L.P. Collin

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)

July to December, 1933

OTTAWA

J. O. PATENAUDE

PRINTER TO THE KING'S MOST EXCELLENT MAJESTY

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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). Semi-annually.

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, JULY TO DECEMBER, 1933

I

REVIEW OF INVESTIGATIONS

W. B. Timm
Chief of Division

During the half-year ending December 31st, 1933, as shown by the foregoing list of investigations, forty reports were issued, of which thirty-three were on the treatment of metallic ores, five on non-metallic minerals, one on ferrous metallurgy, and one on the hazards and precautions in the treatment of radium-bearing ores. Of the investigations on metallic ores five were on ores from the province of British Columbia, three on ores from the province of Manitoba, seventeen on ores from the province of Ontario, six on ores from the province of Quebec, and two on ores from the province of Nova Scotia. Of the investigations on non-metallic minerals, one was on diatomite from the province of British Columbia, one on barite from the province of Ontario, two on garnet from the province of Quebec, and one on the utilization of broken glass from a glass manufacturing plant in the province of Quebec.

Of the investigations on metallic ores from British Columbia, three were on gold ores, one on a gold-silver ore containing antimony and arsenic, and one on the magnetic iron ore from Texada island. From Manitoba, three gold ores were investigated. On Ontario ores the investigations included thirteen gold ores, one silver-gold ore, one copper-gold ore, one copper-nickel ore, and one lead ore. On Quebec ores the investigations included five gold orès and one copper-gold ore; and two gold ores from Nova Scotia were investigated. Twenty-nine of the investigations were on ores in which gold was the mineral of chief economic importance.

The investigations were conducted on ores from many new mining properties under development in consequence of the enhanced price of gold. A number of these properties had reached the stage of development at which milling plants were justified in the opinion of the operators. On some, milling plants have been built and put into operation; on others plants are in course of erection; and on the remainder milling plants are being planned or are under consideration.

The knowledge gained from the experience in conducting the test work on the ores was given to company engineers and consulting engineers engaged on the design and construction of the milling plants and concentrators. The facilities of the laboratories were used to a greater extent than hitherto by consulting engineers and metallurgists representing the mining companies, and the co-operation of the staff was extended to them in the investigation of their ores and ore treatment problems. As it is impossible for the present staff to undertake all the investigative work requested, the use of the laboratory facilities by the mining companies is welcomed.

The investigations in ore dressing and milling of metallic ores on those listed, together with a number not included but which took up considerable time of the staff, were carried out by C. S. Parsons, A. K. Anderson, J. D. Johnston, W. R. McClelland, and W. S. Jenkins.

The investigations in the dressing of non-metallic minerals were carried out by R. K. Carnochan and R. A. Rogers. In addition to those listed, a general investigation into the use of Canadian sandstones and silica sands for sandblasting is being conducted. A number of minor tests were conducted, a quantity of sandstone prepared for the use of the investigators at the Central Experimental Farm, Ottawa, and a quantity of clay samples prepared for the Ottawa Public School Board.

The investigations in the metallurgy of iron and steel were carried out by T. W. Hardy and H. H. Bleakney. Investigative work was conducted for the primary iron and steel producers, for the two large Canadian railway companies, for the steel companies, foundries, and engineering firms, and for the Department of National Defence. The work embodied metallurgical studies and examination of various steels produced or used by the above companies in order to determine their quality, in an effort to improvement. Much of the work conducted for the Department of National Defence was on parts that failed in service and in which the cause of the failure was determined. The reports on these investigations were either of a confidential nature or of particular interest and value only to the parties directly concerned, and so are not published in this report of investigations. The investigation into the use of sponge iron as a base, compared with steel scrap and muck bar and in the production of high quality steels is being continued. A considerable quantity of sponge iron in the form of briquettes was made from magnetic iron concentrate from Texada Island ore for this purpose.

R. J. Traill continued the investigation of the radium-bearing ores of the Great Bear Lake district, Northwest Territories, dealing mainly with problems of plant practice confronting the operators at the treatment plant and refinery at Port Hope, Ontario. In the radium-measuring laboratory, sixty-seven radium measurements were carried out during the year by W. R. McClelland on radioactive ores, concentrates, solutions, and residues. Ninety-nine measurements were also made by the alpha electroscope on test products, and nineteen mineral specimens from various parts of Canada were examined for radioactivity. He also prepared for publication, Investigation No. 550, "Health Hazards in the Production and Handling of Radium—Precautions taken during a laboratory investigation of the treatment of Radium-bearing Ores," which is included in this report of investigations.

In the mineragraphic laboratory during the year seven hundred and sixteen polished sections of ores and mill products, thirty thin sections of non-metallic minerals were prepared and examined and two hundred and twenty-five spectrographic analyses made. M. H. Haycock issued eighty-six reports, giving the results of the microscopic examination and spectrographic analyses of the ores and mill products. These reports were sent separately to those submitting ores for investigation, or the essential features were embodied in the report of investigations.

In the Chemical Laboratories of the Division, H. C. Mabee, Chief Chemist, with a staff of chemists and assayers, made over 11,000 chemical determinations from 3,699 samples received in connection with the investigations undertaken during the year. This was an increase over the previous year of 370 samples and almost three and a half times the number received by the Chemical Laboratories in the year 1925. Of the total number of samples, 70 per cent were of precious metal ores and their test products, 20 per cent of base metal ores and their test products, and 10 per cent in connection with the investigative work on non-metallic minerals.

II

REPORTS OF INVESTIGATIONS

Ore Dressing and Metallurgical Investigation No. 511

GOLD ORE FROM THE CANADIAN PANDORA GOLD MINES, LIMITED, CADILLAC TOWNSHIP, QUEBEC

Shipment. A shipment of four lots of gold ore, total weight 3,500 pounds was received by freight on March 28, 1933. These were marked Lots Nos. 1, 2, 3, and 4.

Tests were desired so as to determine the value of the four samples and also to determine the most suitable process to apply for the recovery of the gold.

After crushing, cutting, and grinding by standard methods, a separate sample of each lot was obtained which showed the lots to contain:—

Lot No.	Gold, oz./ton	Arsenic, per cent
1	0·46	0·08
2	0·02	1·17
3	0·28	0·11
4	0·62	0·33

As Lot No. 2 contained only 0·02 ounce per ton in gold, no tests were made on it. A series of small-scale tests was made on Lot No. 1, followed by mill runs with a feed of 100 pounds per hour on Lots Nos. 1, 3, and 4.

Experimental Tests

The test work included cyanidation, amalgamation, blanket concentration and flotation, singly and in combination.

The results show that over 98 per cent of the gold can be extracted by cyanidation; 96 per cent by traps, blanket concentration, and flotation; 90 to 95 per cent of the gold in the concentrate can be barrel-amalgamated and the residue further reduced by cyanidation.

Lot No. 1

Test No. 1

A sample of the ore ground minus 65 mesh with 63·7 per cent through 200 mesh was amalgamated and the tailing cyanided.

Results:

Feed	0·46 oz. Au/ton
Amalgamation tailing	0·16 "
Recovery by amalgamation	5·2 per cent.
24-hour cyanide tailing	0·02 oz. Au/ton
48-hour cyanide tailing	0·01 "
Cyanide consumed	0·09 lb./ton ore
Total recovery, amalgamation and cyanidation	97·8 per cent.

Test No. 2

A sample ground minus 100 mesh with 93·5 per cent through 200 mesh was treated similarly.

Results:

Feed	0·46 oz. Au/ton
Amalgamation tailing	0·125 "
Recovery by amalgamation	72·8 per cent.
24-hour cyanide tailing	0·015 oz. Au/ton
48-hour cyanide tailing	0·005 "
Cyanide consumed	0·9 lb./ton ore
Lime	2·9 "
Total recovery	98·9 per cent.

CYANIDATION

Test No. 3

Samples of the ore were ground to pass 48, 100, 150, and 200 mesh respectively, and cyanided separately for 48 hours with a 1·0 pound KCN solution, 1 : 3 dilution; 4 pounds of lime per ton was added for protective alkalinity.

Mesh grind	Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
					KCN	CaO
48	24	0·46	0·02	95·7	0·06	2·3
48	48	0·46	0·01	97·8	0·05	2·3
100	24	0·46	0·01	97·8	0·09	2·8
100	48	0·46	0·005	98·9	0·06	2·8
150	24	0·46	0·015	96·7	0·09	3·1
150	48	0·46	0·005	98·9	1·05	3·1
200	24	0·46	0·015	96·7	1·2	3·1
200	48	0·46	0·005	98·9	1·2	3·4

Mill Runs

Test No. 4

The remainder of Lot No. 1 was fed at the rate of 100 pounds per hour to a rod mill that discharged into a trap before entering a classifier in closed circuit with the mill. The classifier overflow, 27 to 30 per cent solids, passed over corduroy blankets having an area of 15 square feet per ton-hour, set at a slope of $2\frac{1}{2}$ inches in 12 inches. The blanket tailing then passed to a conditioning tank where the pulp was conditioned for 15 minutes with 1 pound soda ash and 0·10 pound sodium ethyl xanthate per ton. The conditioning tank overflowed to the first cell of a 6-cell, mechanically agitated flotation machine where a concentrate was removed.

The froth from the last five cells was returned to the head of the machine where 0·08 pound pine oil was added to produce a froth. Throughout the run samples were taken of the feed, mill discharge, classifier overflow, blanket tailing, flotation concentrate and tailing. At the conclusion of the run the blanket concentrate was sampled and assayed.

Assays:

	Gold, oz./ton
Feed	0·46
Mill discharge	0·34
Classifier overflow	0·095
Blanket concentrate	6·24
Blanket tailing	0·05
Flotation concentrate	8·28
Flotation tailing	0·015

Results:

Retained in grinding mill	26·1 per cent.
Retained in trap and classifier	53·3 "
Recovered on blankets	9·8 "
Recovered by flotation	7·5 "
Total recovery	96·7 "
Ratio of concentration by blankets	137·5 : 1
Ratio of concentration by flotation	236 : 1

The trap clean-up was reground and amalgamated. This had a value of 14·53 ounces gold per ton and was reduced to 0·35 ounce, a recovery of 97·6 per cent of the gold.

The blanket concentrate was reground and amalgamated reducing the gold content to 0·33 ounce per ton, a recovery of 94·7 per cent.

Cyanide tests on the blanket tailing reduced its gold content to 0·01 ounce in 24 hours and to 0·005 ounce in 48 hours, a recovery of 92·3 per cent of the gold in this product.

Regrinding and cyaniding the flotation concentrate left a residue of 0·70 ounce gold per ton, an extraction of 90·3 per cent.

Conclusions:

Recovery by blanket concentration, amalgamation, and cyanidation of blanket tailing.

It is assumed that the gold retained in the mill, trap, and classifier responds to amalgamation to the same degree as the trap clean-up.

Amalgam from mill, trap and classifier: 79·4 × 0·976	77·5 per cent.
Amalgam from blanket concentrate: 9·8 × 0·947	9·3 "
Blanket tailing cyanided: 10·8 × 0·923	10·0 "
Total recovery	96·8 "

A slight increase in overall recovery will be effected by cyaniding the residue from the reground trap clean-up and blanket concentrate after amalgamating.

Recoveries by blanket concentration, amalgamation, flotation, and cyanidation of flotation concentrate.

Amalgam from mill, trap and classifier: 79·4 × 0·976	77·5 per cent.
Amalgam from blanket concentrate: 9·8 × 0·947	9·3 "
Flotation concentrate cyanided: 7·5 × 0·903	6·8 "
Total recovery	93·6 "

Screen Analysis of Classifier Overflow:

Mesh	Weight, per cent.
+ 65	0·6
− 65 + 100	4·5
−100 + 150	12·3
−150 + 200	12·8
−200	69·8

Lot No. 3

This lot was milled at the rate of 100 pounds per hour under the same conditions as Lot No. 1.

Assays:

	Au, oz./ton
Feed	0·27
Mill discharge	0·16
Classifier overflow	0·07
Blanket concentrate	3·62
Blanket tailing	0·055
Flotation concentrate	1·25
Flotation tailing	0·01

Results:

Retained in grinding mill	40·7	per cent
Retained in trap and classifier	33·4	"
Recovered on blankets	5·6	"
Recovered by flotation	16·6	"
Total recovery	96·3	"
Ratio of concentration by blankets	237·7 : 1	
Ratio of concentration by flotation	27·6 : 1	

The trap clean-up was reground and amalgamated. This had a gold content of 15·06 ounces per ton and was reduced to 0·465 ounce, a recovery of 96·9 per cent.

The blanket concentrate was reground and amalgamated, reducing the gold content from 3·62 ounces to 0·965 ounce, a recovery of 73·3 per cent.

Cyanidation of the blanket tailing reduced the gold content from 0·055 ounce to 0·005 ounce in 48 hours, a recovery of 91·6 per cent.

Cyaniding the reground flotation concentrate for 24 hours reduced the gold content to 0·07 ounce, an extraction of 95·0 per cent.

Conclusions:

Recovery by blanket concentration, amalgamation and cyanidation of blanket tailing.

Amalgam from mill, trap and classifier: 74·1 × 0·969	71·8	per cent
Amalgam from blanket concentrate: 5·6 × 0·733	4·1	"
Blanket tailing cyanided: 20·3 × 0·916	18·6	"
Total recovery	94·5	"

Recovery by blanket concentration, amalgamation, flotation, and cyanidation of flotation concentrate.

Amalgam from mill, trap and classifier: 74·1 × 0·969	71·8	per cent
Amalgam from blanket concentrate: 5·6 × 0·733	4·1	"
Flotation concentrate cyanided: 16·6 × 0·95	15·8	"
Total recovery	91·7	"

Lot No. 4

This lot was fed at the rate of 100 pounds per hour to the rod mill, grinding minus 48 mesh with 64·7 per cent through 200. The mill discharged over an amalgamating plate into the conditioner where the same reagents were added as in the runs on Lots Nos. 1 and 3. The amalgamation tailing was then floated.

Cyanide tests were made on the amalgamation tailing and flotation concentrate.

Assays:

	Au, oz./ton
Feed	0·65
Mill discharge	0·49
Plate tailing	0·215
Flotation concentrate	4·00
Flotation tailing	0·025

Results:

Retained in mill	24·6 per cent
Recovered on amalgam plates	42·2 "
Flotation concentrate	29·4 "
Total recovery	96·2 "

The amalgamation tailing was cyanided, 1 : 3 dilution, with a 1·0 pound KCN solution for 48 hours. At 24 hours, the tailing contained 0·015 ounce and at 48 hours, 0·01 ounce gold per ton with a cyanide consumption of 0·3 pound KCN per ton ore. An extraction of 95·3 per cent was obtained.

The flotation concentrate was reduced to 0·07 ounce per ton after cyaniding 24 hours with a 5·0 pound KCN solution, an extraction of 97·6 per cent.

CONCLUSIONS

Assuming that the gold retained in the mill would be recovered by amalgamation, the following recovery can be expected:—

By amalgamation	66·8 per cent
By cyanidation of concentrate	28·7 "
Total recovery	95·5 "

In order to cyanide the amalgamation tailing it would be necessary to dewater the product before cyaniding, otherwise large losses of cyanide solution would be necessary.

It is apparent that the ore is easily cyanided. Tests on Lot No. 1 show that 95·7 per cent is extracted from minus 48-mesh material within 24 hours and 98·9 per cent within 48 hours from minus 100 mesh.

Cyanide tests on blanket and amalgamation tailings show a high extraction, a tailing as low as 0·01 to 0·005 ounce per ton being obtained. Approximately 95 per cent of the gold in the flotation concentrates secured in the runs is extracted by cyanide.

Much of the gold can be trapped throughout the circuit as shown in the mill runs.

Amalgamation should recover between 60 and 70 per cent of the gold provided a supply of clear water free from organic matter and mineral acids is obtainable. Brown muskeg water is not suitable as the included gases stain the mercury. Amalgamation, however, leaves a residue too high in gold to discard, thus making additional equipment and treatment necessary.

The process recommended is straight cyanidation of material ground to approximately minus 65 mesh. Traps should be placed between the mill and classifier to remove coarse gold. These traps should be cleaned periodically, and the contained gold recovered by barrel amalgamation. The residue after amalgamation should be added to the classifier overflow for cyanidation.

Ore Dressing and Metallurgical Investigation No. 512

GOLD ORE FROM BUFFALO ANKERITE GOLD MINES, LIMITED, SOUTH PORCUPINE, ONTARIO

Shipment. A shipment of ore, net weight 890 pounds, was received May 8, 1933, from Martin O. Knutson, Manager, Buffalo Ankerite Gold Mines, Limited, Box 533, South Porcupine, Ont.

Characteristics of the Ore. Under the microscope the gangue was seen to consist chiefly of dark greenish grey or black mottled rock containing a large amount of chloritic material with some quartz and carbonate. A distinct parallel schistose structure is apparent in every section, and the carbonate forms fine stringers parallel to this.

The sulphides are disseminated throughout the gangue. Pyrite is the most abundant sulphide, and forms large irregular grains; in some cases a distinct cubic outline is visible. This mineral contains tiny grains, or inclusions, of gangue, chalcopyrite, and pyrrhotite.

Chalcopyrite occurs within the pyrite as noted above, and as irregular grains in the gangue. The mineral is not abundant.

Pyrrhotite was observed only within the pyrite, and is rare.

One tiny grain of native gold was observed within the pyrite, associated with a grain of pyrrhotite.

A hard grey anisotropic mineral negative to all reagents is rather prominent locally. It was not determined, but is possibly ilmenite.

The assay of the ore was as follows:—

Gold	0·215 oz./ton
Silver	0·05 "

Experimental Tests

The ore was being treated at the mine by the cyanide process and as good extraction was not obtained tests were required to find out the cause of the poor recovery and by what means it could be overcome.

A series of small-scale tests was made with the ore ground dry to various sizes. Good extraction was obtained in 24 hours with all the ore ground through 150 mesh or finer. A cycle test was made and little or no sign of fouling was noticed after six cycles. By giving the ore a fairly coarse grind, tabling out the pyrite and regrinding it separately, 96·0 per cent extraction was obtained and considerable grinding saved. By giving the ore the same grind and cyaniding it, then tabling out the pyrite and assaying it separately it is found that 72·1 per cent of the gold in the tailing is contained in the pyrite concentrate, which assays 0·14 ounce per ton in gold. In this test extraction fell off to 86·0 per cent with a calculated average tailing of 0·03 ounce per ton in gold. Details of the tests follow.

