79·40	0·05	3·13

Samples of the concentrate and tailing were mounted, polished, and examined in the mineragraphic laboratory to determine the amounts of free and combined sulphide and gangue. This was done by traversing the mounted products under the microscope.

Results:

Flotation Concentrate from Test No. 4

(Percentages are by weight)

Grain size in mesh	Free sulphide, per cent	Middlings		Free gangue, per cent	Totals
		Sulphide, per cent	Gangue, per cent		
+ 48	5·90	.....	.....	.....	5·90
− 48+ 65	14·90	.....	.....	2·42	17·32
− 65+100	21·4	0·96	.....	1·22	23·58
−100+150	13·30	0·64	0·23	0·97	15·14
−150+200	7·30	.....	0·21	0·86	8·37
−200	23·20	.....	0·16	6·33	29·69
Totals	86·00	1·60	0·60	11·80	100·00
		2·20			

Flotation Tailing from Test No. 4

(Percentages are by weight)

Grain size in mesh	Free sulphide, per cent	Middlings		Free gangue, per cent	Totals
		Sulphide, per cent	Gangue, per cent		
+ 48	.....	.....	.....	2·3	2·3
− 48+ 65	.....	.....	.....	9·4	9·4
− 65+100	.....	.....	1·2	18·3	19·5
−100+150	.....	.....	0·8	15·2	16·0
−150+200	.....	.....	.....	13·7	13·7
−200	0·2	0·4	0·9	37·6	39·1
Totals	0·2	0·4	2·9	96·5	100·0
		3·3			

Minerals. In addition to pyrite and quartz, the only minerals reported previously, a few small grains of chalcopyrite were observed in the concentrate. This mineral is probably present as less than 0·001 per cent of the concentrate however.

MILL RUN

The ore crushed to minus 12 mesh was fed at the rate of 80 pounds per hour to a 12-inch by 24-inch rod mill in closed circuit with a classifier. The classifier overflow passed to a conditioning tank where the pulp was conditioned approximately 15 minutes. The tank discharged into the first cell of a 6-cell flotation machine where a finished concentrate was taken off. The rougher concentrate from the last five cells was returned to the head of the circuit.

Classifier Overflow:

Mesh	Weight, per cent
+ 48	3·9
− 48+ 65	4·9
− 65+100	15·2
−100+150	18·0
−150+200	12·1
−200	45·9

Reagents:

To rod mill, soda ash	6·0 lb./ton
To conditioning tank, sodium xanthate	0·10 "
To cells, pine oil	0·06 "

Assays:

Classifier overflow....Au,	1·28 oz./ton
Flotation concentrate....Au,	6·78 "
" tailing....Au,	0·038 "

Flotation Concentrate:

Fe	41·85 per cent	CaO	trace	As	0·06 per cent
S	46·15 "	MgO	2·17 per cent	Au	6·78 oz./ton
SiO_2	9·25 "	Te	0·025 "	Ag	2·88 "

Results:

Recovery of gold	97·6 per cent
Ratio of concentration	5·4 : 1

SUMMARY AND CONCLUSIONS

1. The ore is not amenable to either amalgamation or cyanidation.

2. Flotation concentration results in a recovery of 97·6 per cent of the gold with a ratio of concentration 5·4 : 1.

3. Coarse grinding is indicated. The following table is of interest:—

Test No.	Grind, per cent −200 mesh	Au, oz./ton in concentrate
1	48·8	7·00
2	34·4	6·78
3	31·2	5·42
Mill run	45·9	6·78

It appears that grinding coarser than 34 per cent minus 200 mesh tends to lower the grade of concentrate. Little difference is noticed in the assay of the concentrate when the grind is 34 to 49 per cent minus 200 mesh. Elimination of gangue middling will fix the high limit. The mineragraphic examination of the concentrate from Test No. 4, where the grind was 31·2 per cent minus 200 mesh, shows only 0·60 per cent gangue middling produced. There is, therefore, no object in grinding finer than 31 to 35 per cent minus 200 mesh.

The grade of concentrate can be raised by the elimination of free gangue in cleaner cells.

The method recommended for this ore is flotation of a coarsely ground product.

Report No. 457

TESTS ON GOLD ORE FROM WHITEWATER, TAKU RIVER AREA, BRITISH COLUMBIA

Shipment. Three sacks of ore marked Whitewater "A" and two marked Whitewater "B" were received at the Ore Dressing and Metallurgical Laboratories, August 8, 1932, from J. R. Williams, Vancouver, B.C. These were sent for the N. A. Timmins Corporation, Montreal, Quebec, and were said to come from the Taku River area, B.C.

Characteristics of the Ore. The ore consists of pyrite and arsenopyrite disseminated irregularly as patches and seamlets through a siliceous gangue with only minor amounts of associated carbonate. The arsenopyrite carried the gold, probably in solid solution. This mineral is finely divided, probably mostly minus 325 mesh; this will probably determine the degree of grinding to which the ore will have to be subjected.

Experimental Tests

The two shipments were sampled by standard methods. The assays of the lots are:—

Whitewater "A".....Au, 0·815 oz./ton; Ag, 0·14 oz./ton; As, 2·59 per cent
Whitewater "B".....Au, 0·82 " Ag, 0·10 " As, 0·88 "

The investigation disclosed that no appreciable recovery of gold was effected by either amalgamation or cyanidation. Flotation concentration recovered 77 to 82 per cent of the gold.

Whitewater "A"

Flotation. A sample of the ore, ground 66 per cent solids in a jar mill loaded with iron balls together with 0·12 pound coal-tar creosote and 6 pounds soda ash per ton of ore, was floated with 0·10 pound amyl xanthate and 0·06 pound pine oil per ton.

Product	Weight, per cent	Assay		Per cent of values	
		Au, oz./ton	As, per cent	Au	As
Heads	100·0	0·79	2·59	100·0	100·0
Concentrate	15·1	4·03	12·44	76·9	72·5
Tailing	84·9	0·215	.....	23·1	27·5

Ratio of concentration, 6·6 : 1.

Whitewater "B"

A test similar to the above was made on the sample marked Whitewater "B".

Product	Weight, per cent	Assay		Per cent of values	
		Au, oz./ton	As, per cent	Au	As
Heads	100·0	0·85	0·88	100·0	100·0
Concentrate	8·3	8·37	8·70	81·7	82·0
Tailing	91·7	0·17	.....	18·3	18·0

Ratio of concentration, 12 : 1.

SUMMARY AND CONCLUSIONS

As the ore is not amenable to amalgamation or cyanidation it will be necessary to concentrate the values in a product suitable for shipment to a smelter; 75 to 80 per cent of the gold can be recovered in a concentrate with a ratio of concentration of from 6 : 1 to 12 : 1, depending on the nature of the ore.

The presence of arsenic will result in some smelters refusing to handle the concentrates. Others will inflict penalties on the arsenic content.

Report No. 458

THE CONCENTRATION OF THE DISSEMINATED COPPER-NICKEL ORE OF THE FALCONBRIDGE NICKEL MINES, LIMITED, FALCONBRIDGE, ONTARIO

Shipment. A small preliminary shipment of disseminated copper-nickel ore was received on November 5, 1932, from the Falconbridge Nickel Mines, Limited, Falconbridge, Ontario. This was followed by a carload shipment of 54,700 pounds which was received on November 15, 1932.

Characteristics of the Ore. The shipments were representative of the disseminated ore of the Falconbridge mine. A microscopic examination of selective samples was made. The minerals present in the polished sections of the ore are various dark silicates, quartz, calcite, pyrrhotite, pentlandite, chalcopyrite, sphalerite, and ilmenite.

The gangue consists chiefly of dark silicates. A small amount of impure quartz is present, and calcite forms minute veinlets in the sulphides.

The sulphides are usually coarsely disseminated, but locally may be very finely divided. Pyrrhotite forms comparatively large, coarsely-granular masses which usually contain pentlandite in irregular patches and small veinlets. Chalcopyrite commonly occurs within the silicates in very small grains and stringers, but may also be associated with the other sulphides. Within some of the chalcopyrite are rare small grains of sphalerite. Ilmenite, which is present in small amount, appears to occur in the dark silicates without regard to the distribution of the sulphides.

Flotation Concentrate No. 2 from Laboratory Test No. 14

—	Free, per cent by volume	Combined, per cent by volume	Totals, per cent by volume
Gangue	59·8	15·3	75·1
Pyrrhotite	16·5	0·3	16·8
Pentlandite	5·2	0·0	5·2
Ilmenite	0·8	0·0	0·8
Chalcopyrite	0·3	1·8	2·1
	82·6	17·4	100·0

Purpose of Experimental Tests. In October 1932, it was definitely decided to erect a 250-ton concentrator to treat disseminated copper-nickel ore by bulk flotation.

The work described in this report was carried out to check previous work and to determine a suitable reagent balance.

Experimental Tests

Small-scale Batch Tests

A small lot of ore, 236 pounds, designated Lot No. 2, was received at the Mines Branch, November 5, 1932. The lot was stage-crushed to 14 mesh and 1,000-gramme samples cut out for laboratory testing. The lot assayed:—

	Per cent		Per cent
Copper	1·58	Iron	21·0
Nickel	1·09	Insoluble	46·5

Test No. 1: Lot No. 2

Results:

Product	Weight	Assay				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads	.....	1·58	1·09	21·0	46·50	100·0	100·0	100·0	100·0
+ 65 mesh	32·4	0·84	0·49	16·1	54·2	17·6	13·7	24·9	39·7
− 65+100	11·0	1·36	1·04	20·9	46·1	9·7	9·9	10·9	11·5
−100+150	10·5	1·56	1·38	23·1	42·9	10·6	12·5	11·6	10·2
−150+200	7·3	1·74	1·48	23·9	40·4	8·2	9·3	8·4	6·7
−200	38·8	2·14	1·63	23·9	36·5	53·9	54·6	44·2	31·9
Heads calc	100·0	1·54	1·15	20·9	44·2	100·0	100·0	100·0	100·0

Grinding:

1,000 grms. ore, 750 c.c. water, 5-minute grind in pebble mill.

Test No. 2

Test to duplicate work of a unit cell in the grinding circuit.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads	.....	1·58	1·09	21·0	46·5	.....	.....	.....	.....
Conc. No. 1	10·6	12·3	4·50	27·9	20·0	79·5	39·2	13·9	4·6
Conc. No. 2	28·0	1·13	2·35	40·8	19·2	19·3	54·2	53·8	11·7
Bulk conc	38·6	4·20	2·95	37·2	19·4	98·8	93·4	67·7	16·3
Tailing	61·4	0·03	0·13	11·1	62·6	1·2	6·6	32·3	83·7
Heads calc	.....	1·64	1·21	21·2	45·7	100·0	100·0	100·0	100·0

Grinding:

Two-stage, 5 and 10 minutes. Approximately 62 per cent −200 mesh.

Reagents, lb./ton:

To mill—		To cell—		Regrind to cell—	
Soda ash	1·0	Soda ash	2·0	Soda ash	4·0
Water-gas tar	0·20	Water-gas tar	0·14	S.E.Z	0·2
Cresylic acid	0·07	Cresylic acid	0·07	Cresylic acid	0·14

NOTE.—Abbreviations used throughout this report:

60-20-20 ... 60 per cent coal-tar creosote, 20 per cent coal tar, 20 per cent cresylic acid.
P.A.Z ... Potassium amyl xanthate.
S.E.Z ... Sodium ethyl xanthate.
Y.P ... Yarmor pine oil.

Test No. 4

Grinding:

Two-stage, 5 and 10 minutes. Approximately 62 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads	.....	1·58	1·09	21·0	46·5	.....	.....	.....	.....
Conc. No. 1	13·8	9·50	4·39	30·5	20·0	79·3	55·2	20·7	5·9
Conc. No. 2	17·5	1·80	2·19	35·2	24·7	19·1	34·8	30·3	9·2
Bulk conc.	31·3	5·19	3·16	33·2	22·6	98·4	90·0	51·0	15·1
Tailing	68·7	0·04	0·16	14·5	58·2	1·6	10·0	49·0	84·9
Calc. heads	.....	1·65	1·10	20·3	47·0	100·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—		To regrind—	
Soda ash	1·0	Soda ash	1·0
Water-gas tar	0·35	Water-gas tar	0·20
Cresylic acid	0·14	Cresylic acid	0·07

Test No. 6a

Grinding:

15 minutes. 62 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads	.....	1·58	1·09	21·0	46·5	.....	.....	.....	.....
Flot. conc.	27·7	5·78	3·55	38·5	17·03	97·4	88·3	49·6	10·9
Tailing	72·3	0·06	0·18	14·8	153·6	2·6	11·7	50·4	89·1
Calc. heads	100·0	1·65	1·12	21·4	43·5	100·0	100·0	100·0	100·0

Reagents, lb/ton:

To mill—		To cell—	
Soda ash	1·5	S.E.Z.	0·05
C.T.C.	0·25	Y.P.	0·20
S.E.Z.	.....		

Total flotation time, 11 minutes. Cell pH 9·5.

Test No. 6b

Grinding:

25 minutes. 80 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads	.....	1·58	1·09	21·0	46·5	.....	.....	.....	.....
Conc.	34·4	4·70	2·96	33·0	22·8	98·8	92·3	53·9	18·5
Tailing	65·6	0·03	0·13	14·7	53·2	1·2	7·7	46·1	81·5
Calc. heads	100·0	.....	.....	.....	.....	100·0	100·0	100·0	100·0

Reagents, lb/ton:

To mill—		To cell—	
Soda ash	1·5	S.E.Z	0·05
C.T.C	0·25	Y.P	0·20
S.E.Z	0·2		

Test No. 7a

Grinding:

15 minutes. 62 per cent —200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads									
Conc	24·0	6·54	3·81	34·9	18·3	96·7	82·3	40·3	10·1
Tailing	76·0	0·07	0·26	16·3	51·4	3·3	17·7	59·7	89·9
Calc. heads	100·0	1·63	1·12	20·8	43·5	100·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—
Lime 1·0
Other reagents same as Test No. 6.
Total flotation time, 11 minutes.

Test No. 7b

Grinding:

25 minutes. 80 per cent —200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads									
Conc	28·6	5·37	3·37	34·8	21·8	97·7	87·1	45·0	14·5
Tailing	71·4	0·05	0·20	16·2	51·1	2·3	12·9	55·0	85·5
Calc. heads	100·0	1·56	1·10	21·0	42·5	100·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—
Lime 1·0
Other reagents same as Test No. 6. Total flotation time, 15 minutes.

Test No. 9

Grinding:

15 minutes. 62 per cent—200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads									
Conc	27·3	6·05	3·57	34·2	20·6	97·4	85·8	43·2	13·2
Tailing	72·7	0·06	0·22	16·9	51·3	2·6	14·2	56·8	86·8
Calc. heads	100·0	1·75	1·17	22·4	44·3	100·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—		To cell—	
Lime	0·5	S.E.Z	0·10
60-20-20	0·28	Y.P	0·10

Total flotation time, 12 minutes.

NOTE.—60-20-20; W.G.T., C.T.C., Cresyl.

Test No. 10

Grinding:

15 minutes. 62 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads									
Conc.	25·6	6·08	3·61	34·2	20·3	98·2	84·5	42·7	11·3
Tailing	74·4	0·04	0·23	15·8	55·2	1·8	15·5	57·3	88·7
Calc. heads	100·0	1·56	1·08	20·2	45·6	100·0	100·0	100·0	100·0

Reagents:

Check on Test No. 9.

Test No. 11

Grinding:

15 minutes. 62 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads									
Conc.	26·8	5·69	3·51	36·9	17·7	98·6	89·0	48·8	9·8
Tailing	73·2	0·03	0·16	14·2	59·7	1·4	11·0	51·2	90·2
Calc. heads	100·0	1·53	1·04	20·0	47·5				

Reagents:

Check on Tests Nos. 9 and 10, except P.A.Z. used replacing S.E.Z.

Test No. 12

Grinding:

15 minutes. 62 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads									
Conc. No. 1	27·5	5·60	3·55	35·8	18·1	96·0	85·5	48·8	10·8
Conc. No. 2	10·1	0·28	0·71	29·3	34·5	1·7	6·3	14·7	7·5
Bulk conc.	37·6	4·10	2·79	34·1	22·5	97·7	91·8	63·5	18·3
Tailing	62·4	0·06	0·15	11·8	60·3	2·3	8·2	36·5	81·7
Calc. heads	100·0	1·62	1·15	20·3	46·5	100·0	100·0	100·0	100·0

Reagents:

Check on Test No. 11. 1st concentrate.

Test No. 13

Grinding:

15 minutes. 62 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads									
Conc	36·1	4·30	2·88	34·9	22·5	98·0	91·5	62·0	17·5
Tailing	63·9	0·05	0·15	12·1	60·0	2·0	8·5	38·0	82·5
Calc. heads	100·0	1·58	1·13	20·2	46·2	100·0	100·0	100·0	100·0

Reagents:

Check on Tests Nos. 11 and 12, except pine oil raised to 0·5 lb./ton.

Test No. 15

Grinding:

62 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads									
Conc	31·7	4·80	3·20	36·7	24·3	99·1	89·8	55·0	15·8
Tailing	68·3	0·02	0·17	14·0	60·3	0·9	10·2	45·0	84·2
Calc. heads	100·0	1·55	1·14	21·4	49·2	100·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—

Cement	1·0
60–20–20	0·28

Test No. 16

Grinding:

62 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads									
Conc. No. 1	31·9	5·08	3·10	32·8	23·5	98·2	87·9	50·5	15·8
Conc. No. 2	5·7	0·34	0·65	31·7	33·0	1·1	3·3	8·7	4·0
Bulk conc	37·6	4·35	2·60	32·6	25·0	99·3	91·2	59·2	19·8
Tailing	62·4	0·02	0·16	13·5	61·0	0·7	8·8	40·8	80·2
Calc. heads	.....	1·64	1·11	20·4	46·8	100·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—

Cement	1·0
60–20–20	0·28

To 2nd flotation—

Sodium sulphate	1·0

Test No. 17

Grinding:

62 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads	.....	.....	.....	.....	.....	.....	.....	.....	.....
Conc.	33·1	4·76	3·06	36·3	19·5	97·9	90·0	58·5	14·2
Tailing	66·9	0·05	0·17	12·7	59·7	2·1	10·0	41·5	85·8
Calc. heads	100·0	1·60	1·12	20·5	45·3	100·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—
Soda ash ..... 0·5
60–20–20 ..... 0·28

Test No. 19

Grinding:

62 per cent −200 mesh, two-stage.
1st float. 7-min. grind.
2nd float. 8 min. grind.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads	.....	.....	.....	.....	.....	.....	.....	.....	.....
Conc. No. 1	18·6	7·00	3·72	35·7	21·1	89·4	64·1	32·3	8·1
Conc. No. 2	21·8	0·65	1·62	35·2	30·1	9·7	32·6	37·4	13·5
Bulk conc.	40·4	3·57	2·59	35·5	25·9	99·1	96·7	69·7	21·6
Tailing	59·6	0·02	0·06	10·5	63·9	0·9	3·3	30·3	78·4
Calc. heads	100·0	1·45	1·05	20·6	48·5	100·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—
Soda ash ..... 1·5
60–20–20 ..... 0·30
60 ..... C.T.C.
20 ..... C.T.
20 ..... Cres.

1st float—
Cresylic acid ..... excess

2nd float—
P.A.Z ..... 0·2
Y.P ..... 0·15

Test No. 20

Grinding:

Two-stage as Test No. 19.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads	.....	.....	.....	.....	.....	.....	.....	.....	.....
Conc. No. 1	22·0	6·47	3·87	39·4	12·5	86·8	77·5	42·8	6·1
Conc. No. 2	12·0	1·17	1·49	30·5	32·8	9·5	16·3	18·1	8·7
Conc. No. 3	7·6	0·30	0·28	16·1	49·3	1·5	1·9	6·0	8·2
Bulk conc.	41·6	3·50	2·53	32·6	24·6	97·8	95·3	66·9	23·0
Tailing	58·3	0·06	0·08	11·5	59·8	2·2	4·3	33·1	77·0
Calc. heads	100·0	1·54	1·13	20·9	46·7	100·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—		1st float.—	
Lime	0·5	P.A.Z	0·1
60-20-20	0·3	Cresylic	0·05
2nd float.—		3rd float.—	
P.A.Z	0·1	Lime	6·0
		P.A.Z	0·1
		Y.P	0·1

Test No. 21

Grinding:

Two-stage, 5 and 10 minutes.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads									
Conc. No. 1	23·5	5·35	3·50	40·9	31·5	80·4	74·4	43·8	16·4
Conc. No. 2	11·7	2·20	1·86	33·9	18·1	16·4	19·7	18·1	4·7
Conc. No. 3	4·4	0·44	0·52	23·5	42·3	1·2	2·1	4·7	4·1
Bulk conc	39·6	3·67	2·78	34·2	25·8	98·0	96·2	66·2	25·2
Tailing	60·4	0·05	0·07	12·2	56·2	2·0	3·8	33·4	74·8
Calc. heads	100·0	1·61	1·14	22·6	46·5	100·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—		Ist float.—	
Soda ash	1·5	P.A.Z	0·1
		Y.P	0·1
2nd float.—		3rd float.—	
P.A.Z	0·1	Soda Ash	5·0
Y.P	0·1	P.A.Z	0·1
		Y.P	0·1

Test No. 22

Grinding:

Two-stage, 5 and 10 minutes.

Product	Weight	Assay, %				Per cent contents			
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
Heads									
Conc. No. 1	20·0	6·50	3·60	38·2	18·3	0 81·0	65·7	35·6	8·5
Conc. No. 2	17·2	1·53	1·82	32·8	28·8	16·3	28·5	26·3	11·5
Bulk conc	37·2	4·21	2·78	35·6	23·2	97·3	94·2	61·9	20·0
Tailing	62·8	0·07	0·10	13·0	55·0	2·7	5·8	38·1	80·0
Calc. heads	100·0	1·65	1·13	22·0	44·5	100·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—		1st float.—		2nd float.—	
Lime	0·5	P.A.Z	0·1	Minerec B	0·1
		Y.P	0·1	Y.P	0·1

*Series to Test Falconbridge Water**

Test No. 32

Grinding:

15 minutes, 62 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents		
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Insol.
Heads								
Conc	41·7	3·32	2·19	.....	28·4	97·4	93·4	25·2
Tailing	58·3	0·06	0·11	.....	60·3	2·6	6·6	74·8
Calc. heads	100·0	1·42	0·97	.....	47·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—
Ottawa water
60-20-20 ..... 0·28
Soda ash ..... 1·5

To float.—
P.A.Z ..... 0·2
Y.P ..... 0·2

Test No. 33

Product	Weight	Assay, %				Per cent contents		
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Insol.
Heads								
Conc	36·2	3·78	2·46	.....	26·2	96·8	94·0	20·3
Tailing	63·8	0·07	0·09	.....	58·8	3·2	6·0	79·7
Calc. heads	100·0	1·41	0·95	.....	46·9	100·0	100·0	100·0

Reagents:

As in Test No. 32 except Falconbridge water used.

Test No. 35

Grinding:

18 minutes, 70 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents		
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Insol.
Heads								
Conc	38·4	3·66	2·69	.....	25·0	98·7	94·4	20·6
Tailing	61·6	0·03	0·10	.....	60·1	1·3	5·6	79·4
Calc. heads	100·0	1·44	1·13	.....	47·0	100·0	100·0	100·0

Reagents:

As in Tests Nos. 32, 33. Falconbridge water used.

*Series includes Tests Nos. 32, 33, 35, 36, 37, 38.

Test No. 36

Grinding:

18 minutes, 70 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents		
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Insol.
Heads	.....	.....	.....	.....	.....	.....	.....	.....
Conc	40·0	3·50	2·28	.....	26·3	98·3	95·0	22·2
Tailing	60·0	0·04	0·08	.....	61·7	1·7	5·0	77·8
Calc. heads	100·0	1·42	0·96	.....	47·5	100·0	100·0	100·0

Reagents:

As in Tests Nos. 32, 33, 35. Ottawa water used.

Test No. 37

Grinding:

18 minutes, 70 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents		
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Insol.
Heads	.....	.....	.....	.....	.....	.....	.....	.....
Conc	40·3	3·52	2·60	.....	26·7	98·8	93·6	23·7
Tailing	59·7	0·03	0·12	.....	58·0	1·2	6·4	76·3
Calc. heads	100·0	1·46	1·15	.....	46·4	100·0	100·0	100·0

Reagents:

Falconbridge water.

To mill—
60-20-20 replaced by 0·05 lb./ton thiocarbanilide.
Mechanical trouble with cell.

To cells.—
As in Test No. 32, etc.

Test No. 38

Grinding:

18 minutes, 70 per cent −200 mesh.

Product	Weight	Assay, %				Per cent contents		
	Per cent	Cu	Ni	Fe	Insol.	Cu	Ni	Insol.
Heads	.....	.....	.....	.....	.....	.....	.....	.....
Conc	43·0	3·16	2·03	.....	30·4	98·4	94·5	27·6
Tailing	57·0	0·04	0·09	.....	60·0	1·6	5·5	72·4
Calc. heads	100·0	.....	.....	.....	.....	100·0	100·0	100·0

Reagents:

As in Test No. 37. Ottawa water.

Experimental Tests: Continuous Large-scale Tests: Lot No. 3

Test No. 1

Feed Rate:

768 lb./hour, ½ inch.

Grinding:

2·3%+65 mesh, 68·4% −200 mesh.
Unit cell in grinding circuit.

Product	Weight, per cent	Assay, %				Per cent contents			
		Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
B.M. feed	.....	1·57	1·14	20·3	44·6	.....	.....	.....	.....
B.M. disch.	.....	1·15	.....	20·4	43·9	.....	.....	.....	.....
Conc. No. 1	11·44	12·34	6·09	32·6	13·9	89·6	69·2	19·2	3·3
Flot. feed	.....	0·18	0·50	18·1	48·9	.....	.....	.....	.....
Conc. No. 2	8·64	1·66	2·28	43·7	13·5	9·1	19·6	19·5	2·4
1st R. tails	.....	0·02	0·14	14·7	59·3	.....	.....	.....	.....
Conc. No. 3	10·60	0·06	0·75	41·6	15·6	0·4	7·5	22·8	3·4
Bulk conc.	30·68	5·09	3·17	38·8	14·4	99·1	96·5	61·5	9·1
Final tails	69·32	0·02	0·05	10·7	63·3	0·9	3·5	38·5	90·9
Calc. heads	100·0	1·57	1·01	19·4	48·3	.....	.....	.....	.....

Reagents, lb/ton:

To mill—		To cells—	
Soda ash	1·7	P.A.Z	0·16
60–20–20	0·24	Y.P	0·08

Test No. 2

Feed Rate:

798 lb./hour. ½ inch.

Grinding:

4·7%+65 mesh, 66·2% −200 mesh.

Product	Weight, per cent	Assay, %				Per cent contents			
		Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
B.M. feed	.....	1·39	0·98	19·6	45·4	.....	.....	.....	.....
B.M. disch.	.....	1·11	0·84	19·4	47·6	.....	.....	.....	.....
Conc. No. 1	10·8	11·34	6·01	32·2	13·8	87·6	66·5	19·3	2·9
Flot. feed	.....	0·18	0·24	16·9	52·8	.....	.....	.....	.....
Conc. No. 2	13·0	1·12	1·76	42·7	16·8	10·4	23·4	29·2	4·2
1st R. tails	.....	0·02	0·13	13·1	60·9	.....	.....	.....	.....
Conc. No. 3	11·0	0·13	0·60	33·0	31·3	1·0	6·8	19·2	6·6
Bulk conc.	34·8	3·98	2·71	36·4	20·4	99·0	96·7	66·7	13·7
Final tails	65·2	0·02	0·05	9·6	68·3	1·0	3·3	33·3	86·3
Calc. heads	100·0	1·40	0·98	18·9	51·6	100·0	100·0	100·0	100·0

Reagents, lb./ton:

To mill—		To cells—	
Soda ash	2·0	P.A.Z	0·15
60–20–20	0·23	Y.P	0·10

Test No. 3

Feed rate, Grinding, and Reagents:

Same as in Mill Test No. 2.

Product	Weight, per cent	Assay, %				Per cent contents			
		Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
B.M. feed	.....	1·42	0·96	19·6	46·3	.....	.....	.....	.....
B.M. disch	.....	1·17	0·89	19·6	47·1	.....	.....	.....	.....
Conc. No. 1	10·2	12·02	6·39	32·7	14·5	85·2	58·4	15·8	3·4
Flot. feed	.....	0·22	0·46	18·5	48·4	.....	.....	.....	.....
Conc. No. 2	19·2	1·04	1·95	41·5	18·0	13·9	33·8	38·0	7·9
1st R. tails	.....	0·02	0·11	14·2	54·5	.....	.....	.....	.....
Conc. No. 3	6·0	0·12	0·58	37·7	24·9	0·5	3·1	10·8	3·4
Bulk conc	35·4	4·54	2·88	37·7	17·7	99·6	95·1	64·6	14·7
Final tails	64·6	0·01	0·08	11·6	57·9	0·4	4·7	35·4	85·3
Calc. heads	.....	1·44	1·11	21·1	43·9	.....	.....	.....	.....

Mechanical trouble during run.

Test No. 4

Feed Rate:

860 lb./hour. ½ inch.

Grinding:

4·4%+65 mesh, 67·4% −200 mesh.

Product	Weight, per cent	Assay, %				Per cent contents			
		Cu	Ni	Fe	Insol.	Cu	Ni	Fe	Insol.
B.M. feed	.....	1·62	1·17	21·0	43·4	.....	.....	.....	.....
B.M. disch	.....	1·23	1·02	.....	50·2	.....	.....	.....	.....
Conc. No. 1	19·1	7·58	4·45	.....	16·4	88·0	72·8	.....	7·0
Flot. feed	.....	0·21	0·33	.....	51·7	.....	.....	.....	.....
Conc. No. 2	26·6	0·67	1·14	.....	30·2	10·8	25·7	.....	17·9
Bulk conc	45·7	.....	.....	.....	.....	98·8	98·5	.....	24·9
Final tails	54·3	0·03	0·03	.....	62·1	1·2	1·5	.....	75·1
Calc. heads	100·0	1·64	1·17	.....	44·9	100·0	100·0	.....	100·0

Reagents, lb./ton:

To mill—		To cells—	
Soda ash	2·1	P.A.Z	0·16
60–20–20	0·26	Y.P	0·15

SUMMARY AND CONCLUSIONS

Problems investigated in the laboratory included:—

1. The grinding necessary to obtain high recoveries of copper and nickel, with a ratio of concentration of 2·5 : 1.
2. Comparative values of alkaline agents.
3. Reagent balance best suited to the problem.
4. Flotation properties of Falconbridge water compared with Ottawa supply.
5. The possibility of using a unit cell in the grinding circuit.