CYANIDATION

Tests Nos. 1 to 8

Four lots of the ore were ground dry to pass through 48, 100, 150, and 200 mesh respectively. Samples of each of the above lots were agitated in cyanide solution, 2 pounds KCN per ton, for 24 and 48 hours. The tailings were filtered, washed, and assayed for gold.

Results:

Feed sample, Au 0·215 oz./ton

Test No.	Agitation period, hours	Tailing assay, Au oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
1	24	0·04	81·4	0·6	4·0
2	24	0·025	88·4	0·9	5·0
3	24	0·02	90·7	0·9	8·5
4	24	0·015	93·0	0·9	8·2
5	48	0·035	83·7	0·6	4·35
6	48	0·02	90·7	0·9	5·10
7	48	0·015	93·0	0·9	8·5
8	48	0·010	95·3	0·9	8·2

CYCLE TEST

Tests Nos. 13 to 18

This series of tests was made to determine whether the solution became foul and lost its dissolving power after a number of contacts with ore. A sample of ore ground dry through 150 mesh was used for this purpose. At the end of each test the solution was filtered off, made up to proper strength and volume and used to treat a fresh batch of ore. Solutions were kept at 2 pounds per ton in KCN and protective alkalinity was maintained by the addition of lime. The period of agitation in each case was 24 hours and pulp dilution was 3 : 1.

Results:

Feed sample, Au 0·215 oz./ton

Test No	Tailing assay, Au oz./ton	Extraction, per cent
13	0·01	95·3
14	0·015	93·0
15	0·02	90·7
16	0·02	90·7
17	0·02	90·7
18	0·02	90·7

Recoveries in Tests Nos. 13 to 18 are similar to that in Test No. 3, but better extraction is obtained in Tests Nos. 13 and 14. This is probably due to a slight irregularity in the head sample used.

Test No. 19

In this test 4,000 grammes of the ore was ground in a ball mill for 15 minutes and then passed over a laboratory concentrating table and a pyrite concentrate was taken off. The table tailing and concentrate were sampled and assayed. Portions of each were then cyanided, the tailing as it came from the table and the pyrite concentrate after being reground for 20 minutes.

Results:

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Table concentrate	14·25	0·88	61·9
Table tailing	85·75	0·09	38·1
Feed (cal.)	100·00	0·203	100·0
Table concentrate cyanided	14·25	0·03	2·1
Table tailing cyanided	85·75	0·005	2·1
Average tailing (cal.)	100·00	0·0086	4·2

Extraction by cyanidation after regrinding pyrite, 100—4·2=95·8 per cent.

Test No. 20

This test differs from the preceding one in that the ore was ground for 15 minutes in a ball mill and then cyanided without further treatment. The cyanidation tailing was then tabled and a pyrite concentrate taken off. The table concentrate and tailing were assayed separately and an average tailing calculated from the two assays.

Results:

Feed sample, Au 0·215 oz./ton

Product	Weight, per cent	Assay, Au oz./ton	Per cent of gold in feed sample
Table concentrate from cyanide tailing	15·6	0·14	10·1
Table tailing from cyanide tailing	84·4	0·01	3·9
Average cyanide tailing	100·0	0·03	14·0

Extraction by cyanidation when pyrite is not reground, 100—14·0=86·0 per cent.

CONCLUSIONS

The results of the tests indicate two methods whereby cyanide tailing lower in grade could be obtained from the ore. The first is to grind all or nearly all the ore through 200 mesh and then cyanide it. The second is to grind all the ore approximately 70 per cent through 200 mesh, table out the pyrite and regrind it to virtually all through 200 mesh, after which it can be re-united with the table tailing in the cyanide plant. As the pyrite represents only about 15 per cent of the weight of the ore it is obvious that considerable grinding will be saved by tabling it out and grinding it separately. Test No. 19 shows that approximately 62 per cent of the gold is associated with the pyrite and a comparison of the results of Tests Nos. 19 and 20 shows the advantage to be gained by regrinding the pyrite. It should, therefore, be good milling practice to include this operation in the flow-sheet.

Ore Dressing and Metallurgical Investigation No. 513

THE CANADIAN MATACHEWAN GOLD MINES, LIMITED, MATACHEWAN, ONTARIO

Shipment. A shipment of 6,475 pounds of ore, consisting of two samples, Lot No. 1 and Lot No. 2, was received on June 2, 1933, from Ventures Limited, Toronto, Canada.

Sampling and Analysis. The whole shipment was crushed to 14 mesh and sampled.

Lot No. 1 assayed	0·205 oz. Au/ton
Lot No. 2 assayed	0·23 "

Character of the Ore. Lot No. 1 consisted of clean ore. Lot No. 2 consisted of ore which showed considerable surface weathering.

The two samples showed great similarity. They consisted of pyrite, magnetite, and native gold disseminated in siliceous gangue of very fine texture, which contains much finely disseminated carbonate and small lenticular stringers of quartz.

The native gold in the sections examined under the microscope was observed as very fine grains in the gangue. Five grains only were observed.

Gold grains	Diameter
2	2 microns
1	6 "
1	8 "
1	10 "

However, a classification test on 1,000 grammes of 14-mesh material disclosed one grain of gold as large as 35 mesh and much fine gold visible as colours in the pan.

A microscopic study was made of the grain size of the pyrite in the ore, and the following table gives an average for both Lot No. 1 and Lot No. 2, six sections being counted.

Mesh	Per cent by volume	Cumulative per cent
− 20+ 28	2·1	
− 28+ 35	5·15	7·25
− 35+ 48	11·5	17·75
− 48+ 65	12·3	31·02
− 65+100	13·35	44·4
−100+150	15·95	60·35
−150+200	14·05	74·4
−200+325	13·0	87·4
−325	12·6	100·0

For the purpose of comparison the following table of the grain sizes of the Young-Davidson Matachewan ore is given.

Mesh	Per cent pyrite	Cumulative per cent
+ 20	7·4	7·4
+ 28	15·6	23·0
+ 35	8·9	31·9
+ 48	21·0	52·9
+ 65	13·4	66·3
+100	13·8	80·1
+150	7·7	87·8
+200	4·9	92·7
−200	7·3	100·0

It will be observed that the pyrite is coarse in the Young-Davidson, and in this connection 92·7 per cent is plus 200 mesh, whereas in the Canadian Matachewan ore only 74·35 per cent is plus 200 mesh. With the Young-Davidson an average of 94·5 per cent recovery was obtained in traps and by flotation when grinding 6 per cent +48 mesh with 40 per cent −200 mesh.

These figures can be used as a basis for comparing the results obtained on samples of this ore ground to various sizes.

Experimental Tests

1. A series of small batch flotation tests was run on samples of each lot ground to various sizes.
2. A series of amalgamation tests was run on samples ground as in the first series.
3. A series of cyanide tests was made on the Lot No. 1 only.
4. A series of continuous flotation tests was made on the bulk sample of Lot No. 1 and Lot No. 2 using a trap to catch the coarse, free gold before flotation.

Lot No. 1

FLOTATION TESTS

Test No. 1

A sample of 1,000 grammes of ore at −14 mesh was ground 20 minutes in a batch ball mill with water to the following sizes:—

Mesh	Weight, per cent
− 48+ 65	0·1
− 65+100	1·1
−100+150	6·5
−150+200	6·3
−200	86·0
Total	100·0

Reagents:

To ball mill:	Soda ash	2·00 lb./ton
	Barrett No. 4	0·12 "
To cells:	Pine oil	0·04 lb./ton
	Z−5 P.A. Xanthate	0·20 "

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Flotation concentrate	10·7	1·52	90·0
Flotation tailing	89·3	0·02	10·0
Total	100·0	0·18	100·0

An amalgamation test was made on the flotation tailing for the purpose of determining the amount of free gold.

Product	Weight, per cent	Assay, Au oz./ton	Recovery of gold, per cent
Feed	89·3	0·02	Nil
Amalgamation tailing	89·3	0·02	

Test No. 2

A sample of 1,000 grammes was ground 15 minutes in a batch ball mill to the following sizes.

Mesh	Weight, per cent
+ 48	0·1
− 48+ 65	0·8
− 65+100	3·8
−100+150	16·5
−150+200	8·0
−200	70·8
Total	100·0

The reagents used were as in Test No. 1.

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Flotation concentrate	13·0	1·42	89·4
Flotation tailing	87·0	0·025	10·6
Total	100·0	0·21	100·0

The flotation tailing was amalgamated to determine the amount of free gold.

Product	Weight, per cent	Assay, Au oz./ton	Recovery of gold, per cent
Feed	87·0	0·025	22·7
Amalgamation tailing	82·0	0·02	

The overall recovery is 91·8 per cent.

Test No. 3

A sample of 1,000 grammes of −14-mesh ore was ground 10 minutes in the batch ball mill.

Mesh	Weight, per cent
+48	0·7
−48+65	3·7
−65+100	7·4
−100+150	13·4
−150+200	8·7
−200	66·1
Total	100·0

The reagents were similar to the other tests.

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Flotation concentrate	12·1	1·34	84·0
Flotation tailing	87·9	0·035	16·0
Total	100·0	0·19	100·0

The flotation tailing was amalgamated to determine the amount of free gold present.

Product	Weight, per cent	Assay, Au oz./ton	Recovery of gold, per cent
Feed	87·9	0·035	58·1
Amalgamation tailing	87·9	0·015	

The overall recovery is 93·3 per cent.

Lot No. 2

FLOTATION

A similar series of tests was run on Lot No. 2, which are summarized as follows.

Grinding, 20 minutes:

Mesh	Weight, per cent
−48+65	
−65+100	1·0
−100+150	5·0
−150+200	5·4
−200	88·6
Total	100·0

Grinding, 15 minutes:

Mesh	Weight, per cent
+65	0·9
−65+100	3·0
−100+150	9·3
−150+200	7·5
−200	79·3
Total	100·0

Grinding, 10 minutes:

Mesh	Weight, per cent
+48	0·6
− 48+65	2·7
− 65+100	6·8
−100+150	12·1
−150+200	7·6
−200	70·2
Total	100·0

Test No.	Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
1	Flotation concentrate	9·4	2·15	88·2
	Flotation tailing	90·6	0·03	11·8
	Total	100·0	0·23	100·0
	Amalgamation tailing	90·6	0·02	33·3
	Overall recovery	92·2		
2	Flotation concentrate	11·8	1·94	94·7
	Flotation tailing	88·2	0·015	5·3
	Total	100·0	0·24	100·0
	Amalgamation tailing	88·2	0·01	38·4
	Overall recovery	96·7		
3	Flotation concentrate	10·5	1·90	88·2
	Flotation tailing	89·5	0·03	11·8
	Total	100·0	0·23	100·0
	Amalgamation tailing	89·5	0·015	50·0
	Overall recovery	94·1		

Composite Sample

AMALGAMATION

A series of amalgamation tests was made, grinding as follows: Test No. 1, 20 minutes; Test No. 2, 15 minutes; and Test No. 3, 10 minutes.

Results:

Lot No. 1—

Test No. 1, gold amalgamated	56·1 per cent
Test No. 2, gold amalgamated	53·7 "
Test No. 3, gold amalgamated	45·3 "

Lot No. 2—

Test No. 1, gold amalgamated	58·6 per cent
Test No. 2, gold amalgamated	58·6 "
Test No. 3, gold amalgamated	54·4 "

CYANIDATION

Tests Nos. 1A and 1B

Cyanide tests were made on Lot No. 1 only. In Tests Nos. 1A and 1B the ore was ground to 50 per cent −200 mesh and cyanided with 2·6 pounds KCN per ton of solution in a 2·5 : 1 pulp for 24 hours, using the Hollinger practice of tabling the cyanide tailing and re-cyaniding the reground table concentrate.

The reagent consumption for the first step in the operation was as follows:—

KCN per ton of ore	0·4 lb.
CaO per ton of ore	4·0 "

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Feed	100·0	0·205	100·0
Cyanide tailing, 24 hour	100·0	0·025	12·2
Recovery by cyanide	87·8		
Table feed	100·0	0·025	100·0
Table concentrate	9·4	0·19	56·7
Table tailing	90·6	0·015	43·3
Total	100·0	0·03	100·0

Ratio of concentration, 10·65 : 1.

The table concentrate was reground to 85 per cent −200 mesh and cyanided for 24 hours in a 3 : 1 pulp with 2 pounds KCN per ton of solution.

KCN consumption per ton of concentrate	1·05 lb.
CaO consumption per ton of concentrate	3·5 "

Products	Weight, per cent	Assay, Au oz./ton	Recovery of gold, per cent
Feed	100·0	0·19	58
Cyanide tailing, 24 hour	100·0	0·08	

Recapitulation of Results

Total cyanide consumption per ton of ore	0·5 lb.
Total lime consumption per ton of ore	4·3 "
Total recovery: 87·8 + 4·0	91·8 per cent

Test No. 2

A batch of 1,000 grammes of −14-mesh ore was ground wet 10 minutes in a ball mill.

Mesh	Weight, per cent
+ 48	0·7
− 48+ 65	3·7
− 65+100	7·4
−100+150	13·4
−150+200	8·7
−200	66·1
Total	100·0

The ore was cyanided for 36 hours in a 2·5 : 1 pulp with 2·55 pounds KCN per ton of solution.

KCN consumption per ton of ore.. 0·5 lb.
CaO consumption per ton of ore.. 5·0 "

Products	Weight, per cent	Assay, Au oz./ton	Recovery of gold, per cent
Cyanide tailing, 36 hours....................................	100	0·02	90·25

The above cyanide tailing was tabled.

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Table concentrate....................................	15·7	0·13	83·0
Table tailing....................................	84·3	0·005	17·0
Total....................................	100·0	0·024	100·0

Using the figure of 58 per cent obtained from the previous test as the amount of gold which can be recovered by cyanidation of the table concentrate, then the results for overall recovery are 90·25 per cent + 4·7 per cent, or 94·95 per cent.

Tests Nos. 3A and 3B

The grinding was the same as used in Test No. 2. Test No. 3A was agitated 24 hours and Test No. 3B, 48 hours in a 2·5 : 1 pulp with 2·5 pounds KCN per ton of solution.

Reagent Consumption—

Test No. 3A—24 hours.

KCN per ton of ore.. 0·4 lb.
CaO per ton of ore.. 4·25 "

Assay of tailing.. 0·02 oz./ton.
Recovery.. 90·25 per cent.

Test No. 3B—48 hours

KCN per ton of ore.. 0·4 lb.
CaO per ton of ore.. 4·75 "

Assay of tailing.. 0·015 oz./ton
Recovery.. 92·75 per cent

Mill Tests

The flow-sheet used was as follows: The ore crushed to −14 mesh was fed to a 6-foot by 24-inch ball mill, which discharged into a hydraulic trap to catch free coarse gold. The overflow of the trap was elevated in a bucket elevator to a Dorr classifier in closed circuit with the ball mill. The overflow of the Dorr went to a conditioning tank where the flotation reagents were added. The flotation was done in a 10-cell mechanical flotation unit.

Cells Nos. 2, 3, and 4 made a concentrate, which was cleaned in cell No. 1 to give a final concentrate. The cells Nos. 5 to 10 were used as roughers which returned a rougher concentrate back to join the feed in cell No. 2.

Run No. 1

Feed rate, 400 pounds per hour.

Reagents used were approximately as follows:—

Soda ash	2·00 lb./ton
Coal-tar creosote	0·08 "
Pot. ethyl xanthate	0·15 "
Pine oil	0·02 "

Screen Test on Flotation Feed

Mesh	Weight, per cent
+ 48	2·8
− 48+ 65	3·4
− 65+100	7·2
−100+150	8·2
−150+200	6·8
−200	71·6
Total	100·0

Results:

Feed to mill, assay	0·205	oz. Au/ton
Classifier overflow assay (feed to flotation)	0·110	"
Recovery in trap and elevator	53·6	per cent of gold
Flotation concentrate	2·74	oz. Au/ton
Flotation tailing	0·02	"
Recovery by flotation	82·4	per cent.

Ratio of concentration, 30·2: 1.
Overall recovery by traps and flotation: 92·0 per cent.

Run No. 2

Feed rate, 400 pounds per hour.

Reagents:

Soda ash	1·3 lb./ton
Barrett No. 4	0·2 "
Amyl xanthate	0·15 "
Pine oil	0·05 "

Screen Test on Flotation Feed:

Mesh	Weight, per cent
+ 48	0·7
− 48+ 65	3·0
− 65+100	8·3
−100+150	11·8
−150+200	9·4
−200	66·8
Total	100·0

Results:

Feed to mill	0·22	oz. Au/ton
Classifier overflow (feed to flotation)	0·13	"
Recovery in trap and elevator	40·9	per cent
Flotation concentrate	1·68	oz. Au/ton
Flotation tailing	0·015	"
Recovery by flotation	89·2	per cent

Ratio of concentration, 14·5 : 1.
Overall recovery by traps and flotation: 93·6 per cent.

Recapitulation:

Product	Weight, lb.	Assay, oz./ton	Units	Per cent of total
Ore feed to mill	6,000·0	0·21	1,260·0	100·0
Flotation concentrate	255·0	2·00	510·0	40·5
Trap clean-up	65·0	1·77	115·1	9·1
Elevator boot clean-up	11·3	17·54	198·2	15·7
Classifier clean-up	393·0	0·53	208·3	16·5
Ball mill and conditioning tank	113·0	1·13	127·6	10·1
Total	....	....	1,159·2	91·9

A study of the above results will show that of the total gold trapped prior to flotation only 17·7 per cent was caught in the hydraulic trap placed in the ball mill discharge. The remainder of the 51·5 per cent was trapped as shown in the foregoing table.

These results serve to point out the inefficiency of the trap on this ore, and further to show that either blankets or amalgam plates must be employed ahead of flotation.

The preliminary small-batch tests show that free gold is present in the flotation tailing when no method is used for its removal to flotation. In these tests between 33 and 50 per cent of the gold in the tailing could be amalgamated. Therefore straight flotation can not be expected to work efficiently on this ore.

Straight cyanidation of the ore ground to the same degree as for flotation will give 92·5 per cent extraction, and if the pulp is tabled in cyanide solution and the table concentrate reground and re-cyanided a total extraction of 95 per cent is obtained.

A series of cyanide tests was made on the flotation concentrate. The concentrates were reground to about 80 per cent minus 200 mesh, and cyanided for 24 and 48 hours.

The average result for 24 hours' treatment was as follows:

Feed	2·00 oz./ton
Tailing	0·12 "
Extraction	94·0 per cent
Cyanide consumption	7·5 lb./ton.

The average for 48 hours gave an extraction of 94 per cent.

SUMMARY

The overall recovery obtained by flotation and cyanidation of the flotation concentrate would be between 86·5 per cent and 88 per cent.

The overall recovery obtained by cyanidation, and regrinding and cyaniding the table concentrate obtained by tabling in cyanide solution was 94·95 per cent.

Ore Dressing and Metallurgical Investigation No. 514

ANTIMONIAL GOLD-SILVER ORE FROM TATLAYOCO LAKE DISTRICT, BRITISH COLUMBIA

Shipment. A shipment of 75 pounds of ore was received on April 11, 1933, from I. T. Morris, 24 Fourth Avenue East, Vancouver, B.C. The sample was taken from claims in the Tatlayoco Lake district, B.C.

Characteristics. The gangue is mottled white to grey quartz. Some of the grey portions appear to be due to very finely-divided stibnite.

The sulphides present are stibnite, arsenopyrite, pyrite, and pyrrhotite. Chalcopyrite is rare, and was observed only in the mill products.

Stibnite occurs throughout the quartz in irregular masses and fine grains, and discontinuous tiny stringers. Arsenopyrite is present as comparatively large well formed crystals, and as disseminated tiny grains and swarms of crystals. The two minerals are commonly associated, in many cases very intimately; the stibnite forms the matrix of shattered arsenopyrite crystals, and occurs as fine veinlets and grains within the arsenopyrite or as an intimate mixture of stibnite and arsenopyrite.

Pyrite and pyrrhotite are comparatively rare, and are sparsely disseminated in the quartz gangue.

Manner of Occurrence of Gold and Silver. Stibnite and arsenopyrite are the only abundant metallic minerals. Spectrographic analyses of these for gold and silver indicated as follows:—

Stibnite	Au—nil. Ag—strong trace.
Arsenopyrite	Au—strong trace. Ag—moderate trace.

As no free gold was observed, it is probable that at least an appreciable proportion of the metal occurs in the arsenopyrite in such a finely-divided state that its particles are sub-microscopic. It will be seen then, that in order to recover the gold, recovery of the arsenopyrite is necessary.

Analysis:

Gold	0·925	oz./ton
Silver	15·26	"
Arsenic	3·90	per cent
Antimony	9·70	"

Experimental Tests

It is well known that a gold ore containing antimony is refractory to treatment by cyanidation. A series of selective flotation tests was therefore made in an endeavour to float out an antimony concentrate containing the silver and to make a tailing containing the bulk of the gold which would be relatively free from antimony or containing amounts that would not interfere with subsequent cyanide treatment.

The results of these selective flotation tests are given below:—

Test No.	Product	Weight, per cent	Analysis				Distribution			
			Au oz./ ton	A oz. /ton	Sb %	As %	Au %	Ag %	Sb %	As %
1......	Concentrate............	19·1	2·0	67·5	43·85	6·93	37·2	76·9	84·7	35·9
	Middling...............	5·1	2·06	24·68	9·55	7·08	10·3	7·6	4·9	9·8
	Tailing................	75·8	0·71	3·43	1·35	2·65	52·5	15·5	10·4	54·2
	Total..........	100·0	1·02	16·7	9·8	3·7	100·0	100·0	100·0	99·9
3......	Concentrate............	14·4	1·5	86·2	48·95	5·53	23·1	74·8	72·7	21·1
	Middling...............	3·2	2·64	23·9	10·95	8·72	9·0	4·6	3·6	7·4
	Tailing................	82·4	0·77	4·15	2·80	3·27	67·9	20·6	23·7	71·5
	Total..........	100·0	0·94	16·6	9·7	3·7	100·0	100·0	100·0	100·0
4......	Concentrate............	14·1	1·4	75·9	48·05	4·95	20·0	67·4	71·9	18·6
	Middling...............	5·3	2·8	30·5	11·2	9·75	14·9	10·1	6·3	13·8
	Tailing................	80·6	0·8	4·46	2·55	3·15	65·1	22·5	21·8	67·6
	Total..........	100·0	0·99	15·9	9·4	3·7	100·0	100·0	100·0	100·0
5......	Concentrate............	13·9	1·24	81·96	48·45	3·9	17·6	69·1	70·5	16·7
	Middling...............	5·9	2·6	32·0	15·0	7·86	15·7	11·5	9·3	14·3
	Tailing................	80·2	0·81	4·0	2·4	2·79	66·7	19·4	20·2	69·0
	Total..........	100·0	0·97	16·5	9·5	3·2	100·0	100·0	100·0	100·0

In Tests Nos. 1 to 4 the grinding was through 65 mesh with 65 per cent minus 200 mesh. In Test No. 5 the grinding was through 100 mesh with 85 per cent through 200 mesh. Two other tests were run on finer ground ore but with no better results.

Difficulty of Freeing Arsenopyrite and Stibnite. A grain analysis carried out microscopically on middling from Test No. 6 is shown in Table I. This shows that 38·2 per cent of the sulphides, represented chiefly by stibnite and arsenopyrite, is not freed in this product.

TABLE I

Grain Analysis of Middling from Test No. 6

Mesh	Approximate per cent by volume			
	Free sulphide, chiefly arseno-pyrite	Combined		Free stibnite
		Sulphide	Stibnite	
− 150+ 200..................................	1·9	4·1	2·1	6·0
− 200+ 325..................................	3·5	11·6	6·8	8·1
− 325+ 560..................................	6·2	4·1	2·4	6·2
− 560+1100..................................	16·2	4·0	2·2	6·2
−1100+2300..................................	3·0	....	0·6	1·4
−2300.......................................	1·2	....	0·3	1·9
Total...........................	32·0	23·8	14·4	29·8

Photomicrographs of the middlings show the manner in which the stibnite still clings to the arsenopyrite, the aggregate apparently possessing a tendency to break along the stibnite; they also show the minute grains of stibnite occurring within the arsenopyrite. This indicates that a fineness of grinding far beyond the realm of practicability would be required to approach a clean separation of the stibnite and arsenopyrite.

CYANIDATION OF FLOTATION TAILING

Test No. 1. A sample of the tailing from the flotation tests was agitated 24 hours with 30 pounds of lime per ton of ore and 10 pounds of lead acetate.

The solutions fouled rapidly, and the extraction obtained was negligible, being only 12 per cent.

Test No. 2. The raw ore was ground with 30 pounds of lime per ton and filtered. The washed material was then cyanided 48 hours with a 2-pound cyanide solution. The solutions fouled badly and the extraction was only 35 per cent.

Test No. 3. Test No. 2 was repeated with the addition of 2 pounds lead acetate per ton. The extraction at the end of 24 hours was 50 per cent.

REMARKS

This ore is very refractory, and although it might be possible to work out a method of treatment, it would require an elaborate plant. Such a method would probably include the selective flotation of an antimony concentrate containing the silver, and the regrinding and re-treatment of the middling obtained from flotation. The tailing from flotation containing about 80 per cent of the gold would then have to be given a roast. The roast would have to be followed in all probability by a second reducing roast with coal to eliminate as much antimony as possible. The roasted product would then have to be cyanided to recover the gold.

Such a complicated treatment does not seem practicable when the location of the property is taken into consideration.

Ore Dressing and Metallurgical Investigation No. 515

MAGNETIC IRON ORE FROM TEXADA ISLAND, B.C.

Shipment. A shipment of 53,000 pounds of magnetic iron ore was received September 7, 1932, from Texada Island, B.C.

Characteristics and Mineralogy of the Ore. The magnetite is quite massive, but when etched shows a finely granular texture. Hematite is distributed unevenly throughout the magnetite as very fine grains which usually do not exceed 0·01 mm. in diameter. Elongated grains of hematite rarely exceeding 0·005 mm. in width are common. They follow the crystallographic directions of the magnetite and are regarded as an alteration product. Silica is also distributed in irregular grains.

Minerals. The minerals observed in the ore are magnetite (Fe_3O_4), pyrite (FeS_2), chalcopyrite ($CuFeS_2$), and a mineral that closely resembles marcasite (FeS_2). Of the sulphides pyrite is the most abundant, marcasite(?) is next, and chalcopyrite assumes a minor position.

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 the sulphides are usually very closely associated with the magnetite.

Grain Size of the Sulphides. Although it is exceedingly difficult to determine the effective grain size of minerals that occur both in individual grains and in veinlets of considerable length, an attempt was made at an approximation by means of the microscope.

Approximate Effective Grain Size of the Sulphides

Mesh	Per cent sulphides
+ 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

Sampling and Analysis. The whole shipment was crushed to ½ inch and sampled.

	Per cent		Per cent
Fe_3O_4	83·54	MgO	0·96
$CuFeS_2$	0·29	Al_2O_3	0·54
FeS_2	3·74	Fe	62·27
MnO_2	0·16	S	2·10
SiO_2	6·45	P	0·015
P_2O_5	0·03	Mn	0·10
CaO	2·78	Cu	0·10

Purpose of Experimental Tests. Tests were run for two purposes: first, to produce a very high-grade, pure concentrate suitable for the production of sponge iron; second, to produce a high-grade product that would be coarse enough to sinter at a low cost so that high-grade sinter would be available for export.

Experimental Tests

Runs Nos. 1 and 2

Pilot runs were made to obtain adjustments on the magnetic separators and in the grinding circuit.

A new design of wet magnetic separator, known as the Roche separator, and similar to those used in the New Jersey iron district, was used.

The flow-sheet was as follows:—

The ore at ½ inch was fed to a 4- by 3-foot ball mill containing 2,500 pounds of 3-inch balls and operated in closed circuit with a Dorr classifier. The overflow from the Dorr went direct to the magnetic separator. In these tests three products were made—a final tailing, a middling, and a concentrate.

Feed rate to the mill	1500·0 lb./hr.
Current on rougher machine	12·0 amperes
Current on finisher machine	3·5 "

Analyses of Products:

Product	Iron, per cent	Sulphur, per cent	Phosphorus, per cent	Fe_3O_4, per cent
Concentrate	72·0	0·12	0·003	
Middling	35·73	6·8		
Tailing	20·9	8·8		0·64

Ratio of concentration	1·23: 1.
Recovery of metallic iron	93·7 per cent
Recovery of magnetic oxide	99·8 "

Run No. 3

Flow-sheet as in Runs Nos. 1 and 2.

Feed rate	1500·0 lb./hr.
Current on rougher machine	12·0 amperes
Current on finisher machine	3·5 "

Analyses of Products:

Product	Iron, per cent	Sulphur, per cent	Phosphorus, per cent	Fe_3O_4, per cent
Feed	61·91			
Concentrate	72·06	0·13	0·003	....
Middling	58·26	3·02	....	....
Tailing	21·72	7·94	....	0·75

Ratio of concentration .... 1·25 : 1.
Recovery of metallic iron .... 93·0 per cent
Recovery of magnetic iron .... 99+ "

Screen Tests on Products:

Feed to Separator

Mesh	Weight, per cent
+ 65	1·3
− 65+100	3·6
−100+150	8·0
−150+200	6·0
−200	81·1
Total	100·0

Magnetic Concentrate

Mesh	Weight, per cent
+ 65	1·1
− 65+100	4·7
−100+150	10·5
−150+200	7·7
−200	76·0
Total	100·0

Run No. 4

The flow-sheet was the same as in the previous tests, with the important exception that the middling was returned to the rougher machine.

Feed rate .... 1500·0 lb./hr.
Current on rougher machine .... 12·0 amperes
Current on finisher machine .... 3·5 "

Middling returned without dewatering to rougher machine.

Analyses of Products:

Product	Iron, per cent	Sulphur, per cent	Phosphorus, per cent	Fe_3O_4, per cent
Feed	59·47			
Concentrate	70·85	0·15	0·003	....
Tailing	21·1	8·4	....	0·88

Ratio of concentration .... 1·3 : 1.
Recovery of metallic iron .... 92·0 per cent
Recovery of magnetic iron .... 99+ "

Screen Tests on Products:

Feed to Separator

Mesh	Weight, per cent
+ 48	1·5
− 48+ 65	1·4
− 65+100	4·6
−100+150	10·6
−150+200	6·6
−200	75·3
Total	100·0

Magnetic Concentrate

Mesh	Weight, per cent
+ 48	0·5
− 48+ 65	0·8
− 65+100	3·3
−100+150	9·1
−150+200	6·4
−200	79·9
Total	100·0

Run No. 5

The flow-sheet was the same as in Run No. 4. The middling was returned to the rougher machine.

Feed rate	1500·0 lb./hr.
Current on rougher machine	12·0 amperes.
Current on finisher machine	3·5 "

Analyses of Products:

Product	Iron, per cent	Sulphur, per cent	Phosphorus, per cent	Fe_3O_4, per cent
Feed	62·52			
Concentrate	71·25	0·14	0·007	....
Tailing	22·13	9·14	....	1·0

Ratio of concentration	1·22 : 1.
Recovery of metallic iron	93·8 per cent
Recovery of magnetic iron	99·5+ "

Screen Tests on Products:

Feed to Separator

Mesh	Weight, per cent
+ 48	1·0
− 48+ 65	1·3
− 65+100	4·4
−100+150	11·7
−150+200	9·4
−200	72·2
Total	100·0

Magnetic Concentrate

Mesh	Weight, per cent
+ 48	0·3
− 48+ 65	0·6
− 65+100	4·9
−100+150	14·5
−150+200	9·7
−200	70·0
Total	100·0

Run No. 6

The flow-sheet was altered for this run by placing a 28-mesh Hummer screen in closed circuit with the ball mill.

The middling was removed from the circuit.

Feed rate	1500·0 lb./hr.
Current on rougher machine	12·0 amperes
Current on finisher machine	3·0 "

Analyses of Products:

Product	Iron, per cent	Sulphur, per cent	Phosphorus, per cent	Fe_3O_4, per cent
Feed	60·90			
Concentrate	70·24	0·19	0·004	
Middling	52·37	2·59		
Tailing	19·48	7·98	....	1·4

Ratio of concentration	1·22 : 1.
Recovery of metallic iron	94·2 per cent
Recovery of magnetic iron	99+ "

Screen Tests on Products:

Feed to Separator

Mesh	Weight, per cent
+ 35	5·4
− 35+ 48	11·3
− 48+ 65	13·0
− 65+100	13·1
−100+150	11·5
−150+200	4·5
−200	41·2
Total	100·0

Magnetic Concentrate

Mesh	Weight, per cent
+ 35	9·2
− 35+ 48	9·9
− 48+ 65	14·5
− 65+100	14·5
−100+150	13·2
−150+200	5·8
−200	32·9
Total	100·0

Run No. 7

The flow-sheet was the same as in Run No. 6 but a 10-mesh screen was used on the Hummer. The middling was taken out of the circuit.

Feed rate	2000·0 lb./hr.
Current on rougher machine	12·0 amperes
Current on finisher machine	2·75 "

Analyses of Products:

Product	Iron, per cent	Sulphur, per cent	Phosphorus, per cent	Fe_3O_4, per cent
Feed	62·52			
Concentrate	70·24	0·23	0·004	
Middling	56·23	2·38		
Tailing	21·11	8·85	....	2·0

Weight of middling made in proportion to the feed was 380 pounds per ton of feed. The middling would have to be reground to lower the sulphur content.

Screen Tests on Products:

Feed to Separator

Mesh	Weight, per cent
+ 14	1·9
− 14+ 20	4·9
− 20+ 28	4·7
− 28+ 35	11·0
− 35+ 48	10·4
− 48+ 65	11·2
− 65+100	10·8
−100+150	9·6
−150+200	3·7
−200	31·8
Total	100·0

Magnetic Concentrate

Mesh	Weight, per cent
+ 14	2·0
− 14+ 20	5·7
− 20+ 28	7·0
− 28+ 35	9·0
− 35+ 48	11·3
− 48+ 65	11·4
− 65+100	12·5
−100+150	10·1
−150+200	4·1
−200	26·9
Total	100·0

The loss of magnetic iron in the tailing was less than 1·0 per cent of the total.

Run No. 8

The flow-sheet was the same as in Run No. 7 but a 20-mesh screen was used on the Hummer. Feed rate was 2,000 pounds per hour. The middling was removed from the circuit.

Analyses of Products:

Product	Iron, per cent	Sulphur, per cent	Phosphorus, per cent	Fe_3O_4, per cent
Feed	62·12			
Concentrate	70·64	0·20	0·004	
Middling	56·64	2·90		
Tailing	22·53	9·14	....	2·0

Ratio of concentration	1·22 : 1.
Recovery of metallic iron	93·6 per cent
Recovery of magnetic iron	99+ "

Screen Tests on Products:

Feed to Separator

Mesh	Weight, per cent
+ 28	2·0
− 28+ 35	12·5
− 35+ 48	12·3
− 48+ 65	11·8
− 65+100	12·0
−100+150	10·2
−150+200	4·0
−200	35·2
Total	100·0

Magnetic Concentrate

Mesh	Weight, per cent
+ 28	3·0
− 28+ 35	14·0
− 35+ 48	13·0
− 48+ 65	13·6
− 65+100	12·6
−100+150	10·5
−150+200	4·8
−200	28·4
Total	100·0

REMARKS

The object of concentrating iron ores is generally for the purpose of enriching their metallic iron content and to eliminate such objectionable elements as phosphorus, sulphur, titanium, copper, and silica.

The suitability of an iron ore for production of sponge iron is determined by the purity of the sponge that can be produced from it. It is now recognized that the sponge should run over 90 per cent in metallic iron and contain less than 10 per cent of slag-forming material.

The results of these tests therefore show that the Texada Island ore is excellent for the production of sponge iron.

As sintering of magnetic concentrate for use in the blast furnace has now become general practice, and as it is also necessary for the production of sponge iron by certain of the more successful processes, sizing and preparation of the concentrate for low-cost sintering is an important consideration. The sizing of the ore for sintering is of great importance as the effect of the porosity of the material to be sintered has a direct bearing on the production of sinter. In sintering, the size of the concentrate directly affects the cost of sintering. For example, a magnetic concentrate that is minus 6 mesh can be sintered for 50 cents per ton while the cost of sintering concentrate that is minus 60 mesh is over $1 per ton.