Results obtained from a series of laboratory tests established the following points:—

1. Grinding to 3 per cent plus 65 mesh and 62 per cent minus 200 mesh makes it possible to recover 98 per cent of the copper values and 93 per cent of the nickel values by floating 35 to 40 per cent of the total ore.

2. Highest copper-nickel recoveries were obtained in a soda ash circuit. Good results were obtained in several tests when using lime and cement as alkaline agents, but the results from soda ash runs were more uniform and, on the average, better.

3. A heavy oil mixture fed to the unit showed considerable advantage over chemical collection. The type of froth produced by coal-tar products is especially suitable for the operation. It was found necessary to feed xanthates to the cells to ensure a low tailing.

4. The possibility of using a unit cell in the grinding circuit was investigated. Samples were ground 5 minutes, subjected to a short flotation treatment, and the tailings reground before the final flotation. These tests yielded encouraging results.

5. Falconbridge water was used in a series of tests. The water as received showed a slightly higher pH. content than Ottawa supply water. This difference resulted in slightly higher ratios of concentration and lower recoveries when using Falconbridge water.

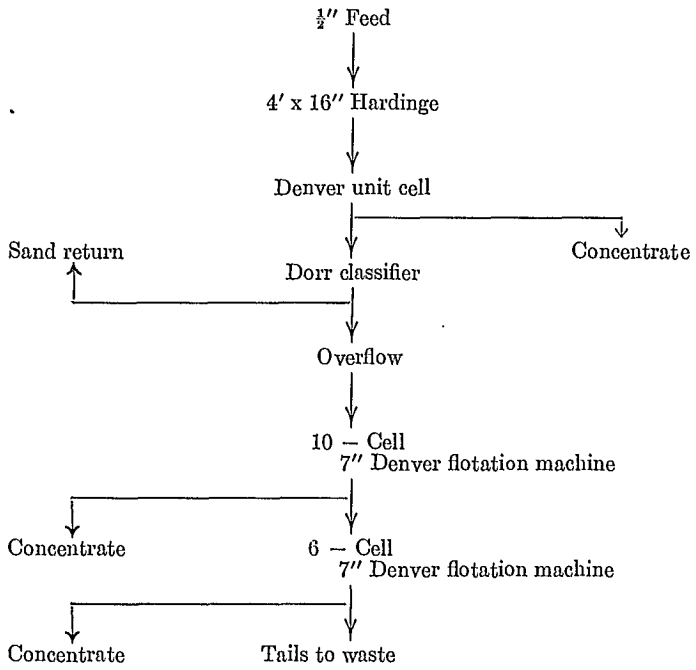
Mill Testing:

A carload shipment of Falconbridge ore, 54,700 pounds, was received at the Mines Branch on November 15, 1932.

The ore was crushed to ½ inch and assayed. The head sample assayed:

Cu, %	Ni, %	Fe, %	Insol. %	S, %	SiO_2 %
1·50	1·07	20·3	50·0	9·9	32·8

Based on the results obtained in the laboratory, large-scale equipment was arranged for the flow-sheet as shown:—



The work may be briefly summarized as follows:—

1. Nine mill runs were completed; the results of four are reported. Large-scale tests were made, floating in lime, neutral and sulphuric acid circuits, but the results obtained did not equal those secured in Tests Nos. 1 to 4 when soda ash was added.

2. Laboratory results were duplicated and improved. Copper recoveries of 99 per cent and nickel recoveries to 98 per cent are reported. Recent investigation indicates that the nickel content of the tailing produced is somewhat higher than reported. The lowest tailing produced assays 0·06 per cent nickel, thus the highest nickel recovery obtained would be 96 per cent.

3. Somewhat finer grinding and classification were responsible for the increased recoveries obtained in the large-scale work. Feeding of reagents in small amounts at several points in the flotation circuit also improved the operation.

GENERAL SUMMARY

As a result of test work previously carried out and the additional work covered in this report, the reagent balance shown is recommended.

	Lb./ton
Soda ash	1·5 to 2·0
Oil mixture: 60–20–20	0·20 to 0·30
P.A. xanthate	0·15 to 0·25
Yarmor pine	0·05 to 0·15

The exact amounts of reagents will be determined by the grinding obtained in the new concentrator and the grade of concentrate desired.

Report No. 459

EXPERIMENTAL TESTS ON TWO SAMPLES OF GOLD ORE FROM BEAUFOR GOLD MINES, LIMITED, PASCALIS TOWNSHIP, NORTHWESTERN QUEBEC

Shipment. A shipment of 11 sacks of ore, net weight 1,507 pounds, was received October 29, 1932, from J. C. R. MacPherson, Manager, Beaufor Gold Mines, Limited, Amos, Quebec. The shipment represented two distinct samples of ore, one from No. 1 vein and the other from No. 4 vein, the weight of each being approximately 750 pounds.

Characteristics of the Ore:

[1]The ore consists essentially of coarse pyrite in a quartz gangue. Locally this sulphide is disseminated in a black rock which is also traversed by veinlets of quartz. The quartz varies from a comparatively clear glassy type to a milky white type, and more rarely a light blue, almost opalescent variety is associated with chloritic material with small amounts of carbonate. The sulphides appear to follow the glassy or milky types, as does the gold.

That the sulphide and gold mineralization took place at a comparatively high temperature is indicated by the associated high-temperature type of quartz and the presence of needles of tourmaline in this quartz, and more rarely within the pyrite itself.

[1] From the report of the Mineragraphic Laboratory, by Maurice Haycock.

An average analysis of each of the samples was as follows:—

		Sample No. 1	Sample No. 4
Au	oz./ton	1·05	0·365
Ag	"	0·06	0·17
Cu	per cent	0·04	0·08
Pb	"	trace	0·03
Zn	"	0·35	0·12
Fe	"	5·40	6·10
Insol	"	77·76	67·10

Experimental Tests

Head samples were cut by standard methods from each of the two types of ore contained in the shipment.

A series of small-scale tests was made on each type of ore as well as a larger scale mill run using a unit of 100 pounds per hour capacity. The small-scale work included in each case flotation tests on both the ore and amalgamation tailings as well as cyanidation and amalgamation tests on both the ore and flotation concentrates.

The flow-sheet for the larger scale unit tests was plate amalgamation followed by flotation of the amalgamation tailing.

In some cases erratic results were obtained, due no doubt to the presence of free gold in varying amounts and sizes.

The results of tests may be summarized as follows:—

Test	Sample No. 1		Sample No. 4	
	Recovery, per cent	Tailing assay, Au, oz./ton	Recovery, per cent	Tailing assay, Au, oz./ton
Flotation	98·0	0·02	97·1	0·01
	90·8	0·185	86·4	0·085
Amalgamation—				
— 35 mesh	50·0	0·525	74·8	0·092
— 65 mesh	70·4	0·311	52·1	0·175
—150 mesh	76·2	0·25	56·7	0·158
Cyanidation—				
— 35 mesh	90·0	0·105	89·0	0·04
— 65 mesh	95·2	0·05	93·2	0·025
—150 mesh	97·6	0·025	97·3	0·01
Amalgamation and cyanidation at —65 mesh	95·7	0·045	91·8	0·03
Amalgamation and flotation at —65 mesh	97·8	0·025	98·4	0·007

Details of the tests follow:—

Sample No. 1

Test No. 1

This is a flotation test on the ore ground 20 minutes in a Denver rod mill.

Charge to Rod Mill:

Ore, 2,000 grammes at —14 mesh.
Water, 1,000 c.c.

Na_2CO_3	2·0 lb./ton
Aerofloat No. 25	0·07 "

Reagents to Flotation Cell:

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

Concentrate and tailing were filtered, washed, and assayed for gold.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Recovery, per cent
Concentrate	5·96	15·18	97·96
Tailing	94·04	0·02	2·04
Head (cal.)	100·00	0·924	100·0

Ratio of concentration, 16·78 : 1.

Screen Analysis of Flotation Tailing

Mesh	Weight, per cent	Assay, Au, oz./ton	Average tailing. Au, oz./ton
+200	52·0	0·02	0·02
−200	48·0	0·02	

Test No. 2

In this test the ore at minus 14 mesh was ground for 20 minutes in a Denver rod mill, then floated. Samples of the flotation concentrate were then treated by cyanidation, amalgamation, and amalgamation followed by cyanidation, the period of agitation being in each case about 40 hours.

Charge to Rod Mill:

Ore	2,000 grammes
Water	1,000 c.c.
Na_2CO_3	3·0 lb./ton
Aerofloat No. 25	0·07 "

Reagents to Cell:

	Lb./ton
Sodium ethyl xanthate	0·10
Pine oil	0·05
Tarol No. 1	0·025

All products were assayed for gold.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Recovery, per cent
Concentrate	7·6	22·24	90·8
Tailing	92·4	0·185	9·2
Head (cal.)	100·0	1·86	100·0

Ratio of concentration, 13·16 : 1.

Summary Tests on Flotation Concentrate

Product	Assay heads, Au, oz./ton	Assay tailing, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
				KCN	CaO
Cyanide tailing	22·24	0·61	97·3	5·3	12·7
Amalgamation tailing	22·24	6·665	70·0	.....	.....
Cyanidation tailing from amalgamation tailing	6·665	0·60	91·0	5·5	13·4

Overall recovery by amalgamation and cyanidation: $70{\cdot}0 + \frac{(91{\cdot}0 \times 30{\cdot}0)}{100} = 97{\cdot}3$ per cent.

AMALGAMATION

Test No. 3

Three lots of the ore were ground to the following sizes, −35, −65, and −150 mesh. Amalgamation tests were made on samples of each of these, 1,000 grammes of the ore in 1 : 1 pulp being amalgamated for 30 minutes with 100 grammes of mercury. The minus 65-mesh amalgamation tails were floated with the following reagents:—

	Lb./ton
Na_2CO_3	3·0
Aerofloat No. 25	0·07
Sodium ethyl xanthate	0·1
Pine oil	0·05

Results:

Head sample: Au, 1·05 oz./ton.

Mesh	Tailing assay, Au, oz./ton	Recovery, per cent
− 35	0·525	50·0
− 65	0·311	70·4
−150	0·25	76·2

Flotation Test Amalgamation Tails −65 mesh

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Concentrate	9·0	3·20	92·7
Tailing	91·0	0·025	7·3
Amalgamation tailing (cal.)	100·0	0·311	100·0

Overall recovery of gold resulting from amalgamation and flotation of the minus 65-mesh ore is 97·8 per cent.

CYANIDATION

Test No. 4

Cyanidation tests were made on samples of the same three lots of ore as used in the amalgamation tests described above. The pulp, at 3 : 1 dilution, was agitated for 24 hours in a solution running 2 pounds per ton in KCN.

Results:

Head sample: Au, 1·05 oz./ton.

Mesh	Tailing assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
− 35	0·105	90·0	0·60	2·17
− 65	0·05	95·2	0·90	2·26
−150	0·025	97·6	1·50	6·80

AMALGAMATION AND CYANIDATION

Test No. 5

In this test 1,000 grammes of the ore at minus 65 mesh was amalgamated in 1 : 1 pulp for 30 minutes with 100 grammes of mercury. The amalgamation tailing was agitated for 24 hours in a solution running one pound per ton in KCN. Dilution was 2·5 : 1, and lime was added at the rate of 3 pounds per ton tailing.

Summary:

Head sample: Au, 1·05 oz./ton.

Product	Assay Au, oz./ton	Recovery by amalgamation	Recovery by cyanidation	Total recovery	Reagents, lb./ton	
					KCN	CaO
Amalgamation tailing	0·311	70·4	25·3	95·7	0·58	2·26
Cyanidation tailing	0·045					

HYDRAULIC CLASSIFICATION

Test No. 6

In this test the ore at minus 65 mesh was put through a hydraulic classifier, the heavy sulphides and coarse gold being allowed to settle against a slowly rising current of water. The classifier concentrate and overflow were assayed for gold.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
Concentrate	1·3	34·70	36·2
Overflow	98·7	0·805	63·8
Head (cal.)	100·0	1·25	100·0

Ratio of concentration, 76·9 : 1.

Mill Run on Sample No. 1

The ore at minus 14 mesh was fed into a rod mill, the discharge from which dropped onto an amalgamation plate. The amalgamation tails were floated with the following reagents:—

	Lb./ton
Na_2CO_3	3·00
Sodium ethyl xanthate	0·1
Pine oil	0·05

Samples were taken at 15-minute intervals of the feed, rod mill discharge, plate discharge, flotation concentrate, and flotation tailing. These products were assayed for gold, the assays being as follows:—

	Au, oz./ton
Mill feed	1·44
Rod mill discharge	0·99
Plate discharge	0·44
Flotation concentrate	4·10
Flotation tailing	0·01
Mill clean-up	2·62

A screen test on the rod mill discharge showed the grinding to be as follows:—

Mesh	Weight, per cent	Cumulative weight, per cent
+100	7·5	7·5
+200	27·2	34·7
−200	65·3	100·0

Summary Mill Run—Sample No. 1

Flow-sheet—Amalgamation and flotation.
Test started, 2.30 p.m.
Test finished, 4.45 p.m.
Power off ten minutes.
Duration of test, 2 hours and 5 minutes.
Feed rate, 90 pounds per hour.
Total feed during test, 187·5 pounds.
Total concentrate produced, 19·7 pounds.
Ratio of concentrate, 9·5 : 1.

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent
Mill feed	100·0	1·44	100·0
Rod mill discharge	100·0	0·90	68·75
Plate discharge	100·0	0·44	30·60
Flotation concentrate	10·5	4·10	29·90
Flotation tailing	89·5	0·01	0·62

	Per cent
Total gold remaining in rod mill	31·25
Total gold recovered on plate	38·15
Total gold recovered in flotation concentrate	29·90
Total recovery by amalgamation and flotation	68·05
Recovery from mill cleanup by amalgamation	23·2
Total recovery, 68·05 + 23·20	91·25
Gold recovered from flotation concentrate by cyanidation	97·1
Gold recovered from flotation concentrate by amalgamation	36·0

Sample No. 4

Test No. 1

This is a flotation test on the ore ground for 20 minutes in a Denver rod mill.

Ore	2,000 grammes — 14 mesh
Water	1,000 c.c.
Na_2CO_3	3·0 lb./ton
Aerofloat No. 25	0·07 "

Reagents to Cell:

	Lb./ton
Sodium ethyl xanthate	0·1
Pine oil	0·05

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Recoveries, per cent
Concentrate	7·6	4·03	97·1
Tailing	91·4	0·01	2·9
Head (cal.)	100·0	0·32	100·0

Ratio of concentration, 13·16 : 1.

Screen Analysis Flotation Tailing

Mesh	Weight, per cent	Assay, Au, oz./ton	Per cent of values	Average tailing
+200	31·8	0·01	31·8	0·01
−200	68·2	0·01	68·2	

Test No. 2

In this test the flotation concentrate produced from 8,000 grammes of ore was treated in the following ways:—

(1) Straight cyanidation.
(2) Amalgamation.
(3) Amalgamation followed by cyanidation.

The ore at minus 14 mesh was ground for 20 minutes in a Denver rod mill, the charge being as follows:—

Ore	2,000 grammes
Water	1,000 c.c.
Na_2CO_3	3·0 lb./ton
Aerofloat No. 25	0·07 "

Reagents to Cell:

	Lb./ton
Sodium ethyl xanthate	0·1
Pine oil	0·05

Results:

Recovery by Flotation of Ore

Product	Weight, per cent	Assay, Au, oz./ton	Recovery, per cent
Concentrate	7·6	6·56	86·4
Tailing	92·4	0·085	15·6
Head (cal.)	100·0	0·58	100·0

Ratio of concentration, 13·15 : 1.

Summary of Tests on Flotation Concentrate

Product	Assay heads, Au, oz./ton	Assay tailing, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
				KCN	CaO
Cyanide tailing	6·56	0·165	97·5	3·0	10·6
Amalgamation tailing	6·56	1·89	71·2	.....	.....
Cyanide tailing from amalgamation tailing	1·89	0·15	92·1	3·0	11·2

AMALGAMATION

Test No. 3

Three lots of the ore were ground to the following sizes −35, −65, and −150 mesh, and amalgamated as was done with Sample No. 1. In each case 1,000 grammes of the ore was amalgamated for 30 minutes in 1: 1 pulp with 100 grammes of mercury. The minus 65-mesh amalgamation tailing was floated using the following reagents:—

	Lb./ton
Na_2CO_3	3·0
Aerofloat	0·07
Sodium ethyl xanthate	0·1
Pine oil	0·05

Results:

Head sample: Au, 0·365 oz./ton.

Mesh	Tailing assay, Au, oz./ton	Recovery, per cent
− 35	0·092	74·8
− 65	0·175	52·1
−150	0·158	56·7

Flotation −65 mesh Amalgamation Tailing

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
Concentrate	15·7	1·08	96·6
Tailing	84·3	0·007	3·4
Amalgamation tailing (cal.)	100·0	0·175	100·0

Screen Analysis Flotation Tailing from −65 mesh Amalgamation Tailing

Mesh	Weight, per cent	Assay, Au, oz./ton	Per cent of values	Average tailing, Au, oz./ton
+200	52·6	0·005	35·7	0·007
−200	47·4	0·010	64·3	

CYANIDATION

Test No. 4

Cyanidation tests were made on samples of the ore crushed dry to pass through the following meshes, 35, 65, and 150. The pulp was agitated for 24 hours at 3: 1 dilution and KCN 2 pounds per ton solution.

Results:

Head sample: Au, 0·365 oz./ton.

Mesh	Tailing assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
− 35	0·04	89·0	0·3	2·00
− 65	0·025	93·2	0·3	2·00
−150	0·01	97·3	0·9	6·80

AMALGAMATION AND CYANIDATION

Test No. 5

The ore dry crushed to pass through a 65-mesh screen was amalgamated with mercury and the amalgamation tailing agitated for 24 hours in cyanide solution running one pound per ton in KCN. Dilution was 2·5 : 1.

Results:

Head sample: Au, 0·365 oz./ton.

Product	Assay, Au, oz./ton	Recovery by amalgamation	Recovery by cyanidation	Total recovery	Reagents consumed, lb./ton	
					KCN	CaO
Amalgamation tailing	0·175	52·1	39·7	91·8	0·33	2·00
Cyanidation tailing	0·03	.....	.....	.....	.....	.....

HYDRAULIC CLASSIFICATION

Test No. 6

In this test, as in Test No. 6, Sample No. 1, the ore at minus 65 mesh was put through a hydraulic classifier, the coarse gold and heavy sulphides being allowed to settle against a slowly rising current of water. The classifier concentrate and overflow were assayed for gold.

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
Concentrate	1·95	7·49	39·3
Overflow	98·05	0·23	60·7
Head (cal.)	100·00	0·37	100·0

Ratio of concentration, 51·3 : 1.

Mill Run on Sample No. 4

The flow-sheet for the mill run was exactly the same as it was for Sample No. 1. The pine oil used for Sample No. 4 was slightly less than that used for Sample No. 1, but otherwise the reagent combination was exactly the same.

A similar set of samples was taken and gave the following assays:—

	Au, oz./ton
Mill feed	0·39
Rod mill discharge	0·38
Plate discharge	0·215
Flotation concentrate	2·36
Flotation tailing	0·015

A screen test on the rod mill discharge showed the grinding to be as follows:—

Mesh	Weight, per cent	Cumulative weight, per cent
+100	1·5	1·5
+200	15·7	17·2
−200	82·8	100·0

Summary Mill Run Sample No. 4:

Flow-sheet—Amalgamation and flotation.
Test started, 3.30 p.m.
Stopped, 4.30 p.m.
Restarted, 9.30 a.m.
Finished, 10.45 a.m.
Duration of tests, 2 hours 15 minutes.
Feed rate, 97·5 pounds per hour.
Total feed during test, 219·4 pounds.
Total concentrate produced, 18·9 pounds.
Ratio of concentration, 11· 6 : 1.

Product	Weight, per cent	Assay, Au, oz./ton	Units	Gold, per cent
Mill feed	100·0	0·39	39·0	100·0
Rod mill discharge	100·0	0·38	38·0	97·4
Plate discharge	100·0	0·215	21·5	55·1
Flotation concentrate	8·6	2·36	20·3	52·0
Flotation tailing	91·4	0·015	1·4	3·5

	Per cent
Total gold remaining in rod mill	2·6
Total gold recovered on plate	42·3
Total gold recovered in flotation concentrate	52·0
Total recovery by amalgamation and flotation	94·3
Gold recovered from flotation concentrate by cyanidation	87·7
Gold recovered from flotation concentrate by amalgamation	47·7

CONCLUSIONS

This ore might be treated by any one of a number of flow-sheets. Straight cyanidation will give excellent recoveries on both samples of ore when ground to 85 per cent minus 200 mesh. Approximately 75 per cent of the gold in either sample of ore can be recovered by amalgamation, but to accomplish this Sample No. 1 should be ground to 85 per cent minus 200 mesh and Sample No. 4 much coarser at not more than 41 per cent minus 200 mesh. Flotation alone would not be practical with an ore that contains so much free gold, and some of it coarse.

A simple flow-sheet would be to grind the ore all through 65 mesh and amalgamate on plates. The amalgamation tailing would be concentrated by flotation. The flotation concentrate would then be reground if necessary or desirable and cyanided. This flow-sheet would save a lot of grinding as compared to straight cyanidation since in that case the ore would all have to be crushed through 150 mesh or 85 to 90 per cent minus 200 mesh.

Report No. 460

EXPERIMENTAL TESTS ON GOLD ORE FROM THE SULLIVAN CONSOLIDATED MINES, LIMITED, DUBUISSON TOWNSHIP, ABITIBI COUNTY, QUEBEC

Shipment. A shipment of 700 pounds of gold ore was received at the Ore Dressing and Metallurgical Laboratories on November 12, 1932. The shipment consisted of one sample of mine ore and was submitted by Mr. A. K. Muir, Superintendent, Sullivan Consolidated Mines, Limited, Siscoe, Quebec.

Characteristics of the Ore. The gangue consists chiefly of glassy grey quartz which commonly contains small veinlets or disseminated grains of carbonate. Locally a greenish grey chloritic phase is prominent, and in places dark silicates are present; the latter probably include tourmaline. Small grains of a mineral which is light grey in reflected light, and which shows bright internal reflections between crossed nicols, are common, but form a very small portion of the aggregate. This mineral was not identified.

Pyrite is sparingly disseminated throughout this gangue, but the grains rarely attain sizes of over 2 millimetres.

A few grains of native gold were observed within the quartz; these were all less than 0·5 millimetre in diameter. It is highly probable that most, if not all, of the gold occurs in the native state.

Experimental Tests

On sampling the lot by standard methods the shipment of mine ore was found to contain 0·285 ounce gold per ton; 0·095 ounce silver per ton; copper, nil.

Under the direction of Mr. R. C. Mott of Ventures Limited, a series of tests was made to establish the most suitable method for the recovery of the contained values. The investigation included amalgamation, flotation, and cyanidation separately and in combination. The results indicate that 49 per cent of the gold can be recovered by amalgamation of minus 48-mesh material. Cyanidation of the amalgamation tailing increases the recovery to 95 per cent. Straight cyanidation at minus 200 mesh gives an extraction of 96 per cent.

Additional tests carried out included: blanket concentration; a small-scale mill run of plate amalgamation followed by flotation; amalgamation and cyanidation of the flotation concentrate produced from the mill run.

A list of the tests conducted is as follows:—

1. Amalgamation and cyanidation of ore ground to 48 mesh.
2. Amalgamation and cyanidation of ore ground to 100 mesh.
3. Straight cyanidation of ore ground to 48, 100, 150, and 200 mesh.
4. Determination of the amount of free gold by hydraulic classification.
5. Straight flotation.
6. Amalgamation and flotation of ore ground to 48 mesh.
7. Amalgamation followed by regrinding and flotation.
8. Blanket concentration at table slope of 4 inches to the foot.
9. Blanket concentration at table slope of $1\frac{5}{16}$ inches to the foot.
10. Mill run—plate amalgamation and flotation.
11. Amalgamation of flotation concentrate from mill run.
12. Cyanidation of flotation concentrate from mill run.

AMALGAMATION AND CYANIDATION

Test Nos. 1 and 2

Samples of the ore were ground dry to pass 48 and 100 mesh, and amalgamated. The amalgamation tailings were agitated in a pulp containing three parts of solution to one of ore. The strength of the sodium cyanide solution used was equivalent to 1·0 pound potassium cyanide per ton of solution, and lime equivalent to 6 pounds per ton of ore was added to supply protective alkalinity.

Mesh grind	Amalgamation			Cyanidation			Total recovery
	Heads, Au, oz./ton	Tailing, Au, oz./ton	Recovery, per cent	Agitation, hours	Tailing, Au, oz./ton	Recovery, per cent	
48	0·285	0·145	49·12	24	0·02	86·21	92·98
48	0·285	0·145	49·12	48	0·015	89·65	94·74
100	0·285	0·11	61·40	24	0·015	86·36	94·74
100	0·285	0·11	61·40	48	0·01	90·91	96·49

Cyanidation

Mesh	Reagent consumption		
	Agitation, hours	Lb./ton ore	
		KCN	CaO
48	24	0·20	4·14
48	48	1·22	5·70
100	24	0·35	4·17
100	48	1·73	5·58

Screen analyses of the amalgamation tailing from the ore ground to 48 mesh—

Mesh	Weight, per cent	Assay, Au, oz./ton	Per cent of values
+ 65	6·56	0·135	6·1
− 65+100	18·67	0·14	18·0
−100+150	17·34	0·095	11·4
−150+200	12·21	0·10	8·4
−200	45·22	0·18	56·1

The large amount of values not recovered by amalgamation is readily soluble in cyanide solution.

STRAIGHT CYANIDATION

Test No. 3

Representative samples of the ore were ground to pass 48-, 100-, 150-, and 200-mesh screens. The ore was treated by cyanidation in a pulp containing three parts of solution to one part of ore. The strength of the sodium cyanide solution used was equivalent to 1·0 pound of potassium cyanide per ton of solution, and lime equivalent to 6 pounds per ton of ore was added to supply a protective alkalinity.

Results

Head: Au, 0·285 oz./ton.

Mesh	Agitation period, hr.	Tailing, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton	
				KCN	CaO
− 48	24	0·02	92·98	0·15	4·05
− 48	48	0·02	92·98	0·90	5·9
−100	24	0·02	92·98	0·30	4·41
−100	48	0·03	89·47	0·75	5·66
−150	24	0·01	96·49	0·15	4·05
−150	48	0·015	94·74	1·05	6·20
−200	24	0·01	96·49	0·15	4·95
−200	48	0·015	94·74	1·05	6·56

CONCENTRATION BY HYDRAULIC CLASSIFIER

Test No. 4

A sample of minus 14-mesh ore was crushed in a ball mill and concentrated in a hydraulic classifier. This test was made to indicate the recovery of gold which would be obtained by the use of a trap in the ball mill circuit.

Results:

Products	Weight, per cent	Assay, Au, oz./ton	Per cent of values	Ratio of concentration
Heads	100·0	0·344	100·00	
Classifier concentrate	0·15	82·81	36·12	667 : 1
Tailing	99·85	0·22	63·87	

Screen analyses of the tails show:—

Mesh	Weight, per cent	Assay, Au, oz./ton
+100	10·90	0·11
−100+200	30·19	0·19
−200	58·91	0·25

FLOTATION

Test No. 5

A representative sample of the minus 14-mesh ore was ground in a ball mill with soda ash, 4 pounds per ton; coal-tar creosote, 0·14 pound per ton; sodium ethyl xanthate, 0·2 pound per ton; pine oil, 0·05 pound per ton; and concentrated in a small flotation machine with the following results:—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values	Ratio of concentration
Heads	100·00	0·282	100·0	
Flotation concentrate	1·15	19·75	80·67	86·96 : 1
Flotation tailing	98·85	0·055	19·33	

Total recovery of gold, 80·67 per cent.

Screen test on the flotation tailing:—

Mesh	Weight, per cent
+100	0·40
−100+150	6·66
−150+200	14·76
−200	78·18

AMALGAMATION AND FLOTATION

Test No. 6

A sample of the ore was ground dry to pass a 48-mesh screen and amalgamated. The amalgam was separated and the residue was floated with soda ash, 4 pounds per ton; coal-tar creosote, 0·14 pound per ton; sodium ethyl xanthate, 0·2 pound per ton; pine oil, 0·05 pound per ton, with the following results:—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values	Ratio of concentration
Heads	100·00	0·285	100·00	.....
Amalgamation tailing	.....	0·149	.....	.....
Recovery by amalgamation	.....	.....	47·72	.....
Flotation concentrate	1·24	8·10	35·13	80·65 : 1
Flotation tailing	98·76	0·05	17·15	.....

	Per cent
Recovery by amalgamation	47·72
Recovery by flotation	35·13
Total recovery	82·85

Screen test on the flotation tailing:—

Mesh	Weight, per cent
+ 65	9·80
− 65+100	21·76
−100+150	14·61
−150+200	8·89
−200	44·94

AMALGAMATION, REGRINDING, AND FLOTATION

Test No. 7

A sample of the ore was ground to pass 48 mesh and amalgamated. The residue was ground in ball mill with soda ash, 4 pounds per ton, and coal-tar creosote, 0·14 pound per ton; and floated with sodium ethyl xanthate, 0·2 pound per ton, and pine oil, 0·05 pound per ton, with the following results:—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values	Ratio of concentration
Heads	100·00	0·285	100·00	.....
Amalgamation tailing	.....	0·18	.....	.....
Recovery by amalgamation	.....	.....	38·11	.....
Flotation concentrate	1·78	7·15	44·67	56·18 : 1
Flotation tailing	98·22	0·05	17·22	.....