For use in the blast furnace, the sinter must be hard and resist crushing while handling. The use of sinter in the blast furnace has resulted in lower costs owing to increased capacity of the furnace and because less coke and limestone are required. The cost of a ton of pig iron made from sintered ore or concentrate assaying 60 per cent Fe and 10 per cent silica is about $2 per ton less than pig iron made from a crude ore assaying 55 per cent Fe and 15 per cent silica.

The object of producing a high-grade concentrate with coarser grinding, as was done in Runs Nos. 6, 7, and 8, is, therefore, evident. Although the actual tests were not carried out, there is no doubt that a concentrate 10 per cent of which is plus 10 mesh can be produced and the grade of the total concentrate maintained at 68 to 69 per cent iron.

It will be observed, however, that the sulphur is not eliminated to the same extent as with finer grinding, but sintering tests on the raw ore have demonstrated that there is no difficulty in reducing the sulphur in the sinter to 0·10 per cent or lower.

Ore Dressing and Metallurgical Investigation No. 516

GOLD ORE FROM THE GUELPH MINE, WAWA, MICHIPICOTEN, ONT.

Shipment. A shipment of 18 samples of ore was received July 11, 1933, from R. N. Bond, in care of A. P. Earle, Montreal Life Insurance Company, Montreal, Que.

Characteristics of the Ore. The ore was chiefly white to yellowish quartz containing a small amount of pyrite, some of which has been altered to limonite.

Experimental Tests

After the samples had been assayed individually a composite sample was made by mixing together eight of them, viz. Nos. 1, 2, 3, 4, 9, 10, 15, and 16. Some small-scale tests were made on this composite sample, the work being limited to amalgamation and cyanidation.

An assay of the composite sample showed the gold content to be 0·26 ounce per ton.

AMALGAMATION AND CYANIDATION

Tests Nos. 1 to 4

Four samples of the ore were ground dry to pass respectively through the following screens, 48, 100, 150, and 200 mesh; 1,000 grammes of each of these was amalgamated for 30 minutes in 1 : 1 pulp. The tailings were sampled and assayed for gold. A portion of the tailing was agitated for 24 hours in cyanide solution, 2 pounds KCN per ton; the cyanide tailing also being assayed for gold.

Summary:

Feed sample, Au 0·26 oz./ton

Test No.	Amalgamation tailing, Au, oz./ton	Amalgamation tailing cyanided, Au, oz./ton	Recovery by amalgamation	Recovery by cyanidation	Reagents consumed, lb./ton	
					KCN	CaO
1	0·045	0·005	82·7	15·4	2·25	5·5
2	0·04	0·005	84·6	13·5	3·15	6·3
3	0·04	0·005	84·6	13·5	3·15	6·3
4	0·03	0·005	88·5	9·6	3·15	6·3

The test numbers 1, 2, 3, and 4 correspond to the different sized products in the order mentioned above. The same applies to Tests Nos. 5 to 12.

CYANIDATION

Tests Nos. 5 to 12

Samples of the raw ore were ground as described under the preceding tests and were agitated in cyanide solution, 2 pounds KCN, per ton for 24 and 48 hours. The tailings were assayed for gold.

Summary:

Test No.	Period of agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
5	24	0·015	94·2	2·1	6·1
6	24	0·01	96·2	3·0	6·1
7	24	0·01	96·2	3·0	6·1
8	24	0·01	96·2	3·0	6·1
9	48	0·005	98·1	3·0	6·1
10	48	0·005	98·1	3·6	6·1
11	48	0·005	98·1	3·6	6·1
12	48	0·005	98·1	3·6	6·1

CONCLUSIONS

The ore appears to be a simple one to treat. With a comparatively coarse grind (all through 48 mesh) 98·1 per cent of the gold can be recovered either by cyanidation for 48 hours or by amalgamating and then cyaniding the amalgamation tailing for 24 hours. From the high recovery by amalgamation it is evident that most of the gold occurs in the free state, but it must be quite fine as it cyanides readily.

It would seem that straight cyanidation is the most practical method for treating this ore, but it would be advisable to have the ball mill discharge into a hydraulic trap so that any large particles of free gold in the ore would be kept out of the cyanide circuit. The trap cleaning could be barrel-amalgamated and then united with the main body of ore in the cyanide circuit.

Ore Dressing and Metallurgical Investigation No. 517

GOLD ORE FROM NORTHERN METALS, LIMITED, LARDER LAKE, KATRINE TOWNSHIP, ONT.

Shipment. Two shipments of ore were received from the Northern Metals, Limited, 1502 St. Catherine St. W., Montreal, consigned from their property at Larder Lake, Katrine township, Ontario. The first weighing 3,000 pounds arrived on June 24, 1933, and the other shipment of 300 pounds on July 5.

Characteristics of the Ore. The larger sample consisted of a small amount of sulphides in a gangue of white quartz, pink calcite, and a greenish grey chloritic material. The constituents were very irregular in distribution imparting a mottled appearance to the ore.

The metallic minerals consisted mainly of iron pyrite with a very small amount of chalcopyrite, magnetite, hematite, and native gold. The gold is very fine and occurs both in the quartz and in the pyrite. The chalcopyrite also is fine-grained, minus 325 mesh, and is less than 1 per cent by volume of the total sulphides.

Sample No. 2 was a high-grade material, and consisted of white milky quartz with patches of chloritic material and a small amount of disseminated pink calcite.

The same metallic minerals are present in more abundance than in Sample No. 1. The pyrite is disseminated as well formed cubes and irregular grains and contains veinlets and tiny grains of chalcopyrite and native gold.

Experimental Tests

After crushing, grinding, and sampling, the shipments were found to contain:—

Shipment No. 1	Au 0·11 oz./ton
Shipment No. 2	Au 1·87 "

For test purposes Lot No. 2 was mixed with approximately one-half of Lot No. 1, giving a product having an assay value of 0·445 ounce gold per ton. Amalgamation, cyanidation, and flotation tests were made on this composite lot.

It was found that 49·2 per cent of the gold could be recovered by amalgamating minus 48-mesh material with 40 per cent minus 200-mesh. Finer grinding liberates more gold, 69 per cent being amalgamated when ground 80 per cent minus 200-mesh.

Cyaniding extracts 96·7 per cent of the gold at minus 200 mesh.

Flotation recovers 87 per cent of the gold and gives a high-grade concentrate carrying 11·8 ounces of gold per ton, with a ratio of concentration of 41 : 1.

AMALGAMATION AND CYANIDATION

Test No. 1

A sample of the ore was ground to pass 48 mesh and amalgamated.

A portion of the amalgamation tailing was cyanided for 48 hours, 1 : 3 dilution, with a 1 pound KCN per ton solution; 7 pounds of lime per ton was added for protective alkalinity.

Results:

Feed	Au 0·445 oz./ton
Amalgamation tailing	Au 0·23 "
Recovery	49·2 per cent

Cyanidation of the amalgamation tailing reduced the gold content to 0·06 ounce in 24 hours and to 0·04 ounce in 48 hours. Overall recoveries by amalgamation and cyanidation were 86·5 per cent and 91 per cent respectively.

A screen analysis of the amalgamation tailing shows:

Mesh	Weight, per cent	Assay, Au oz./ton
− +65	8·9	0·24
− 65+100	23·8	0·30
−100+150	18·0	0·28
—150+200	9·4	0·24
−200	39·9	0·15

Test No. 2

A similar test was made on minus 100-mesh material, 62 per cent minus 200 mesh.

Results:

Feed	Au 0·445 oz./ton
Amalgamation tailing	Au 0·19 "
Recovery	57·3 per cent
24-hour cyanide tailing	0·03 oz.
Total recovery in 24 hours	93·3 per cent
48-hour cyanide tailing	0·025 oz.
Total recovery	94·4 per cent

CYANIDATION

Test No. 3

Samples of the ore ground to pass 48, 100, 150, and 200 mesh were cyanided for 48 hours, 1 : 3 dilution, with a 1 pound KCN per ton solution; 7 pounds lime per ton of ore was added for protective alkalinity.

Mesh	Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
					KCN	CaO
− 48	24	0·445	0·085	80·9	1·2	4·2
− 48	48	0·445	0·055	87·7	1·2	5·6
−100	24	0·445	0·035	92·2	1·2	4·2
−100	48	0·445	0·03	93·3	2·1	5·8
−150	24	0·445	0·02	95·5	1·5	4·5
−150	48	0·445	0·02	95·5	2·1	6·2
−200	24	0·445	0·015	96·7	1·2	4·5
−200	48	0·445	0·015	96·7	2·1	6·4

FLOTATION

Test No. 4

A sample of the ore was ground 80 per cent minus 200 mesh with 4 pounds soda ash per ton and floated with 0·10 pound sodium xanthate and 0·16 pound pine oil per ton.

Product	Weight, per cent	Assay Au, oz./ton	Distribution of gold, per cent
Feed	100·0	....	100·0
Concentrate	2·4	11·80	87·9
Tailing	97·6	0·04	12·1

Test No. 5

A sample of the ore was ground to pass 80 per cent through 200 mesh and amalgamated. After removing the amalgam, the pulp was conditioned with 4 pounds soda ash per ton and a concentrate removed by the addition of 0·10 pound sodium xanthate and 0·06 pound pine oil per ton.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·00	0·445	100·0
Amalgam (cal.)	....	....	68·9
Amalgam. tailing (cal.)	....	0·138	
Flotation concentrate	2·1	4·72	22·3
Flotation tailing	97·9	0·04	8·8

SUMMARY AND CONCLUSIONS

Amalgamation of the ore ground 40 per cent minus 200 mesh and all through 48 mesh gives a recovery of 49·2 per cent. This is raised to 57·3 per cent when the ore is ground 61·8 per cent minus 200 mesh.

Cyanidation of a minus 48-mesh product gives 80·9 per cent extraction. This increases with the fineness of grinding until 96·7 per cent is recovered by cyaniding minus 200-mesh ore for 24 hours.

Flotation produces a high-grade concentrate with a tailing loss of 0·04 ounce per ton. Approximately 47 tons of ore would be required to produce one ton of concentrate. This concentrate would require additional treatment to recover the gold.

Fine grinding is indicated in all tests. Microscopic examination shows the gold to be finely divided.

The process indicated is cyanidation of the ore ground to approximately 80 per cent minus 200 mesh.

Ore Dressing and Metallurgical Investigation No. 518

GOLD ORE FROM LAKELAND GOLD MINES, LIMITED, BOURKES, ONT.

Shipment. A shipment of 2,000 pounds of ore contained in 31 bags was received by freight on July 14, 1933, from the Lakeland Gold Mines, Limited, Bourkes, Ont., Oscar L. Knutson, Manager.

Characteristics of the Ore. The shipment consisted of greenish grey, altered rock of a siliceous nature. This is traversed by a network of fine veinlets of white carbonate. The carbonate is also finely disseminated throughout the gangue.

Metallic minerals to about 2 per cent the volume of the ore are present mainly as iron pyrite. Magnetite, hematite, and native gold were observed in polished sections. The pyrite occurs as finely disseminated cubes and irregular grains; chalcopyrite is rare. Extremely fine grains of native gold —12 microns in diameter—were observed within the pyrite.

Experimental Tests

The entire shipment was crushed, quartered, ground, and sampled, and found to contain 0·29 ounce gold per ton.

The experimental work carried out included tests by amalgamation, flotation, and cyanidation. The results obtained show that 35·9 per cent of the gold can be recovered by amalgamating ore ground minus 48 mesh with 43·7 per cent minus 200 mesh; 51·7 per cent is recovered from ore ground 90 per cent minus 200 mesh.

Flotation recovered 86·4 per cent of the gold in a concentrate assaying 3·66 ounces gold per ton with a ratio of concentration of 15·4 : 1.

Cyanidation of minus 48-mesh material extracts 81 per cent of the gold. The recovery increases with the fineness of grinding until 94·7 per cent is extracted from a minus 200-mesh product.

AMALGAMATION AND CYANIDATION

Test No. 1

A sample of the ore ground minus 48 mesh was amalgamated. After removing amalgam, the tailing was cyanided for 48 hours, 1 : 3 dilution with a 1 pound KCN per ton solution and 9 pounds lime per ton of ore.

Results:

Feed	Au	0·29	oz. /ton
Amalgamation tailing	Au	0·186	"
Recovery		35·9	per cent

Screen Analysis of Amalgamation Tailing:

Mesh	Weight, per cent	Assay, Au, oz./ton
− 48+ 65	15·1	0·185
− 65+100	22·9	0·22
−100+150	13·8	0·27
−150+200	4·5	0·27
−200	43·7	0·135

24-hour cyanide tailing .... 0·055 oz. Au/ton

Total recovery:—
Amalgamation and cyanidation .... 81·0 per cent
48-hour cyanide tailing .... 0·05 oz. Au/ton
Total recovery .... 82·0 per cent

Test No. 2

A test similar to Test No. 1 was made on material ground to pass 100 mesh.

Results:

Feed .... 0·29 oz. Au/ton
Amalgamation tailing .... 0·14 "
Recovery .... 51·7 per cent
24-hour cyanide tailing .... 0·025 oz. Au/ton

Total recovery:—
Amalgamation and cyanidation .... 91·4 per cent
48-hour cyanide tailing .... 0·02 oz. Au/ton
Total recovery .... 93·1 per cent

CYANIDATION

Test No. 3

Samples of the ore were ground to pass 48, 100, 150, and 200 mesh and cyanided 1 : 3 dilution with a 1 pound KCN solution for 48 hours; 7 pounds lime was added for protective alkalinity.

Mesh	Agitation, hours	Feed, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
					KCN	CaO
− 48	24	0·29	0·055	81·0	0·3	5·7
− 48	48	0·29	0·055	81·0	0·3	6·7
−100	24	0·29	0·025	91·4	0·6	6·3
−100	48	0·29	0·025	91·4	0·9	6·9
−150	24	0·29	0·025	91·4	0·6	6·3
−150	48	0·29	0·02	93·1	0·9	6·9
−200	24	0·29	0·015	94·7	0·9	6·7
−200	48	0·29	0·02	93·1	0·9	7·7

Test No. 4

A sample of the ore was ground 82 per cent minus 200 mesh with 4 pounds soda ash per ton and floated with 0·10 pound potassium ethyl xanthate and 0·10 pound pine oil per ton.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·0	0·29	100·0
Flotation concentrate	6·5	3·66	86·4
Tailing	93·5	0·04	13·6

Ratio of concentration, 15·4 : 1.

SUMMARY AND CONCLUSIONS

Approximately 52 per cent of the gold is free and can be amalgamated and 35·9 per cent should be recovered from material crushed to pass 48 mesh.

Flotation recovers 86 per cent of the gold, leaving 0·04 ounce per ton in the tailing.

Cyanidation of the raw ore ground minus 200 mesh leaves a residue of 0·015 ounce per ton, a recovery of 94·7 per cent.

Fine grinding is necessary to secure a high recovery.

The process indicated is cyanidation. Grinding should be done in cyanide solution to produce a classifier overflow approximately minus 200 mesh, followed by agitation for about 24 hours.

Ore Dressing and Metallurgical Investigation No. 519

GOLD ORE FROM ONAMAN LAKE AREA, THUNDER BAY DISTRICT, ONT.

Shipment. A shipment consisting of 18 samples, weighing 40 pounds, was received from E. D. Loney, Sudbury, Ont., on July 28, 1933.

Characteristics of the Ore. The shipment consisted of glassy to milky vein quartz with a small amount of carbonate and some scattered grains of pyrite.

Sampling and Analysis. The ore was crushed minus 100 mesh and carefully mixed. A representative portion was cut out for a feed sample. The remainder was used in the following tests:—

Assay of composite sample.................................. 0·20 oz. Au/ton.

Summary of Experimental Tests. The recovery by cyanidation for 24- and 48-hour tests was the same, namely 95 per cent. The consumption of reagents was considerably greater in the 48-hour test. The extraction by amalgamation was 50 per cent. The recovery by flotation was 86 per cent and the grade of concentrate was 3·71 ounces gold per ton.

Experimental Tests

CYANIDATION

A representative sample of ore was cyanided by agitating in a Winchester bottle with a solution of sodium cyanide equivalent in strength to 1 pound of KCN per ton of solution. The dilution of the pulp was 1 part of ore to 3 parts of solution. Lime was added at the rate of 5 pounds per ton of ore to give a protective alkalinity.

Results:

Test No.	Time, hours	Assay Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton	
		Feed	Tailing		KCN	CaO
1..................	24	0·20	0·01	95·0	0·36	4·16
2..................	48	0·20	0·01	95·0	0·66	4·25

AMALGAMATION

A representative sample of the ore was amalgamated with 10 per cent of its weight of mercury in a pulp dilution of 1 : 1. The amalgam was separated and the tailing assayed.

Results:

Assay Au, oz./ton		Extraction
Feed	Tailing	Per cent
0·20	0·10	50

FLOTATION

A representative sample of the ore was crushed in a ball mill with the following reagents:—

Soda ash.. 4·0 lb./ton
Aerofloat No. 25.. 0·035 "

The reagents added to the flotation cell were:—

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

Results:

Products	Weight, per cent	Assay, Au, oz/ton	Distribution of gold, per cent
Feed	100·00	*0·2045	100·0
Concentrate	4·74	3·71	86·0
Tailing	95·26	0·03	14·0

*Feed assay calculated.

Ore Dressing and Metallurgical Investigation No. 520

GOLD ORE FROM NORGOLD MINES, LIMITED, BOUSQUET TOWNSHIP, ABITIBI COUNTY, QUEBEC

Shipment. A shipment of 32 bags of ore, net weight 2,420 pounds, was received July 22, 1933, from H. S. Denny, Consulting Engineer, 2529 Canadian Bank of Commerce Building, 25 King St. West, Toronto 2, Ontario.

Characteristics of the Ore. The gangue consists chiefly of grey to greyish white glassy vein quartz, with local portions containing a small amount of greenish grey chloritic material. Considerable carbonate is present both as stringers and as small masses and crystals in the quartz.

The metallic minerals present are pyrite, arsenopyrite, pyrrhotite, and chalcopyrite. The pyrite and pyrrhotite occur in poorly-formed crystals and irregular grains.

The pyrrhotite and chalcopyrite show a tendency toward mutual association, usually in the carbonate. The grains of these minerals vary greatly in size.

No free gold was observed, and it is not known how this metal occurs in the ore.