	Per cent
Recovery by amalgamation	38·11
Recovery by flotation	44·67
Total recovery	82·78

Screen test on the flotation tailing:—

Mesh	Weight, per cent
+ 65	0·00
− 65+100	3·62
−100+150	15·90
−150+200	14·84
−200	65·64

BLANKET CONCENTRATION

Test No. 8

A sample of the ore was ground in a ball mill in a pulp dilution of 2 ore to 1 water. The blanket cloth used was the standard corduroy blanket set on a table with a slope of 4 inches to the foot. Water used was sufficient to wash away practically all fine gangue material. The results were as follows:—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values	Ratio of concentration
Heads	100·00	0·285	100·00	
Concentrate	1·75	9·54	58·63	57·14 : 1
Tailing	98·25	0·12	41·37	

Screen test on the blanket tailing:—

Mesh	Weight, per cent
+ 48	
− 48+ 65	0·06
− 65+100	2·44
−100+150	7·10
−150+200	11·40
−200	79·00

Test No. 9

A sample of the ore was ground in a ball mill in a pulp dilution of 2 : 1 as in Test No. 8. The blanket table slope was $1\frac{5}{16}$-inch to the foot. Using a minimum of water considerable gangue was left on the blanket. The results were as follows:—

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values	Ratio of concentration
Heads	100·00	0·285	100·00	
Concentrate	7·71	2·38	64·39	12·97 : 1
Tailing	92·29	0·11	35·61	

Screen test on the blanket tailing:—

Mesh	Weight, per cent
+ 48	.....
− 48+ 65	0·26
− 65+100	3·00
−100+150	10·52
−150+200	13·14
−200	73·08

AMALGAMATION AND FLOTATION

Test No. 10

The ore ground minus 14 mesh was fed at the rate of 99 pounds per hour to a 12-inch by 24-inch rod mill. The mill discharged directly on to an amalgamating plate. The tailing from amalgamation was pumped to a conditioning tank and thence to the flotation unit.

Reagents added to conditioning tank:—

Na_2CO_3	1·5 lb./ton
Sodium amyl xanthate	0·10 "

Oil Mixture:

60 per cent coal-tar creosote 20 per cent coal tar 20 per cent cresylic acid	0·10 lb./ton

Reagents to Cells:

Pine oil	0·06 lb./ton

Grind: Rod mill discharge.

Mesh	Weight, per cent
−100	3·4
−100+200	19·2
−200	77·4

Assays:

Product—	Au, oz./ton
Feed	0·355
Mill discharge	0·26
Plate discharge	0·14
Flotation feed	.....
Flotation concentrate	3·39
Flotation tailing	0·02

Summary of Results

Product	Per cent of total Au	Recovered by amalgamation, per cent	Per cent of total Au recovered
Mill clean-up	26·76	91·03	24·36
Amalgam	33·80	.....	33·80
Flotation concentrate	33·80	.....	33·80
Flotation tailing	5·64	.....	.....
Totals	100·00	.....	91·96

Ratio of concentration, 28·1: 1.

Cyanidation of Flotation Concentrate

Test No. 10	Agitation, hr.	Assay		Extraction, per cent	Reagent consumption, lb./ton	
		Heads, Au, oz./ton	Tailing, Au, oz./ton		KCN	CaO
Straight 1	24	3·39	0·635	81·27	6·68	17·6
Cyanide 2	48	3·39	1·25	63·13	7·02	18·1
Regrind 3	24	3·39	0·50	85·25	8·12	23·3
" 4	48	3·39	0·565	83·33	9·03	23·8

AMALGAMATION OF FLOTATION CONCENTRATE FROM MILL RUN

Test No. 11

A sample of the concentrate was amalgamated. A similar sample was ground in a ball mill and amalgamated.

Results:

Product	Assay		Extraction, per cent
	Heads,* Au, oz./ton	Tailing, Au, oz./ton	
Straight amalgamation	4·56	3·52	22·81
Grind and amalgamation	4·56	2·97	34·87

*Sample of flotation concentrate used in Test 11 was not the same as that used in Test 12, hence difference in head assays.

Screen Test on the Amalgamation Tailing from Sample No. 2

Mesh	Weight, per cent
+ 65	0·04
− 65+100	0·36
−100+150	1·60
−150+200	3·54
−200	94·46

CYANIDATION OF FLOTATION CONCENTRATE FROM MILL RUN

Test No. 12

A sample of the wet flotation concentrate was treated by cyanidation in a pulp containing three parts of solution to one part of concentrate. The strength of the sodium cyanide solution used was equivalent to 5·0 pounds of potassium cyanide per ton of solution, and lime equivalent to 10 pounds per ton of ore was added to supply a protective alkalinity. The results were as follows:—

Product	Agitation period, hr.	Assay		Extraction, per cent	Reagent consumption, lb./ton	
		Heads, Au, oz./ton	Tailing, Au, oz./ton		KCN	CaO
Tailing	24	3·39	0·635	81·27	6·68	17·6
"	48	3·39	1·25	63·13	7·02	18·1

NOTE.—The cyanide pulp frothed badly during the agitation period.

Test No. 12A

A sample of the wet flotation concentrate was ground in a ball mill in a dilution of 2 : 1. The pulp was dewatered by settling with lime and siphoning off excess. The pulp was tested by cyanidation in a pulp containing $2\frac{1}{2}$ parts of solution to one part of concentrate. The strength of the sodium cyanide solution used was equivalent to 5·0 pounds of potassium cyanide per ton of solution, and lime equivalent to 10 pounds per ton of ore was added to supply a protective alkalinity. The results were as follows:—

Product	Agitation period, hr.	Assay		Extraction, per cent	Reagent consumption, lb./ton	
		Heads, Au, oz./ton	Tailing, Au, oz./ton		KCN	CaO
Tailing	24	3·39	0·50	85·25	8·12	23·3
"	48	3·39	0·565	83·33	9·03	23·8

NOTE.—The cyanide pulp frothed very badly during the agitation period.

Screen test of reground flotation concentrate:—

Mesh	Weight, per cent
+100	0·3
−100+150	0·5
−150+200	0·9
−200	98·3

CONCLUSIONS

The results of the tests conducted on this ore show that there is free gold which can be caught by amalgamation or on blankets. The recovery varies between 50 and 60 per cent, depending on how fine the ore is ground. The continuous flotation tests made on amalgamation tailing give a final tailing of 40 cents, which is equivalent to a loss of 5·64 per cent in the final flotation tailing. The grinding in this case was 77·4 per cent minus 200 mesh, and about 58 per cent of the gold was extracted by amalgamation before flotation. The flotation concentrate assays 3·4 ounces in gold with a ratio of concentration of 28 : 1 and contained 38·8 per cent of the total values in the ore.

The results of the cyanide tests made on this concentrate are disappointing inasmuch as they show a comparatively poor extraction and consequently a high tailing. The disturbing feature of these tests was the erratic results obtained, and when considered in conjunction with the results of the straight cyanide tests made on the ore, which show a very good extraction, this was surprising because these tests showed clearly that there were no refractory conditions present in the ore. The accuracy of the results of the cyanide tests on the flotation concentrate was therefore open to question. The details of the procedure followed in making these tests were checked and it was found that an excess of lime was used during the cyanidation period, and that at the end of the agitation period there was the equivalent of over 1 pound of lime per ton of solution present.

The cyanidation of the concentrates, therefore, was repeated, reducing the amount of lime. A sample of the concentrate used in Test No. 11 was given a short grind, 1 : 1 dilution with a 5·0-pound KCN solution. Lime at the rate of 17 pounds per ton ore was added to the mill. The pulp was then made up to 3 : 1 dilution and the solution strength brought to 5·0 pounds KCN per ton. No additional lime was added.

After 24 hours' agitation, additional cyanide was added to bring the solution back to 5 pounds per ton.

At the end of 48 hours, the test was concluded, the solutions titrating 5·6 pounds KCN and 0·2 pound CaO per ton.

Results:

Heads	Au,	4·56	oz./ton
Tailing	Au,	0·405	"
Recovery		91·1	per cent

The test shows that a lower tailing is made from the concentrate when the lime in solution is kept low. An excess of this reagent over that necessary to produce a clear thickener overflow retards solution of gold.

Report No. 461

EXPERIMENTAL TESTS ON GOLD ORE FROM THE MACKEY POINT SYNDICATE, MICHIPICOTEN AREA, ONT.

Shipment. A shipment of 266 pounds of gold ore was received on November 9, 1932, from Dr. Eng. F. Esser, President, Metallum, Limited, Coristine Building, Montreal, Canada. The shipment consisted of three samples designated 1, 2, and 3.

Characteristics of the Ore. Polished sections were made from specimens selected from these samples, and were examined under the microscope. In addition, hand specimens were studied by means of the binocular microscope.

Sample No. 1. The minerals observed in Sample No. 1 are: pyrrhotite, pyrite, chalcopyrite; quartz, dolomite (or ankerite (?)), and calcite; "limonite."

The gangue consists chiefly of light grey glassy quartz; considerable amounts of dolomite (or ankerite (?)) and small amounts of calcite occur as stringers and small irregular veinlets in the quartz.

The sulphides are associated with the carbonate stringers. Both pyrrhotite and pyrite are common and both, but especially the former, have undergone considerable alteration to "limonite". The pyrrhotite occurs in granular masses, as does the pyrite, and is sometimes veined by the latter. Certain grains of pyrite, due to the fact that they contain few inclusions of gangue minerals, take a smooth polish; these appear as "islands" in another type of pyrite which does not take a good polish because of numerous minute inclusions of other minerals. Of these two generations of pyrite, the "island" type is the older.

Chalcopyrite is not uncommon but occurs in small quantities as individual grains associated with the other sulphides.

No native gold was seen either in the hand specimens or in the polished section.

Sample No. 2. The minerals observed are: pyrrhotite, pyrite, chalcopyrite; quartz, carbonate; a black, hard silicate (possibly tourmaline).

The gangue is almost wholly composed of light grey glassy quartz. Very small amounts of carbonate and occasional small grains of a hard, black silicate were observed, but very little of the sulphide is accompanied by carbonate as in Sample No. 1.

Pyrrhotite occurs in granular masses and probably forms a considerably larger portion of the sulphides than does pyrite.

Pyrite forms discontinuous irregular stringers in the quartz, and is very rarely accompanied by minute quantities of chalcopyrite.

No gold was seen either in the hand specimen or in the polished section.

Sample No. 3. The minerals observed are: pyrrhotite, pyrite, chalcopyrite, "limonite"; quartz, tourmaline.

The gangue is chiefly smoky to grey glassy quartz. Commonly, small needles of tourmaline are present in the quartz, and occasionally they are numerous.

Rather prominent granular masses of pyrrhotite occur in the quartz, and in extremely rare instances, minute quantities of chalcopyrite are associated with this mineral. "Limonite" is present as an alteration product of the pyrrhotite.

No gold was seen in either the hand specimens or the polished section.

Comparison of the Characteristics of the Samples. The material of Sample No. 1 exhibits characteristics which indicates that part of the mineralization represented took place at a definitely lower temperature and from solutions of considerably different chemical composition than did the mineralization characteristic of Samples No. 2 and No. 3. What

may be termed the "pyrite-carbonate association" is prominent in Sample No. 1, is almost absent in Sample No. 2, and was not seen in Sample No. 3. Conversely, in proportion to the "pyrite-carbonate association," pyrrhotite is more prominent in Sample No. 2 than in Sample No. 1, and in Sample No. 3 it comprises nearly the total of the sulphides. Moreover, tourmaline was not seen in Sample No. 1, is rare in Sample No. 2, and is present in considerable amount in Sample No. 3. This proportionate increase in pyrrhotite, and the presence of tourmaline, or in other words, the presence of the "tourmaline-pyrrhotite association," indicates a higher temperature of formation for No. 2 and No. 3.

Assays[1] show that Sample No. 1 carries 1·74 ounces, Sample No. 2 carries only 0·112 ounce, and Sample No. 3 carries only 0·36 ounce to the ton in gold. These assays indicate that a large proportion of the gold was deposited with the "pyrite-carbonate association," or at least that this "association," rather than the higher temperature "tourmaline-pyrrhotite association," formed the locus for the major gold mineralization.

Purpose of Experimental Tests. The shipment was made for the purpose of determining the most efficient method to apply for the recovery of the contained gold.

Sampling and Analysis. Each of the three lots of ore was sampled separately. After sampling the composite sample was made by mixing all three shipments.

The average assays were as follows:—

Sample	Au, oz./ton
No. 1	1·74
No. 2	0·112
No. 3	0·36
Composite	0·39

Experimental Tests

Sample No. 1 was treated by the following methods:—

1. Amalgamation and cyanidation at −48 mesh.
2. Amalgamation and cyanidation at −100 mesh.
3. Straight cyanidation at −48, −100, −150, −200 mesh.

Composite Sample:

The mixed ore was treated by the following methods:—

4. Blanket concentration.
5. Blanket concentration followed by amalgamation of the blanket concentrate and flotation of the blanket tailing.
6. Blanket concentration followed by flotation of the blanket tailing and cyanidation of the flotation concentrate.

[1]Assays from the Assay Laboratory, Ore Dressing and Metallurgical Laboratories, Mines Branch, Ottawa.

AMALGAMATION AND CYANIDATION

Test No. 1

A representative portion of the ore was ground to pass 48 mesh and amalgamated. After removing the amalgam the residue was treated by cyanidation in a pulp containing three parts of solution to one of ore. The strength of the sodium cyanide solution used was equivalent to 1·0 pound of potassium cyanide per ton of solution, and lime equivalent to 5 pounds per ton of ore was added to supply a protective alkalinity.

Results:

Head assay	Au, 1·74 oz./ton
Amalgamation tailing assay	Au, 0·17 "
Recovery by amalgamation	90·2 per cent
Cyanide tailing after 24-hr. agitation	0·06 oz./ton
Cyanide tailing after 48-hr. agitation	0·04 "
Total recovery after 24-hr. agitation	96·55 per cent
Total recovery after 48-hr. agitation	97·70 "

Reageant consumption, lb./ton of ore:

	KCN	CaO
24-hr. period	0·84	3·60
48-hr. period	0·91	3·85

Screen Analyses of the Amalgamation Tailing

Mesh	Weight, per cent	Assay, Au, oz./ton	Per cent of values
+ 65	10·0	0·235	14·11
− 65+100	23·97	0·17	24·43
−100+150	19·31	0·125	14·47
−150+200	12·81	0·135	10·38
−200	33·91	0·18	36·61

Test No. 2

A representative sample of the ore was ground to pass a 100-mesh screen and treated as in Test No. 1.

Results:

Head assay	Au, 1·74 oz./ton
Amalgamation tailing assay	Au, 0·14 "
Recovery by amalgamation	91·95 per cent
Cyanide tailing after 24-hr. agitation	0·03 oz./ton
Cyanide tailing after 48-hr. agitation	0·03 "
Total recovery after 24-hr. agitation	98·28 per cent
Total recovery after 48-hr. agitation	98·28 "

Reagent consumption, lb./ton of ore:

	KCN	CaO
24-hr. period	1·54	4·10
48-hr. period	1·68	4·06

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A screen test on the amalgamation tailing shows the ore to be ground to the following sizes:—

Mesh	Weight, per cent
+100	0·15
+200	35·00
−200	64·85

STRAIGHT CYANIDATION

Test No. 3

Representative samples of the ore were crushed to pass 48-, 100-, 150-, 200-mesh screens. The ore was treated by cyanidation in a pulp containing three parts of solution to one part of ore. The strength of the sodium cyanide solution used was equivalent to 1·0 pound of potassium cyanide per ton of solution, and lime equivalent to 5 pounds per ton of ore was added to supply a protective alkalinity.

Results:

Mesh grind	Agitation period, hr.	Assay		Extraction, per cent	Reagent consumption, lb./ton	
		Heads, Au, oz./ton	Tails, Au, oz./ton		KCN	CaO
− 48	24	1·74	0·25	85·63	0·90	3·56
− 48	48	.....	0·05	97·13	0·60	3·5
−100	24	.....	0·05	97·13	1·05	3·95
−100	48	.....	0·03	98·28	1·05	3·65
−150	24	.....	0·025	98·56	0·75	4·90
−150	48	.....	0·02	98·85	0·90	5·11
−200	24	.....	0·02	98·85	1·50	5·41
−200	48	.....	0·01	99·43	1·35	5·8

BLANKET CONCENTRATION

Test No. 4

A representative sample of minus 14-mesh ore from the composite sample was crushed in a rod mill, dilution 2 : 1, and passed over a standard corduroy blanket. The blanket concentrate was assayed, the blanket tailing was screen-tested and assayed. A screen test on the blanket tailing shows the ore was crushed to the following sizes:—

Mesh	Weight, per cent
+ 65	0·38
− 65+100	10·52
−100+150	23·61
−150+200	16·05
−200	49·44

Results:

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Ratio of concentration
Heads	100·0	0·39	100·0	
Blanket concentrate	3·25	9·92	82·67	30·77 : 1
Blanket tailing	96·75	0·07	17·33	

BLANKET CONCENTRATION

Test No. 5

A representative sample of minus 14-mesh ore from the composite sample was crushed in a rod mill, dilution 2 : 1, and passed over a standard corduroy blanket. The blanket concentrate was amalgamated. The blanket tailing was floated with soda ash, 4 pounds per ton; coal-tar creosote, 0·07 pound per ton; sodium ethyl xanthate, 0·1 pound per ton; and pine oil, 0·05 pound per ton.

A screen test on the flotation tailing shows the ore was ground to the following sizes:—

Mesh	Weight, per cent
+ 65	0·90
− 65+100	17·92
−100+150	21·30
−150+200	12·64
−200	47·24

Results:

Blanket Concentration

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Recovery, per cent	Ratio of concentration
Heads	100·0	0·39	100·00	.....	
Blanket concentration	2·60	10·95	73·00	73·00	38·46 : 1
Blanket tailing	97·40	0·094	27·00	27·00	

Amalgamation of Blanket Concentrate

Amalgamation heads	.....	10·95	97·61	71·26	.....
Amalgamation tailing	.....	0·26	2·39	1·74	.....

Flotation of Blanket Tailing

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Recovery, per cent	Ratio of concentration
Blanket tailing	97·40	0·094	100·0	.....	
Flotation concentrate	2·00	3·39	73·94	19·96	50 : 1
Flotation tailing	95·40	0·025	26·06	7·04	

Summary:

	Per cent
Recovery in blanket concentration	73·00
Recovery in flotation concentrate	19·96
Total	92·96
Recovery of gold in the form of bullion by amalgamation of blanket concentrate	71·26

Test No. 6

A representative sample of minus 14-mesh ore from the composite sample was ground in a rod mill, and concentrated on a standard corduroy blanket. The blanket concentrate was screened on a 100-mesh screen to take out the metallics. The blanket tailing was floated as in Test No. 5. The flotation concentrate was treated by cyanidation in a pulp containing three parts of solution to one part of concentrate. The strength of the sodium cyanide solution was equivalent to 5·0 pounds of potassium cyanide per ton of solution, and lime at the rate of 10 pounds per ton of concentrate was added to supply a protective alkalinity. A screen test on the flotation tailing shows the ore was crushed to the following sizes:

Mesh	Weight, per cent
+ 65	1·43
− 65+100	16·21
−100+150	22·81
−150+200	13·43
−200	46·12

Results:

Blanket Concentration

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Recovery, per cent	Ratio of concentration
Heads	100·0	0·39	100·0	.....	
Blanket concentrate	2·97	12·12	92·31	92·31	33·67 : 1
Blanket tailing	97·03	0·09	7·69	7·69	

Flotation of Blanket Tailing

Blanket tailing	97·03	0·09	100·0	.....	.....
Flotation concentrate	2·17	2·77	67·83	5·22	46·08 : 1
Flotation tailing	94·86	0·03	32·17	2·47	

Cyanidation of Flotation Concentrate

Product	Weight, per cent	Assay, Au, oz./ton	Gold, per cent	Recovery, per cent	Reagent consumption lb./ton	
					KCN	CaO
Flotation concentrate	.....	2·77	98·38	5·13	.....	.....
Cyanide tailing	.....	0·045	1·62	2·56	37·12	38·45

SUMMARY

	Per cent
Recovery in blanket concentrate	92·31
Recovery in flotation concentrate	5·22
Total	97·53

It was determined in Test No. 5 that 97·61 per cent of the gold in the blanket concentrate could be secured as bullion. Therefore the extraction of gold indicated by this test is:

	Per cent
Gold recovered in the form of bullion by barrel amalgamation of blanket concentrate	90·1
Gold recovered by cyanidation of flotation concentrates	5·1
Total	95·2

Report No. 462

EXPERIMENTAL TESTS ON A SHIPMENT OF GOLD-SILVER-COPPER-LEAD-ZINC ORE FROM TEDDY GLACIER MINE AT CAMBOURNE, B.C.

Shipment. A shipment of seven sacks of ore, net weight 700 pounds, was received October 13, 1932, from A. S. MacCulloch, Director, Teddy Glacier Mines, Limited, 804 Standard Bank Building, Vancouver, B.C.

Characteristics of the Ore. [1]The ore from Teddy Glacier mine consists of heavy sulphides with lesser amounts of quartz gangue. The sulphides include pyrite, sphalerite, galena, chalcopyrite, and tetrahedrite(?) of which the first three are abundant, the fourth is present in minor amount, and the last is comparatively rare.

Pyrite is the earliest formed sulphide, and is much fractured and brecciated. The remaining sulphides are mutually associated in varying proportions and form comparatively large masses and stringers as well as the aggregate of minerals which form the veinlet fillings and the matrix for the shattered pyrite.

The sphalerite commonly contains numerous dots of chalcopyrite; the latter mineral may also form larger masses as the matrix of the brecciated pyrite fragments. Tetrahedrite(?) was observed only with the galena.

No free gold was seen, and its mode of occurrence is not known. From the resemblance of the ore to others of the same type, it is highly probable that the silver occurs in the tetrahedrite(?) and in solid solution in the galena.

An average analysis of the ore was as follows:

Au	0·66 oz./ton
Ag	9·78 "
Cu	1·07 per cent
Pb	15·54 "
Zn	23·00 "
Fe	16·64 "
S	29·90 "
Insol	11·10 "

[1]From report of Mineragraphic Laboratory by Maurice Haycock.

Experimental Tests

The shipment was sampled and a small-scale flotation test made on it. A copper-lead concentrate was produced containing 61 per cent of the gold, 84 per cent of the silver, 86 per cent of the copper, and almost 97 per cent of the lead. Over 94 per cent of the total gold is contained in the three concentrates produced. Details of the test follow:—

FLOTATION

Test No. 2

The ore at minus 14 mesh was crushed in a Denver rod mill for 30 minutes, then floated.

Charge to rod mill:	
Ore	2,000 grammes
Water	1,000 c.c.
Thiocarbanalide	0·1 lb./ton
Sodium cyanide	0·2 "
Soda ash	9·0 "
Copper-lead float:	
Cresylic acid	0·2 lb./ton
Zinc float:	
$CuSO_4$	1·0 lb./ton
Sodium Aerofloat	0·1 "
Cresylic acid	0·07 "
Zinc concentrate recleaned:	
Cresylic acid	0·07 lb./ton
Pyrite float:	
Potassium amyl xanthate	0·2 lb./ton
Pine oil	0·1 "
Tarol No. 1	0·02 "

Results:

Product	Weight, per cent	Analyses					Recoveries, per cent				
		Au oz. ton	Ag oz. ton	Cu %	Pb %	Zn %	Au oz./ton	Ag oz./ton	Cu %	Pb %	Zn %
Cu-Pb concentrate..	25·8	1·42	26·24	3·64	48·80	11·30	61·2	84·1	85·8	96·7	12·0
Zn concentrate.....	33·6	0·22	2·02	0·33	0·60	60·50	12·3	8·5	10·2	1·5	83·8
Zn middlings.....	5·4	0·54	3·12	0·43	1·86	11·80	4·9	2·1	2·1	0·8	2·6
Fe concentrate.....	23·2	0·54	1·50	0·08	0·30	1·50	20·9	4·3	1·7	0·5	1·4
Tailing.....	12·0	0·035	0·68	0·02	0·55	0·25	0·7	1·0	0·2	0·5	0·2

CONCLUSIONS

The test carried out shows that the ore concentrates quite readily. No further work was done because it was felt that results obtained on a sample so rich in sulphides as this one might not be duplicated on milling grade ore. It is, therefore, suggested that another sample, more representative of milling grade, be submitted and more detailed work will be undertaken.

Report No. 463

MAGNETIC SEPARATION OF VANADIUM-BEARING TITANIFEROUS MAGNETITE FROM MINE CENTRE, ONTARIO

Shipment. A shipment of four lots of ore samples was received October 27, 1931, from W. M. Goodwin, Ste. Anne-de-Bellevue, Quebec. The details of the shipments were as follows:—

Pit A	30,881 pounds
" B	29,666 "
" C	29,434 "
" D	3,320 "

Magnetite and ilmenite occur in the ore in all of the three pits, A, B, and C, and are everywhere intimately associated with each other. The ilmenite occurs in three ways:—

(a) As fine lamellæ within the magnetite.
(b) As small, irregular blebs within the magnetite.
(c) As distinct, interlocking grains associated with the magnetite.

Observations indicate that with comparatively low percentages of ilmenite, occurrence *(a)* is most common, and that with increasing amounts of this mineral, occurrences *(b)* and *(c)* become prominent.

No replacement of the magnetite by silica (or silicate) was noted in sections from Pits A or B, but a small amount was noted in ore from Pit C.

The only metallic minerals observed other than magnetite and ilmenite were pyrite, pyrrhotite, and chalcopyrite in ore from Pit B, and pyrite in ore from Pit C.

The titanium occurs mainly in the mineral ilmenite ($FeTiO_3$).

No information was obtained concerning the manner of occurrence of the vanadium. Spark spectra were produced of pure ilmenite, and a mixture composed essentially of magnetite with some ilmenite in the hope of obtaining vanadium lines in one or the other. This mode of excitation was, however, not sufficiently sensitive to disclose the presence of the small amounts of vanadium present.

Analyses of Samples:

—	Lot No. 3186	Lot No. 3187	Lot No. 3188	Lot No. 3189
	Pit A No. 3217	Pit B No. 3122	Pit C No. 3088	Pit D No. 356
	%	%	%	%
Fe	38·31	32·66	38·89	44·76
Fe_2O_3	27·78	19·70	26·87	21·02
FeO	24·29	24·16	25·86	38·67
TiO_2	5·73	5·78	9·00	25·60
SiO_2	20·31	27·20	17·75	3·40
V_2O_5	0·45	0·27	0·51	0·23
P_2O_5	trace	trace	nil	nil
MnO_2	0·49	0·33	0·38	0·50
CaO	0·08	3·04	0·20	0·12
MgO	11·39	9·52	8·90	1·30
Al_2O_3	4·15	4·25	6·05	4·45
S	0·08	0·41	0·42	0·05
Loss on ignition	2·24	1·24	1·15	nil

Experimental Tests

The shipments used in these tests were the samples from Pit A, Pit B, and Pit C. No work was done on the sample from Pit D, on account of the small amount received.

The following weights of samples were used in the making of the concentration tests:—

Pit A	15,217 pounds
" B	14,420 "
" C	10,085 "

Tests on Ore from Pit A and Pit C

These two samples were found by preliminary work to be very similar, hence the same procedure was carried out on both of them.

They were crushed to one inch in a jaw crusher and then ground in a Marcy rod mill to pass a 1/16-inch screen. The minus 1/16-inch product was

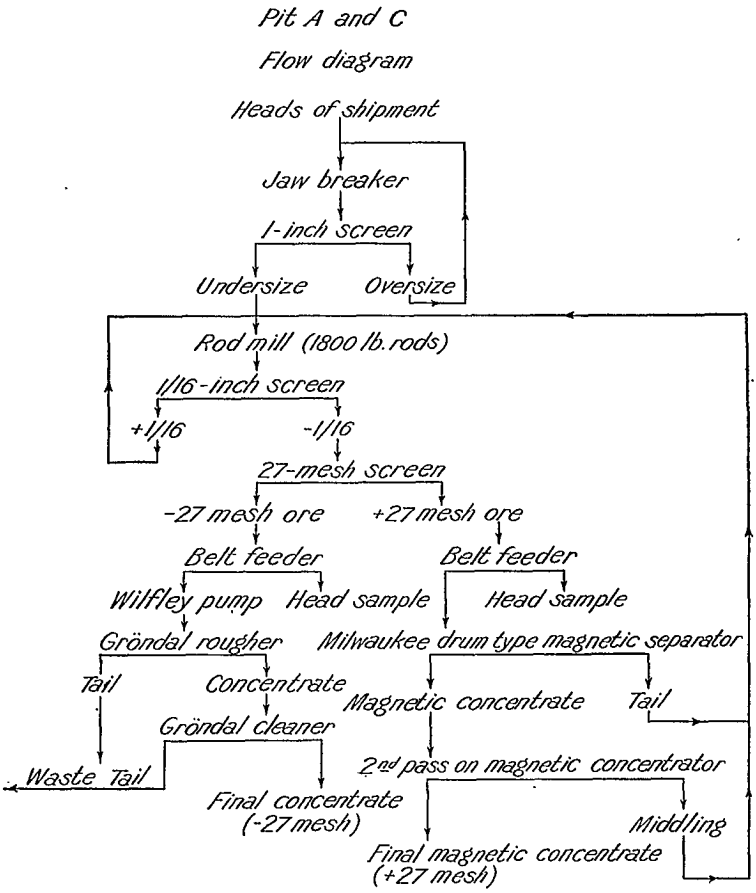


Figure 9. Flow-sheet No. 1, for treatment of magnetite ore from Mine Centre, Ontario.

screened on a vibratory type of screen to obtain a plus 27-mesh product and a minus 27-mesh product. Both products were sampled.

The 27-mesh product was fed to a dry type Milwaukee drum magnetic separator from which a final tailing and a low-grade concentrate were made. This concentrate was again passed over the separator and a final concentrate and tailing classed as a middling were obtained. The middling was re-crushed and added to the minus 27-mesh screen product.

The minus 27-mesh material was treated in a wet type Gröndal drum magnetic separator. This separator consisted of two drums arranged so that the second drum re-treated the concentrate from the first drum. A financial tailing was obtained from both drums. A flow-sheet giving the details of the run is shown as Figure 9.