An average assay of the ore was as follows:—

Au	0·29 oz./ton
Ag	0·05 "

Experimental Tests

A series of small-scale tests was made on the ore to determine how best it might be treated in practice. The work included tests by amalgamation, cyanidation, and flotation. Recoveries of 90 per cent were obtained by flotation and amalgamation, and 99 per cent by cyanidation.

Details of the tests follow.

AMALGAMATION AND CYANIDATION

Tests Nos. 1 to 4

Four samples of the ore were ground dry to pass through the following screens: 48, 100, 150, and 200 mesh. Tests were made on each of the different sized products, the test numbers in each series corresponding to the different sizes in the above-mentioned order.

In this series, 1,000 grammes of the ore was amalgamated with mercury in 1 : 1 pulp for thirty minutes. The amalgamation tailing was sampled and assayed for gold and a portion of each agitated in cyanide solution, 2 pounds per ton KCN, for 24 hours. The cyanide tailing was also assayed for gold.

Summary:

Feed sample, Au 0·29 oz./ton.

Test No.	Amalgamation tailing, Au, oz./ton	Amalgamation tailing cyanided, Au, oz./ton	Recovery by amalgamation	Recovery by cyanidation	Total net recovery	Reagents consumed, lb./ton	
						KCN	CaO
1.......	0·04	0·010	86·2	10·4	96·6	0·75	7·4
2.......	0·03	0·005	89·7	8·6	98·3	0·75	7·9
3.......	0·04	0·005	86·2	12·1	98·3	0·75	8·1
4.......	0·075	0·010	74·1	22·5	96·6	0·75	9·0

Tests Nos. 5 to 12

In this series of tests samples of the ore, ground dry as described above, were agitated in cyanide solution, 2 pounds KCN per ton, for periods of 24 and 48 hours. The tailings were assayed for gold.

Summary:

Feed sample, Au 0·29 oz./ton

Test No.	Mesh	Period of agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
					KCN	CaO
5..................	− 48	24	0·01	96·6	0·45	8·00
6..................	−100	24	0·005	98·3	0·60	8·00
7..................	−150	24	0·003	99·0	0·60	8·25
8..................	−200	24	0·010	96·6	0·75	8·70
9..................	− 48	48	0·01	96·6	0·60	8·70
10..................	−100	48	0·005	98·3	0·60	9·2
11..................	−150	48	0·005	98·3	0·60	9·2
12..................	−200	48	0·005	98·3	0·80	9·8

FLOTATION

Test No. 13

A sample of 1,000 grammes of the ore at −14 mesh was ground in a ball mill for 20 minutes and then floated.

Charge to Ball Mill:

Ore..	1000	grammes
Water..	750	c.c.
Na_2CO_3..	15	lb./ton
Aerofloat No. 25..	0·07	"

Reagents to Cell:

Potassium amyl xanthate..	0·16 lb./ton
Pine oil..	0·16 "

A screen test on the flotation tailing showed the grinding to be as follows:—

Mesh	Weight, per cent
+ 65	0·2
− 65+100	2·6
−100+150	9·8
−150+200	8·6
−200	78·8

The concentrate and tailing were assayed for gold.

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Concentrate	3·6	5·34	90·1
Tailing	96·4	0·02	9·9
Feed (cal.)	100·0	0·21	100·0

SUMMARY AND CONCLUSIONS

The ore responds readily to cyanidation, and this is the process to be recommended. In Test No. 5 with the ore ground to approximately 66 per cent minus 200 mesh a recovery of 96·6 per cent was obtained in 24 hours leaving a tailing assaying 0·01 ounce per ton in gold. Finer grinding reduces the tailing assay, but is of doubtful economic value.

There is apparently too much free gold in the ore to float it in its raw state. By referring to Test No. 13 it will be seen that the feed sample calculated back from the products is low, some of the free gold, no doubt, having been lost in the machine and so being unaccounted for.

The higher recovery by cyanidation, as compared with amalgamation, makes this process more desirable.

Amalgamation followed by flotation and cyanidation of the flotation concentrate may have possibilities.

Ore Dressing and Metallurgical Investigation No. 521

GOLD ORE FROM THE SUMMIT LAKE DISTRICT, ONTARIO

Shipment. A shipment of 40 pounds of ore was received on August 14, 1933, from Ventures, Limited, 100 Adelaide Street West, Toronto.

Analysis of Ore:

Gold	0·66 oz./ton
Arsenic	0·43 per cent

Character of the Ore. Microscopically, the ore is seen to consist chiefly of mottled grey to white vein quartz, locally including portions rich in chloritic material or small amounts of carbonate, and containing rather coarse, sparsely disseminated arsenopyrite. The microscope, however, shows in addition to arsenopyrite, galena, chalcopyrite, pyrrhotite(?), and native gold.

Arsenopyrite occurs as angular and poorly-formed disseminated crystals, the size of which averages plus 35 mesh. Some of the crystals contain a small amount of galena, occasional chalcopyrite usually associated with native gold, and many small inclusions of gangue.

Galena is in small amount, as described above. Chalcopyrite was observed as tiny grains in the gangue and within the arsenopyrite, but its amount is practically negligible. A few tiny grains of a mineral closely resembling pyrrhotite were seen within the arsenopyrite.

Mode of Occurrence of the Gold. Gold was seen only within the arsenopyrite. Its most common occurrence in the sections studied is as tiny irregular grains; less commonly the metal is associated with chalcopyrite or with inclusions of gangue.

It is noteworthy that a network of extremely fine fractures exists in the arsenopyrite. It is highly probable that these fractures carry a large proportion of the gold grains and thus minimize the degree to which grinding need be carried, as the arsenopyrite tends to break along these fractures readily, thus exposing the gold for attack of cyanide solutions; and also it is probable that these fractures would in many cases provide channels by means of which the cyanide solutions could penetrate to the gold grains. While the above condition exists in all of the arsenopyrite observed in the sections examined, it is not possible to predict uniformity of this condition throughout the ore.

Experimental Tests

No flotation tests were made on this ore, as arsenopyrite is the predominating sulphide and it is shown by the microscopic examination to contain the gold. Inasmuch as arsenopyrite is difficult to float and in addition is refractory in contact with cyanide solutions, particularly as a concentrate, it does not seem good practice to make a concentrate high in arsenic that would subsequently have to be cyanided.

The experimental tests, therefore, consisted of a series of straight cyanide tests at different sizes of feed, and of a series of amalgamation tests followed in one test by table concentration and barrel amalgamation of the table concentrate.

CYANIDATION

Tests Nos. 1 to 6

Test No.	Assay, Au, oz./ton		Extraction, per cent	Reagents consumed, lb./ton		Time of agitation, hours
	Feed	Tailing		KCN	CaO	
1	0·66	0·04	94·0	0·10	3·7	30
2	0·66	0·085	87·1	0·30	4·15	30
3	0·66	0·045	93·2	0·50	4·2	30
4	0·66	0·015	96·3	0·32	4·0	30
5	0·66	0·015	96·3	0·32	4·3	30
6	0·66	0·015	96·3	0·56	4·1	30

Screen Analyses:

Test No. 1

Mesh	Weight, per cent
+ 48	14·1
− 48+ 65	17·5
− 65+100	19·2
−100+150	14·5
−150+200	5·0
−200	29·7
Total	100·0

Test No. 2

Mesh	Weight, per cent
+ 48	1·5
− 48+ 65	7·6
− 65+100	16·9
−100+150	19·6
−150+200	8·6
−200	45·8
Total	100·0

Test No. 3

Mesh	Weight, per cent
+ 48	0·5
− 48+ 65	4·6
− 65+100	13·7
−100+150	19·8
−150+200	9·7
−200	51·7
Total	100·0

Test No. 4

Mesh	Weight, per cent
+ 48	0·12
− 48+ 65	1·54
− 65+100	8·64
−100+150	17·80
−150+200	10·80
−200	61·10
Total	100·00

Test No. 5

Mesh	Weight, per cent
+ 48	0·04
− 48+ 65	0·22
− 65+100	3·14
−100+150	11·88
−150+200	9·56
−200	75·16
Total	100·00

Test No. 6

Mesh	Weight, per cent
+ 48	0·02
− 48+ 65	0·08
− 65+100	1·88
−100+150	8·20
−150+200	7·74
−200	82·08
Total	100·00

The erratic results in Tests Nos. 2 and 3 are undoubtedly due to coarse free gold that remains undissolved. The grinding in these tests was done wet from 14 mesh to the size shown in the screen tests. The excellent extractions shown can be accounted for by the extremely fine fractures in the arsenopyrite that lead into the enclosures of native gold. The best results were obtained with a grind of 60 per cent −200 mesh and the tests show conclusively that it is unnecessary to grind any finer for cyanidation.

AMALGAMATION

Test No. 1

A batch of 1,000 grammes of ore, crushed to 14 mesh, was ground wet in a ball mill and then amalgamated in an amalgamated copper gold pan. This amalgamation was carried out in the same manner as for panning a sample of gold gravel.

Screen Test:

Mesh	Weight, per cent
+ 48	14·5
− 48+ 65	17·4
− 65+100	18·5
−100+150	14·6
−150+200	5·0
−200	30·0
Total	100·0

Results:

Amalgamation gave a tailing of 0·10 ounce gold per ton, which is equivalent to a recovery of 84·9 per cent.

Test No. 2

A lot of 2,000 grammes of ore at 14 mesh was ground and amalgamated by being agitated in a bottle with 200 grammes of mercury for one-half hour.

Screen Test:

Mesh	Weight, per cent
+ 48	1·5
− 48+ 65	7·6
− 65+100	16·9
−100+150	19·6
−150+200	8·6
−200	45·8
Total	100·0

After amalgamation the ore was passed over a small Wilfley table and a concentrate removed. The table concentrate was barrel-amalgamated. Owing to the high arsenic content of the concentrate there was some mercury loss due to sickening but not enough to be serious.

Results:

Gold recovered by amalgamation of the ore and barrel amalgamation of the concentrate	90·3	per cent
Table tailing:—		
Gold	0·037	oz./ton
Arsenic	0·08	per cent
Table concentrate after barrel amalgamation:—		
Gold	0·05	oz./ton
Arsenic	5·37	per cent
Ratio of concentration on table	15 : 1.	

Ore Dressing and Metallurgical Investigation No. 522

GOLD ORE FROM MASKWA GOLD MINES, CENTRAL MANITOBA AREA

Shipment. A sample shipment of two bags of ore, approximate weight 278 pounds, was received on August 8, 1933, from J. A. McVicar, 700 Great West Permanent Building, Winnipeg, Man.

Characteristics. The sample consisted of glassy to grey vein quartz and a microscopic examination showed pyrite to be the only abundant metallic mineral in the ore. Small amounts of chalcopyrite, petzite $(Au, Ag)_2Te$, tetradymite, $Bi_2(Te, S)_3$, native gold, and "limonite" occur. Most of the gold and all of the petzite observed occur within the pyrite. Most of the pyrite is coarse-grained; the other metallic minerals are very fine-grained.

Experimental Tests

The ore was sampled by standard methods and assayed.

Gold	2·73 oz./ton
Silver	4·66 "

Amalgamation, cyanidation, blanket, and flotation tests were carried out to determine a suitable method for treating the ore.

AMALGAMATION AND CYANIDATION

Test No. 1

One thousand grammes minus 48-mesh material was barrel-amalgamated in a 1 : 1 pulp with 7·5 grammes mercury for one hour. The tailing was filtered and 500 grammes were reserved for a screen analysis. On the remainder two cyanide tests were run, one for 24 hours and one for 48 hours.

Screen Analysis of Amalgamation Tailing:

Mesh	Weight, per cent	Assays, oz./ton	
		Au	Ag
+ 65	4·5	1·635	3·42
+100	20·2		
+150	23·4	1·550	3·35
+200	6·6	1·415	3·23
−200	45·2	2·055	5·30
	100·0		

Calculated Assay of Amalgamation Tailing:

Gold.. 1·79 oz./ton
Silver.. 4·24 "

Recoveries by amalgamation: Au, 34·4 per cent; Ag, 9 per cent.

Results of Cyanidation of Tailing:

Time	Product	Assays, oz./ton		Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent	
		Au	Ag	KCN	CaO	KCN	CaO	Au	Ag
24 hours.........	−48	0·45	1·69	1·25	0·625	1·75	7·125	73·2	60·1
48 hours.........	−48	0·41	1·43	1·3	0·5	1·60	7·5	77·1	66·2

Overall Recovery of Gold and Silver:—

Gold...................... 34·4% + (77·1% of 65·6%) = 84·9%
Silver...................... 9·0% + (66·2% of 91%) = 69·2%

Test No. 2

This test was carried out on minus 100-mesh ore and was similar to Test No. 1.

Screen Test Amalgamation Tailing

Mesh	Weight, per cent
+100	0·05
+150	9·55
+200	9·05
−200	81·35
	100·00

Assay of Amalgamation Tailing:

Gold.. 1·64 oz./ton
Silver.. 4·13 "

Recoveries by amalgamation: Au, 39·9 per cent; Ag, 11·3 per cent.

Results of Cyanidation of Amalgamation Tailing:

Time	Product	Assays, oz./ton		Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent	
		Au	Ag	KCN	CaO	KCN	CaO	Au	Ag
24 hours.........	−100	0·25	1·03	1·2	0·575	1·89	7·275	84·7	75·0
48 hours.........	−100	0·25	0·91	1·1	0·475	2·19	7·575	84·7	77·9

Overall Recovery of Gold and Silver:—

Gold......................39·9% + (84·7% of 60·1%) = 90·8%
Silver......................11·3% + (77·9% of 88·7%) = 80·4%

The results of these tests show that a better recovery is made on the finer product. But the recovery of free gold by amalgamation is low, due, no doubt, to the close association of the gold with the grains of pyrite.

CYANIDATION

Tests Nos. 3 and 4 embraced a series of cyanidation tests on sized samples of ore. Samples weighing 200 grammes were agitated in bottles in a pulp ratio of 1 : 3 and a cyanide concentration of 1 pound KCN per ton.

Test No. 3

24-hour agitation

Product	Assays, oz./ton		Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent	
	Au	Ag	KCN	CaO	KCN	CaO	Au	Ag
− 48+100	0·445	1·83	1·55	0·775	0·84	6·68	83·7	60·7
−100+150	0·310	1·28	1·45	0·60	1·14	7·20	88·6	72·5
−150+200	0·215	1·27	1·45	0·55	1·14	7·35	92·1	72·7
−200	0·190	1·22	1·4	0·575	1·29	7·28	93·0	73·8

Test No. 4

48-hour agitation

Product	Assays, oz./ton		Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent	
	Au	Ag	KCN	CaO	KCN	CaO	Au	Ag
− 48+100	0·38	1·42	1·6	0·55	0·69	7·35	86·0	69·5
−100+150	0·145	0·81	1·5	0·45	0·99	7·65	94·6	82·6
−150+200	0·215	0·95	1·5	0·45	0·99	7·65	92·1	79·6
−200	0·155	0·875	1·35	0·425	1·44	7·73	94·3	81·2

The results indicate that good recoveries are made by cyanidation of the raw ore, but that fine grinding and time of agitation are important factors in treating this ore. The tailings, however, are high.

BLANKET CONCENTRATION AND FLOTATION

Test No. 5

This test was a blanket concentration in which the tailing was reground and floated.

One thousand grammes of −14-mesh ore was ground in a 1 : 1 pulp for 15 minutes and then run over a corduroy blanket. The blanket tailing was reground for 15 minutes with 2 pounds per ton soda ash and 0·4 pound per ton potassium xanthate and then floated using 0·28 pound per ton Aerofloat No. 25.

The results of these tests are tabulated below:—

Product	Weight, per cent	Assays, oz./ton		Distribution of metals, per cent	
		Au	Ag	Au	Ag
Blanket concentrate	18·2	5·92	9·09	38·2	33·9
Blanket tailing	81·8	....	....	61·8	66·1

Product	Weight, per cent	Assays, oz./ton		Distribution of metals, per cent	
		Au	Ag	Au	Ag
Flotation concentrate	5·1	21·76	41·19	63·8	65·2
Flotation tailing	94·9	0·665	1·18	36·2	34·8

From the results obtained by blanket concentration it does not appear that this ore is suitable for such a form of concentration.

The overall recovery of gold in the blanket and flotation concentrates is only 77·6 per cent.

CYANIDATION AND FLOTATION

Test No. 6

This test consisted of cyaniding 2,000 grammes of ore, ground to pass through a 200-mesh screen, in a cyanide solution 1 pound KCN per ton, and a dilution of 2·5: 1. Agitation was carried out in two lots in bottles for 48 hours. The cyanide tailing was filtered and washed. A sample was cut for assay and the remainder reground for 15 minutes with 1 gramme soda ash and 0·2 gramme potassium xanthate and then floated using 2 drops pine oil as frothing reagent. The flotation concentrate was further ground for 30 minutes and then agitated in a 1 pound KCN per ton cyanide solution for 48 hours.

Cyanidation of Raw Ore:

Product	Assays, oz./ton		Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent	
	Au	Ag	KCN	CaO	KCN	CaO	Au	Ag
Tailing	0·165	0·855	0·95	·387	0·97	6·03	93·9	81·6

Flotation of Cyanide Tailing:

Product	Weight, per cent	Assays, oz./ton		Distribution of metals, per cent	
		Au	Ag	Au	Ag
Concentrate	3·1	1·258	14·45	23·6	52·4
Tailing	96·9	0·13	0·42	76·4	47·6

Cyanidation of Flotation Concentrate (Dilution 12 : 1):

Product	Assays, oz./ton		Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent	
	Au	Ag	KCN	CaO	KCN	CaO	Au	Ag
Tailing....................	0·07	0·36	1·4	0·75	7·2	31·0	94·4	97·5

Gold Recovery:

	Per cent
Recovery by cyanidation of raw ore....................................	93·9
Recoverable gold in flotation concentrate..............................	1·44
Recovery by cyanidation flotation concentrate..........................	1·36
Overall recovery, 93·9+1·36...	95·26

Silver Recovery:

	Per cent
Recovery by cyanidation raw ore.......................................	81·6
Recoverable silver in flotation concentrate............................	9·64
Recovery by cyanidation flotation concentrate..........................	9·4
Overall recovery, 81·6+9·4..	91·0

The recoveries are good by this method although the flotation tailing is still high.