Tests on Ore from Pit B

This ore was more difficult to concentrate than the samples from Pits A and C. The magnetic properties of the mineral were not so pro-

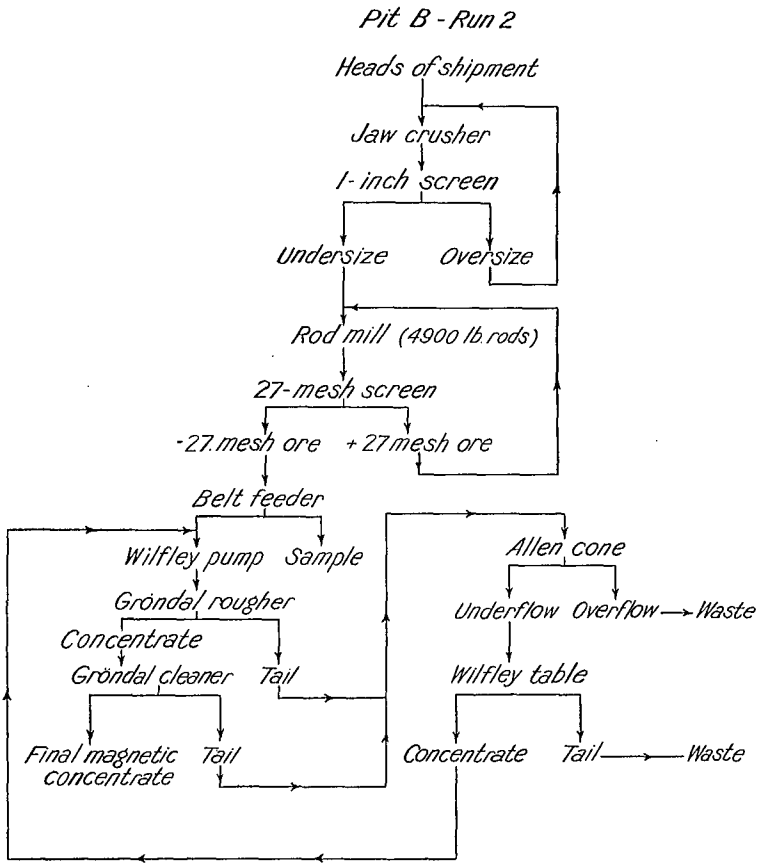


Figure 10. Flow-sheet No. 2, for treatment of magnetite ore from Mine Centre, Ontario.

nounced and it was difficult to make a low tailing. Finer grinding was also required to free the mineral. Two runs were made on this sample. In Run No. 1 the ore was crushed to pass a $\frac{1}{16}$-inch screen and fed to the Gröndal wet separators. The results were not satisfactory as the magnetic field of the separator was not strong enough to pick up the magnetite.

A second run was made on the sample crushed to pass a 27-mesh screen. The finer crushing did not improve the recovery and it was necessary to re-treat the tailings. This was accomplished by passing them over a Wilfley table after first thickening in an Allen cone. The table tailing was sent to waste and the concentrate was returned to the magnetic separator. Tabling was not very successful because most of the sulphides in the ore reported in the table concentrate and in order to eliminate this element a table concentrate had to be re-treated by magnetic separation. A recovery made by the magnetic separation when re-treating this product was low for the reason that the field was not strong enough to pick up the titaniferous magnetite.

A flow-sheet of the second run is shown as Figure 10.

Pit A, Mine Centre

	Weight	Analyses, per cent					Recovery, per cent				Grade				
	Tons	Fe	TiO_2	V_2O_5	SiO_2	S	Fe	TiO_2	V_2O_5	SiO_2	Fe	TiO_2	V_2O_5	SiO_2	Sulphur
															Wet + Dry
Heads	100·0	38·31	6·83	0·45	19·50	.....	100·00	100·00	100·0	100·0	.....	.....	.....	.....	−27 +27
Con. +27 mesh	36·10	53·60	10·75	0·64	6·10	0·04	31·75	27·12	51·45	11·30	.....	.....	.....	.....	
Con. −27 mesh	21·04	57·81	8·80	0·82	5·33	0·08	50·51	56·85	38·42	5·75	55·11	10·07	0·70	5·83	0·07
Con. re-treat	2·60	54·20	10·86	0·66	6·23	0·15	3·68	4·14	3·83	0·83	.....	.....	.....	.....	
Tailing	40·26	13·38	2·02	0·07	39·77	.....	14·06	11·89	6·30	82·12	.....	.....	.....	.....	

Ratio of concentration, 1·67 : 1.

Pit C Mine Centre

	Weight	Analyses, per cent					Recovery, per cent				Grade				
	Tons	Fe	TiO_2	V_2O_5	SiO_2	S	Fe	TiO_2	V_2O_5	SiO_2	Fe	TiO_2	V_2O_5	SiO_2	Sulphur
															Wet + Dry
Heads	100·00	38·88	7·50	0·49	19·50	.....	100·00	100·00	100·0	100·0	.....	.....	.....	.....	−27 +27
Con. +27 mesh	17·22	56·91	13·28	0·77	4·29	0·11	25·21	30·49	27·35	3·79	.....	.....	.....	.....	
Con. −27 mesh	33·47	59·48	11·21	0·86	3·36	0·11	51·20	50·02	59·35	5·76	58·48	11·88	0·62	3·75	0·15
Con. re-treat	1·46	54·20	10·86	0·66	6·23	0·15	2·03	2·12	1·98	0·47	.....	.....	.....	.....	
Tailing	47·85	17·51	2·72	0·11	36·68	.....	21·56	17·37	11·32	89·98	.....	.....	.....	.....	

Ratio of concentration, 1·92 : 1.

Pit B, Mine Centre, Runs Nos. 1 and 2

	Weight	Analyses, per cent				Recovery, per cent				Grade			
	Tons	Fe	TiO_2	V_2O_5	SiO_2	Fe	TiO_2	V_2O_5	SiO_2	Fe	TiO_2	V_2O_5	SiO_2
Run No. 1													
Heads	100·00	32·66	6·57	0·28	26·09	100·00	100·00	100·0	100·00	.....	.....	.....	.....
Concentrate	37·90	55·62	10·64	0·57	5·72	64·54	61·38	77·14	8·31	55·62	10·64	0·57	5·72
Tailing	62·10	18·65	4·09	0·100	38·52	35·46	38·62	22·86	91·69	.....	.....	.....	.....
Run No. 2													
Heads	100·00	32·66	6·57	0·28	26·09	100·00	100·00	100·0	100·00	.....	.....	.....	.....
Concentrate	36·71	56·61	11·31	0·60	5·06	63·63	63·19	78·68	7·12	56·61	11·31	0·60	5·06
Tailing	63·29	18·77	3·82	0·09	38·29	36·37	36·81	21·32	92·88	.....	.....	.....	.....

NOTE.—Sulphur in concentrate, 0·12 per cent.

NOTE.—Sulphur in concentrate, 0·09 per cent. Ratio of concentration:

Run No. 1, 2·64 : 1.
Run No. 2, 2·72 : 1.

III

REPORTS OF INVESTIGATIONS; NON-METALLIC MINERALS SECTION

Report No. 464

THE CONCENTRATION OF GARNET FROM RIVER VALLEY, ONTARIO

Shipments. Two shipments of garnet rock were received from Mr. A. G. Chew, River Valley, Ontario. The first, designated as Lot No. 1, consisted of 19 bags, gross weight 2,112½ pounds, and was received February 12, 1932. The second, designated as Lot No. 2, consisted of 9 bags, gross weight 1,238 pounds, and was received February 18, 1932. The garnet occurrence from which the shipments were made is situated on Lot No. 12, Concession VI, Gibbon Township, Nipissing District. The property is 5 miles north of the River Valley Station, on the Canadian National Railway, 41 miles northwest from North Bay.

Purpose of Experimental Tests. Concentration tests were requested on each lot to determine the quantity of recoverable garnet and a suitable method of concentration. Samples were to be submitted to the garnet paper makers for tests as to quality of the garnet produced.

Arrangements for Experimental Tests. V. L. Eardley-Wilmot of the Mineral Resources Division of the Mines Branch co-operated in the test work and agreed to arrange for the submittal of samples to a garnet paper maker.

Characteristics of the Garnet Rock. Both lots were similar in character. Lot No. 2 contained a greater percentage and better quality of garnet than Lot No. 1. The garnet, which occurs in gneiss, consists of crystals ranging in size up to over 2 inches in diameter, most being about 1½ inches. The crystals are embedded in biotite mica and chlorite, with considerable quartz and a small amount of feldspar. Some pyrite is present, usually as an intergrowth in the garnet crystals, which also contain inclusions of quartz.

Experimental Tests

The lots were crushed separately to all pass 4 mesh by means of a gyratory crusher, Symons disk crusher, and a small set of rolls which were in closed circuit with a Newaygo 4-mesh screen.

After crushing to 4 mesh each lot was sampled and the remainder was sized on a Hummer screen.

The following tables give the results of this work:—

Lot No. 1:	Pounds
Gross weight	2,112½
Net weight	2,094½
Weight crushed	2,090½
Sample	11¼
Used in test	2,079¼
− 4 + 5½ mesh	135
− 5½ + 7 "	534
− 7 +10 "	370
−10 +14 "	141
−14 +20 "	208
−20 +26 "	98
−26 +35 "	142
−35 +42 "	123
−42 +60 "	57
−60 "	185
Total weight of sizes	1,993

Lot No. 2:	Pounds
Gross weight	1,238
Net weight	1,231
Weight crushed	1,226
Sample	9
Used in test	1,217
− 4 + 5½ mesh	76
− 5½ + 7 "	281
− 7 +10 "	221
−10 +14 "	104
−14 +20 "	135
−20 +26 "	67
−26 +35 "	72
−35 +42 "	74
−42 +60 "	36
−60 "	103
Total weight of sizes	1,169

The sizes down to and including −10+14 were jigged in a two-compartment James jig. The jig beds were re-treated in a small Harz jig, the bed and hutch of which were put with the tailing. The James jig hutches, which were all minus 35 mesh, were collected together for each lot, dried and put with the original −35+42.

The original sizes from −10+14 down were tabled on a large Wilfley table having a middling return. At the end of each run on the large table any middling remaining was re-tabled on a small Wilfley table.

The following tables give the results of the jig and table concentration tests:—

Lot No. 1: Size	Tailing lb.	Concentrate lb.	Concentrate oz.
− 4 + 5½	90	35	13
− 5½ + 7	356	173	9
− 7 +10	279	90	4
−10 +14	110	31	4
−14 +20	160	48	10
−20 +26	77	23	2
−26 +35	105	36	9
−35 +42	88	40	1
−42 +60	37	18	9
−60	85	57	4
Total	1,387	555	1

Lot No. 2: Size	Tailing lb.	Concentrate lb.	oz.
− 4 + 5½	64	19	5
− 5½ + 7	170	109	0
− 7 +10	144	78	9
−10 +14	71	33	11
−14 +20	83	41	6
−20 +26	52	21	7
−26 +35	49	24	7
−35 +42	53	28	7
−42 +60	24	12	15
−60	42	31	13
Total	752	401	0

The jig and table concentrates of each lot were treated on an Ullrich magnetic machine. Each size was run separately, first to lift the garnet and leave sand, and then the garnet was re-run with a lower current to lift out any strongly magnetic material, such as pyrrhotite and metallic iron, from the crushers.

The following tables give the data obtained from the magnetic concentration:—

Lot No. 1:

Size	Amperes to lift garnet	Sand product, lb.	oz.	Amperes to lift iron	Iron product, oz.	Garnet, lb.	oz.
− 4 + 5½	5	0	2	½	½	35	10
− 5½ + 7	4	2	1	½	10	170	11
− 7 +10	4	0	8	½	8	89	4
−10 +14	3½	1	0	½	1	30	3
−14 +20	3½	3	0	½	1	45	8
−20 +26	3½	1	4	½	½	21	14
−26 +35	3½	1	1	½	½	35	7
−35 +42	3½	1	12	½	2	38	3
−42 +60	4	0	7	½	1	18	1
−60	4½	3	6	½	4	53	8
Total		14	9		28½	538	5

Lot No. 2:

Size	Amperes to lift garnet	Sand product lb.	oz.	Amperes to lift iron	Iron product oz.	Garnet lb.	oz.
− 4 + 5½	5	0	2	½	1	19	1
− 5½ + 7	4	1	12	½	11	106	8
− 7 +10	4	1	13	½	9	76	1
−10 +14	3½	1	6	½	3	32	2
−14 +20	3½	2	6	½	4	38	12
−20 +26	3½	1	2	½	2	20	2
−26 +35	3½	0	15	½	3	23	3
−35 +42	3½	2	3	½	4	25	15
−42 +60	4	0	10	½	2	12	2
−60	3	4	4	½	8	27	0
Total		16	9		47	380	14

70362—15

In some sizes the products add up to more than the feed to the jig or table. In the −35+42 size this is owing to the addition of the jig hutch. In other sizes it is owing to leaks in the machines, the clean-up from these leaks having been put into the size that they most closely resembled.

SUMMARY OF EXPERIMENTAL TESTS

From 2,079¼ pounds of Lot No. 1, 538 pounds 5 ounces of garnet concentrates were recovered, equivalent to 25·9 per cent of the feed.

From 1,217 pounds of Lot No. 2, 380 pounds 14 ounces of garnet was recovered, equivalent to 31·3 per cent of the feed.

The amount of iron product removed from the garnet by magnetic separation is very small, being about 0·5 per cent.

The amount of sand product removed from the jigged garnet by magnetic separation is 1½ per cent, whereas the amount removed from the tabled garnet is from 5 to 8 per cent.

The final garnet concentrates from Lot No. 2 are of higher grade than those from Lot No. 1 when similar sizes are compared, there being less gangue.

The final garnet concentrates from the four jig sizes of both lots contain quite a percentage of gangue, the concentrate from Lot No. 1 containing more than that from Lot No. 2.

The final garnet concentrates from the table sizes of Lot No. 2 appear to be of good quality and those of Lot No. 1 of fair quality.

CONCLUSIONS

1. The garnet rock received contains a high percentage of garnet, very much higher than deposits which are worked at present.

2. The garnet can easily be recovered by crushing, sizing, jigging, and tabling, and magnetic separation as described in the test work.

3. Although desirable it is not necessary to use magnetic treatment for the removal of iron, as the amount of this product is very small.

4. If the garnet paper makers accept the concentrates from the table sizes, but do not desire the concentrates from the jig sizes on account of the gangue present, it would be necessary to crush finer and table all the sizes.

5. It will be necessary to have the different sizes of garnet concentrates tested for abrasive-paper making before going any further with the development of the property.

Report No. 465

THE TREATMENT OF DIATOMITE FROM MARTIN SIDING, ONTARIO

Shipments. Two shipments of crude wet diatomite were received from the property of Diatomite Products, Limited, at Martin Siding, Muskoka, Ontario. The first shipment, designated Lot No. 1, was received June 22, 1931, and consisted of 9 bags having a net weight of 413 pounds. The second shipment, designated Lot No. 2, was received September 21, 1931; it contained 145 pounds of wet diatomite in two bags.

V. L. Eardley-Wilmot of the Mineral Resources Division of the Mines Branch arranged with the Diatomite Products, Limited for these shipments, and co-operated in the test work.

Purpose of Experimental Tests. Tests were desired to determine what products could be secured from the diatomite by calcining, crushing, and the removal of grit by air separation.

Occurrence of the Diatomite. The diatomite occurs in the bottom of Slocombe Lake from where it is pumped to a storage dam. The diatomite settles behind the dam and the water overflows. The samples received had been taken from behind this dam and were very wet and had the appearance of black muck.

Experimental Tests

The test work was divided into three parts, namely, calcining, crushing, and separating the grit from the fine diatomite.

CALCINING

Lot No. 1 was dried before calcining. A sample of the dried material contained 0·57 per cent Fe and the loss on ignition was 62·34 per cent. When the dried diatomite was calcined the resulting product was quite hard, which is an objectionable quality.

Lot No. 2 was calcined without pre-drying, and gave a much softer calcined product.

Calcining was done in an electric, T-grid furnace. This furnace consists of a horizontal iron cylinder, 16 inches in diameter and 6 feet long, closed at one end and open at the other. An electric T-grid heating element is placed outside the cylinder, and insulated outside; a means of slowly revolving the cylinder, and tilting the furnace up or down, and an automatic electric heat control are also provided.

The furnace was preheated to 500° F. and the diatomite was charged into the cylinder which was set with the closed end downwards so as to keep the charge in. The automatic control was set at 1200° F. and this temperature was reached in 1½ hours at which point it was maintained. In four hours from the time of charging the diatomite was completely calcined and the furnace was tilted and the contents discharged.

Lot No. 2, which had been taken from behind the dam at Martin Siding on September 7 and which weighed 145 pounds when received on September 21, had dried out to some extent by standing in the laboratory so that before calcining it weighed 133 pounds. This was divided into two parts of 66½ pounds each. The work on the first half was called Test No. 1 and that on the second half, Test No. 2. The calcining of Test No. 1 was done October 2, and that of Test No. 2, October 3. Six and one-half pounds of calcined material was obtained in Test No. 1 and five and one-half pounds in Test No. 2. The calcined material from both tests was soft, and buff in colour.

CRUSHING

In both Tests Nos. 1 and 2 the calcined diatomite was put through a Raymond No. 0000 pulverizer. The fineness regulator was not used. This allowed the fan to remove the diatomite readily from the beater chamber so that the action of the machine was more one of separating the particles than grinding them.

70362—15½

SEPARATING

The separation was done in a 30-inch Gayco centrifugal air separator set to give as fine a product as possible. In Test No. 1 the crushed diatomite was passed through the Gayco separator, giving a first fine product and a first oversize. The first oversize was passed through the machine again giving a second fine and a second oversize. The second oversize was passed through again and this process was repeated until five passes had been made, resulting in five final fine products and an oversize. After the fifth pass the machine was taken apart and cleaned; this gave a small amount of clean-up.

Results

Pass No.	Fines lb.	Fines, −325, per cent	Oversize lb.	Oversize, −325, per cent	Clean-up lb.
1	1	97·99	5	.....	.....
2	1½	98·28	3	.....	.....
3	1	98·91	2½	.....	.....
4	1	98·85	1	.....	.....
5	½	98·95	½	98·12	½

In Test No. 2 the calcined diatomite was passed through the Gayco separator only once, giving a fine and a coarse product. After the run the machine was cleaned. This gave a small amount of clean-up. Two screen tests were made on both the fines and oversize. This was because the fines contained more plus 325 mesh than the oversize and a check was desired. The plus 325 mesh in the fines is composed of large diatoms and the plus 325 mesh in the oversize contains large diatoms and grit. The plus 325 mesh in the clean-up also contains large diatoms and grit. The following table gives the results of the test:—

Product	Lb.	Oz.	Per cent −325 mesh
Fines	2	10	95·96 96·86
Oversize	2	0	98·53 98·22
Cleanout	0	8	96·62

Samples of the different products from Tests Nos. 1 and 2 were examined by Mr. Eardley-Wilmot. The following is extracted from his report:—

Test No. 1

The calcined product was pinkish buff in colour and somewhat harsh.

Micro Examination of Products

Gayco Oversize. Although fines greatly predominate, there are a few large diatoms such as Pinnularia, Neidium, Surirella, large Cymbella and Eunotia, Nitzchia and some Stauroneis. Amongst the small diatoms Melosira predominates, some being in fairly long chains, also Tabellaria, some Anomœoneis, etc. There is considerable fines and "ash" material, some sponge spicules and some grit.

Gayco Fines, Pass No. 1. About the same as oversize, but slightly more fines and less large diatoms, slightly more sponge spicules but less grit.

Gayco Fines, Passes Nos. 2-5. All very similar to pass No. 1, but less large diatoms in Nos. 4 and 5.

There is very little difference between the oversize and the various passes, but appreciably more grit in the former and more sponge spicules in the passes. There is only a very slight tendency for the largest diatoms to diminish in size through the passes.

Passes Nos. 1 and 5 were given a water settlement test. Pass No. 5 residue in a ¾-minute settling was small but contained about 50 per cent grit, about 15 per cent sponge spicules and 15-20 per cent spongy, brown masses and the rest very large diatoms and large diatom fragments of all the larger types.

Pass No. 1 settlings for the same period was very similar to those of pass No. 5, but contained perhaps less spongy "ash" material.

It is to be concluded therefore that no benefit was derived by keeping the different passes separate. A satisfactory separation could be made with straight oversize and fine products.

Test No. 2

Micro Examination of Products

Gayco Fines. About 75-80 per cent diatoms, some large, but mainly small and fragments, many types being represented; about 5-8 per cent grit and a few sponge spicules (*See* Test No. 1—Gayco Oversize for Diatoms).

Gayco Oversize. Diatoms mainly small and broken, with a few large and well preserved; some grit.

The Gayco fines may be regarded as a good product, while the Gayco oversize might be stored and subsequently run again when the fines obtained would be added to the original fines and the oversize might be used as a cheap filler.

On the whole these Gayco products are fairly satisfactory and with further adjustments in the machine may be improved upon.

CONCLUSIONS

1. Satisfactory products can be prepared from the crude diatomite by calcining, crushing, and separating the grit from the fine diatomite.
2. When the wet diatomite is fed to the calcining furnace a softer product is obtained than by drying before calcination.
3. The colour of the calcined diatomite was pinkish buff which might be improved by better methods of calcination. A sample calcined at Martin Siding by means of an oil burner had a lighter colour. This sample was not quite completely calcined but was very soft.
4. A pulverizer similar to the one used in the tests could be employed to crush the diatomite.
5. Air separators similar to the one employed in the test work or cyclone separators could be used to separate the diatomite into different grades. In all air separators, including cyclones, a lot of fine material is left in the oversize, and it might be necessary to re-run some of these oversizes to obtain the proper proportion of the different grades. In re-running the oversizes another separator could be used or part of the oversize could be automatically returned to the separator from which it came.

Report No. 466

THE SEPARATION OF GYPSUM AND DOLOMITE FROM AMARANTH, MANITOBA

Shipment. Three bags of gypsum containing dolomite, gross weight 200 pounds, were received on November 26, 1931, from the Western Gypsum Products Company, Amaranth, Manitoba.

Purpose of Experimental Tests. The Western Gypsum Products Company required assistance in developing a method to remove the dolomite from the gypsum in order to produce a better material.

Arrangements for Experimental Tests. Mr. L. Heber Cole of the Mineral Resources Division of the Mines Branch arranged with the Western Gypsum Products Company for the shipment to be sent in, and co-operated in the test work.

Characteristics of the Gypsum. The shipment consisted of white gypsum containing buff-coloured dolomite. The gypsum and dolomite are not intimately mixed. The gypsum forms the bulk of the rock with the dolomite scattered through it in sizes ranging from $\frac{1}{4}$ inch up to several inches.

Sampling and Analysis. After removing 3 pounds of specimens, the balance of the shipment, 195 pounds, was crushed to $\frac{1}{2}$ inch. A head sample was obtained from the $\frac{1}{2}$-inch material by repeatedly cutting with a Jones riffle and crushing the portion cut out to a smaller size. The sample gave on analysis the following:—

	Per cent
H_2O	15·64
CaO	31·74
MgO	4·36

The analysis indicates the shipment to be about 74·8 per cent gyspum and 20·1 per cent dolomite.

Experimental Tests

The following tests were conducted:—

Jigging. A jigging test was tried on a small quantity of the gypsum $-\frac{1}{2}$ inch $+$ 6 mesh with poor results.

Another test on the same material sized on $\frac{3}{8}$ inch, 3 and 4 mesh, gave fair results. The best concentrate was 96·7 per cent gypsum and the poorest 79·2 per cent. The best results were secured in the finer sizes $-\frac{3}{8}$ inch $+$ 3 mesh and $-3+4$ mesh.

There was an appreciable loss of gypsum by dissolution in the water used.

Tabling. Two tests were made on a small Wilfley table. In these tests a quantity of gypsum was ground to 20 mesh and screened on 48 and 100. The three sizes were then tabled separately. The results were between poor and fair. The best concentrate obtained ran 19·01 per cent water and the poorest 15·22 per cent water. These figures correspond to 90·9 per cent and 72·8 per cent gypsum. The tails varied from 11·95 per cent to 17·03 per cent water, equal to 57·2 per cent to 81·5 per cent gypsum. As in jigging the best results were obtained with the finer sizes. In tabling some gypsum is lost by dissolution.

Flotation. A small flotation test was made with very poor results.

Calcining and Screening. A lot, $21\frac{3}{4}$ pounds, of the gypsum crushed to pass $\frac{1}{2}$ inch was calcined in a small, electrically heated kettle. After calcining, the charge weighed $19\frac{1}{2}$ pounds and it was screened on 20 mesh. The plus 20 mesh was run one hour in a jar without any pebbles and then screened on 20 mesh. The two minus 20-mesh products obtained were put together and called "first minus 20."

The plus 20 was then run 1 hour in a jar with some rubber stoppers and screened on 20 mesh. The minus 20 obtained was called "second minus 20."

The following table gives the results of this work:—

Product	Lb.	Oz.	H_2O per cent	MgO per cent	CaO per cent
+20 (dolomite)	8	4	5·68	8·00	33·69
1st −20 (gypsum)	8	2	6·54	1·42	37·15
2nd −20 (gypsum)	2	7	8·47	0·87	36·89

As indicated by the results given in the above table, a gypsum product fairly low in dolomite was obtained, but quite an amount of gypsum remained in the dolomite product.

Another lot of gypsum was calcined and screened on 20 mesh. The plus 20 was run for 1-hour periods in a jar with rubber stoppers and screened on 20, the plus 20 being put back into the jar for another hour run. This was continued for 11 runs in all. It was found that after the first run the minus 20-mesh product was high in dolomite.

Calcining, Grinding, and Air Separating. A lot of 17¼ pounds of calcined gyspum, obtained by mixing the products of the last test described, was crushed in a Raymond No. 0000 pulverizer without the regulator. This allowed the material to be carried out of the beater chamber when ground to about 80 mesh. The ground gypsum was run through a 30-inch Gayco air separator set at its coarsest setting. The oversize from the Gayco was run through the separator again, and the second oversize obtained was re-run. The third oversize and clean-up of the Gayco were reground in the Raymond. The Raymond product was run three times through the Gayco as before.

The first three fine products from the Gayco contained about 3·25 per cent magnesia, the second three about 6·25 per cent, and the oversize 7·40 per cent. This shows that the first grinding and separation give fair results, and that the second grinding and separation give poor results. The first grinding and separation alone do not recover a high percentage of the gypsum.

Calcining and Grinding in a Raymond Pulverizer Fitted with a Throw-out Attachment. A quantity of gypsum, 43 pounds 6 ounces in weight, was calcined. After calcining, it weighed 37 pounds 4 ounces, and was then shipped to the Raymond Bros. Impact Pulverizer Company, Chicago, Illinois, who ground it in one of their No. 00 pulverizers fitted with a throw-out attachment, which automatically throws out of the grinding chamber any hard material such as dolomite so that the ground product is purer than the feed. After running the gypsum the Raymond Company shipped back samples of the fine and oversize, which were analysed with the following results:—

Product	H_2O per cent	MgO per cent	CaO per cent
Fine	5·49	2·25	28·22
Oversize	2·41	11·39	31·00

This method of treatment gives a gypsum product low in dolomite and a fair dolomite product with some gypsum.

SUMMARY AND CONCLUSIONS

Fair grades of products were obtained by jigging and tabling, but there are two objections to such operations, mainly, loss of gypsum by dissolution, and the necessity of drying the products. Flotation gives very poor results.

Although fair grades of products were obtained by calcining and screening, the method is complicated and would need either a charge system or a great number of machines if made continuous. The same considerations apply to calcining, grinding, and air separation.

Calcining and grinding in a Raymond pulverizer fitted with a throw-out attachment gives good grades of products and employs a fairly simple method of treatment.

The method of treatment recommended for the separation of the dolomite from the gypsum is calcination followed by grinding in a Raymond pulverizer fitted with an automatic throw-out attachment.

This method will give a gypsum product low in dolomite and a dolomite product containing some gypsum. Some use might be found for the dolomite product, as when ground and mixed with water it sets and develops a fair strength upon drying.

Report No. 467

THE TESTING OF QUARTZ FROM LAROUCHE, CHICOUTIMI COUNTY, QUEBEC

Shipment. A box, shipping weight 41 pounds 6 ounces, containing two pieces of quartz was received on October 3, 1932, from Mr. George Dupont, 4313a rue Bordeaux, Montreal, Quebec. The quartz came originally from Larouche, Chicoutimi County, Quebec.

Mr. Dupont desired that the quartz be tested to see if products suitable for making glass, paint, and pottery could be made from it.

The shipment consisted of 31 pounds 6 ounces net of quartz, part white and part slightly rose in colour. A few rust stains were noticeable.

After selecting two specimens the balance, 29 pounds 10 ounces, of the shipment was crushed in a small jaw crusher and a small set of rolls to all pass 20 mesh, care being taken to crush the quartz gradually and to screen on a 20-mesh screen after each crushing. After crushing there was 29 pounds 6 ounces of minus 20 mesh, from which a sample was taken for chemical analysis. The analysis gave:—

	Per cent
SiO_2	99·12
Fe_2O_3	0·09

TESTING QUARTZ FOR GLASS-MAKING

For the preparation of a glass sand part of the remaining minus 20 mesh, 14 pounds 11 ounces, was washed by stirring it with water in a small tub and then decanting the water and fine quartz. After washing in this way four times, the washed quartz was dried. The four washings were combined, filtered, and dried. The dried products were weighed and sampled for analysis. The results of this work are as follows:—

Washed quartz, 12 pounds 9 ounces	99·74 % SiO_2 0·06 % Fe_2O_3 0·006% TiO_2
Fine quartz, 1 pound 15 ounces (869 grms.)	0·27 % Fe_2O_3

Screen tests were made on 134·8 grammes of the original minus 20-mesh product, on 171·9 grammes of the washed quartz, and on 205·1 grammes of the fine quartz with the following results:—

Product	Original −20 mesh		Washed quartz		Fine quartz	
	Grms.	Per cent	Grms.	Per cent	Grms.	Per cent
+ 20	0·6	0·5	0·0	0·0	0·0	0·0
− 20+ 28	29·8	22·1	23·8	13·9	0·0	0·0
− 28+ 35	26·9	20·0	41·3	24·0	0·1	0·1
− 35+ 48	18·6	13·8	30·6	17·8	0·4	0·2
− 48+ 65	19·3	14·3	23·6	13·7	0·5	0·2
− 65+100	16·2	12·0	20·9	12·2	2·9	1·4
−100+150	6·7	5·0	16·7	9·7	11·7	5·7
−150+200	2·6	1·9	7·1	4·1	19·3	9·5
−200	14·0	10·4	7·8	4·6	169·4	82·9
Total	134·7	100·0	171·8	100·0	204·3	100·0

As the washed quartz contained 18·4 per cent of minus 100-mesh material, 136·4 grammes of it was cut out and screened on 100 mesh giving:—

+100, 118·3 grammes..... 0·046% Fe_2O_3
−100, 17·9 grammes..... 0·14 % Fe_2O_3

This test showed that the washed quartz would be improved by washing so that less minus 100-mesh material would be in the washed product.

Sand for making clear glass should be 0·06 per cent or less Fe_2O_3 plus TiO_2. The washed quartz would be suitable if it were washed so as to leave very little minus 100 mesh in it, or if it were washed and screened on a screen as near 100 mesh as commercially possible, say 80 mesh.

TESTING QUARTZ FOR CERAMICS

Two lots of the minus 20-mesh quartz, the first 1,604 grammes and the second 1,581 grammes, were each ground for four hours in a small pebble jar with about three times their weight in pebbles. The pebbles with the first lot were 4,812 grammes and with the second lot 4,749 grammes. After grinding the first lot was 99·63 per cent minus 200 mesh, and the second lot was 99·40 per cent minus 200 mesh.

A 110·1-gramme sample was cut out of the first lot and sent to be tested to the Division of Ceramics and Road Materials of the Mines Branch. This Division reported as follows:—

> The sample was formed into fired colour determination cones and fired to cone 12 (2390° F.). After firing, the cones were noted to be very dirty with large, black iron spots.
>
> This material is considered much too dirty to permit of its being suitable for use as potter's flint in the ceramic industry.

TESTING QUARTZ FOR SAND-BLASTING

A lot of 1,000 grammes of the washed minus 20-mesh quartz was ground for one hour with about 3,000 grammes of pebbles in a small jar. After grinding, the quartz was screened on 100 mesh and enough washed minus 20-mesh quartz added to the plus 100 mesh to make it up to 1,000

grammes. This was ground for another hour, screened, and sweetened up as before and ground for a third hour, after which it was screened. The results of this test are as follows:—

Hours grinding	Grammes −100	Grammes +100	Grammes pebbles
0	.....	.....	3,006·1
1	638·8	359·2	3,004·2
2	631·8	366·0	3,002·2
3	628·3	369·8	3,000·6

These results indicate that the quartz would make only a fair blast sand, as a good blast sand will show only about 530 grammes of minus 100 mesh on the third run and a loss of pebbles in the three hours' grinding of over 7 grammes. The test gives a loss of 5·5 grammes of pebbles for the three hours.

TESTING QUARTZ FOR PAINT-MAKING

Two lots of the minus 20-mesh quartz, the first one 1,586 grammes and the second one 1,615 grammes, were ground for 8 hours in a small pebble jar, the first lot with 4,758 grammes of pebbles and the second lot with 4,849 grammes of pebbles. After grinding, the first lot was 99·48 per cent minus 325 mesh and the second lot was 99·30 per cent minus 325 mesh.

The colour of these two ground lots was compared with a pure white product and they were found to be slightly off colour.

The quartz could be used in paint-making, but not for a pure white paint.

Report No. 468

THE CRUSHING AND WASHING OF SANDSTONE FROM ST. CANUTE, QUEBEC

Two small lots of sandstone were submitted by the Canadian Carborundum Company, Limited, Shawinigan Falls, Quebec. These lots came from the company's sandstone quarry at St. Canute, Quebec, and were received in the latter part of August, 1932. One lot was of white sandstone and weighed 24½ pounds gross; the other was of buff-coloured sandstone and weighed 23½ pounds gross.

Tests were desired on both lots of sandstone to determine if sand suitable for making clear glass could be made from either lot.

After removing a specimen from each lot the remainders were crushed separately to all pass 20 mesh by means of a small jaw crusher, small rolls, and a hand screen. A sample for chemical analysis was cut out of each of the minus 20-mesh products. The remaining 20-mesh material of each lot was washed, one quarter at a time, by stirring with water in a tub and then decanting the water. Each quarter was washed four times. After washing, the washed quarters of each lot were put together and dried. The washings from each lot were also put together and then filtered and

dried. Samples for analysis were cut from the dried products. The following tables give the results of the test work:—

White Lot:	Lb.	Oz.	Per cent Fe_2O_3
Received	23	0	
Specimen	1	8	
To crush	21	8	
−20 mesh	21	1	
Sample −20 mesh	0	11	0·076
To wash	20	6	
Washed	19	1	0·066
Washings	1	4	0·35

Buff Lot:	Lb.	Oz.	Per cent Fe_2O_3
Received	22	0	
Specimen	1	8	
To crush	20	8	
−20 mesh	19	15	
Sample −20 mesh	0	11	0·10
To wash	19	4	
Washed	17	13	0·069
Washings	1	7	0·84

Sand for making clear glass should be 0·06 per cent Fe_2O_3 or less. Both lots when washed are over this limit by a very small amount.

Report No. 469

EXPERIMENTAL TESTS ON MADOC TALC FOR THE SEPARATION OF DOLOMITE

Shipment. One bag of crude talc, weight 200 pounds, was received August 6, 1931, from the Canada Talc Company, Limited, Madoc, Ontario.

Purpose of Experimental Tests. The shipment of crude talc was submitted to determine whether the quality of the ground talc being supplied the European market could be improved and thus command a wider market.

Characteristics of the Talc. The crude talc consisted of large pieces of soft white talc, containing small rounded aggregates of harder dolomite about $\frac{1}{4}$ inch in diameter of a slightly grey colour.

Sampling and Analysis. The crude talc was crushed in a small jaw crusher to pass a 1-inch screen. A small portion was cut out for analysis which gave the following:—

	Per cent
CaO	5·70
MgO	26·58
SiO_2	51·75
H_2O	7·90
$Al_2O_3 + Fe_2O_3$	1·75
CO_2	5·90

Experimental Tests

The experimental tests conducted comprise: (1) Crushing, grinding, and screening; (2) Grinding and air separation; (3) Grinding in a Raymond pulverizer fitted with an automatic throw-out; (4) Wet concentration on Wilfley table; and (5) Wet concentration by flotation. Flotation was the only method that reduced the lime content any appreciable amount.

CRUSHING, GRINDING, AND SCREENING

A representative sample of six pounds of the crude talc, cut out by sampling the shipment after crushing to 1-inch size, was crushed in a small jaw crusher set at $\frac{1}{4}$-inch opening. At this setting the rounded masses of dolomite were crushed to some extent. The crushed material was ground in a large pebble jar with 18 pounds of pebbles. The ground product was screened on 10 mesh and the oversize ground for another hour and screened. The results of the test were as follows:—

Product	Weight, lb.	Assay
First −10 mesh	4·6	6·08% CaO
Second −10 mesh	1·3	5·32% "
Oversize +10 mesh	0·1	25·34% "

Very little improvement was made in the grade of the talc.

A second test was made by crushing a sample of six pounds in a small jaw crusher set at $\frac{5}{16}$-inch opening. This setting did not crush the masses of dolomite. The crushed material was ground for two hours in a large pebble jar with 18 pounds of pebbles and the ground product screened on 10 mesh. The results of this test were as follows:—

Product	Weight, lb.	Assay
−10 mesh	4·3	5·23% CaO
+10 mesh	1·7	7·47% "

Little effect was shown on the grade of the talc by this operation.

GRINDING AND AIR SEPARATION

A sample of six pounds was crushed in a No. 0000 Raymond pulverizer set to give as fine a product as possible. The ground talc was then passed through a 30-inch Gayco air separator, the oversize being passed through twice. The results of the test were as follows:—

Product	Weight, lb.	Assay
Separator fines-pass No. 1	3·0	3·23% CaO
" " No. 2	0·5	3·33% "
" " No. 3	0·25	3·69% "
" oversize " No. 3	0·25	9·20% "

A second test was made with the pulverizer set to give as coarse a product as possible. The oversize in this case was passed through the air separator three times. The results were as follows:—

Product	Weight, lb.	Assay
Separator fines-pass No. 1	1·5	3·03% CaO
" " No. 2	0·5	3·50% "
" " Nos. 3 and 4	0·5	3·84% "
"oversize" No. 4	2·5	9·76% "

These tests indicate that the lime content in the talc can be reduced about 3 per cent.

GRINDING IN THE RAYMOND PULVERIZER FITTED WITH AUTOMATIC THROW-OUT

A 100-pound lot was shipped to Raymond Bros., Chicago, for test purposes. The talc was ground in a No. 0000 pulverizer to about 90 per cent through 200 mesh. Eighty pounds of the ground talc was passed through a No. 00 pulverizer fitted with an automatic throw-out. The results were as follows:—

Product	Weight, lb.	Assay
Fines	63·0	4·31% CaO
Throw-out product	17·0	12·98% "

This test shows a reduction of 1·4 per cent of the lime content.

WET CONCENTRATION ON WILFLEY TABLE

A sample of 11·5 pounds was crushed to pass 20 mesh and sized on 48- and 100-mesh screens. The three sizes were run over a small Wilfley table, the dolomite reporting over the concentrate end and the talc over the tailing side of the table. The assays of the talc tailings were as follows:-

− 20+ 48 size	4·03% CaO
− 48+100 "	2·44% "
−100 "	2·50% "

A considerable reduction in the lime content is shown especially on the finer sizes, but the talc would have to be dried, which item of cost would probably offset any improvement made.

WET CONCENTRATION BY FLOTATION

Fifteen small flotation tests were made. In the best test 500 grammes of the minus 20-mesh talc was ground for one hour in a small pebble jar with 0·25 gramme of soda ash and 1,000 c.c. of water. The charge was then transferred to a small Ruth flotation machine, and mixed for 5 minutes

with 0·1 gramme of potassium xanthate and 3 drops of water glass. Flotation was carried on for 5 minutes. The rougher concentrate obtained was recleaned by agitating for 5 minutes and floating for 5 minutes. The results obtained were as follows:—

Product	Weight, grammes	Per cent −200 mesh	Assay
Concentrate	239·9	43·0	0·32% CaO
Middling	48·6	71·0	
Tailing	204·9	77·0	

The flotation tests showed that the lime content in the talc could be reduced to less than 0·5 per cent. The finished talc product would require drying, which would add materially to the cost of production.

SAMPLES SUBMITTED FOR EXAMINATION

Six flotation concentrates from the best tests were ground in a small pebble jar to about 88 per cent minus 200 mesh, the fineness of a sample of Italian talc received from the Canadian Trade Commissioner at Rotterdam. The six concentrates were then submitted to him, to obtain the opinion of the trade on their suitability for the European market. The following extract is taken from a letter of the Trade Commissioner at Rotterdam.

. . . . You will recall that previously the principal difficulty, as far as the Canadian product is concerned, was the high lime content. The flotation test to which reference was made by the Mines Branch has had the effect of overcoming this. The importers with whom we are in touch contend, however, that the physical properties of the Canadian article are not equal to those of the competing product which they are now buying in Spain.

CONCLUSIONS

1. The lime content of the talc sample received for test purposes can be reduced from 5·7 to 3·5 per cent by grinding in a Raymond pulverizer and separating in a Gayco air separator.

2. The lime content can be reduced to below 0·5 per cent by flotation of the crude talc, but flotation and drying operations would increase materially the cost of production.

3. European buyers, to whom samples of the talc product obtained by flotation and ground to the same fineness as Italian and Spanish talcs were submitted, contend that the Canadian talc does not possess the slip qualities of either the Italian or Spanish talcs.

Report No. 470

EXPERIMENTAL TESTS ON BARITE FROM STRATHLORNE, N.S.

Shipment. A shipment of 40 bags of barite, shipping weight 4,000 pounds, was received March 11, 1932, from Strathlorne, N.S. The barite was submitted by Mr. John W. Shaw, New Liskeard, Ontario, and consisted of 11 bags marked Scotsville Lot, 10 bags marked T. Campbell Lot, 16 bags of McMillan Lot, and 3 bags that had lost their tags in transit.

Purpose of Experimental Tests. Mr. Shaw wanted as large a sample as possible prepared by mixing the lots and grinding so that the ground product would be 90 per cent $BaSO_4$ and nearly all minus 325 mesh. He also desired concentration tests made on the barite not used for grinding.

Characteristics of the Barite. The shipment consisted principally of white barite. There were also present in varying amounts in the different lots green fluorite and white calcite. The minerals were all in fairly coarse pieces.

Sampling and Analysis. The first two lots were crushed in a gyratory crusher and disk crusher and screened on a 5½-mesh Hummer screen. The oversize after being passed once through rolls was combined with the undersize. The third lot was put through the gyratory and disk crushers and then once through the rolls.

Samples for analysis were cut from each lot by a Jones riffle sampler. The samples gave:—

—	$BaSO_4$ per cent	CaO per cent	SiO_2 per cent	CaF_2 per cent	Sp. gr.
Scotsville Lot	95·38	0·14	1·87	0·54	4·4
Campbell Lot	86·44	2·55	1·25	7·03	4·2
McMillan Lot	75·30	2·97	0·91	18·09	4·0

Fine Grinding. All the Scotsville Lot after sampling and parts of the other two lots were mixed:—

	Pounds
Scotsville Lot	1,083½
Campbell Lot	400
McMillan Lot	100
Total	1,583½

The mixed lot was fed to a 4-foot by 6-foot cylindrical pebble mill in closed circuit with a Sturtevant air separator. The fines were regarded as finished product. After all the mixed lot had been fed in, the mill was run for one hour. It was then cleaned out and the material fed to the Sturtevant separator. The oversize was fed back to the separator after each pass. In all, four passes were made. The last oversize was ground in the mill for three hours, and separated as before in the Sturtevant, seven passes being made. The products obtained from this test were:—

—	Pounds	$BaSO_4$ per cent	−325 per cent
Sturtevant fines	1,259	92·02	99·1
Sturtevant oversize	43	.....	.....

The fines were bagged and shipped by Mr. Shaw to Trinidad. Later he reported that they had been tested and found satisfactory for use in oil drilling.

CONCENTRATING TESTS

Some tests were made on the remainder of the Campbell and McMillan lots. In these tests a comparison of jigging with tabling and classifying with screening was made.

Classifying and Tabling

One half of the remainder of the McMillan lot, 708 pounds, was crushed to 10 mesh and run through a Richards launder classifier making three products. These products were run on a large Wilfley table with a middling return. At the end of each tabling some middlings were kept separate. The following are the results.

—	Pounds	$BaSO_4$ per cent
Coarse concentrate	212·0	87·44
Medium "	259·7	78·50
Fine "	78·3	93·92
Coarse middlings	8·1	66·20
Medium "	1·7	35·02
Fine "	1·2	58·66
Coarse tailing	11·2	30·36
Medium "	43·4	25·48
Fine "	25·2	21·20

This test shows that if the barite is classified before tabling, only the fine table concentrate will be over 90 per cent $BaSO_4$.

Screening and Jigging Combined with Classifying and Tabling

The balance of the McMillan Lot, 723 pounds, was crushed to pass 3 mesh and screened on 6 and 10 mesh.

	Pounds
− 3+ 6	169·0
− 6+10	192·5
−10	355·5
	717·0

The −3 +6 was jigged in a pair of James jigs. The tail of the first jig went into the second jig and a concentrate was taken from each. At the end of the test the first jig was run down as far as possible and then the bed was put into the second jig. The second jig was then run down. The following table gives the results:—

—	Pounds	$BaSO_4$ per cent
Jig Concentrate No. 1	46·7	89·48
" " No. 2	37·8	83·42
" " No. 1 from running down bed	10·0	84·26
" " No. 2 " " "	23·2	80·48
Hutch	0·9	81·66
Bed	17·1	41·96
Tailing	29·5	37·72

The −6 +10 was treated in a similar manner to the −3 +6. The following table gives the results:—

—	Pounds	$BaSO_4$ per cent
Jig Concentrate No. 1	82·0	90·12
" " No. 2	38·0	85·26
" " No. 1 from running down bed	12·2	78·88
" " No. 2 " "	6·9	74·04
Hutch	2·2	80·26
Bed	18·5	50·68
Tailing	29·0	22·96

The −10 was classified into three sizes by means of a Richards launder classifier and each size was tabled on a large Wilfley table with a middling return. At the end of each tabling some of the middlings were kept separate. The products were:—

—	Pounds	$BaSO_4$ per cent
Coarse concentrate	110·4	86·86
Medium "	113·0	84·12
Fine "	53·0	93·42
Coarse middling	6·7	49·60
Medium "	9·1	48·60
Fine "	7·0	71·60
Coarse tailing	3·1	11·26
Medium "	9·6	18·58
Fine "	17·1	26·20

This test shows that jigging gives products running from 83 to 90 per cent $BaSO_4$, and that classifying and tabling gives products from 84 to 93 per cent $BaSO_4$, with only the fine table concentrates over 90 per cent $BaSO_4$.

Screening and Tabling

The balance of the Campbell Lot, 538 pounds, was crushed to pass 10 mesh and screened by means of a Hummer vibrating screen on 26 and 60 meshes. This gave:—

	Pounds
−10 +26	244
−26 +60	151
−60	139
	534

The three sizes were tabled on a large Wilfley table fitted with a middling return. At the end of the tabling of the two coarse sizes some of the middlings were kept separate. The following are the products:—

—	Pounds	$BaSO_4$ per cent
−10 +26 concentrate	189·9	96·00
−26 +60 "	85·4	96·28
−60 "	84·5	97·12
−10 +26 middling	8·6	93·00
−26 +60 "	7·1	90·00
−10 +26 tailing	42·4	57·10
−26 +60 "	54·9	78·80
−60 "	28·2	66·00

This test shows that screening before tabling gives much better concentrates than classifying before tabling.

CONCLUSIONS

1. The barite can be readily ground to nearly all minus 325 mesh by the use of a pebble mill and centrifugal air separator.

2. If a product not too high in barite is desired, jigging of the coarser sizes, and classifying and tabling of the fines could be employed.

3. If a product high in barite is desired it will be necessary to screen and table.

4. If it were found impracticable to screen as fine as 60 mesh classifying could be used for this separation.

Report No. 471

THE CONCENTRATION OF CYANITE FROM DEATH RAPIDS, B. C.

Shipment. One bag, shipping weight 65 pounds, of crude cyanite was received October 1, 1931, from Mr. B. T. O'Grady, Resident Mining Engineer, Nelson, B.C. The cyanite came originally from Death Rapids, B.C., in the Big Bend district north of Revelstoke.

Purpose of Experimental Tests. Mr. O'Grady suggested that preliminary tests be made. However it was decided to make not only concentration tests but also to test the cyanite for ceramic use.

Characteristics of the Cyanite. The shipment consisted of long, flat crystals of sky-blue cyanite in white quartz. A small amount of brown mica was also present.

Experimental Tests

Several methods of separation were tried; those giving poor results were tabling, magnetic separation, electrostatic separation, and screening with square mesh screens and fine grizzlies. However, the Ullrich machine used in the magnetic separation tests removed the brown mica from the cyanite and quartz.

Hand-picking

To secure some clean cyanite three-quarters of the lot which had been crushed to 1 inch was screened on $\frac{1}{2}$-inch and $\frac{1}{4}$-inch screens and the -1 inch and $+\frac{1}{2}$ inch and $-\frac{1}{2}$ inch $+$ $\frac{1}{4}$ inch were carefully hand-picked. This resulted in the following products:—

		Pounds
Cyanide	-1 inch $+\frac{1}{2}$ inch	4·1
"	$-\frac{1}{2}$ inch $+\frac{1}{4}$ inch	2·1
Reject	-1 inch $+\frac{1}{2}$ inch	16·2
"	$-\frac{1}{2}$ inch $+\frac{1}{4}$ inch	4·0
"	$-\frac{1}{4}$	15·5

Flotation

Three small flotation tests were made on the cyanite as received. The best results were obtained by grinding 500 grammes of minus 20-mesh material in a pebble jar for 30 minutes with one gramme of soda ash. The ground material was then put into a small Ruth flotation machine, mixed for 5 minutes with 9 drops of oleic acid and floated for 5 minutes. The concentrates were re-run by mixing for 5 minutes and floating for 5 minutes.

Results:

Product	Grms.	Al_2O_3 per cent	Al_2O_3 grms.
Concentrate	169·5	61·45	104·15
Middling	44·7	51·29	22·93
Tailing	282·2	18·41	51·95
Total	496·4	36·07	179·03

The concentrates were quite high grade. A piece of cyanite picked out of the shipment ran 61·75 per cent Al_2O_3. The recovery in the concentrates alone is 58·2 per cent. If allowance were made for the cyanite in the middling the recovery would be higher. Besides with further work, doubtless, much better recoveries could be obtained.

In order to obtain more flotation concentrates for testing for ceramic use the discard from hand-picking was crushed to 20 mesh and then ground in charges of 1,000 grammes in pebble jars and floated. Two charges were floated separately and the concentrates from them were combined and re-floated. The middlings from the re-run were put into the next two charges before floating. In this way eight concentrates were obtained, the Al_2O_3 content of which varied from 55·74 per cent to 61·31 per cent. However, recoveries were poor.

Testing for Ceramic Use

The hand-picked cyanite and all flotation concentrates were sent to the Division of Ceramics and Road Materials for testing. Their report was as follows:—

This material is similar to several other cyanites which have been studied for use in the ceramic industry. It converts to mullite ($3\ Al_2O_3\ 2SiO_2$) with a drop in specific gravity from 3·62 to 3·04, and specimens containing from 88 to 98 per cent cyanite showed linear expansions of from 8 to 16 per cent.

Microscopic examination showed that the material upon burning maintained its original platy structure to a marked extent.

The material is worthy of further study, for use in the production of refractory products. The difficulty in the use of this material has been not that it requires calcination before use but due to the poor success met with in developing binders for the calcined material.

CONCLUSIONS

1. The only methods of concentration found suitable were hand-picking and flotation.

2. Hand-picking could not be used commercially unless the cyanite occurred in larger pieces than those in the sample received.

3. The concentrates made by flotation were very good but the recoveries were low. It is believed that this could be improved by further test work.

Report No. 472

EXPERIMENTAL TESTS ON QUARTZ AND CHINA CLAY FROM LAC RÉMI, QUEBEC

Shipments. Seven shipments of quartz and china clay, comprising in all nine lots, were received from May 1931 to December 1932. These shipments varied from a few pounds to nearly four tons. The material came from a deposit at Lac Rémi, Quebec, and was sent in by the owners, the Canadian Kaolin Silica Products, Limited, Montreal, Quebec.

Purpose of Experimental Tests. The test work involved many different investigations. These were preparation of quartz for use in glass-making and sand blasting, preparation of china clay, air separation tests, grinding quartz to 200 mesh for ceramic use, washing dark clay out of quartz and grinding for ceramic use, and grinding to 20 mesh with a minimum of minus 100 mesh. Besides making the tests some large samples were shipped to commercial firms to be tested for ceramic use.

Arrangements for Experimental Tests. Mr. H. J. Emery, General Manager of the Canadian Kaolin Silica Products, Ltd., was present for many of the tests and co-operated in the work.

Characteristics of the Quartz and Clay. Many of the shipments were of white quartz carrying a small amount of white china clay. Some of the shipments had been crushed and some of the clay had been removed by air separation. A small amount of the removed clay was also sent in but no work was done on it.

PREPARATION OF QUARTZ FOR GLASS-MAKING AND RECOVERY OF CHINA CLAY

Two small samples of quartz and clay, Lots Nos. 1 and 2, were tested to see if a product suitable for glass-making could be prepared from them and also to determine the quantity and quality of clay that could be recovered.

In the first test on each lot the quartz and clay were crushed to 20 mesh and blunged in a small pebble jar with rubber stoppers. Three products were made by classifying, viz., coarse sand, fine sand, and clay.

The clay was buff in colour. The coarse sand when sized gave no size 0·06 per cent Fe_2O_3 plus TiO_2 or less, which is the specification for sand for making clear glass.

In the second test on each lot the sand and clay were treated as in the first test and the coarse sand was tabled.

The clay was buff in colour as before, for the first lot it equalled 5·13 per cent of the feed, and for the second 3·17 per cent. The tabled coarse sand of the first lot on being sized gave the −20 +48 mesh, 47·5 per cent of the feed, suitable for making clear glass. For the second lot the −35 +65, 22·3 per cent of the feed was suitable for making clear glass.

PREPARATION OF QUARTZ FOR SAND-BLASTING

Lot No. 3 consisted of 11 bags of quartz −8 +16 mesh. The net weight received was 1,016 pounds. Blast sand was prepared from 763 pounds of the lot by crushing to 20 mesh, washing four times in an Akins classifier, drying the sand and screening $\frac{9}{16}$ of the washed and dried sand on 42 mesh. This gave two grades of blast sand −20 and −20 +42. The weights of some of the final products were:—

Product	Pounds
−20 mesh sand	232
−20 +42 "	266
−42 "	28

No opportunity has occurred to try out the sand for blasting, but small grinding tests in a pebble jar indicate that the quartz will make a fair blast sand.

The head sample of Lot No. 3 ran 0·05 per cent FeO_2O_3, 0·036 per cent TiO_2, and 1·30 per cent Al_2O_3.

Some work was also done on the lot to see if a sand suitable for glass-making could be prepared. The results were better than with Lots Nos. 1 and 2 as washing and tabling without sizing gave 71·1 per cent of the feed as a product 0·03 per cent Fe_2O_3 and 0·02 per cent TiO_2, which is suitable for making clear glass.

Air-Separation Tests

The remaining part of Lot No. 3, 247 pounds, was used for air-separation tests.

The air separator used in the mill at Lac Rémi gave a small amount of coarse particles in the fines although every effort possible had been made to stop leaks. Tests were desired to see if these coarse particles could be eliminated.

Two 8-foot separators of different makes and a 30-inch separator were all tried out and all gave coarse particles in the fines.

The small separator was tried out again with long blades and also with long blades and a very close fitting ring around the outside of the blades. In both these tests coarse grains went into the fines.

Apparently it is characteristic of air separators that if there are coarse particles in the feed there will be a few of these coarse particles in the fines.

GRINDING QUARTZ FOR CERAMIC USE

Lot No. 4, 3,000 pounds of minus 8-mesh quartz, was ground in a Hardinge air-swept pebble mill fitted with rotary and superfine classifiers. The excess air was vented through an auxiliary collector. The ground quartz from the regular and auxiliary collectors was mixed as produced. The fines were over 99 per cent minus 200 mesh.

There was a small amount of charcoal in the quartz before grinding. This was quite likely from a fire made on or near the deposit. However, the charcoal would make no difference for ceramic use.

Burning tests were made by the Division of Ceramics and Road Materials on the fines from the regular collector, the fines from the auxiliary collector, and a mixture of 90 per cent regular fines and 10 per cent auxiliary fines. All these tests show a few spots and a slight colour. The auxiliary fines although the darkest before burning were the lightest after burning.

A lot of 2,811 pounds of the ground quartz was shipped out to be tried for ceramic use in a pottery.

A second lot of quartz was prepared for ceramic use. This was Lot No. 5 of which there was received 1,000 pounds. This lot was about 14 mesh in size and contained a small amount of dark green clay in small specks.

To remove the clay the sand was washed twice in an Akins classifier and as the treatment was insufficient it was agitated in a 4-foot by 8-foot cylindrical mill without pebbles for one hour with water, and then washed twice more in the Akins. The washed sand was then dried and ground for five hours in a 4-foot by 6-foot pebble mill. At the end of this time the quartz was 98·4 per cent minus 200 mesh and weighed 820 pounds. The quartz used for the test weighed 981 pounds, and the quartz put in the mill weighed 802 pounds.

A burning test was made on the fine ground quartz with the result that the burned quartz contained a few fine spots and had a slight colour.

Seven hundred pounds of the fine quartz was shipped out to be tested in a pottery for ceramic use.

A third lot of quartz was prepared for ceramic use. This was Lot No. 8 which was about minus 14 mesh and weighed 2,906 pounds, all of which was used in grinding with the Hardinge air-swept mill fitted with rotary and superfine classifiers. Pebbles were used in the mill and the auxiliary collector fines and the regular fines were kept separate. The regular fines were 98·4 per cent minus 200 mesh.

Four separate shipments were made from the regular fines to different manufacturers.

So far no report has been received from any one to whom a shipment was sent for trial in ceramic use.

CRUSHING QUARTZ TO MINUS 20 MESH

Tests were made on Lots Nos. 6 and 7 to determine if a rod mill, ball mill, or pebble mill could be made to give a 20-mesh quartz product as low in minus 100-mesh material as that given by rolls. Lot No. 6 weighed 7,576 pounds net, and Lot No. 7 weighed 6,680 pounds net.

Both lots when received were screened on a 20-mesh Hummer screen and the grinding tests were made on the plus 20 mesh from the screen.

Two roll tests were made with a small set of rolls. The first with six crushings, one in a small jaw crusher, two with rolls just touching, and three with rolls tight, gave a minus 20-mesh product 20 per cent minus 100 mesh.

The second roll test with eight crushings, one in a jaw crusher and seven in the rolls, gradually varied from $\frac{1}{8}$ inch apart to tight for the last three passes, gave a minus 20-mesh product 17·5 per cent minus 100 mesh.

Tests with the ball and pebble mills gave very poor results because when operated so as to give a suitable product their capacities were small.

Tests were made with a 3-foot by 6-foot Marcy rod mill. It was found that the feeder would not feed quite fast enough to give very high circulating loads so the mill was operated by putting the feed in through the discharge door, running for 2 minutes, screening the discharge and placing it back in the mill with fresh feed for another 2-minute run.

Operated in this way the mill with 3,500 pounds of $1\frac{5}{16}$-inch and $1\frac{7}{8}$-inch rods gave a 20-mesh product 18·7 per cent minus 100 mesh. The capacity was 4,980 pounds per hour and the circulating load 1,072 per cent.

Shortly after all fresh feed had been used up a 20-mesh product 9·3 per cent minus 100 mesh was obtained, the capacity was 3,240 pounds an hour and the circulating load 969 per cent.

Perhaps with no fresh feed the quartz fed in this last-mentioned run was harder than if there had been fresh feed added and the results are better than would be obtained with a proper amount of fresh feed. However, it is believed that results would lie between the two runs given.

CONCLUSIONS

1. Washing, tabling, and sizing give 47·5 per cent of Lot No. 1 and 22·3 per cent of Lot No. 2 as sand suitable to be used in making clear glass, and washing and tabling gave 71·1 per cent of Lot No. 3. It is believed that if sand is not over 0·06 per cent Fe_2O_3, then a little TiO_2 would not be objectionable in making clear glass. If this is so the quartz could be used for making clear glass by crushing and removal of the clay and fines.

2. The first two lots of silica ground to 200 mesh for ceramic use did not give good burning tests. However, the third one seems much better and it is possible the silica may have a ceramic use.

3. The silica will make a fair blast sand.

4. The clay recovered from Lots Nos. 1 and 2 was buff in colour, however Lot No. 7A, which was clay removed at Lac Rémi from the quartz has a much better colour and it is probable that this clay will have a ceramic use.

5. The air-separation tests show that if there is coarse in the feed a small amount of it will go into fines.

6. Rolls and a rod mill would be suitable machines to reduce the quartz to 20 mesh with a minimum of fines, but a pebble or ball mill would not.

Report No. 473

THE TESTING OF GYPSUM FROM OTTAWA BROOK, NOVA SCOTIA

Shipment. One bag of gypsum rock, net weight 24 pounds, was received on August 13, 1932, from L. H. Cole, Mines Branch, Ottawa. This sample came from the north quarry of the Newark Plaster Company at Ottawa Brook, Nova Scotia.