CYCLE TESTS

Test No. 7

A cyanidation cycle test was tried in which fresh cyanide solution was added after each 24-hour period of agitation. Total time of the test was 71 hours. A lot of 1,000 grammes of −200-mesh ore was agitated in KCN solution 1 pound per ton and 5 pounds lime per ton at a dilution of 2·5: 1. After 24 hours the pulp was filtered and a fresh solution of cyanide 1 pound KCN per ton and 5 pounds lime per ton added to the ore and agitation continued for another 24 hours. The third cycle was carried out similarly to the above and continued for 23 hours.

Results:

Cycle	Product	Assays, oz./ton		Solution, lb./ton		Recoveries, per cent	
		Au	Ag	KCN	CaO	Au	Ag
1....................	....	....	....	0·6	0·10	....	....
2....................	....	....	....	1·5	0·75	....	....
3....................	Tailing	0·115	0·335	1·0	0·60	95·7	92·8

By bringing fresh cyanide solution in contact with the ore it was found possible to obtain a very good recovery of the gold and silver. The tailing still remains high. But considering the high gold content of the feed and the fact that tellurides are present, a 0·115 ounce per ton tailing from 2·73 ounces per ton feed represents a good recovery.

SUMMARY

The tests made on this ore indicate that a very satisfactory recovery of gold and silver may be obtained by cyanidation of the raw ore. Certain conditions, however, are necessary to obtain the highest results. Very fine grinding, practically 100 per cent through 200 mesh is essential. Contact of the ore with fresh cyanide solution at frequent intervals is another factor that tends to promote better dissolution of the gold and silver.

The presence of petzite, a gold-silver telluride, will tend to retard the action of cyanide upon the gold tied up in this mineral. The microscopic examination has shown that the bulk of the gold occurs within the pyrite. The blanket concentration test confirms this, as the small recoveries in the concentrates would indicate little free gold in the ore. The ore is not amenable to amalgamation.

Ore Dressing and Metallurgical Investigation No. 523

SILVER ORE FROM KOZAK MINE, GOUDREAU, ONTARIO

Shipment. The shipment, consisting of 50 pounds of ore, was submitted by J. H. Neill, representing Orecana Trust, Kozak mine, and was received August 31, 1933.

Characteristics of the Ore. The gangue is light in colour and consists chiefly of quartz with considerable carbonate and sericitic material.

In their order of abundance in the section examined the minerals are pyrite, sphalerite, galena, freibergite (?), native silver, chalcopyrite, dyscrasite (?), and pyrrhotite.

The native silver and sphalerite, galena, freibergite (?), dyscrasite (?), and chalcopyrite are usually very intimately associated, and recovery of the silver by flotation will entail floating all of the metallic minerals present. Perhaps the least important metallic mineral is chalcopyrite, some disseminated grains of which are isolated in the gangue and are apparently not associated with any silver.

Sampling and Analysis. After removing several specimens of ore for the microscopic examination, the sample was crushed to 14 mesh and by repeated sampling and grinding a representative sample was obtained for analysis, which gave the following:

Gold	0·13 oz./ton
Silver	5·19 "

EXPERIMENTAL TESTS

Three flotation tests were made on the ore using the same reagents but varying the time of grinding.

A representative sample of ore was used for each charge and ground in a pulp dilution of 2 : 1. Soda ash and ethyl xanthate were added to the mill at the rate of 3 pounds per ton and 0·1 pound per ton respectively.

Pine oil was used in the cell at the rate of 0·05 pound per ton.

A screen test was made on the tailing from each test to determine the amount of grinding. One concentrate was assayed for copper, lead, and zinc, and all concentrates for gold and silver.

Results of Flotation Tests:

Test No.	Products	Weight, per cent	Grind, per cent −200	Assay, oz./ton		Distribution of metals, per cent		Ratio of concentration
				Au	Ag	Au	Ag	
1	Feed	100·00	....	0·14	5·36	100·00	100·00	
	Concentrate	3·15	....	3·72	149·38	85·80	87·70	31·7 : 1
	Tailing	96·85	98·5	0·02	0·68	14·20	12·30	
2	Feed	100·00	....	0·13	5·33	100·00	100·00	
	Concentrate	3·00	....	4·15	163·64	92·80	92·20	33·3 1
	Tailing	97·00	99·4	0·01	0·43	7·20	7·80	
3	Feed	100·00	....	0·13	5·29	100·00	100·00	
	Concentrate	3·20	....	3·84	154·10	92·70	93·20	31·3 : 1
	Tailing	96·80	99·9	0·01	0·37	7·30	6·80	

Flotation Concentrate from Test No. 2

Assay:

Gold	4·16 oz./ton
Silver	163·64 "
Copper	0·92 per cent
Lead	4·84 "
Zinc	9·25 "

SUMMARY AND CONCLUSIONS

The ore was very easy to crush and the flotation concentrate was made without difficulty. The amount of acid-forming salts in the ore was low. The consumption of soda ash to maintain a slightly alkaline pulp was reasonable.

Practically all the gold was recovered in the concentrate and the amount of silver lost in the tailing averages about 0·5 ounce. The ratio of concentration is 33 : 1.

Ore Dressing and Metallurgical Investigation No. 524

TAILING FROM THE BOSTON-RICHARDSON MINE, NOVA SCOTIA

Shipment. A shipment of about 25 pounds, consisting of 29 small samples, was received August 31, 1933, from Jas. G. MacGregor, 605 Northern Ontario Building, Toronto, Ont.

Character. The samples consisted of tailing taken from a tailing dump near the old mill at the Boston-Richardson property. An examination by the microscope showed that nearly all the mineral was free. The sulphides consisted principally of arsenopyrite with a little pyrite and were slightly tarnished from surface oxidation.

An assay showed gold 0·11 ounce per ton, arsenic 2·11 per cent.

A screen analysis of the crude tailing was made and is shown in the following table.

Mesh	Weight, per cent	Gold, oz./ton
+ 35	10·0	0·03
− 35+ 48	15·4	0·015
− 48+ 65	17·9	0·06
− 65+100	17·1	0·08
−100+150	12·0	0·135
−150+200	5·0	0·137
−200	22·6	0·21
Total	100·0	0·11

It can be seen from this analysis that most of the gold-bearing arsenopyrite is in the finer sizes. No free gold was observed in any of the screen sizes.

Experimental Tests

The amount of material available for test work was small. Tests consisted first of a table concentration test and then of a series of flotation tests on the material graded to various sizes. The flotation tailings in every case were re-treated on a table to save any pyrite not recovered by flotation. This was necessary owing to the tarnished condition of the sulphides.

TABLING

A lot of 2,000 grammes of ore was fed to a small laboratory Wilfley table.

Results:

Product	Weight, per cent	Assays		Distribution of metals, per cent	
		Au oz./ton	As per cent	Au	As
Concentrate	7·15	0·86	22·88	59·5	22·6
Tailing	92·85	0·045	0·5	40·5	22·4
Total	100·00	0·10	2·11	100·0	100·0

It can be seen that straight table concentration will not make a high recovery.

FLOTATION AND TABLE CONCENTRATION OF FLOTATION TAILING

Test No. 1

In this test the crude sand tailing was ground for a short time in a ball mill, the object being to brighten the sulphide rather than to grind the sand.

Reagents:

To Ball Mill:

Soda ash.......... 2·0 lb./ton
Barrett No. 4.......... 0·1 "

To Flotation Cell:

Potassium ethyl xanthate.......... 0·2 lb./ton
Pine oil.......... 0·08 "
Copper sulphate.......... 0·05 "

Results:

Product	Weight, per cent	Assays		Distribution of metals, per cent	
		Au oz./ton	As per cent	Au	As
Flotation concentrate	5·5	1·26	22·55	75·9	64·0
Flotation tailing	94·5	0·025	....	....	....
Table concentrate	6·6	0·20	6·69	14·5	20·9
Table tailing	87·9	0·01	....	9·6	15·1
Total	100·0	....	....	100·0	100·0

Screen Analysis:

Mesh	Weight, per cent	Assay, Au oz./ton
+ 48	2·4	0·01
− 48+ 65	9·0	
− 65+100	21·8	0·02
−100+150	18·9	0·02
−150+200	7·1	0·035
−200	40·8	0·025
Total	100·0	0·025

The screen analysis of the flotation tailing shows the necessity of using tables in order to make high recovery. It should be kept in mind that the microscopic examination of the tailing shows that the sulphides are free. The reason they did not float was probably due to two causes: first, that arsenopyrite is always difficult to float and, second, that there is considerable tarnish on the surface of the sulphides. The overall recovery of the gold is 90·4 per cent and the ratio of concentration, 8·27: 1. Without doubt the grade of the table concentrate could be raised without causing any additional loss, and it would be safe to assume a ratio of concentration of 10 : 1 in practice.

Test No. 2

The tailing was ground somewhat finer as shown by the accompanying screen analysis.

The reagents used in this test were the same as in Test No. 1.

Product	Weight, per cent	Assays		Distribution of metals, per cent	
		Au oz./ton	As per cent	Au	As
Flotation concentrate	8·9	1·0	20·21	86·4	
Flotation tailing	91·1	0·02			
Table concentrate	2·94	0·17	2·34	8·7	
Table tailing	88·16	0·01		4·9	
Total	100·0			100·0	

Total recovery on combined concentrate, 95·1 per cent.
Ratio of concentration, 8·5 : 1.

Screen Analysis:

Mesh	Weight, per cent	Assay, Au oz./ton
+ 48 / − 48+ 65	0·7	
− 65+100	4·1	0·01
−100+150	12·9	0·015
−150+200	11·3	0·015
−200	71·0	0·02
Total	100·0	0·02

CYANIDATION OF FLOTATION AND TABLE CONCENTRATE

Two series of cyanide tests were made on mixed flotation and table concentrate.

The feed assay of the samples cyanided were:—

Gold .... 0·846 oz./ton

Test No. 1

The concentrate was reground in a ball mill to approximately 80 per cent −200 mesh, filtered and cyanided for 72 hours by agitation.

Results:

Feed assay	0·846 oz./ton
Tailing assay	0·03 "
Extraction	96·5 per cent
Cyanide consumption	13·5 lb./ton of ore
Lime consumption	27·1 "

Tests Nos. 2 and 3

Some of the concentrate was given a dead roast. During the first period of the roast the temperature was kept low in order that a minimum of arsenates would be formed and a better elimination of arsenic be effected.

Results:

Test No.	Time, hours	Tailing assay, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
2	48	0·025	97·0	0·96	27·3
3	48	0·025	97·0	0·62	26·8

The cyanide tests show that the flotation and table concentrate can be cyanided without roasting.

SUMMARY

Table concentration of the tailing will not recover over 60 per cent of the gold. Flotation followed by table concentration will recover 90 to 95 per cent of the gold. In order to obtain these results the tailing should be passed through a ball mill, not so much for the purpose of regrinding but in order to brighten the sulphides so that they float more readily. The flotation and table concentrates cyanide readily with high extraction of the gold without roasting. In practice the ratio of concentration will be almost 10 : 1. The overall recovery will be about 90 per cent.

CONCLUSIONS

The flow-sheet recommended for the treatment of this tailing would consist of a small ball mill, a surge tank, flotation machines of the mechanical type, and concentrating tables. The flotation and table concentrate to be reground in water to 80 per cent −200 mesh, filtered, and repulped in cyanide solution, and given at least 48 hours' treatment.

Ore Dressing and Metallurgical Investigation No. 525

GOLD ORE FROM THE BEARDMORE MINE OF NORTHERN EMPIRE MINES, LTD., JELLICOE, ONT.

Shipment. A shipment of 32 sacks of ore, net weight 3,234 pounds, was received August 21, 1933, from Robert J. Hendricks on instructions from Fred Searls, Jr., President, Northern Empire Mines Company, Ltd., 14 Wall Street, New York, N.Y.

Characteristics of the Ore. The gangue material is chiefly a light greenish grey fine-textured rock, rich in chlorite, and showing an indistinct schistosity in some portions. Patches and stringers of glassy vein quartz associated with considerable carbonate are present.

The metallic minerals in their order of abundance are, pyrrhotite, arsenopyrite, pyrite, sphalerite, chalcopyrite, and native gold.

All of the sulphides tend to be very intimately associated, particularly in the larger aggregates of grains, but a very considerable portion of their total volume occurs as tiny irregular grains scattered throughout both the quartz and the chloritic gangues.

An average assay of the ore was as follows:—

Gold	0·505	oz./ton
Silver	0·12	"
Iron	5·10	per cent
Sulphur	0·80	"
Arsenic	0·18	"
Insoluble	73·65	"

Experimental Tests

A series of small-scale tests was made on the ore, consisting of amalgamation, cyanidation, and flotation tests on samples of the ore both wet and dry ground.

A series of larger scale mill runs, using a unit of 100 pounds per hour capacity, was also made. The flow-sheet was altered from run to run, details of which will be given later with the results of the tests.

By amalgamation with the ore ground dry to roughly 90 per cent minus 200 mesh, a recovery of 70 per cent of the gold was obtained. With the ore ground wet to the same size, recovery by amalgamation increased to 87 per cent. The ore responded remarkably well to cyanidation, a tailing assaying 0·01 ounce per ton in gold being obtained in 24 hours from ore dry ground through 48 mesh and 76·5 per cent through 200 mesh. With finer grinding this tailing assay was reduced to 0·005 ounce per ton in gold.

Amalgamation followed by flotation produced a flotation tailing assaying 0·015 ounce per ton in gold with the ore ground to 80·5 per cent minus 200 mesh, 85·9 per cent of the gold being recovered as amalgam and 11·2 per cent of it in the flotation concentrate, which assayed 1·34 ounces per ton in gold and represents a ratio of concentration of 23·8 : 1. With finer grinding, recovery by amalgamation was slightly increased with a corresponding decrease in recovery in the flotation concentrate. The net recovery in both cases was exactly the same, 97·1 per cent.

AMALGAMATION

Test Nos. 1 to 4

In this series of tests samples of four lots of the ore, ground dry through 48-, 100-, 150-, and 250-mesh screens, were amalgamated with mercury for thirty minutes in a jar mill. The amalgamation tailings were filtered, washed, and assayed for gold.

Summary:

Feed sample, Au 0·505 oz./ton.

Test No.	Mesh	Tailing assay, Au oz./ton	Recovery, per cent
1	−48	0·175	65·3
2	−100	0·155	69·3
3	−150	0·23	54·5
4	−200	0·22	56·4

Screen tests showed the grinding to be as follows:—

All through	Weight, per cent +65	Weight, per cent −65+100	Weight, per cent −100+150	Weight, per cent −150+200	Weight, per cent −200	Total
48 mesh	0·3	2·1	12·6	8·2	76·8	100·0
100 "			5·0	6·0	89·0	100·0
150 "				5·9	94·1	100·0
200 "					100·0	100·0

CYANIDATION

Tests Nos. 5 to 12

In this series of tests samples of the same four lots of ore as were used for Tests Nos. 1 to 4 were agitated in cyanide solution, 2·0 pounds per ton KCN, for periods of 24 and 48 hours. The cyanide tailings were filtered, washed, and assayed for gold.

Summary:

Feed sample, Au 0·505 oz./ton.

Test No.	Mesh	Period of agitation, hours	Tailing assay, Au oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
					KCN	CaO
5	−48	24	0·01	98·0	0·6	4·8
6	−100	24	0·005	99·0	0·9	5·1
7	−150	24	0·005	99·0	0·9	5·1
8	−200	24	0·005	99·0	0·9	5·2
9	−48	48	0·01	98·0	0·6	5·25
10	−100	48	0·005	99·0	0·9	5·25
11	−150	48	0·01	98·0	0·9	5·25
12	−200	48	0·01	98·0	0·9	5·25

AMALGAMATION AND FLOTATION

Test No. 14

In this test 2,000 grammes of the ore at −14 mesh was ground in a ball mill for 20 minutes. The pulp was then amalgamated with mercury for 30 minutes at 1 : 1 density in a jar mill and the amalgamation tailing floated.

Charge to Ball Mill:

Ore.......................................2,000 grammes
Water.......................................1,500 c.c.
Na_2CO_3....................................... 2·0 lb./ton

Reagents to Cell:

Potassium amyl xanthate....................................... 0·10 lb./ton
Pine oil....................................... 0·05 "

The flotation concentrate and tailing were filtered, washed, and assayed for gold. The amalgamation tailing assay was calculated from these.

Summary:

Feed to amalgamation, Au 0·505 oz./ton.

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Flotation concentrate	4·2	1·34	11·2
Flotation tailing	95·8	0·015	2·9
Amalgamation tailing (cal.)	100·0	0·071	14·1

Recovery by amalgamation.......................................85·9 per cent
Recovery in flotation concentrate.......................................11·2 "

Net recovery.......................................97·1 "

A screen test showed the flotation tailing to be 80·5 per cent minus 200 mesh.

Test No. 15.

This test was a duplicate of Test No. 14, except that ore was ground for 30 minutes instead of 20 minutes.

Summary:

Feed to amalgamation, Au 0·505 oz./ton.

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Flotation concentrate	4·1	1·25	10·0
Flotation tailing	95·9	0·015	2·9
Amalgamation tailing (cal.)	100·0	0·065	12·9

Recovery by amalgamation	87·1 per cent
Recovery in flotation concentrate	10·0 "
Net recovery	97·1 "

HYDRAULIC CLASSIFICATION

Test No. 17

In this test 1,000 grammes of the ore at −14 mesh was ground for 20 minutes in a ball mill and the pulp passed through a hydraulic classifier where the gold and heavy minerals were allowed to settle out against a slowly rising current of water.

Summary:

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Classifier oversize	2·37	15·09	71·6
Classifier overflow	97·63	0·145	28·4
Feed (cal.)	100·00	0·50	100·0

Mill Runs

A series of mill runs was made on this ore, using a unit of one hundred pounds per hour capacity, in order to get some idea of the results to be expected under operating conditions. When the series of runs was finished the rod mill, classifier, and conditioning tank were cleaned out and the cleaning from each was weighed and assayed separately. Samples of the various products in the flow-sheets were taken at regular intervals.

The total feed to the rod mill during the series of runs was 2,950 pounds of ore at −14 mesh assaying on an average 0·505 ounce per ton in gold.