Purpose of Tests. The experimental tests were made to determine the suitability of the gypsum for plaster manufacture.

Sampling and Analysis. The crude rock was white and had a massive, compact texture. There were very narrow dark bands running through it.

The rock was crushed in a small jaw crusher and rolls to 20 mesh, and then ground in a burr mill to 99 per cent through 100 mesh. A sample cut from this product gave the following results on chemical analysis.

Sample dried at 45° C. for 2 hours	Per cent
Insoluble	2·62
Ferric oxide (Fe_2O_3)	0·04
Alumina (Al_2O_3)	0·12
Lime (CaO)	32·64
Magnesia (MgO)	0·04
Sulphur trioxide (SO_3)	45·04
Combined water (H_2O)	19·18
Carbon dioxide (CO_2)	0·69
	100·37

Recalculation of these results gives:—

Gypsum ($CaSO_4 \cdot 2H_2O$)	91·65
Anhydrite ($CaSO_4$)	4·11
Magnesium carbonate ($MgCO_3$)	0·08
Calcium carbonate ($CaCO_3$)	1·48
Other ingredients	3·05
	100·37

Experimental Tests

Twenty pounds of the ground gypsum was calcined in a small, electric, batch kettle to the first settle. The calcined product was tested as usual for water, testing consistency, time of setting, and tensile and compressive strengths. The water content of the dried briquettes was also determined.

Screen Analysis of Calcined Gypsum

	Per cent
−200 mesh	98·6
−325 "	68·0

Summary of Tests

Water after calcination / Water before calcination	0·308
Water before calcination, per cent	19·18
Water after calcination, per cent	5·90
Final temperature of calcination, °F	315
Testing consistency, c.c. per 100 grms	63
Time of setting in minutes	11
Tensile strength, lb./sq. in	323
Compressive strength, lb./sq. in	2,125
Water in air-dried briquettes, per cent	19·72
Colour of ground gypsum	White
Colour of calcined product after set	White

CONCLUSIONS

The tests show that this gypsum will make excellent plaster suitable for the manufacture of structural materials having a gypsum base.

Report No. 474

THE TESTING OF ANHYDRITE FROM BADDECK BAY, NOVA SCOTIA

Shipment. Two bags of anhydrite rock, net weight 73 pounds, were received on August 13, 1932, from the property of the North American Gypsum Company at the head of Baddeck Bay, Victoria County, Nova Scotia. This sample was submitted by L. H. Cole, Mines Branch, Ottawa.

Purpose of Tests. It was requested that this anhydrite be tested for its plaster-making qualities.

Sampling and Analysis. The crude rock was massive and compact in texture, and coloured grey with medium brown streaks. It had a pearly lustre.

The rock was crushed to pass 20 mesh by means of a small jaw crusher, rolls, and hand screen.

Screen Analysis

Mesh	Weight, per cent
+100	49·6
+150	2·8
+200	0·8
−200	46·8

A sample was cut from the ground anhydrite for chemical analysis.

Sample dried at 45° C. for 2 hours	Per cent
Insoluble	0·60
Ferric oxide (Fe_2O_3)	0·03
Alumina (Al_2O_3)	0·13
Lime (CaO)	41·30
Magnesia (MgO)	0·05
Sulphur trioxide (SO_3)	56·20
Combined water (H_2O)	0·10
Carbon dioxide (CO_2)	1·54
	99·95

Hypothetical Combination:

Anhydrite ($CaSO_4$)	95·18
Gypsum ($CaSO_4 \cdot 2H_2O$)	0·48
Magnesium carbonate ($MgCO_3$)	0·10
Calcium carbonate ($CaCO_3$)	3·39

Experimental Tests

Four tests were made using the catalytic agents, zinc sulphate and potassium sulphate in combination. Lots of 2,000 grammes of the 20-mesh anhydrite with 1½ per cent of zinc sulphate and 1 per cent of potassium sulphate were ground in a pebble jar with 6,000 grammes of pebbles for periods of 4 hours, 2 hours, 1 hour, and 12 minutes. These were tested for fineness, time of setting, and tensile strength. The water in the air-dried briquettes was determined. Specimen squares, 3 inches by 3 inches by ½ inch, were made.

Results:

Test No.	1	2	2	3	4
Screen analysis	Per cent	Per cent	Per cent	Per cent	Per cent
+ 28 mesh					0
+ 35 "					0
+ 48 "					0
+ 65 "					0·49
+ 80 "					0·66
+ 90 "					1·15
+100 "	0·2	0·4	0·4	0·6	1·1
+150 "	0·06	0·04	0·04	0·2	4·4
+200 "	0·02	0·04	0·04	0·2	5·8
−200 "	99·72	99·52	99·52	99·0	86·4
−325 "					65
Time of grinding	4 hours	2 hours	2 hours	1 hour	12 minutes
Agent	1½ zinc sulphate ($Zn\,SO_4 \cdot 7H_2O$) 1 per cent potassium sulphate (K_2SO_4)				
Water used, c.c. per 100 grms	31	31	27	28	21
Age in days when broken	9	9	7	9	8
Weight of briquette, grms	105·5	104·7	111·7	110·3	121·7
Tensile strength, lb./sq. in	300	286	323	281	333
Water in plaster when broken, per cent	10·53	10·19	9·10	9·43	7·30
Time of setting	57 mins.	1¼ hrs.	57 mins.	1½ hrs.	1½ hrs.
Area shrinkage, per cent	1	1	0·2	0·4	0·4
Efflorescence	none	none	none	none	very slight
Remarks	hard top	hard top	hard top	hard top	hard top
Colour of set plaster	white	white	white	white	grey-white

CONCLUSIONS

These tests show that a good plaster may be made from this anhydrite by the addition of the chemicals zinc sulphate and potassium sulphate, as the set plaster shows practically no shrinkage nor efflorescence. The sample ground for only 12 minutes appears to give as good if not better plaster than those ground for a longer time. It is not quite so smooth working nor so white.

Report No. 475

THE TESTING OF GYPSUM FROM WINDSOR, NOVA SCOTIA

Shipment. One box of gypsum rock, net weight 82 pounds, was received on December 7, 1932, from the Windsor Plaster Company, Windsor, Nova Scotia.

Purpose of Tests. Tests were requested to determine the quality of the gypsum for the manufacture of plaster.

Sampling and Analysis. The crude rock was white, streaked with grey, and had a massive, compact texture, with some narrow, finely crystalline seams.

The rock was crushed in a small jaw crusher and rolls to 20 mesh, and then ground in a burr mill to 99 per cent through 100 mesh. A sample cut from this product gave the following analysis:—

Sample dried at 45° C. for 2 hours	Per cent
Insoluble	1·42
Ferric oxide (Fe_2O_3)	0·08
Alumina (Al_2O_3)	0·14
Lime (CaO)	32·77
Magnesia (MgO)	0·11
Sulphur trioxide (SO_3)	45·98
Combined water (H_2O)	18·82
Carbon dioxide (CO_2)	0·45
	99·77

Recalculation of these results gives:—

Gypsum ($CaSO_4 \cdot 2H_2O$)	89·93
Anhydrite ($CaSO_4$)	7·07
Magnesium carbonate ($MgCO_3$)	0·23
Calcium carbonate ($CaCO_3$)	0·75
Other ingredients	1·79
	99·77

Experimental Tests

Fifty pounds of the ground gypsum was calcined in a small, electric, batch kettle to the first settle. The calcined product was tested as usual for water, testing consistency, time of setting, and tensile and compressive strengths.

Screen Analysis:

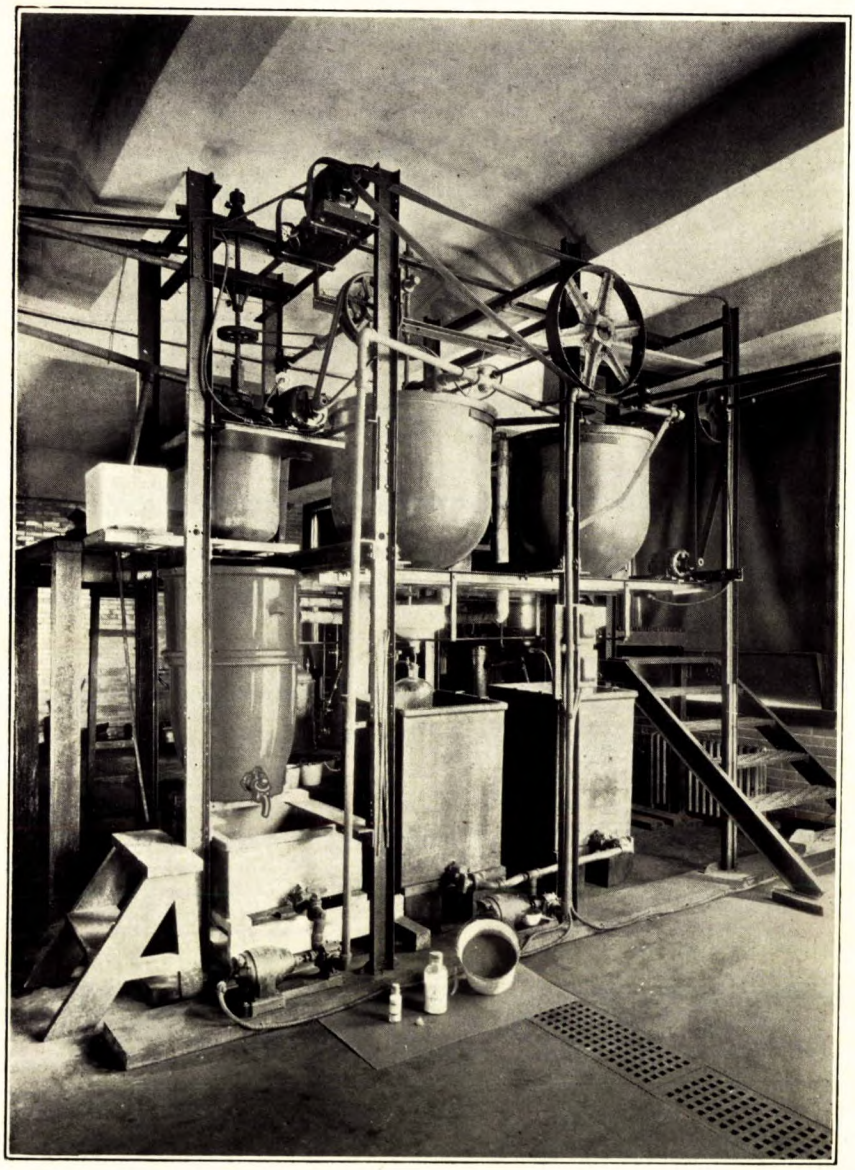
Mesh	Burr mill product	Calcined product
+100 per cent	0·8	0·4
+150 "	1·2	1·4
+200 "	3·2	0·8
−200 "	94·8	97·4
	100·0	100·0

Summary of Tests:

$\frac{\text{Water after calcination}}{\text{Water before calcination}}$	0·287
Water before calcination, per cent	18·82
Water after calcination, per cent	5·40
Final temperature of calcination, °F	290
Testing consistency, c.c. per 100 grms	66
Time of setting, minutes	22
Tensile strength, lb./sq. in	400
Compressive strength, lb./sq. in	2,081
Colour of ground gypsum	Slightly grey
Colour of calcined product after set	Slightly grey

CONCLUSIONS

The tests show that this gypsum is quite suitable for the manufacture of structural materials having a gypsum base. It gives an excellent plaster with high tensile and compressive strengths. The colour, however, is slightly greyish due to specks of the dark material in the gypsum.



Pilot plant for investigation of the treatment of Great Bear Lake pitchblende ores for the extraction of radium. Note: in foreground, the charge of ore, 10 kilograms, in pail; the bottle of lead chloride recovered; the small button of silver recovered; the small bottle of crude radium salt.

IV

SECTION OF HYDROMETALLURGY AND ELECTROCHEMISTRY

Report No. 476

METHODS OF TREATING GREAT BEAR LAKE PITCHBLENDE FOR EXTRACTION OF RADIUM

Introduction

In the "Report of Investigations" of this Division for last year a preliminary report was included outlining the initial experimental work carried out with the purpose of selecting a process suitable to the treatment of the pitchblende from the above area for the extraction of radium.

The following report is a continuation of that work embracing experiments on a larger scale. Minor alterations in the procedure suggested in the former report were found to be necessary in order to minimize certain losses of radium that became apparent in the larger scale tests. A 20-ton shipment, representing ore taken from several pits, was submitted by the Eldorada Gold Mines, Ltd., for the purpose of determining the best methods of preparing the ore for teatment, and for determining the suitability of the suggested process for the extraction of radium from representative samples of the ore.

From an examination of the shipments it developed that two distinct types of ore occur in the deposit sufficiently dissimilar in chemical composition to require separate methods of treatment for the most efficient extraction of radium.

Two methods of treatment are, therefore, suggested in the following text, with results indicating satisfactory extraction of radium combined with a minimum of operation. These methods are designated as Procedure I for the treatment of the high-silica-gangue type pitchblende, and Procedure II for the carbonate-barite-gangue type pitchblende which also contains silver.

Large Laboratory Test Unit for Radium Extraction

Due to the excessively corrosive action of the solutions, chemical, acid-proof stoneware was selected as the most suitable material for plant construction. For the first few tests wooden precipitation tanks were tried but were replaced by stoneware on account of their tendency to absorb fine precipitate and their failure to withstand the action of the hot acid solutions of the ore. Hard rubber, rotary gear pumps connected to $\frac{1}{4}$ h.p. motors were employed for transferring solutions through hard rubber pipes, the temperature of solutions being kept below 55° C.

The accompanying photograph illustrates the setup of the plant used. The leaching tank (upper left) is of 15-gallon capacity and is heated by a steam-line entering through cover. Agitation is maintained by means

of a stoneware agitator. The cover also has a charge opening, and another opening is connected to a suction fan for the purpose of drawing off acid and other fumes.

Underneath the leaching tank is a suction type stoneware filter, with a 15-gallon capacity upper section and 25-gallon capacity lower section. The filter medium comprises blue asbestos cloth over a porous stoneware plate. Suction is provided by a Crowell rotary pump.

Below the filter is a rectangular receiving tank used principally for cooling the leach filtrate before pumping through the hard rubber pump.

The large upper tanks are the precipitating tanks, each having a capacity of 75 gallons and provided with suitable agitators, and the lower tanks are the radium-free filtrate receiving tanks of equal capacity.

A 12-inch, Buchner-type, porcelain funnel with filter paper is used as the filtering medium between the upper and lower tanks.

The leaching capacity of this unit is equal to a 10-kilogram (22-pounds) charge of 55 per cent U_3O_8 ore, or a 15-kilogram (33-pounds) charge of 35 to 37 per cent of U_3O_8 ore.

Preparation of Ore

In preparing the ore for treatment two important factors have to be considered. Firstly, a minimum amount of fines is desirable to prevent a slow filtration condition, and, secondly, dusting has to be eliminated as completely as possible to minimize the loss of valuable material and, more important, to minimize the health hazards to workmen from inhaling the fine dust.

Investigation into the treatment of the ore showed that for quick leaching, the ore should be crushed and ground to pass a 35-mesh screen; for good filtration, a minimum of fines was desirable so that the problem presented in its preparation was to determine the methods of crushing and grinding that would fulfil these conditions.

The treatment method also showed that roasting was necessary for the pitchblende containing the carbonate gangue minerals, to prevent excessive effervescence when brought into contact with the hot acid.

It was found that the ore should be crushed to about 4 mesh, the carbonate-type pitchblende roasted at this size and the final grinding done in a rod mill to give a 35-mesh product. Grinding in a rod mill gave the more desirable product (more uniform with less fines) than any other method investigated.

Wet grinding was considered but was not deemed necessary nor economical for a small plant where the operation would be intermittent. The ore would have to be crushed and roasted in the dry state and provision made for taking care of the dust problem for these operations. Accessory equipment such as classifiers, thickeners, filters, etc., would be required for wet grinding and the loss in fine slime would probably be greater than the dust loss of a dry grinding plant. Efficient dust-collecting systems are now available for dry grinding operations.

Tests were conducted on the various lots comprising the 20-ton shipment, brought out from the property in the fall of 1931.

The various lots were crushed in a jaw crusher to about 1½-inch size, and in rolls to about ½-inch size. In the case of the carbonate gangue pitchblende, a 4-mesh roll product was found a satisfactory size for roasting. Batch charges were roasted in a muffle furnace for one hour with the temperature maintained at 750° C. A satisfactory product when ground to 35 mesh was obtained for leaching. Sufficient of this type of ore was not available to determine whether the dust loss from a multiple-hearth furnace with revolving rabble arms, or from a revolving kiln type of furnace, would be excessive.

Grinding tests on the 4-inch calcined product and the ½-inch roll product were carried out using the following grinding mills:—

No. 0000 Raymond Pulverizer in circuit with 35-inch Hummer screen on the 4-mesh calcined and ½-inch roll product.

Traylor, 4-foot, free-screening, ball mill fitted with 40-mesh screen and in circuit with 35-mesh, Moto-Vibro screen on the ½-inch roll products.

Marcy, 3- by 6-foot open-door discharge, rod mill in circuit with 35-mesh Moto-Vibro screen, on the ½-inch roll product. Feed rate—2,040 pounds per hour. Rod charge—4,013 pounds.

The following comparative screen tests give the grinding obtained from the three types of grinding mills on the ½-inch roll product:—

Mesh	Pulverizer, per cent	Ball mill, per cent	Rod mill, per cent
+ 35	All through	0·1	1·4
− 35+ 48	2·7	6·1	7·2
− 48+ 65	9·4	23·4	34·5
− 65+100	11·2	4·8	12·2
−100+150	8·5	10·7	2·7
−150+200	6·8	8·6	8·0
−200	61·4	46·3	34·0

The hammer consumption for the pulverizer was 3 pounds per ton.
The ball consumption for the ball mill was 3 pounds per ton.
The rod consumption for the rod mill was 1 pound per ton.

The results of the above tests show rod mill grinding gives the most desirable product. By decreasing the rod charge and carrying a greater circulating load the results given in the screen test can be improved upon.

The flow-sheet recommended for the preparation of the pitchblende ore, is as follows: Crushing in a jaw crusher and rolls in the circuit with 4-mesh screen. Calcining of the 4-mesh product in case of the carbonate-gangue pitchblende in a suitable roasting furnace. Grinding the calcines or the 4-mesh ore in a rod mill in circuit with a 35-mesh screen.

Provision made for an efficient dust-collecting system to take care of the dust from all operations.

Analyses of the Lots from 20-ton Shipment

The following table gives the analyses of the various lots comprising the 20-ton shipment.

—	Vein 1, Pit 1	Vein 2, Pit 1	Vein 2, Pits 2-6	Vein 2, Pit 9	Vein 2, Pit 9
Uranium, U	37·82	33·92	35·19	38·96	27·47
U_3O_8	44·6	44·0	41·50	45·94	32·40
Lead, Pb	7·89	8·14	7·23	10·62	6·88
Iron, Fe	2·21	2·06	2·26	5·37	7·00
Copper, Cu	0·85	0·82	0·80	2·80	2·90
Manganese, Mn	0·18	0·17	0·17	4·90	9·06
Nickel, Ni	nil	nil	nil	nil	nil
Cobalt, Co	0·26	0·35	0·22	0·43	0·16
Antimony, Sb	0·13	0·06	0·07	0·13	0·06
Barite, $BaSO_4$	trace	nil	nil	2·18	8·10
Silica, SiO_2	34·10	31·50	35·90	1·76	3·26
Arsenic, As	0·28	0·25	0·18	0·47	0·44
Sulphur, S	1·31	1·05	0·88	2·20	3·01
Lime, CaO	0·95	0·90	0·75	3·00	3·85
Magnesia, MgO	0·67	0·47	0·41	0·54	0·68
Alumina, Al_2O_3	1·00	1·75	1·37	1·03	1·16
Water, H_2O	1·20	1·95	1·68	1·84	2·10
Alkalis, $Na_2O + K_2O$	1·30	1·56	1·45	1·81	2·12
Phosphorus, P	0·074	0·17	0·15	0·17	0·17
Molybdenite, MoS_2	0·23	0·10	0·17	0·10	trace
Zinc, Zn	trace	trace	trace	0·22	0·10
Carbon dioxide, CO_2	1·67	1·50	1·23	7·72	11·72
Silver, Ag oz./ton	4·20	3·71	10·97	301·67	360·98
Specific gravity	4·32	4·19	4·08	4·75	4·38

It will be noted in the above table that there are two samples from Vein No. 2, Pit No. 9. The reason for this is that on the material taken from this pit an attempt was made before shipment to cob out high-grade pitchblende ore and high-grade silver ore separately.

Treatment of High-Silica Type Pitchblende

The initial large-scale tests on this type of ore indicated on examination of residues that satisfactory radium extractions were obtainable by the procedure suggested and described in the report for 1931.

Minor difficulties resulting from oxidation, filtration, and washing had to be overcome to improve extraction and to carry the primary operations through in the shortest time.

The procedure previously suggested embraced washing the residue with brine to remove lead chloride, combining these washings with the main filtrate, and allowing the lead chloride to precipitate out in the diluted solution. The lead chloride was then separated by filtration and the filtrate treated with barium chloride and sodium sulphate for precipitation of radium. In this procedure it was found that the lead chloride showed a decidedly high radioactivity due to the presence of elemental radium. This radium content, it was proved, could be recovered by re-leaching the lead chloride with brine, indicating that some element in solution was precipitated by dilution and had the property of carrying with

it important amounts of radium. The element responsible appeared to be principally silica, probably present as a soluble product in the hot-acid solution, but becoming insoluble on cooling. The procedure, therefore, was altered to overcome this condition. Instead of using a brine wash on the residue, hot water was substituted, the amount of lead chloride contained in the residue being readily removed by this means. The washings were combined with the main filtrate and instead of permitting the lead chloride to precipitate and separate as previously, the precipitation of the radium by the addition of barium chloride and sodium sulphate was made directly, the whole being agitated for a total of 24 hours. A precipitate was thus obtained consisting of radium-bearing sulphate and lead chloride. This precipitate was next filtered off and treated with a strong brine solution whereby the lead chloride was dissolved and the radium-barium sulphate remained insoluble.

The brine solution upon dilution gave a lead chloride comparatively free from radium. This altered procedure saves the loss of radium occurring in the lead chloride, or saves a secondary treatment of the lead chloride for recovery of contained radium and in addition reduces the time for the first step of the process, namely concentration of the radium, by a full day.

Procedure I, therefore, may be briefly described as follows:—

The ore ground to 35 mesh is leached with 13° Bé. hydrochloric acid (low in sulphate content) for three hours at a temperature above 90° C., the quantity of acid used varying with the grade of ore. A solution of sodium nitrate is then added in small amounts at short intervals so that oxidation will not be too rapid nor frothing too excessive. The sodium nitrate has the effect of oxidizing and thereby making more readily soluble the highly refractory uranous oxide (UO_2) content of the ore.

The UO_2 content in these silica pitchblende ores apparently varies distinctly, ranging in the ratio of $2\frac{1}{2}$ to 4 parts UO_2 for each part UO_3, with the higher UO_2 ratio following higher silica content.

The oxidation operation requires about $1\frac{1}{2}$ hours. With the leach completed the residue is separated by filtration. Filtration should be carried out while the liquor is hot, as lead chloride separates out rapidly with cooling and makes filtration more difficult. The residue is well washed with hot water, the washings and main leach solution are combined and treated with measured quantities of barium chloride and sodium sulphate while warm, to precipitate the radium with barium as sulphates. Agitation for about 24 hours ensures practically complete precipitation of radium, and after allowing the precipitate to settle, it is separated by filtration and washed with acidulated water to remove the metal salts. The filtrate containing uranium and other metal chlorides is treated for recovery of uranium. The lead chloride-radium-barium sulphate precipitate is now treated with a hot solution of brine, whereby the lead chloride is dissolved, the radium-barium sulphate remaining insoluble and being collected on a filter. This precipitate constitutes the radium concentrate and is sent to the refining plant for final recovery of radium.

The above operations to produce a concentrate can be readily completed in 4 days.

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The accompanying flow-sheet illustrates the various steps of this procedure.

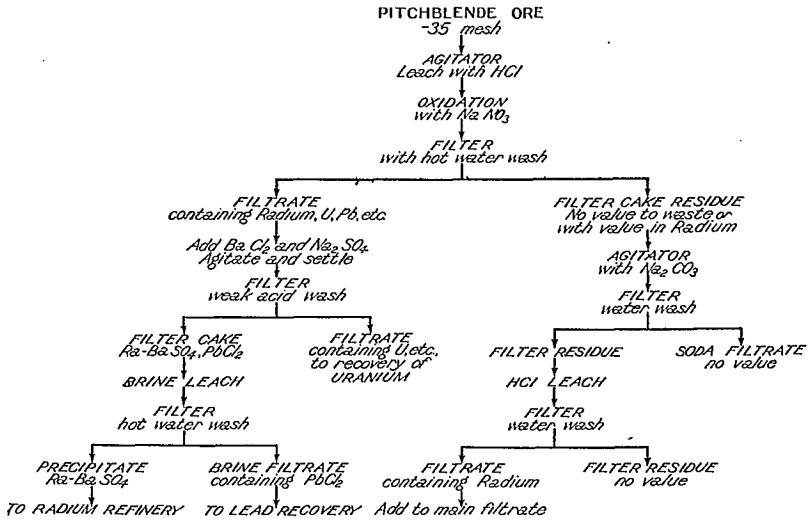

Figure 11. Flow-sheet for treatment of silica gangue pitchblende ore.

Results of Tests. Numerous tests were made in the manner described, with results as shown in the following examples:—

First Shipment

Composition: U_3O_8, 56·8 per cent; SiO_2, 19·92 per cent; S, 0·86 per cent.

Ore charge, kg.	Radium content, mg.	Residue, grm.	Radium content, mg.	Extraction, %
10	1·637	2,910	0·106	93·5
10	1·637	2,486	0·0296	98·2
10	1·637	2,535	0·0513	96·8

Vein No. 1, Pit No. 1.
Composition: U_3O_8, 44·6 per cent; SiO_2, 34·1 per cent; S, 1·31 per cent.

Ore charge, kg.	Radium content, mg.	Residue, grm.	Radium content, mg.	Extraction, %
10	1·2854	3,900	0·0506	96

Vein No. 2, Pits Nos. 2 to 6.
Composition: U_3O_8, 41·5 per cent; SiO_2, 35·9 per cent; S, 0·88 per cent.

Ore charge, kg.	Radium content, mg.	Residue, grm.	Radium content, mg.	Extraction, %
10	1·196	4,240	0·02	98·3

Included in the 20-ton lot was a 2,500-pound shipment from Vein No. 2 Pit No. 1.

This ore has the same general appearance and chemical composition as those above, but on test does not show extractions quite as satisfactory by direct leaching. The reason for this cannot at present be satisfactorily stated as a complete analysis of the ore is not available, and time has not permitted to make a definite investigation of the problem.

Direct leaching by the procedure described above shows an average extraction of about 80 per cent of the radium, the remainder being found in the residue. The residue contains practically no uranium and the radium is in a form that is insoluble in acids. Barium would naturally be suspected as the cause of the retention of radium in the residue but this element is present in an exceedingly small amount and may therefore be eliminated as the disturbing factor.

As a means of increasing the radium extraction, re-treatment of the residue is the only plan that can be suggested for the present, and the following method has been used with good results.

The residue is heated with a solution of soda ash for 3 hours with agitation, diluted with water and allowed to settle. The clear solution is then siphoned off and the residue transferred to a pressure filter and washed with water. The washed residue cake is then treated with 1 : 1 hydrochloric acid (low in SO_3 content) and after thorough agitation, the insoluble is separated by filtration. The filtrate contains the radium which may then be precipitated by the addition of barium chloride and sodium sulphate or sulphuric acid.

Filtration of both the alkaline and acid solutions is usually slow, and washing is likewise tedious and time-consuming.

By this double treatment, extractions have been increased to over 95 per cent. The following examples are typical of the results obtained:—

Vein No. 2, Pit No. 1

Composition:—U_3O_8, 44·0 per cent; SiO_2, 31·5 per cent; S, 1·05 per cent.

Ore charge, kg.	Radium content, mg.	Residue, grm.	Radium content, mg.	Extraction, %
10	1·268	3,850	0·2046	83·9
By re-treatment of residue..	.....	3,653	0·0438	96·5
10	1·268	3,780	0·2056	83·8
By re-treatment of residue..	.....	3,047	0·0437	96·5
10	1·268	3,770	0·2025	84·0
By re-treatment of residue..	.....	3,016	0·092	92·7
10	1·268	3,776	0·34	73·2
By re-treatment of residue..	.....	3,227	0·03	97·6

In all of the tests made the uranium extraction was higher than the radium extraction, the majority of the tests showing 99·9 per cent extraction.

The main filtrates, after precipitation and separation of the radium barium sulphate and the lead chloride, contained an average of 0·00145 milligram radium, or less than 0·10 per cent of the total radium.

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The separated lead chloride from the brine solution, representing 70 to 85 per cent of the total lead content of the ore, contained an average of 0·00057 milligram radium, or less than 0·04 per cent of the total radium content, while the brine solutions, after separation of the lead chloride, contained an average of 0·005 milligram or approximately 0·40 per cent of the total radium. The concentrates contained 86 to 97 per cent of the total radium content in concentrations varying from 1 part radium in 90,000 to 1 part in 125,000.

Treatment of Carbonate-Barite-Silver Type Pitchblende

The 20-ton shipment from the Eldorado property included two lots of this type of ore from the same pit.

The partial analyses of these lots are shown on page 252 (Preparation of Ore). Initial laboratory tests showed that, by direct leaching, the radium content would be divided between the insoluble residue and the leach solution, the greater proportion being held in the insoluble residue by reason of the presence of barium sulphate. In addition to this direct leaching with acid was attended with difficulties owing to the high carbonate content of the ore causing excessive frothing. The addition of acid to the ore in the most careful manner proved unsuccessful due to an apparently delayed reaction of the rhodochrosite with the acid, which caused an overflowing of the charge in the leaching tank.

The difficulty was overcome by roasting the ore at a temperature of 750° C. to 800° C. with good access of air, whereby the carbonates were materially broken up or deadened. Acid leaching could then be carried out with complete safety.

Leaching with hydrochloric acid (13° Bé), the radium remained in the insoluble residue with the silica, barium, silver, etc., hot water washing removing practically all of the lead. The presence of a small amount of sulphate, formed during the roasting (or added), ensures almost complete precipitation of radium in the residue. The filtrate contains the uranium, lead chloride and some silver chloride. When filtration of the leach is carried out hot, there is a greater amount of silver chloride carried to the filtrate than when the leach is allowed to cool before filtration. The latter procedure is therefore recommended: the leaching charge being allowed to cool and settle, the clear liquor siphoned off through the filter and the residue finally transferred to the filter with hot water and thoroughly washed.

The removal of silver from the residue would appear to be the next logical step. This is readily accomplished by leaching with sodium cyanide solution, the silver, present for the most part in the form of chloride, being easily soluble. The silver in solution may be recovered by any of the standard processes such as precipitation by sodium sulphide, aluminium, or zinc. The silver-free residue is next treated with a solution of sodium carbonate at boiling temperature and with continuous agitation for several hours, whereby the radium-barium sulphates are changed to carbonates. Filtration and careful washing eliminate the excess alkali and the sodium sulphate.

The residue of silica and carbonates is then treated with hydrochloric acid, the carbonates dissolving to chlorides, and the silica remaining insoluble is eliminated by filtration.

To the filtrate of the chlorides sodium sulphate is added in excess to precipitate the radium with the barium, the precipitate constituting the radium concentrate ready for refining. Preliminary tests following this procedure indicated that 95 per cent of the total radium content of the ore could be extracted as radium barium sulphate.

Large-scale tests using 10-kilogram (22 pounds) samples were made with equally satisfactory results as shown below.

The equipment used was as follows.

The ore was roasted in a gas-fired, muffle-hearth furnace at 750°C. to 800° C. for about an hour. For acid leaching of the roasted ore and filtration of the leach, similar equipment to that used in the leaching of the silica type ore was employed. Cyanidation of the residue was carried out in porcelain pebble jars containing a small quantity of pebbles, the jars being rotated. Pressure filtration was used for separation of residue and cyanide solution. Conversion of the sulphates to carbonates by soda ash was accomplished in equipment similar to that employed in the primary leach, filtration being done in a pressure filter. Final acid leaching was done in glass for convenience, but would ordinarily be done in stoneware.

The low-silica content of this type of ore assists in permitting quick filtering at all stages.

The various steps in the recommended procedure for ores of this type are shown in Figure 12.

Typical examples of results obtained by this procedure are as follows:—

Roasted ore: U_3O_8, 51·15 per cent; Ag, 339·9 oz./ton; $BaSO_4$, 3·00 per cent.

Ore charge, kg.	Radium content, mg.	Final residue, grm.	Radium content, mg.	Extraction, %
10	1·473	682	0·0543	96·3
10	1·473	610	0·0464	96·8
10	1·473	591	0·0365	97·5

Roasted ore: U_3O_8, 42·0 per cent; Ag, 223·6 oz./ton; $BaSO_4$, 5·10 per cent; SiO_2, 18·7 per cent.

Ore charge, kg.	Radium content, mg.	Final residue, grm.	Radium content, mg.	Extraction, %
10	1·21	2·160	0·0776	93·6

Roasted ore: U_3O_8, 50·5 per cent; Ag, 342·6 oz./ton; $BaSO_4$, 2·75 per cent; SiO_2, 2·30 per cent

Ore charge, kg.	Radium content, mg.	Final residue, grm.	Radium content, mg.	Extraction, %
10	1·456	390	0·027	98·1
10	1·456	308	0·0236	98·4

Roasted ore: U_3O_8, 38·8 per cent; Ag, 380·9 oz./ton; $BaSO_4$, 7·70 per cent; SiO_2, 6·55 per cent.

Ore charge, kg.	Radium content, mg.	Final residue, grm.	Radium content, mg.	Extraction, %
10	1·118	997	0·0368	96·7

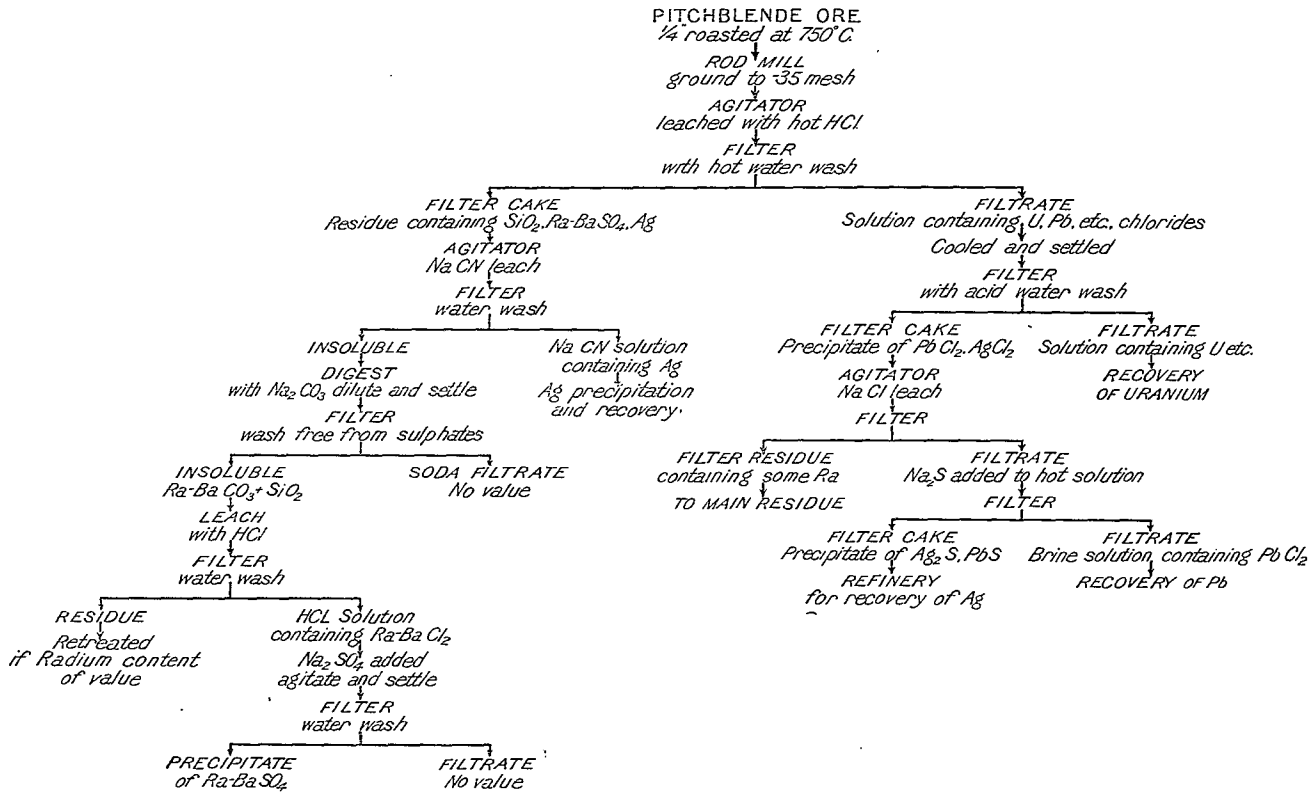

Figure 12. Flow-sheet for treatment of carbonate-silver pitchblende ore containing barite.

The uranium extractions in all cases were well above 99 per cent.

The uranium filtrates varied in total radium content between 0·003 and 0·015 milligram, the lead chloride between 0·004 and 0·02 milligram, the sodium cyanide solutions between 0·00004 and 0·0025 milligram, the sodium carbonate solutions between 0·0002 and 0·002 milligram, and the final acid filtrate from the radium barium sulphate precipitation averaged 0·00005 milligram.

The total radium losses in these solutions was in all tests under 2 per cent. The silver recoveries from the cyanide solution varied from 84 to 93 per cent of the total silver content of the ore. The silver was here recovered by sodium sulphide precipitation and reduction of the silver sulphide to metallic silver. The radium-barium concentrates varied in grade, being governed by the barium content of the ore. All the barium in solution must be precipitated to obtain complete precipitation of the radium, and so the higher the barium content in the ore the lower the grade of concentrate obtainable, and this is reflected further in the crystallization laboratory where increased fractionations will be required.

The operations entailed in this method of procedure should not require more than 8 to 10 days to produce a concentrate of radium ready for refining.

Refining of Radium-Barium Sulphate

In the treatment of pitchblende for the production of radium two main steps may be said to be involved, first, the concentration of the radium and, second, the refining of the concentrate to obtain radium salts.

The first of these two steps, relating to the ores under consideration, has been accomplished by the methods shown above, and a concentrate of radium in barium sulphate has been obtained.

The refining of this concentrate has not been carried out in this present investigation, mainly because the method used is already a well established practice, and was first proposed and used by Madame Curie some thirty years ago and has not been improved upon to any marked degree.

The refining of the concentrate entails mainly the separation of radium and barium and the only economical method known is by fractional crystallization of the soluble salts of the two elements.

To obtain the soluble salts the radium-barium sulphate is converted to carbonate by boiling with sodium carbonate. The soluble sulphates and excess alkali are leached out and the residue thoroughly washed with sulphate-free water.

The carbonates in the residue are then leached with C. P. hydrochloric acid, the chloride solution filtered, and the insoluble washed, and returned for re-treatment with sodium carbonate to convert any remaining sulphate to carbonate and thence to chloride.

The chloride solution sometimes contains appreciable quantities of lead which are removed by passing hydrogen sulphide gas through the solution and the lead sulphide is separated by filtration. Sometimes in presence of other impurities it is advisable to pass the hydrogen sulphide into an ammoniacal solution of the chlorides.

A departure from the above procedure was made by the U.S. Bureau of Mines investigators in their work on refining the radium concentrate obtained from the carnotite ores. The method consisted of reducing the

barium sulphate by means of charcoal to barium sulphide, the radium also changing to sulphide. The sulphides were then leached with hydrochloric acid, the solution filtered and the unconverted sulphates re-treated.

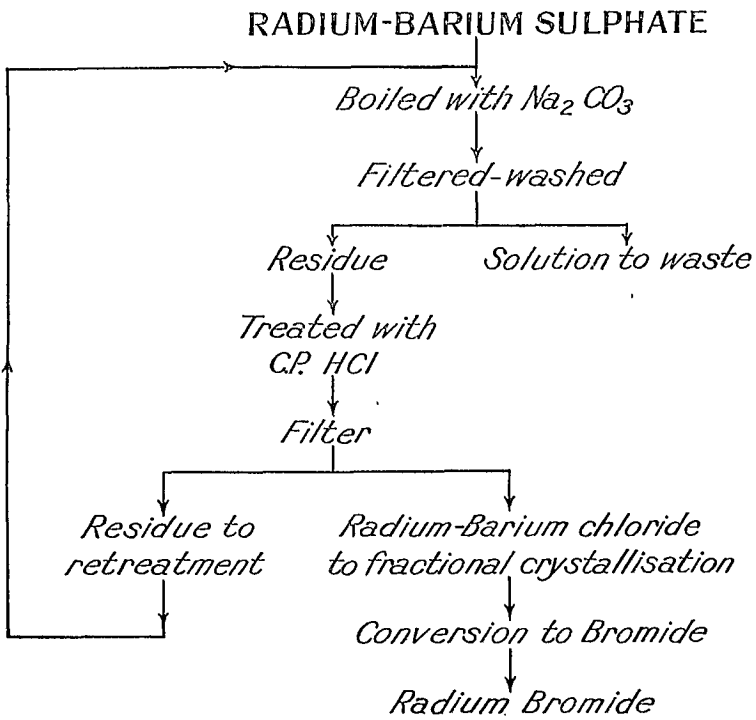

Figure 13. Flow-sheet for the refining of radium.

The process, however, has not been adopted elsewhere and the original method of Curie is generally adhered to.

The solution of the chlorides is now evaported to the crystallizing point. Radium chloride is less soluble than barium chloride and concentrates in the crystal fractions. This insolubility is greater in acid solutions than in neutral, and crystallizations are therefore carried out in acid solutions.

The crystal fraction of a chloride solution contains approximately four times as much radium as remains in the mother liquor.

These first crystals are now put into solution, the solution brought to the crystallizing point and a second crop of crystals obtained much higher in radium content. This scheme is continued until the concentration has reached about 50 milligrams of radium element per kilogram, at which point the chlorides are usually converted to bromides, the bromide of radium being still less soluble than the chloride, and the fractionations continued in hydrobromic acid.

The mother liquors are all treated in the same way, the crystal fractions being put into the system at the most convenient points. The system of

crystallization is established according to the ratio of radium to barium in the concentrate obtained from the plant. The final radium bromide is tubed and stored in lead-lined compartments, and checked several times for radium content. A purity of 95 to 98 per cent is usually called for. A careful account of the radium content of all liquors, precipitates, and crystals is maintained to determine the efficiency of the various steps.

Recovery of Uranium

No attempt has been made so far in this investigation to conduct large-scale tests on the recovery of uranium, but small-scale tests indicate that this phase should present no serious difficulty.

Methods already established are quite applicable and the general scheme may be briefly outlined as follows:

The main acid leach filtrate carries, as shown in test results, practically all of the uranium in acid chloride solution, together with iron, copper, manganese, and other base metals.

Most uranium compounds are soluble in solutions of alkaline carbonates and the methods commonly employed use this as the basis of separation from the other metals present.

Treatment of the acid leach filtrate with excess sodium carbonate precipitates most of the base metals, the uranium remaining in solution as sodium uranyl carbonate. The precipitate of base metals is separated

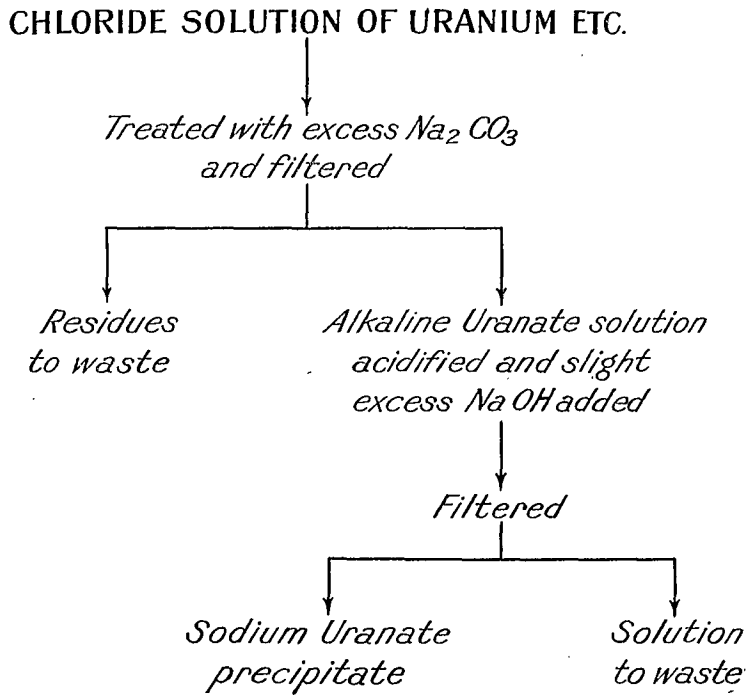

Figure 14. Flow-sheet for recovery of uranium.

by filtration and the filtrate is neutralized with acid, a yellow precipitate consisting of sodium uranate and uranyl carbonate being formed, which is soluble in excess of acid.

This yellow precipitate may be filtered off and the small amount of unprecipitated uranium remaining in solution recovered by addition of caustic soda or the main solution may be carried to the acid condition and all of the uranium precipitated as sodium uranate giving an orange-coloured compound.

The procedure is governed by the nature of the uranium compound required and to some extent also by the impurities present.

The uranium compounds obtained are separated by filtration, washed and dried, and prepared for market according to the colour, purity, and grade required.

CONCLUSIONS

Investigation of the present available pitchblendes from Great Bear Lake indicates that these ores can be readily treated for the recovery of radium.

The ores examined comprise two distinct types, differing widely in gangue-forming elements, one being a highly siliceous gangue pitchblende, and the other a carbonate-barite gangue pitchblende carrying silver.

No single process would appear to be economically suited to the treatment of the two ores for recovery of radium, and for reason of greater efficiency and better economic operation two separate processes are suggested.

Plant operation on the unit principle is advisable in working such high-grade and valuable material, and, where for the most part similar chemicals and equipment are applicable to both processes, the two-process plan should be quite feasible.

Units capable of treating 100 kilograms (220 pounds) ore per charge, based on present grade of ore, are suggested. Laboratory tests on a scale one-tenth this capacity have given satisfactory results, the uranium and radium extractions being well over 90 per cent in all tests, and all operations such as filtrations and washing have been performed without difficulty.

The procedure recommended for the high-silica type ore is simple and direct, requiring a minimum of operations and time to produce a high-grade radium concentrate.

The procedure for the carbonate-barite-silver pitchblende, while involving a greater number of operations, and longer time, is equally efficient and permits of a high recovery of the silver content.

No attempt has been made to estimate costs of operation as this is dependent upon too many variables which cannot be determined with sufficient accuracy on a laboratory-scale operation.

Report No. 477

REPORT OF RADIUM-MEASURING LABORATORY

In the Report of Investigations for this division for 1931 a brief account was given of the methods employed in radium measurements and the principles upon which these determinations are based.

During the year under review 381 measurements were made by the emanation method on the numerous products involved in the operations of the radium investigation. A compilation of the products tested is as follows:—

Pitchblende ore	25
Leached residue	72
Radium-barium concentrate	53
Lead chloride	47
Leach liquor (uranium solution)	14
Spent solutions (uranium)	23
Concentrate filtrate	27
Brine solution	18
Sodium carbonate solution	20
Sulphide cyanide solution	8
Miscellaneous determinations	17
Calibration of chambers	57
	381

Apart from the above, 34 tests were made under the alpha electroscope on mineral samples submitted to this division for radioactive determinations. These comprised rock specimens, uraninite concentrate, pitchblende ore, volcanic, ash, etc.

In the summer a small building near the Ore Dressing and Metallurgical Laboratory was taken over by this division and fitted up as a measuring laboratory and the quipment was moved up from the room which had been used for this purpose at the Mines Branch Building. The largest room was equipped with tables, hood and benches, and the necessary air and gas lines. A Hyvac pump and small blower were installed. This room was used for preparing the samples and boiling off and collecting the radon (radium emanation). A small adjoining room was used for the electroscopic readings and the storage of ionization chambers and the electroscope. A third room served as an office.

The methods of test consisted essentially of the two methods, boiling and fusions as outlined in the Summary Report of 1931.

All solutions were run by boiling off the radon and collecting it over hot sodium hydroxide solution in a gas burette and thence drawing it off into an evacuated ionization chamber. Ores, residues, lead chloride, and radium-barium-sulphate concentrates were run by the potassium bi-sulphate fusion method. This is the standard method developed by Howard H. Barker. A method using mixed chlorides, developed by a French investigator, A. Karl, was also tried. This consists of using an equimolecular mixture of sodium, potassium, and lithium chlorides instead of bi-sulphate. The melting point is higher than in the bi-sulphate process but the fusion is quiescent. Numerous tests were made all of which gave lower results than those obtained by the bi-sulphate method.

The calibration of the chambers constitutes one of the most important features of this work, for upon the accuracy of the calibration depends the accuracy of the final results. Any change in the leaf of the electroscope or change in alignment of the reading microscope will alter the calibration factor. It is necessary to read over the same scale divisions on a determination as when calibrating the chamber. Re-calibration should be carried out every few months. A standard radium solution is required and the one used by this laboratory was supplied by the kindness of Dr. Herman Schlundt of the University of Missouri. It is known as the University of Missouri Standard and was made up as follows:—

A specimen of radium bromide containing 0·978 milligram of radium element was dissolved in 2,200 c.c. of 1·2 normal hydrochloric acid solution containing 1 gramme of barium chloride per litre. Forty c.c. of this solution was supplied to the writer and from it suitable working standard solutions were made up. The original solution known as "A" has a radium content of 0·4445 microgram of radium per c.c. Ten c.c. of solution "A" was carefully and accurately pipetted and diluted to 500 c.c. in 1·2 normal hydrochloric acid containing 0·5 gramme barium chloride. This solution called "B" had a radium content of 0·00889 microgram per c.c. Solution "C" was prepared by diluting 50 c.c. of "B" in 500 c.c. 1·2 normal hydrochloric acid containing 0·5 gramme barium chloride. The quantity of solution used in the calibration measurements was:—

5 c.c. solution "B"	$4{\cdot}445 \times 10^{-5}$	milligrams radium
or 25 c.c. solution "C"	$2{\cdot}222 \times 10^{-5}$	" "

Three methods of removing the radon from the solution were employed: (1) Boiling in a flask, (2) Aspirating cold in a Curie bottle, and (3) A modification of (2) developed by the writer in which the solution is heated almost to boiling during the period that air is drawn through the solution into the chamber. Suitable tubes for drying and absorbing any acid fume are set in the tram ahead of the chamber. This method proved satisfactory and is slightly more accurate than aspirating the solution cold.

After the radon has been transferred to the chamber a definite period of time must elapse before a reading is made. Coincident with the decay of radon is the growth of the products radium A, B, and C. When equilibrium is established between the decay of radon and the growth of the disintegration products ionization will be at a maximum. This period of equilibrium is usually given at between the third and fourth hour. Series of readings were made over a period of six hours and it was found that at the third hour the maximum ionization was apparent. The results of these readings are tabulated below.

Time	Time of leaf fall for 10 divisions, in seconds	Rate of leaf fall in divisions per minute	—
10.05 a.m.	0	0	
10.35	17·40	34·412	
10.50	16·32	36·694	
11.05	15·68	38·195	1 hour
11.20	15·32	39·094	
11.35	14·96	40·036	
11.50	14·68	40·801	
12.05 p.m.	14·44	41·481	2 hours
12.20	14·00	42·787	
12.35	14·00	42·787	
1.05	13·84	43·282	3 hours
1.35	14·38	41·654	
2.20	14·40	41·596	
2.35	14·68	40·801	
3.05	14·70	40·746	5 hours
3.35	14·88	40·252	
4.05	14·90	40·198	6 hours

The natural drift of the electroscope leaf is always taken prior to a test. The instrument is charged for fifteen minutes and the fall of the leaf then noted across one or two divisions, the time for which may vary from ten to twenty minutes. It is essential that the natural fall should be as low as possible; and atmospheric conditions play an important part in this regard. Humidity presents a serious problem especially in the summer and certain precautions have to be adopted to guard against traces of moisture accumulating in and around the insulation of the chamber heads. It was found that a liberal use of calcium chloride in the cupboards in which the chambers were stored contributed in a large measure in keeping the humidity of the air down, thus maintaining the natural leak within limits suitable for normal determinations.

The calibration factor is calculated from the following equation:—

$$C = x \frac{(1-e^{-\lambda t})}{N-n}$$

C = Calibration factor = milligrams of radium in equilibrium giving one division fall per minute.

x = Milligrams of radium in standard sample used.

N = Divisions per minute leaf fall given by emanation from solution.

n = Divisions per minute leaf fall due to natural drift of electroscope.

$1-e^{-\lambda t}$ = Fraction of equilibrium amount of emanation, formed in t days (found from Kolowrats table).

The measurement of radium by the radon (radium emanation) method presents a multiplicity of factors all of which have a very direct bearing on the final results obtained. Each determination must be carried out with extreme care. The method is limited to solutions and materials of low radium concentration and difficulty is experienced in obtaining concordant results on materials having a radium concentration of 1 to 100,000 or 150,000. However, as a means of detecting traces of radium, the method is extremely sensitive and accurate.

Report No. 478

PRECAUTIONS FOR WORKERS IN THE TREATING OF RADIUM ORES

The precautions to be taken in guarding against the hazards involved in the treating of radium-bearing materials were briefly outlined in the Report of Investigations for this division in 1931.

During the year under review, all those engaged on the radium work were subjected to a regular examination each week for presence of radon in the lungs, and to a blood test every month. Through co-operation with the Department of Pensions and National Health, the blood tests were made by Mr. B. W. Culyer of the Laboratory of Hygiene, and early in September the making of expired air tests was transferred from the laboratory to the Food and Drugs Laboratory and the tests carried out by Mr. R. D. Whitmore.

It was soon apparent that the high uranium content of the pitchblende under investigation increased the scope of the usual hazards. This conclusion was arrived at only after the tests carried out on the workers and the tests on the atmosphere of the laboratories showed very definitely the presence of radioactivity. Despite the artificial and natural ventilation of the laboratories during the summer months, an increase in the radon (radium emanation) content of the atmosphere was found.

Expired Air Tests

During the year 107 tests were made on the personnel engaged in the radium investigation. The test is carried out quite simply and furnishes an excellent check on the individual. A 2·7-litre capacity ionization chamber, of which the natural leak has been obtained, is used. The subject exhales air from his lungs for three minutes, through a series of drying bottles containing calcium chloride and soda lime, into the ionization chamber. The calcium chloride is to dry the exhaled air, and the soda lime to remove carbon dioxide. At the end of three minutes' blowing, the taps on the chamber are closed and it is set aside for three hours, after which a reading is made. The results are then calculated from the calibration factor of the chamber in curies per litre of radon.

Some interesting information relative to the time taken to eliminate radon from the lungs was obtained. The following figures in curies per litre show the gradual elimination over a period of seven days, and serve as an illustration:—

August 2	$46{\cdot}8 \times 10^{-11}$
" 3	$15{\cdot}26 \times 10^{-11}$
" 4	$12{\cdot}22 \times 10^{-11}$
" 5	$7{\cdot}26 \times 10^{-11}$
" 8	0

Radon (radium emanation) is the heaviest gas known and consequently it will have a tendency to collect at the bottom of the lungs. This fact retards its elimination by normal breathing and accounts for the time taken to completely free it from the lungs.

A mill operator, who for a week was grinding pitchblende ore, was tested and showed a definite radon content in his exhaled air.

A small test was made on 50 grammes of pitchblende, about $\frac{1}{2}$-inch size, in order to determine the radon given off from coarse pieces of ore. The results showed that about 7 millicuries of radon per 100 kilos of ore were evolved in thirteen days.

Test of Laboratory Air

The testing of the atmosphere of the laboratories was also carried out at different times and a description of these tests and their results will be discussed.

A glass funnel, packed loosely with cotton wool and attached to a rubber tube, was suspended five feet from the floor in the leaching laboratory. The rubber tubing was carried into another room and attached to a jar containing calcium chloride and a wash bottle containing sulphuric acid. A calibrated ionization chamber was placed in the train, the off-tap

being connected to a vacuum pump. A slow current of air was drawn through the chamber for twenty-three minutes. The reading for this test gave a radon content of the atmosphere of 1,220 x 10^{-11} curies per cubic metre. This test was made on March 11. By May 4 a similar test was carried out and the radon in the laboratory air had increased to 1,826 x 10^{-11} curies per cubic metre. A further test made on September 30 gave the radon content in the leaching laboratory air at 2,167 x 10^{-11} curies per cubic metre, and that of the small laboratory, where ore charges were weighed out, as 1,734 x 10^{-11} curies per cubic metre.

The results show that, despite ventilation, there was a gradual increase in the radon content of the air, due probably to the normal splashing or dropping of active solutions on the floor and, further, to the storage of pitchblende ore. While the figures reported above are extremely small, they show a gradual increase, despite the fact that the tests were made during the season when natural ventilation can be utilized to the fullest extent.

Blood Tests

Blood tests were made on those engaged in radium work at approximately monthly intervals. These tests were begun on June 27 and were carried out at the Laboratory of Hygiene, Department of Pensions and National Health, under the direction of Dr. N. MacL. Harris. A complete chart is maintained for each worker, and any irregularity is checked against the expired air tests.

Conclusions

In view of the importance of all necessary precautions being taken to minimize the inherent hazards in the treating of radium ores and products, a very complete record has been kept and the information so obtained constitutes valuable data on this important subject.

V

REPORTS OF INVESTIGATIONS: SECTION OF FERROUS METALLURGY

The systematic investigation of Canadian iron ores from the viewpoint of their utilization in the blast furnace as well as from that of their suitability for the production of sponge iron has been continued as far as possible during the year. Unfortunately, it is becoming increasingly difficult to obtain suitable representative samples of our known deposits of iron ore and for this reason the number of ores investigated during 1932 was less than in the previous year.

In our report for the year 1931 the results of laboratory concentration tests on a small shipment of magnetite from Texada Island, B.C., were given and the conclusions there drawn from these tests were that the ore was excellent for the production of sponge iron as well as for pig iron. A report covering larger scale work on the concentration of this ore and its actual conversion into sponge iron will be found in the following pages and it will be noted that the results anticipated in the light of the laboratory concentration tests have been realized in the actual reduction tests.

In view of the favourable results obtained from the first small shipment of Texada Island magnetite and of the possibility that the deposits as a whole might run considerably higher in sulphur than did this lot, arrangements were made for a carload shipment. As anticipated the sulphur content of this shipment was much higher than that of the first lot, but the results of sintering tests and laboratory concentration tests presented elsewhere in the report indicate that this high-sulphur magnetite can readily be made suitable for the production of both pig iron and sponge iron. The results of the laboratory concentration tests on this high-sulphur magnetite will be confirmed by large-scale tests which will include the actual production of sponge iron and its conversion into steel.

The operations carried out at the Moose Mountain, Limited property, near Sellwood, Ontario, during the period 1917-1921, have made it clear that in this property Canada possesses a valuable reserve of low-grade magnetite, and that this ore may readily be beneficiated to yield an exceptionally high-grade concentrate. That this concentrate is well suited to the production of a high-grade sponge iron will be seen from the accompanying report of metallization tests carried out on a small shipment of screenings from briquetted concentrates made on a semi-commercial scale at the property. These briquettes were made for blast furnace use and not for conversion into sponge iron and do not, it is believed, represent the highest grade of concentrate that can be produced from this ore. It is, therefore, reasonable to expect that a sponge iron containing 93 to 96 per cent iron and low in phosphorus and sulphur can be produced from Moose Mountain ore.

During the year various problems were undertaken at the request of several manufacturers and of the Department of National Defence. This work was largely concerned with the investigation of materials that failed

to stand up in service, and the development of better materials. Since this work is closely associated with the intimate affairs of the organization for which it was carried out the reports covering these investigations are not reproduced here. An idea of the nature of these special investigations may be formed from the following list of reports that have been prepared during the year but which are not included in this summary report.

The structure and compressive strength of some samples of cold-drawn tubing used in aircraft construction.

The comparative properties and constitution of some samples of aluminium alloy used in airscrews and the investigation of a propeller blade which failed in service.

McQuaid-Ehm and other tests on certain samples of hollow drill steel.

An investigation to determine the possibility of controlling coarse carbide formation in cast chromium-molybdenum rolls of high-carbon content.