Run No. 1

The ore at minus 14 mesh was fed into a 12- by 24-inch rod mill containing 276 pounds of steel rods. The mill discharged onto an amalgamation plate, the tailing from which was pumped into a conditioning tank and overflowed from there to a battery of six flotation cells. The conditioning tank overflow was fed into cell No. 2 and flowed progressively over to No 6, where the tailing was discharged. The rougher concentrate taken from cells Nos. 2 to 6 was returned to cell No. 1, from which a clean concentrate was taken and the cleaner tailing flowed into cell No. 2, where it united with the conditioning tank overflow.

Reagents:

Na_2CO_3	2·0 lb./ton
Potassium amyl xanthate	0·10 "
Pine oil	0·08 "

Pine oil was discontinued and cresylic acid 0·5 pound per ton used in its place shortly after the run started.

Assays:

	Au, oz./ton
Rod mill feed	0·55
Rod mill discharge	0·55
Amalgamation tailing	0·55
Flotation tailing	0·03
Flotation concentrate	6·71

Results:

Recovery by plate amalgamation	Nil
Recovery in flotation concentrate	94·9 per cent
Ratio of concentration	13 : 1

A screen test of the rod mill discharge showed the grinding to be 64·8 per cent minus 200 mesh with a feed rate of 105 pounds per hour.

Run No. 2

This was an uninterrupted continuation of Run No. 1 and was carried out under the same conditions. The higher ratio of concentration in this test as compared with No. 1 is no doubt due to the use of cresylic acid in place of pine oil.

Assays:

	Au, oz./ton
Rod mill feed	0·49
Rod mill discharge	0·54
Amalgamation tailing	0·54
Flotation tailing	0·03
Flotation concentrate	9·43

Results:

Recovery by amalgamation	Nil
Recovery in flotation concentrate	94·7 per cent
Ratio of concentration	18·4 : 1

A screen test of the rod mill discharge showed the grinding to be 65·5 per cent minus 200 mesh with a feed rate of 105 pounds per hour.

Run No. 3

In this test corduroy blankets were placed on the lower half of the amalgamation plate, the rod mill discharging on the upper half of it. The blanket tailing was floated with the same reagent combination as was used in Nos. 1 and 2.

Assays:	Au, oz./ton
Rod mill feed	0·51
Rod mill discharge	0·53
Blanket tailing	0·31
Blanket concentrate	38·87
Flotation tailing	0·03
Flotation concentrate	8·86

Results:	Per cent
Recovery by amalgamation and blanketing	39·5
Recovery in flotation concentrate	54·8
Net recovery	94·3
Ratio of concentration by blanketing	192·8 : 1
Ratio of concentration by flotation	31·5 : 1

A screen test of the rod mill discharge showed the grinding to be 69 per cent minus 200 mesh, with a feed rate of 97 pounds per hour.

Run No. 4

This was a duplicate of Run No. 3 except that the feed rate was reduced to 79 pounds per hour.

Assays:	Au, oz./ton
Rod mill feed	0·475
Rod mill discharge	0·55
Blanket tailing	0·19
Blanket concentrate	66·17
Flotation tailing	0·03
Flotation concentrate	9·14

Results:	Per cent
Recovery by amalgamation and blanketing	65·6
Recovery in flotation concentrate	29·1
Net recovery	94·7
Ratio of concentration by blanketing	183·3 : 1
Ratio of concentration by flotation	56·9 : 1

A screen test of rod mill discharge showed the grinding to be 71·5 per cent minus 200 mesh.

Run No. 5

In this run blankets replaced amalgamation completely. The rod mill pulp was discharged directly to the blankets and the blanket tailing was floated with the same reagent combination as was used in the previous test.

Assays:	Au, oz./ton
Rod mill feed	0·525
Rod mill discharge	0·56
Blanket tailing	0·32
Blanket concentrate	71·12
Flotation tailing	0·025
Flotation concentrate	8·04

Results:

	Per cent
Recovery in blanket concentrate	43·0
Recovery in flotation concentrate	52·7
Overall recovery	95·7
Ratio of concentration by blanketing	295 : 1
Ratio of concentration by flotation	27·2 : 1

Run No. 6

In this run the rod mill pulp discharged into a hydraulic trap. The trap overflow went to a classifier the overflow from which was blanketed and the oversize returned to the rod mill for regrinding. The blanket tailing was floated still using the same reagent combination.

Assays:

	Au, oz./ton
Rod mill feed	0·49
Rod mill discharge	0·57
Trap overflow	0·22
Trap cleaning	17·10
Blanket tailing	0·11
Blanket concentrate	3·72
Flotation tailing	0·02
Flotation concentrate	7·18

Results:

	Total gold, per cent
Recovery in hydraulic trap	62·2
Recovery in blanket concentrate	3·3
Recovery in flotation concentrate	14·4
Overall recovery	79·9
Retained in classifier	16·9
Loss in flotation tailing	3·2
Ratio of concentration in trap	48·2 : 1
Ratio of concentration by blanketing	200 : 1
Ratio of concentration by flotation	79·6 : 1

Samples of the trap cleaning and the blanket concentrate from this run were amalgamated and then cyanided with and without regrinding. Samples of the flotation concentrate were cyanided.

The trap cleaning assayed 17·10 ounce per ton in gold. Amalgamation, regrinding and 48 hours' agitation in cyanide solution produced a tailing assaying 0·70 ounce per ton in gold.

This represents a recovery of 95·9 per cent of the gold in the trap cleaning, or 59·7 per cent of the total gold in the ore.

The blanket concentrate after the same treatment assayed 0·11 ounce per ton in gold. This represents a recovery of 97·0 per cent of the gold in the blanket concentrate or 3·2 per cent of the total gold in the ore.

The cyanide tailing from the flotation concentrate after 48 hours' agitation assayed 1·01 ounces per ton in gold representing a recovery of 85·9 per cent of the gold in the flotation concentrate, or 12·4 per cent of the total gold in the ore.

The net recovery, therefore, is $59 \cdot 7 + 3 \cdot 2 + 12 \cdot 4 = 75 \cdot 3$ per cent.

The classifier, which was used for this test only, was found to contain 16·9 per cent of the total gold contained in the ore treated during this test. Eighty-four per cent of this gold was recoverable by barrel amalgamation, a tailing assaying 0·21 ounce per ton in gold being left.

In the foregoing tests where amalgamation, blanketing, or trapping followed the rod mill directly, the recoveries credited to such operations were calculated on the assumption that no gold had accumulated in the rod mill. However, when the runs were all finished and the mill cleaned out it was found that 4·1 per cent of the total gold fed to the rod mill had accumulated there. The accumulation of this gold, however, may not have been uniform throughout the series of runs, because the classifier, which was used during the last run only, may have effected considerable change in the nature of the contents of the rod mill.

Recovery in flotation concentrate was calculated using the feed to the conditioning tank as the feed to flotation. Approximately 4·0 per cent of the gold that went to the conditioning tank accumulated there. Owing to the wide variation in the assay of this product from test to test, it is impossible to arrive at a correction factor that would suit all cases. It is indeed probable that most of this gold accumulated in the conditioning tank during Runs Nos. 1 and 2 when the feed to it was highest in gold content. These facts must, therefore, be borne in mind and reasonable consideration be given to them in each test.

SUMMARY AND CONCLUSIONS

Straight cyanidation seems to be the simplest and most efficient method that can be used to treat this ore. As may be seen from Test No. 5, 98 per cent of the gold can be extracted in 24 hours by grinding the ore all through 48 mesh and 76·8 per cent through 200 mesh.

Good recovery was obtained by amalgamation in the barrel amalgamation tests, but when plates were used in the larger scale mill runs no recovery at all was obtained. Recovery might be improved by using a stamp mill with inside amalgamation plates.

In Mill Run No. 6 a hydraulic trap recovered 62 per cent of the gold from the rod mill discharge with a ratio of concentration of roughly 48 : 1. This step is to be strongly recommended if rod mills or ball mills are used.

Corduroy blankets also proved efficacious in the treatment of this ore, 65·6 per cent of the gold being recovered in this way in Run No. 4 in an extremely high-grade blanket concentrate with a ratio of concentration of 183·3 : 1.

Although from a metallurgical point of view straight cyanidation is the process to be recommended, local conditions may favour another method. The results of the mill runs show that at no time did the gold content in the flotation tailings show any tendency to rise, but remained either constant or showed a definite falling off as in Runs Nos. 5 and 6 when the coarse gold was more thoroughly removed by blankets and traps before flotation.

Ore Dressing and Metallurgical Investigation No. 526

LEAD ORE FROM GODFREY, ONTARIO

Shipment. A sample shipment of lead ore weighing 100 pounds was received on August 3, 1933, from the Dominion Mining and Smelting Company, Godfrey, Ontario. The property from which the ore sample was obtained is situated on the west half of lot 18, concession 7, Bedford township, Frontenac county, Ontario.

Characteristics of the Ore. The ore consisted of crystals of galena up to ½ inch in size, in a gangue of calcite. Associated with the galena is a small amount of sphalerite

Sampling and Analysis. The ore sample was crushed to 14 mesh and by repeated sampling and grinding a representative sample was obtained for analysis.

Gold	Nil
Silver	0·16 oz./ton
Copper	0·04 per cent
Zinc	0·60 "
Lead	12·64 "

EXPERIMENTAL TESTS

Two methods are applicable. First: Gravity concentration on jigs and tables of a sized and classified feed. This was the method practised at the Kingdon Mining and Smelting Company, Galetta, Ontario, on a similar type of ore. A coarse high-grade lead concentrate, suitable for reduction to pig lead on a Newman hearth, is produced. Second: Flotation concentration of the ore, ground to about 48 mesh. This is the simplest method, but for reduction the concentrate requires roasting or sintering for the lead blast furnace.

GRAVITY CONCENTRATION

There was not sufficient of the ore to conduct jig tests, so that only table tests were run on a sized feed. About 50 pounds of the ore crushed to −14 mesh was sized as follows:—

Mesh	Weight, lb.
−14+28	20·0
−28+65	16·0
−65	14·5

Each size was run separately over a Wilfley table and the products, weighed, sampled, and assayed. The results were as follows:—

—14+28 mesh:

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Ag oz./ton	Pb per cent	Zn per cent	Ag	Pb	Zn
Feed	100·00	....	12·00	0·53	....	100·0	100·00
Concentrate	13·48	1·29	86·22	0·57	....	96·9	14·41
Middling	5·08	....	7·27	4·81	....	3·1	45·85
Tailing	81·44	....	trace	0·26	....	....	39·74

—28+65 mesh:

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Ag oz./ton	Pb per cent	Zn per cent	Ag	Pb	Zn
Feed	100·00	....	13·00	0·625	....	100·0	100·00
Concentrate	15·20	1·00	83·05	1·34	....	97·1	32·64
Middling	3·07	....	8·80	6·24	....	2·1	30·72
Tailing	81·73	....	0·13	0·28	....	0·8	36·64

—65 mesh:

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Ag oz./ton	Pb per cent	Zn per cent	Ag	Pb	Zn
Feed	100·00	....	10·80	0·845	....	100·00	100·00
Concentrate	9·62	0·96	77·79	1·76	....	69·31	20·00
Middling	2·22	....	22·34	8·00	....	4·59	21·07
Sand tailing	29·88	....	1·55	0·75	....	4·29	26·50
Slime tailing	58·28	....	4·04	0·47	....	21·81	32·43

The tests show that a sand tailing can be discarded from the gravity concentration of the coarser sizes, but that the middling would require regrinding. The reground middling from the coarse sizes together with the slime could be concentrated by flotation with a high recovery of the lead. The lead loss in the minus 65-mesh size is considerable. In the above tests over 90 per cent of the lead was recovered in a concentrate assaying 83 per cent lead.

FLOTATION

A representative sample of the minus 14-mesh ore was used for flotation tests.

The ore was ground in a ball mill with soda ash at the rate of 2 pounds per ton and Aerofloat No. 25, at the rate of 0·06 pound per ton. By means of a screen test on the tailing, it was found to be 66 per cent minus 200 mesh.

Pine oil was used as a frother in the cell and was added at the rate of 0·05 pound per ton.

The concentrate was assayed for silver, lead, and zinc, and the tailing for lead and zinc.

Test No. 2 was similar to Test No. 1 except that no soda ash was added to the pulp.

Test No. 1

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Ag oz./ton	Pb per cent	Zn per cent	Ag	Pb	Zn
Feed	100·00	....	12·79	0·60	....	100·00	100·00
Concentrate	17·16	0·96	74·41	1·98	....	99·87	56·95
Tailing	82·84	....	0·02	0·31	....	0·13	43·05

Test No. 2

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Ag oz./ton	Pb per cent	Zn per cent	Ag	Pb	Zn
Feed	100·00	....	12·95	0·62	....	100·00	100·00
Concentrate	16·47	....	77·57	1·14	....	98·65	30·23
Tailing	83·53	....	0·21	0·52	....	1·35	69·77

A screen test of the flotation tailing was as follows:—

Mesh	Weight, per cent
+ 48	0·3
− 48+ 65	2·4
− 65+100	9·2
−100+150	15·9
−150+200	6·4
−200	65·8

A screen test of the flotation concentrate was as follows:—

Mesh	Weight, per cent
− 48+ 65	0·5
− 65+100	6·1
−100+150	20·0
−150+200	7·3
−200	66·1

CONCLUSIONS

Flotation of the ore is simple. A lead concentrate containing over 75 per cent lead can be produced with a recovery of 98 per cent of the lead content in the ore.

Ore Dressing and Metallurgical Investigation No. 527

GOLD ORE FROM DEMARA MINES, LIMITED,
BARRAUTE, QUEBEC

Shipment. A shipment consisting of 1,300 pounds of ore, contained in 20 bags, was received on July 31, 1933, from the Demara Mines, Limited, Barraute township, Quebec, with head office at 276 St. James St., Montreal.

Characteristics of the Ore. Microscopic examination of the ore showed it to consist of milky-white vein quartz through which are scattered grains and masses of sulphides, and fine grains of native gold. The sulphides are pyrite, chalcopyrite, and their oxidation products, limonite and covellite.

Experimental Tests

The entire shipment was crushed to ½ inch and sampled by standard methods; analysis showed the ore to contain 0·43 per cent copper, 0·02 per cent arsenic, 0·34 ounce gold, and 0·28 ounce silver per ton.

Amalgamation indicates that 70·8 per cent of the gold is freed at minus 48 mesh, and 75 per cent when the ore is ground 81·7 per cent minus 200 mesh.

Cyanidation of the raw ore, owing to the presence of copper, results in a high cyanide consumption.

Flotation recovers 84 per cent of the gold in a high-grade concentrate. Cyanidation of the flotation tailing reduces the gold to 0·01 ounce per ton, giving an overall recovery of 97·1 per cent.

AMALGAMATION AND CYANIDATION

Test No. 1

A sample of the ore ground minus 48 mesh was amalgamated. After removing amalgam, the tailing was cyanided with a 3·0 pound cyanide solution and sufficient lime to maintain protective alkalinity.

Results:

Feed	0·34	oz. gold/ton.
Amalgamation tailing	0·096	" "
Recovery	70·8	per cent.
24-hour cyanide tailing	0·02	oz./ton
Total recovery, amalgamation and cyanidation	94·1	per cent.
48-hour cyanide tailing	0·01	oz. gold/ton.
Total recovery	97·1	per cent.
Cyanide consumed	6·3	lb. KCN/ton.
Lime consumed	6·8	lb./ton.

Screen Analysis of Amalgamation Tailing:

Mesh	Weight, per cent	Assay, Au oz./ton
− 48+ 65	2·1	0·115
− 65+100	20·6	0·115
−100+150	21·6	0·115
−150+200	7·7	0·09
−200	48·0	0·08

The test shows that after amalgamable gold is removed the remainder can be extracted with cyanide.

Test No. 2

A similar test was made on a sample ground to 81·7 per cent minus 200 mesh.

Amalgamation recovers 75 per cent of the gold. Cyanidation of the amalgamation tailing, which had a gold content of 0·085 ounce per ton, reduces this to 0·01 ounce in 24 hours and to 0·005 ounce in 48 hours; recoveries were 97·1 per cent and 98·5 per cent. Eight pounds KCN per ton of ore was consumed.

CYANIDATION

Test No. 3

Samples of the ore ground to pass 48, 100, 150, and 200 mesh were cyanided 1 : 3 dilution with a 3·0 pound KCN solution for 48 hours. Nine pounds lime per ton was added for protective alkalinity.

Mesh	Agitation, hours	Feed Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
					KCN	CaO
− 48	24	0·34	0·185	45·6	5·3	5·8
− 48	48	"	0·04	88·2	7·2	7·5
−100	24	"	0·12	64·7	6·3	5·8
−100	48	"	0·03	91·2	8·7	8·1
−150	24	"	0·215	36·8	7·2	6·1
−150	48	"	0·04	88·2	10·2	7·5
−200	24	"	0·15	55·9	7·8	6·6
−200	48	"	0·045	86·8	10·5	8·1

FLOTATION

Test No. 4

A sample of the ore was ground in water with 6 pounds soda ash and 0·06 pound Aerofloat No. 25 per ton until 75 per cent passed 200 mesh and then floated with 0·10 pound sodium xanthate and 0·15 pound pine oil.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·0	0·39	100·0
Concentrate	5·0	6·54	84·1
Tailing	95·0	0·065	15·9

Flotation does not recover all the gold, 0·065 ounce gold per ton remaining in the tailing.

Test No. 5

In this test an attempt was made to produce a concentrate of small bulk containing most of the copper, leaving gold in the tailing for subsequent cyanidation.

The ore was ground in water with 4 pounds lime per ton until 90 per cent passed 200 mesh. Following the addition of 0·10 pound potassium xanthate and 0·14 pound pine oil, a concentrate was removed.

The flotation tailing was then cyanided 1 : 3 dilution with a 1 pound KCN solution and 3 pounds lime per ton.

Product	Weight, per cent	Assay		Distribution of metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Feed	100·00	0·43	0·34	100·0	100·0
Concentrate	3·27	12·13	8·85	93·2	84·5
Tailing	96·73	0·03	0·055	6·8	15·5

24-hour cyanide tailing	0·015	oz. gold/ton.
Total recovery, flotation and cyanidation	95·6	per cent.
48-hour cyanide tailing	0·01	oz. gold/ton.
Total recovery	97·1	per cent.
Cyanide consumed	1·8	lb./ton.

Flotation in a lime circuit does not depress any large amount of gold, 84·5 per cent being found in the copper concentrate.

Test No. 6

A sample of the ore ground dry to pass 65 mesh with 62 per cent through 200 mesh was amalgamated. The tailing was conditioned with 4 pounds soda ash and floated with 0·10 pound potassium xanthate and 0·06 pound pine oil per ton.