McQuaid-Ehm and other tests on sixteen samples of hollow drill steel.

The use of our special equipment and the co-operation of our staff have been requested by several organizations in the preparation of special alloys and the heat treatment of various articles. These services have not been the subjects of formal reports but they constitute an important phase of our work. A list of the more important services so rendered follows:—

1. The preparation of a number of special experimental austenitic manganese steel castings.
2. The preparation of a number of special experimental austenitic steel ingots.
3. The production of two experimental ingots of nickel-iron staybolt material.
4. The heat treatment of seven airplane tail forks.
5. The heat treatment of a number of nickel steel eyebolts.

Report No. 479

THE PRODUCTION OF SPONGE IRON FROM TEXADA ISLAND (B.C.) IRON ORE

Object of Investigation. To produce sponge iron, on a semi-commercial scale, from Texada Island iron ore and to determine the suitability of the sponge iron so produced for the manufacture of steel.

Shipment. A shipment of 2,496 pounds of this ore was received on November 4, 1930, from Mr. John D. Galloway, Provincial Mineralogist, Province of British Columbia.

Nature of the Ore and Chemical Analysis of Shipment. The ore is a high-grade magnetite. The shipment was crushed and carefully sampled. This sample, on chemical analysis, gave the following results:—

(Dried at 105° C.)

Fe_3O_4	85·59	(Fe = 62·06)
$CuFeS_2$	0·13	(S = 0·15)
FeS_2	0·20	(Mn = 0·10)
MnO_2	0·16	(P = 0·056)
SiO_2	8·41	(Cu = 0·045)
P_2O_5	0·13	
CaO	2·80	
MgO	1·17	
Al_2O_3	1·31	
	99·90	

Experimental Tests

Laboratory Tests

With the object of determining the extent to which the gangue content of the ore could be removed by concentration small-scale magnetic concentration tests were carried out in the Davis magnetic tube. The results of these tests have been presented in Report No. 420 entitled "The Laboratory Concentration of Texada Island Iron Ore."[1] These tests, showed that the ore responds very well to magnetic concentration, and that by grinding to minus 40 mesh or finer the gangue, including the sulphur-bearing constituents, is liberated to an extent sufficient to make it possible to produce from this concentrate, a sponge iron containing over 90 per cent iron and low in sulphur and phosphorus.

Large-Scale Tests

In the light of the information obtained from the laboratory concentration tests, the following series of steps for the commercial production of sponge iron from Texada Island ore, was indicated as the most suitable:

1. The wet magnetic concentration of the ore ground to minus 100 mesh.
2. The sintering of this concentrate to produce a product more physically suitable for reduction and briquetting.
3. The low-temperature reduction of this sintered material, crushed to minus 10 mesh.
4. The briquetting of the sponge iron so reduced to yield a product suitable for conversion into steel in electric or open-hearth furnaces.

Concentration

In this run, 1,500 pounds of ore were crushed to minus 6 mesh and fed to rod mill at the rate of 100 pounds per hour, the rod mill being operated in closed circuit with an Aikens classifier. A screen test on the classifier overflow gave the following results:—

	Per cent
+100 mesh (Tyler screen)	2·00
−100 +150 mesh	7·65
−150 +200 mesh	11·45
−200 mesh	78·90
	100·00

From the screen test it will be seen that the material was ground much finer than necessary, as preliminary small-scale tests indicated that the desired results could be obtained with 100-mesh material.

The concentration was carried out in two Gröndal magnetic separators, so arranged that the second of these machines served to clean the tailing from the first. The results obtained in this concentration are summarized in Table I. This table shows that the results obtained in the preliminary small-scale tests can be duplicated in large-scale operations.

[1]Invest. Ore Dressing and Metallurgy, 1931, Mines Branch, Dept. of Mines, Cannda.

TABLE I

Wet Magnetic Concentration of Texada Island Ore

Feed:

	Per cent
Weight	100·0
Iron	62·06
Copper	0·045
Sulphur	0·150
Phosphorus	0·056
Insoluble	11·54

Concentrate:

	Per cent
Weight	85·2
Iron	70·25
Copper	nil
Sulphur	0·01
Phosphorus	0·02
Insoluble	3·39

Tailing:

	Per cent
Weight	14·8
Iron	14·90
Copper	0·30
Sulphur	0·95
Insoluble	58·40

	Per cent
Iron recovery	96·4
Sulphur rejected	94·3
Phosphorus rejected	69·6
Insoluble rejected	74·9

Ratio of concentration, 1 : 1·17.

Sintering

This operation was carried out for two reasons: First, low-temperature reduction or metallization is very difficult to carry out on fine material that packs tightly and is impervious to the reducing gases. Second, a much better briquette is obtained from comparatively coarse material than from very fine sponge. The finely divided concentrate was, therefore, agglomerated by sintering.

The successful sintering of finely divided material is rather difficult due to imperviousness of the bed to the air blast and the tendency of the material to run through the grates. In commercial practice, however, there is generally, if not always, sufficient so-called "returns," that is, semi-sintered material and comparatively fine particles of sintered material available to mix with the raw material, thus correcting these undesirable conditions to a large extent.

In these tests a total of 1,000 pounds of concentrate was sintered on a 12- by 48-inch Dwight-Lloyd continuous sintering machine. The mix yielding the most physically satisfactory sinter was as follows:—

	Per cent
Concentrate	70
"Returns"	17
Coke breeze	5
Water	8

In addition to the above "returns" a layer of coarse sinter (about 4 mesh) was placed directly on the grates to prevent the fine material from being drawn through.

The sinter obtained was well fused and was fairly hard. If possible the mix should be changed to yield a sinter with cells that are smaller and more numerous since such a sinter would be mechanically stronger and more suitable for reduction in stack furnaces. However, the sinter obtained was quite satisfactory for the required purpose as the plan included crushing the sinter to 10 mesh and reducing it in an externally heated rotary retort.

Low-Temperature Reduction (Metallization)

The only equipment available for this operation consisted of an electrically heated rotary retort that has been used in all our work on the low-temperature reduction of ores. This alloy retort, which forms the heating chamber of a 60 k.w. furnace has an inside diameter of $14\frac{3}{4}$ inches and an effective loading length of 48 inches.

In carrying out this work, a batch of approximately 100 pounds of the sintered concentrate, crushed to pass a 10-mesh Tyler screen, was charged into the retort. The retort and its contents were heated to 1,800° F. and held at this temperature for periods of time varying from 6 to $7\frac{1}{2}$ hours with different charges. During this entire period a stream of city gas was passed through the retort. At the end of the heating operation, the material was discharged into a special container provided with a tightly fitting cover.

In all, nine batches, totalling 1,000 pounds of sintered concentrate, were metallized in this manner. The results obtained with each batch and the general average result of the total amount metallized are summarized in Table II.

TABLE II

Metallization Tests on Sintered Concentrate

Batch No.	Total iron, per cent	Metallic iron, per cent	Metallization, per cent	Sulphur, per cent
1	87·35	82·63	94·6	0·02
2	88·26	87·07	98·6	
3	91·16	90·96	99·7	
4	90·81	90·23	99·3	
5	91·62	90·03	99·3	
6	91·01	90·42	99·3	
7	83·92	74·76	89·1	
8	84·90	80·08	94·3	
9	89·63	88·85	99·1	
Average of mixture of 9 batches	89·17	86·79	97·5	0·02

This material was given one pass on a drum-type dry magnetic separator and a concentrate of the following composition was obtained:—

	Per cent
Total iron	90·03
Metallic iron	88·26
Sulphur	0·020
Phosphorus	0·025

It will be noted in Table II that the metallization in two of the runs was considerably below that of the others and that this has resulted in the metallic iron content of the mixture of all 9 batches being somewhat lower than it would otherwise have been.

Briquetting

Sponge iron in the loose powdery form is most unsuitable for charging into steel-melting furnaces for two reasons: first, its density is so low that in order to get any considerable weight into the furnace, repeated rechargings involving great expense would be necessary; and second, such finely divided material is subject to excessive oxidation loss in melting. By means of briquetting the material may be formed into compact briquettes the density of which will be about 50 to 60 per cent of that of solid steel and these briquettes due to their greatly reduced surface area do not oxidize during melting.

In this work some 660 pounds of minus 10-mesh sponge iron was briquetted in a 70-ton Southwark briquetting machine. The material briquetted very satisfactorily. The briquettes were 2 inches in diameter and averaged about $1\frac{1}{8}$ inches long. One of these briquettes weighed about 55 per cent of the weight of a solid steel slug of the same dimensions.

Melting Tests

To demonstrate that these briquettes are quite suitable for conversion into commercial steel by melting and alloying with carbon and manganese, melting tests were carried out. Melting was done in a 50-pound Ajax-Northrup, high-frequency induction furnace and the steel produced was cast into $3\frac{1}{2}$-inch ingots.

The results of these tests are summarized in Table III which gives for each ingot made the materials used, the weight and chemical analysis of the ingot produced, and the approximate recovery of the metallic constituents of the charge.

SUMMARY AND CONCLUSIONS

This work has confirmed the conclusion drawn from small-scale laboratory concentration tests that Texada Island magnetite is entirely suitable for the manufacture of sponge iron. Approximately 700 pounds of sponge iron of excellent quality was produced from this ore, and the suitability of this sponge iron for the manufacture of low-sulphur, low-phosphorus steel was demonstrated by actual melting tests.

TABLE III

Summarized Results of Melting Tests

Ingot No.	Charge, pounds				Ingot						Recovery %
	Briquettes	Coke	Ferro-manganese	Ferro-silicon	Weight, lb.	C %	Mn %	P %	S %	Si %	
1.....	45·0	0·09	0·40	0·40	37·5	0·11	0·55	0·028	0·017	0·16	92·3
2.....	45·0	0·13	0·40	0·40	37·0	0·09	0·49	0·028	0·013	0·15	91·0
3.....	45·0	0·10	0·40	0·40	36·5	0·22	0·66	0·032	0·017	0·25	90·0
4.....	45·0	0·26	0·40	0·40	37·0	0·29	0·67	0·026	0·028	0·25	91·0
5.....	45·0	0·19	0·40	0·40	36·5	0·43	0·73	0·028	0·022	0·31	90·0
6.....	45·0	0·39	0·40	0·40	36·6	0·99	0·75	0·029	0·024	0·30	90·0

Report No. 480

THE PRODUCTION OF SPONGE IRON FROM MOOSE MOUNTAIN (ONT.) CONCENTRATE

Object of Investigation. To produce sponge iron, on a semi-commercial scale, from Moose Mountain concentrate, and to determine the suitability of the sponge iron so produced for the manufacture of steel.

Shipment. A shipment of 1,000 pounds of briquette screenings made from Moose Mountain concentrate was received on August 4, 1932. This shipment was made from Sellwood, Ontario, at the request of Mr. W. Rowland Cox.

Physical and Chemical Characteristics of Shipment. This shipment consists of screenings from briquettes made some years ago on a fairly large scale at Sellwood, Ontario, from concentrated Moose Mountain magnetite. The average grade of the ore as mined is about 34 per cent iron and about 0·080 per cent phosphorus, and large-scale tests have demonstrated that by fine grinding and wet magnetic separation a concentrate averaging 68 to 69 per cent iron and under 0·012 per cent phosphorus may be obtained. This concentrate yielded a sinter of approximately the same composition. Briquettes made from this concentrate averaged 66 to 67 per cent iron and less than 0·012 per cent phosphorus.

The shipment submitted for test consisted of screenings from briquettes made during the years 1917-20, when the property was operating. The material was about quarter-inch size and finer, as received, and a carefully prepared sample yielded the following results:—

	Per cent
Iron	64·13
Phosphorus	0·018
Sulphur	0·006
Silica	6·96
Moisture	0·02

Experimental Tests

Low-Temperature Reduction (Metallization)

For the conversion of this material into sponge iron there was available the electrically heated, rotary retort that has been used in all previous experimental work of this nature. This alloy retort, which forms the heating chamber of a 60 k.w. furnace, has an inside diameter of $14\frac{3}{4}$ inches and an effective loading length of 48 inches.

In these experiments, batches, usually consisting of 60 pounds of briquette screenings and 20 pounds of 3-mesh charcoal, were charged into the retort. The retort and its contents were heated to 1,700° F. and held at this temperature for three hours. The temperature was then raised to 1,800° F. and maintained there until a total of four hours had elapsed after the retort had reached the temperature of 1,700° F. At the end of this 4-hour period, the reduced material was discharged into a special container provided with a tightly fitting cover. When cool the sponge iron was given three passes over a drum type dry magnetic separator in order to remove the excess charcoal and any other mechanically free non-metallic matter that might be present. A total of 975 pounds of ore was treated in this manner and the yield was 685 pounds of concentrated sponge iron.

The analyses of the sponge iron concentrate from the various runs made are as follows:—

Results of Metallization Experiments

Batch No.	Total iron	Metallic iron	Metallization	Phosphorus	Sulphur
	%	%	%	%	%
1	88·26	87·23	98·8		
2	89·18	88·46	99·2		
3	88·70	88·40	99·6		
4	88·80	88·50	99·6		
5	88·86	88·67	99·7		
6	88·45	87·82	99·3		
7	86·19	85·98	99·7	0·020	
8	85·77	85·37	99·5	.....	0·010
9	85·98	85·56	99·5	0·019	
10	86·81	86·60	99·7	.....	0·006
11	87·84	87·42	99·5	0·019	
12	85·99	84·95	98·8	.....	0·010
13	84·95	84·54	99·5	0·023	
14	87·64	86·60	98·8	.....	0·010
15	88·25	87·22	98·8	0·021	
16	85·96	85·37	99·3	.....	0·010
17	87·63	86·60	98·8	0·021	
Average	87·37	86·76	99·3	0·020	0·010

Iron charged into retort..... = 975 x 0·6413 = 625 pounds
Iron recovered as metallic iron..... = 685 x 0·8676 = 594 pounds

Iron recovery..... = 95·0 per cent

Briquetting

Since sponge iron in the loose powdery form is most unsuitable for charging into steel-melting furnaces because of its low density and its susceptibility to excessive oxidation during melting, briquetting is most desirable if not absolutely necessary.

In this work 680 pounds of concentrated sponge iron was briquetted in a 70-ton Southwark briquetting machine. The material briquetted very satisfactorily. These briquettes were 2 inches in diameter and averaged about 1 inch in length. Their density was about 5·3 or approximately 67 per cent of solid iron.

Melting Tests

That these briquettes are physically and chemically suitable for direct conversion into steel by simple melting and recarburizing is fairly evident from the data already submitted. In order to demonstrate the suitability of these briquettes for direct conversion into steel, actual melting tests were carried out in a 50-pound Ajax-Northrup, high-frequency, induction furnace. The steel produced was cast into 3½-inch ingots.

The results of the melting tests are tabulated below. This tabulation shows for each of the ingots made, the materials used, the weight and chemical analyses of the steel produced, and the approximate recovery of the metallic constituents of the charge.

Summarized Results of Melting Tests

Ingot No.	Charge in pounds				Steel produced						Recovery %
	Briquettes	Coke	Ferro-manganese	Ferro-silicon	Weight	C %	Mn %	P %	S %	Si %	
					lb.						
1.....	56	Nil	0·37	0·25	47	0·10	0·47	0·031	0·106	0·18	95·5
2.....	56	0·15	0·37	0·25	47	0·34	*0·16	0·030	0·017	0·17	95·5

* Low manganese due to faulty melting practice.

SUMMARY AND CONCLUSIONS

The tests demonstrate that a sponge iron suitable for the production of high-grade steels may be produced from Moose Mountain ore. The concentrate submitted for test is not so high grade as may be produced from Moose Mountain ore since it is a matter of record that a concentrate containing 68 to 69 per cent iron was produced from this ore on a fairly large scale during the years 1917-1920. Since the concentrate submitted, which contained under 65 per cent iron, yielded a satisfactory sponge iron containing almost 87 per cent metallic iron and low in sulphur and phosphorus, it is clear that a sponge containing over 90 per cent of iron and equally low in sulphur and phosphorus would be obtainable from the high-grade concentrate produced from Moose Mountain ore.

Report No. 481

THE LABORATORY CONCENTRATION OF A HIGH-SULPHUR MAGNETITE FROM TEXADA ISLAND, B.C.

Shipment. A shipment of 29½ tons of Texada Island iron ore was received on September 7, 1932, from Mr. John D. Galloway, Provincial Mineralogist, Province of British Columbia. Report No. 420, "The Laboratory Concentration of Texada Island Iron Ore"[1] and the subsequent beneficiation of that ore on a semi-commercial scale as detailed in Report No. 479, "The Production of Sponge Iron from Texada Island (B.C.) Iron Ore,"[2] were so favourable as to warrant further work on a larger scale. It was also believed that further work should be desirable in view of the fact that the average sulphur content of the ore-body was thought to be considerably higher than that of the first shipment.

Object of Investigation. To determine the degree to which this shipment may be beneficiated, particularly with respect to the elimination of sulphur.

Character of the Ore. The minerals observed to occur in the ore are magnetite (Fe_3O_4), pyrite (FeS_2), chalcopyrite ($CuFeS_2$), and a mineral which in most respects closely resembles marcasite (FeS_2). Of the sulphides pyrite is the most abundant, marcasite(?) is next, and chalcopyrite in small quantities.

[1] Mines Branch, Dept. of Mines, Canada. Invest. Ore Dress. & Met., 1931, pp. 156-158.
[2] *See* page 269 of this volume.

The pyrite occurs as small grains and veinlets in the magnetite. Chalcopyrite is closely associated with the pyrite, and is finely divided. The marcasite(?) is present as incrustations and fillings in cavities both in the magnetite and in the gangue. All of the sulphides are usually very closely associated with the magnetite.

Although it is exceedingly difficult to determine the effective grain size of minerals that occur in both individual grains and in veinlets of considerable length, an attempt was made at an approximation by means of the microscope. The results are given in Table I.

TABLE I

Approximate Effective Grain Size of the Sulphides

Mesh	Sulphides, per cent
+ 48	27·0
− 48+ 65	26·0
− 65+100	29·0
−100+150	9·0
−150+200	6·0
−200+325	2·0
−325	0·7
Total	99·7

Analysis of Shipment. This high-grade magnetite was crushed and carefully sampled to provide a head sample which, upon analysis, gave the following results:—

	Per cent		Per cent
Fe_3O_4	83·54	Fe	62·27
$CuFeS_2$	0·29	S	2·10
FeS_2	3·74	P	0·015
MnO_2	0·16	Mn	0·10
SiO_2	6·45	Cu	0·10
P_2O_5	0·034		
CaO	2·78		
MgO	0·96		
Al_2O_3	0·54		

Except for the high-sulphur content this would be an excellent ore for direct use in the blast furnace. The high-sulphur content, however, makes some form of desulphurization necessary before the ore can be commercially utilized either for the production of pig iron or of sponge iron. To effect this desulphurization, two methods suggest themselves. The first is the sintering of the ore after comparatively coarse grinding; this would probably yield a product satisfactory for blast furnace use, but too low in iron for the production of sponge iron. The second is the wet magnetic concentration of the ore after crushing to a degree sufficient to liberate the sulphides and at the same time the gangue, thus making possible the production of a concentrate not only low in sulphur but also high enough in iron to be suitable for the production of sponge iron as well as for pig iron. This report deals with the second method only.

Experimental Method

The method employed in these small-scale tests has been fully explained in the report[1] on "Laboratory Concentration of Wabana Iron Ore." The Davis tube was used, and separations made on samples crushed to −14, −20, −40, −60, −100, and −150 mesh.

Results Obtained

	Size of ore fed to magnetic tube (Tyler mesh)					
	−14	−20	−40	−60	−100	−150
	%	%	%	%	%	%
Tube Feed—						
Iron	61·86	61·86	61·86	61·86	61·86	61·86
Insoluble	11·05	11·05	11·05	11·05	11·05	11·05
Sulphur	2·10	2·10	2·10	2·10	2·10	2·10
Phosphorus	0·015	0·015	0·015	0·015	0·015	0·015
Tube Concentrate—						
Weight	85·5	84·5	83·0	81·5	81·0	80·9
Iron	68·46	69·18	70·11	71·14	71·45	71·55
Insoluble	5·01	4·43	2·85	1·50	1·14	1·02
Sulphur	0·44	0·40	0·30	0·12	0·09	0·08
Tube Tailing—						
Iron	23·0	21·9	21·6	21·0	21·0	20·8
Insoluble	46·66	47·1	51·1	53·1	53·3	53·5
Sulphur	11·9	11·4	10·9	10·8	10·7	10·6
Ratio of Concentration	117 : 1	118 : 1	121 : 1	123 : 1	124 : 1	124 : 1
Iron Recovery	94·6	94·5	94·1	93·7	93·6	93·6
Insoluble Rejected	62·1	66·1	78·6	88·9	91·6	92·5
Sulphur Rejected	82·0	83·9	88·1	95·4	96·5	96·9
Theoretical iron content of sponge iron producible from concentrate	92·7	94·0	95·8	97·7	98·3	98·4

SUMMARY AND CONCLUSIONS

The phosphorus in the head sample was so low as to render unnecessary any consideration of that element in the tests. The results obtained indicate that even at minus 14 mesh a product is obtained suitable for conversion to sponge iron. However, grinding to minus 60 mesh appears to be the best practice with this shipment, since a marked improvement in the iron content and a very appreciable lowering of the sulphur and insoluble content of the concentrate is obtained at that point. Finer grinding is not warranted since the improvement in the concentrate is apparently insufficient to compensate for the extra grinding.

This shipment appears to be nearly ideal for the production of sponge iron, since the iron content can be raised almost to that of pure Fe_3O_4, whereas the sulphur remaining after concentration should be readily reduced to a low value by the subsequent sintering operation which would be necessary to put the concentrate in a physical condition suitable for blast furnace or direct reduction.

[1]Mines Branch, Dept. of Mines, Canada. Invest. Ore Dress. & Met. 1930, pp. 195-198.

Report No. 482

SINTERING TESTS ON HIGH-SULPHUR MAGNETITE FROM TEXADA ISLAND, B.C.

Shipment. A carload shipment of 29½ tons of Texada Island magnetite was received on September 7, 1932, from Mr. John D. Galloway, Provincial Mineralogist, Province of British Columbia.

Object of Investigation. This shipment was submitted in order that the favourable results obtained with a previous shipment of 2,496 pounds, recorded in Report No. 420, "The Laboratory Concentration of Texada Island Iron Ore,"[1] and also in Report No. 479, "The Production of Sponge Iron from Texada Island (B.C.) Iron Ore,"[2] might be confirmed by tests on a larger and possibly more representative shipment. The belief that the sulphur content of the ore-body as a whole might be considerably higher than that of the first shipment made additional tests particularly desirable.

This report, which covers one of several investigations designed to show the potential value of this ore to the iron and steel industry and to indicate the general procedure by which commercial products may be obtained, is concerned only with the preparation of suitable blast furnace feed from this ore by sintering, since the ore is too high in sulphur for use as mined.

Nature of Ore and Analysis of Shipment. Like the first shipment, this shipment is made up of a high-grade magnetite ore in which much of the gangue present exists in comparatively coarse particles readily visible to the naked eye. The sulphur content of this shipment however is considerably higher than that of the first lot, as shown by the following analysis made from a carefully prepared sample.

(Dried at 105° C.)

	Per cent		Per cent
Fe_3O_4	83·54	Fe	62·27
$CuFeS_2$	0·29	S	2·10
FeS_2	3·74	P	0·015
MnO_2	0·16	Mn	0·10
SiO_2	6·45	Cu	0·10
P_2O_5	0·034		
CaO	2·78		
MgO	0·96		
Al_2O_3	0·54		

Except for the high-sulphur content this would be an excellent ore for direct use in the blast furnace. The high-sulphur content, however, makes some form of desulphurization necessary before the ore can be considered satisfactory for blast furnace use.

Sintering Tests Carried Out

Since sintering affords a convenient and cheap method of desulphurization, a number of sintering tests were carried out on this ore. The tests carried out may be classified under two headings, viz., tests on the raw ore after crushing to suitable sizes and tests on the ore after dry magnetic concentration. The tests on the raw ore were carried out with both minus

[1] Mines Branch, Dept. of Mines, Canada. Invest. Ore Dress. & Met. 1931, pp. 156-158.
[2] *See* page 269 of this volume.

12- and minus 3-mesh material, but, as these tests indicated that there was no necessity for crushing finer than minus 3 mesh, tests on the concentrated material were carried out on the minus 3-mesh size only. All sintering was done on a 12- by 48-inch Dwight & Lloyd continuous sintering machine.

Results. The results obtained in these tests are tabulated in Tables I, II, III, and IV. These results show that for both the raw ore and the concentrate the best results are obtained from a mix containing about 2·5 per cent coke. They also show that a mix of these proportions yields a sinter of excellent chemical and physical characteristics, the grade of the sinter from the concentrate being naturally appreciably higher than that from the raw ore.

SUMMARY

Sintering tests were carried out on Texada Island magnetite of high-sulphur content. Tests on the raw ore showed that the sinter from material crushed to minus 3 mesh was just as satisfactory physically and chemically as that from minus 12-mesh material. They also indicated that a mix made up of about 94·0 per cent ore, 2·5 per cent coke, and 3·5 per cent moisture yielded excellent results.

The tests also showed that by crushing to minus 3 mesh a sufficient amount of gangue is liberated to warrant passing the crushed material over a magnetic separator before sintering since by so doing a sinter higher in iron by about 4·0 per cent and lower in insoluble by about 3·0 per cent is obtained.

TABLE I

Dry Magnetic Concentration of Minus 3-Mesh Ore

(1 Pass over Stearns Drum Type Separator)

	Per cent
Separator Feed:	
Weight	100·0
Iron	62·27
Sulphur	2·10
Insoluble	11·05
Concentrate:	
Weight	90·0
Iron	66·39
Sulphur	1·30
Insoluble	7·09
Tailing:	
Weight	10·0
Iron	25·00
Sulphur	9·35
Insoluble	46·70
Ratio of Concentration	1·11 : 1·00
Iron Recovery	95·9
Sulphur Rejected	44·5

TABLE II

Sintering Tests on Raw Ore Crushed to Minus 12-Mesh

Mix No.	Charge in pounds			Charge, per cent			Bed thickness	Weight of sinter produced	Analysis of sinter			Remarks
	Ore	Coke*	Water	Ore	Coke*	Water			Fe	S	Insol.	
							inches		%	%	%	
1.....	209·0	10·5	16·8	88·5	4·4	7·1	3¼	188	61·55	0·37	11·10	Too much fusion; sinter sticks to grates.
2.....	205·0	4·2	13·4	92·1	1·9	6·0	3¼	198	61·45	0·07	10·84	Good strong sinter; fine cellular structure.
3.....	192·0	6·0	12·6	91·2	2·8	6·0	3¼	189	61·65	0·07	10·32	Good strong sinter; slightly stronger than No. 2.

TABLE III

Sintering Tests on Raw Ore Crushed to Minus 3-Mesh

Mix No.	Charge, in pounds			Charge, per cent			Bed thickness	Weight of sinter produced	Analysis of sinter			Remarks
	Ore	Coke*	Water	Ore	Coke*	Water			Fe	S	Insol.	
							inches		%	%	%	
4.....	290·0	5·8	10·4	94·7	1·9	3·4	3¼	286	61·65	0·05	10·30	Well fused; strong sinter.
5.....	288·0	8·7	12·4	93·2	2·8	4·0	3¼	281	61·86	0·06	10·10	Good strong sinter; similar to No.4

TABLE IV

Sintering Tests on Minus 3-Mesh Magnetic Concentrate

Mix No.	Charge in pounds			Charge, per cent			Bed thickness	Weight of sinter produced	Analysis of sinter			Remarks
	Ore	Coke*	Water	Conct.	Coke*	Water			Fe	S	Insol.	
							inches		%	%	%	
6.....	190·0	7·6	8·2	92·3	3·7	4·0	3¼	185·0	65·88	0·11	7·04	Hard dense sinter; tendency to stick to grate.
7.....	19·0	5·7	7·3	94·1	2·3	3·6	3¼	185·0	66·09	0·06	6·84	Very hard dense sinter; did not stick to grate.

*20-mesh coke.

VI

REPORT OF THE CHEMICAL LABORATORY SECTION

H. C. Mabee
Chief Chemist

The activities of the Chemical Laboratories during the year, although somewhat restricted on account of the need of the greatest economy, show that excellent progress has been made and that the usual efficiency and co-operation of the staff have been maintained in the performance of the chemical duties in connexion with the investigative work undertaken in all the various branches of the division.

The list that follows shows that 3,329 samples were received for analysis, and reports including well over 10,000 chemical determinations have been issued.

The list, as previously stated, represents the samples originating from experimental work in the various laboratory divisions, including samples of the original material as received, together with the concentration of metallic products, and are arranged in the following list on the basis of the essential mineral or metal classification and not on the entire chemical character of the material. In many cases ores and products of a highly complex nature require to be given a complete analysis.

	Number of samples		Number of samples
Aluminium alloys	7	Gypsum	54
Anhydrite	3	Iron	435
Barite	59	Titaniferous iron	5
Bleach solutions	10	Metallic iron and steel	109
Chromite	20	Manganese	3
Copper	8	Mercury	1
Copper-iron	31	Mica	1
Clay	6	Nickel-copper	263
Cyanite	20	Phosphate	103
Coal	34	Pitchblende	134
Cobalt	10	Plaster	1
Cobalt-copper	16	Sand and sandstone	91
Garnet	1	Salt	1
Gold	1,235	Talc	103
Gold-copper	158	Tungsten	36
Gold-platinum	28	Ore for classification and valuation	34
Gold-silver	..	Zinc-lead	6

The continued interest in gold mining operations was again reflected throughout the year in the new high record as shown in the number of samples on which the gold or precious metal values were required.

Special mention may be made also of chemical investigational work carried on in connexion with the various classes of pitchblende, silver-bearing ores, samples which were received from Great Bear Lake district during the early part of the year. In addition to the complete examina-

tion of the sample of the original ore, the staff also examined and reported on the numerous residues and products of concentration resulting from the experimental treatment of these ores in determining a suitable method for the recovery of their valuable constituents.

The regular analytical work was again arranged in sections as in previous years. B. P. Coyne, H. L. Beer, and R. W. Cornish conducted the work on metallic ores; A. E. La Rochelle, on ferrous products; L. Lutes, J. S. McCree, assisted by J. Cullen, performed the fire assay work; and R. A. Rogers was again placed at the disposal of the non-metallic section.

In addition to his general supervisory duties the Chief Chemist performed the planning of the equipment and the direction of the chemical laboratory procedure as well as the preparation of reports, etc.

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