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Feed	100·0	0·34	100·0
Amalgamation tailing	....	0·12	
Amalgam	....	....	63·3
Flotation concentrate	4·1	2·34	28·2
Flotation tailing	95·9	0·03	8·5

Gold recovered by amalgamation and flotation, 91·5 per cent.

SUMMARY AND CONCLUSIONS

The ore contains enough copper to cause an excessive consumption of cyanide. Test No. 5 shows that 93·2 per cent of the copper can be recovered by flotation in a concentrate assaying 12·13 per cent copper and carrying 84·5 per cent of the gold. The gold content of the tailing is reduced from 0·055 ounce per ton to 0·01 ounce by cyanidation.

Tests Nos. 1 and 2 show that from 70 to 75 per cent of the gold is freed with ordinary grinding.

Flotation fails to make a high recovery, the coarser particles of gold apparently passing out in the tailing.

The treatment indicated is flotation in a lime circuit preceded by blankets or followed by blankets to catch any coarse particles of gold not floated. The tailing from this operation could then be dewatered and cyanided to secure maximum recovery.

The blanket concentrate should be barrel-amalgamated. The flotation concentrate could also be amalgamated and the residue from amalgamation shipped to a smelter.

The shipment on which this test work was made is partly oxidized, for this reason it does not float as readily as should be expected. Before any definite conclusions are made, fresh ore from below the zone of surface alteration should be investigated to determine the results obtained from the true mine ore.

Ore Dressing and Metallurgical Investigation No. 528

GOLD ORE FROM WEST CALEDONIA, N.S.

Shipment. A shipment of 53 pounds was received from Charles MacKenzie, Falmouth, Hants county, N.S., on August 8, 1933.

Characteristics of the Ore. The ore is a typical Nova Scotia gold-quartz ore containing free gold and a small amount of sulphides.

Sampling and Analysis. The ore was crushed minus 14 mesh and a representative sample cut out. By repeated sampling and grinding a feed sample was obtained which assayed 0·486 ounce gold per ton.

Experimental Tests

AMALGAMATION

Representative samples of the ore were crushed −28, −35, −48 mesh, two tests being run on each size. The ore was amalgamated with mercury and after separating the amalgam the tailing was assayed for gold.

The tailing from each test was tested on a set of standard screens. The individual tests are shown below.

−28-mesh ore:

Mesh	Test No. 1	Test No. 2
	Weight, per cent	Weight, per cent
− 28+ 35	3·45	1·50
− 35+ 48	18·15	3·75
− 48+ 65	16·35	21·00
− 65+100	13·25	16·85
−100+150	14·80	12·50
−150+200	2·95	4·60
−200	31·05	39·80

Test No.	Assay, Au oz./ton		Extraction, per cent
	Feed	Tailing	
1	0·486	0·115	76·34
2	0·486	0·125	74·28

—35-mesh ore:

Mesh	Test No. 3	Test No. 4
	Weight, per cent	Weight, per cent
— 35+ 48	7·6	2·15
— 48+ 65	20·0	6·15
— 65+100	17·6	23·15
—100+150	15·1	17·90
—150+200	2·9	5·70
—200	36·8	44·95

Test No.	Assay, Au oz./ton		Extraction, per cent
	Feed	Tailing	
3	0·486	0·07	85·60
4	0·486	0·125	74·28

—48-mesh ore:

Mesh	Test No. 5	Test No. 6
	Weight, per cent	Weight, per cent
— 48+ 65	6·9	1·4
— 65+100	21·8	10·6
—100+150	18·2	18·4
—150+200	6·2	8·3
—200	46·9	61·3

Test No.	Assay, Au oz./ton		Extraction, per cent
	Feed	Tailing	
5	0·486	0·14	71·19
6	0·486	0·13	73·25

Summary:

Test No.	Assay, Au oz./ton		Extraction, per cent	Mesh
	Feed	Tailing		
1	0·486	0·115	76·34	—28
2	0·486	0·125	74·28	—28
3	0·486	0·07	85·60	—35
4	0·486	0·125	74·28	—35
5	0·486	0·14	71·19	—48
6	0·486	0·13	73·25	—48

AMALGAMATION AND TABLING

A representative sample of minus 35-mesh ore was amalgamated. The amalgam was removed and the tailing sampled. The tailing was concentrated on a Wilfley table, and the concentrate was amalgamated with mercury. The amalgam and the tailing were assayed and the gold in the table concentrate calculated. The calculated results agree closely with the assay values. The total recovery is shown to be about 81 per cent.

Results:

Products	Weight, per cent	Assay, Au oz./ton	Extraction, per cent	Recovery, per cent
Feed	....	0·486	....	....
Amalgam	....	....	76·34	76·34
Amalgamation tailing	....	0·115	....	....
Table feed	100·0	0·115	....	....
Table concentrate	1·5	2·41	31·48	7·45
Table tailing	98·5	0·08	....	....

The table concentration recovered 31·48 per cent of the gold left after amalgamation (23·66 per cent), hence the recovery by table concentration was 23·66 × 31·48 = 7·45 per cent. The table concentrate was treated by barrel amalgamation and 70·95 per cent of the gold was recovered, or 70·95 × 7·45 = 5·29 per cent of the gold left after the first amalgamation. The overall recovery by amalgamation is 81·63 per cent.

Summary of Test:

Products	Weight, per cent	Assay, Au oz./ton	Extraction, per cent	Ratio of concentration	Recovery, per cent
Amalgamation of −35 Mesh Ore					
Feed	....	0·486	....	....	....
Amalgam	....	....	76·34	....	76·34
Amalgamation tailing	....	0·115	....	....	....
Tabling of Amalgamation Tailing					
Table feed	100·0	0·115	....	....	....
Table concentrate	1·5	2·41	31·48	66·67 : 1	7·45
Feed tailing	98·5	0·08	....	....	....
Barrel Amalgamation of Table Concentrate					
Table concentrate	....	2·41	....	....	....
Amalgam	....	....	70·95	....	5·29
Amalgamation tailing	....	0·70	....	....	....

Total overall recovery by amalgamation .... 76·34+5·29=81·63

CYANIDATION

Two tests were carried out to determine if the ore was amenable to treatment by cyanidation.

Two representative samples of −48-mesh ore were agitated in a solution of sodium cyanide equivalent in strength to 1 pound of potassium cyanide per ton of solution. Lime was added at the rate of 5 pounds per ton of ore to maintain a protective alkalinity in the pulp. The agitation was carried out for 24 and 48 hours.

The results of the test show that 98 per cent of the gold was extracted by cyanidation.

Results:

Test No.	Time, hours	Assay, Au oz./ton		Extraction, per cent	Reagents consumed, lb./ton ore	
		Feed	Tailing		KCN	CaO
1	24	0·486	0·01	97·9	0·39	4·25
2	48	0·486	0·01	97·9	0·69	4·10

This result shows that the ore is amenable to cyanidation.

SUMMARY

Amalgamation will recover about 75 per cent of the gold when the ore is crushed to 30 mesh.

Amalgamation followed by concentration on tables will recover an additional 7·45 per cent of the gold in a table concentrate. Approximately 1·5 tons of concentrate would be obtained for every 100 tons of ore milled. The table concentrate can be treated by barrel amalgamation and over 70·0 per cent of the gold contained can be recovered. An overall recovery of better than 80 per cent of the gold can be expected by this procedure. Straight cyanidation will extract more then 95 per cent of the gold when the ore is ground to 48 mesh.

CONCLUSIONS AND RECOMMENDATIONS

For a small mill of 25 tons a day capacity the method recommended for milling would be by stamps using inside and outside amalgamation. The tailing from the amalgam plates should be concentrated on tables. A grinding pan or amalgam barrel should be provided for the amalgamation of the table concentrate. The concentrate residue should be stocked for future treatment.

Ore Dressing and Metallurgical Investigation No. 529

GOLD ORE FROM KIRKLAND GOLD BELT MINES, KING KIRKLAND, ONT.

Shipment. A shipment of ore consisting of 6 bags, weight 550 pounds, was received on July 8, 1933, from Kirkland Gold Belt Mines, King Kirkland, Ont. The ore was submitted by Geo. W. Morris, 325 Jackson Building, Buffalo, N.Y.

Characteristics. Under the microscope the gangue is seen to consist of a fine-textured, dense, light greenish grey siliceous material containing indistinct grains and small patches of pink dolomite. This is traversed by a network of small veinlets of glassy quartz.

The metallic minerals are pyrite, chalcopyrite, galena, sphalerite, and magnetite.

Pyrite is by far the most abundant sulphide, and occurs as disseminated imperfectly-formed cubes and irregular grains. It commonly contains numerous tiny inclusions of gangue material, and may occasionally contain small irregular grains of galena, chalcopyrite, or sphalerite.

The remaining sulphides are rare and in general are found as sparsely disseminated individual grains.

The ore was ground to minus 14 mesh and a sample cut by standard methods. The assays of the feed sample show:—

Gold	0·33 oz./ton
Silver	0·60 "

Experimental Tests

Tests embracing cyanidation, flotation, amalgamation, and blanket concentration were run in order to determine a suitable method of treatment.

FLOTATION

Tests Nos. 1, 2, and 3 were small-scale flotation runs using charges of 1,000 grammes each. The minus 14-mesh ore was ground wet with soda ash and potassium xanthate for 30 minutes. The pulp carried 66 per cent solids.

Test No. 1

Charge to Mill:

Ore,—14 mesh	1000·0 grammes.
Water	500·0 c.c.
Soda ash	2·0 lb./ton
Potassium xanthate	0·2 "

To Cell:

Aerofloat No. 25 .... 0·35 lb./ton.

Results:

Product	Weight, per cent	Assays, Au, oz./ton	Distribution of gold, per cent
Concentrate	15·3	1·95	83·4
Tailing	84·7	0·07	16·6

Ratio of concentration, 6·5 : 1.

Screen Test on Flotation Tailing:

Mesh	Weight, per cent
+100	1·5
+150	5·9
+200	3·7
−200	88·9
	100·0

Tests Nos. 2 and 3

Charge to Mill:

Ore, −14 mesh .... 1000·0 grammes.
Water .... 500·0 "
Soda ash .... 2·0 lb./ton.
Potassium xanthate .... 0·4 "

To Cell:

Sodium Aerofloat .... 0·10 lb./ton.
Aerofloat No. 25 .... 0·28 "

Results:

Test No.	Product	Weight, per cent	Assays, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
2	Concentrate	14·3	2·04	85·0	7 : 1
	Tailing	85·7	0·06	15·0	
3	Concentrate	12·8	2·20	84·3	7·8 : 1
	Tailing	87·2	0·06	15·7	

A fair concentration of gold in the concentrate was obtained but the tailing was high.

CYANIDATION

Tests Nos. 4 and 5

A sample of ore was ground to four sizes and 200-gramme charges were agitated in bottles in 0·1 per cent KCN solution with a pulp ratio of 1 : 3.

Test No. 4, 24-hour agitation:

Product size	Assay of tailing	Recovery, per cent	Reagents consumed, lb./ton	
	Oz./ton		KCN	CaO
− 48	0·08	75·7	0·90	7·90
−100	0·05	84·8	0·45	8·35
−150	0·05	84·8	0·45	8·50
−200	0·04	87·8	0·90	8·80

Test No. 5, 48-hour agitation:

Product size	Assay of tailing	Recovery, per cent	Reagents consumed, lb./ton	
	Oz./ton		KCN	CaO
− 48	0·08	75·7	0·75	8·35
−100	0·055	83·3	0·90	8·80
−150	0·055	83·3	0·60	9·03
−200	0·045	86·3	0·60	8·65

Test No. 6

Test No. 6 was a cyanidation test on the combined flotation concentrates of Tests Nos. 1, 2, and 3. The concentrates were ground for 30 minutes and agitated for 48 hours in a bottle containing 0·1 per cent KCN solution with a pulp ratio of 3·97 : 1.

Results:

Product	Assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
Tailing	0·12	94·5	16·51	21·76

Cyanidation of the flotation concentrate shows a good recovery but the tailing is high, having a value of over 0·12 ounce per ton.

BLANKET CONCENTRATION

Test No. 7

Five hundred grammes of −14-mesh ore was ground in a pebble jar for 30 minutes and the pulp run over a corduroy blanket. The results showed a poor recovery of gold.

Product	Weight, per cent	Assays, oz./ton	Distribution of gold, per cent
Concentrate	9·3	1·09	28·8
Tailing	90·7	0·275	71·2

AMALGAMATION

Test No. 8

Five hundred grammes of −14-mesh ore was ground in a 2 : 1 pulp for 15 minutes and then barrel-amalgamated with 10 grammes of mercury and 0·25 gramme NaCN for 1 hour. The results indicate that almost 40 per cent of the gold present is not free gold.

Product	Weight, grammes	Assay, oz./ton	Recovery, per cent
Tailing	504·4	0·125	62·1

CLASSIFICATION

Test No. 9

Five hundred grammes −48 + 100-mesh ore was pulped and run through a hydraulic classifier. The object of the test was to find what recoveries might be expected in a trap in the grinding circuit. Three products were obtained: No. 1, material passing down the tube into the collecting bottle; No. 2, material held in suspension in the tube; and a tailing consisting of the overflow. The results with assays are tabulated below.

Product	Weight, per cent	Assays, oz./ton	Distribution of gold, per cent	
No. 1	3·5	1·011	10·4	27·3
No. 2	14·2	0·405	16·9	
Tailing	82·3	0·30	72·7	

CYANIDATION

Test No. 10

Two hundred grammes of flotation tailing (assaying 0·06 ounce per ton) was agitated in a bottle for 24 hours in a 1 pound per ton KCN solution with 5 pounds per ton lime (CaO) at a pulp density of 1 : 3.

Product	Assay, Au, oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
			KCN	CaO
Tailing	0·02	66·6	0·1	4·275

The results so far obtained indicate that an appreciable amount of the gold is very closely associated with the pyrite. Several tests were run with the object of verifying this assumption and if possible to effect higher extractions by cyanidation.

Test No. 11

This test consisted of grinding 1,000 grammes of minus 14-mesh ore for 30 minutes in 500 c.c. of solution equivalent in strength to 1 pound KCN per ton and 10 pounds of lime. After grinding, the pulp was transferred to a bottle and diluted to a pulp ratio of 2·5 : 1. The solution was titrated for KCN, and further KCN added to bring the cyanide strength up to 1 pound KCN per ton. Agitation was carried out for 24 hours. The cyanide tailing was then filtered and run over a laboratory Wilfley table. The results of the test and assays are as follows:—

Grinding and Cyanidation Results:

Product	Assay, oz./ton	Recovery, per cent	Final solution, lb./ton		Reagents consumed, lb./ton	
			KCN	CaO	KCN	CaO
Cyanide Tailing	0·051 (cal.)	84·5	0·90	0·975	0·55	7·56

Results of Tabling Cyanide Tailing:

Product	Weight, per cent	Assays, oz./ton	Distribution of gold, per cent
Cut 1	10·1	0·28	55·56
Cut 2	15·0	0·04	11·78
Middling	16·4	0·03	9·67
Slime	58·5	0·03	22·99

The results indicate that over 55 per cent of the gold in the cyanide tailing is associated with the sulphides, which represents roughly about 10 per cent of the material treated.

Test No. 12

This test consisted of grinding 1,000 grammes of −14-mesh ore in a 1 pound per ton KCN solution with 10 pounds per ton CaO for 30 minutes. The pulp was transferred to a bottle, diluted to pulp density of 1 : 2·5 and the cyanide strength made up to 1 pound per ton. Agitation was continued for 24 hours after which the cyanide tailing was filtered, washed, and a portion cut for assay and the balance reground for 15 minutes with 1 gramme soda ash and 0·2 gramme potassium xanthate. The pulp was then floated using 0·28 pound per ton Aerofloat No. 25. The concentrate was reground for 30 minutes and then cyanided for 48 hours. The flotation tailing was assayed for gold and a screen test made.

Results of Cyanidation on Raw Ore:

Product	Assay, Au oz./ton	Recovery, per cent	Final solution, lb./ton		Reagents consumed, lb./ton	
			KCN	CaO	KCN	CaO
Tailing.............	0·05	84·8	1·2	0·45	0·78	8·05

Results of Flotation of Cyanide Tails:

Product	Weight, per cent	Assays, oz./ton	Distribution of gold, per cent
Concentrate..	15·8	0·103	32·6
Tailing..	84·2	0·04	67·4

Screen Test on Flotation Tailing:

Mesh	Weight, per cent
+ 65..	0·0
+100..	0·1
+150..	0·3
+200..	1·0
−200..	98·6
	100·0

Results of Cyanidation on Flotation Concentrate:

Product	Weight, grammes	Assays, oz./ton	Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent
			KCN	CaO	KCN	CaO	
Tailing...............	131·5	0·07	0·9	1·4	3·34	39·4	32·0

Dilution ratio..	4·5 : 1.
Gold recovery by cyanidation....................................	84·8 per cent
Gold in flotation concentrate from cyanide tailing................	4·96 "
Recovery of gold by cyanidation from flotation concentrate.......	32·0 "
Overall recovery of gold = 84·8 + 32 per cent of 4·96 =..........	86·4 "

Test No. 13

This test consisted of cyaniding 3,000 grammes of raw ore in three lots of 1,000 grammes each. The combined tailings were then run over a small Wilfley table and the sulphide concentrate again ground and re-cyanided.

Three lots of 1,000 grammes −14 mesh were ground in a pebble jar, in a 2 : 1 pulp containing 0·46 pound NaCN and 10·0 pounds CaO per ton. After 30 minutes' grinding the pulp was transferred to bottles and further diluted to a dilution of 2·5 : 1. Sufficient cyanide was added to bring the concentration to 1 pound KCN per ton.

Results of Cyanidation on Raw Ore, 24-hour agitation:

Product	Weight, grammes	Assay, Au oz./ton	Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent
			KCN	CaO	KCN	CaO	
Tailing.........	Sample cut for assay 387·5	0·04	0·9	0·7	0·9	8·25	87·8

Screen Test on Cyanide Tailing:

Mesh	Weight, per cent
+100	0·1
+150	1·8
+200	3·2
−200	94·9
	100·0

The tailings were filtered and combined. A sample was cut for assay and screen test and the remainder run over a laboratory Wilfley table.

Results of Tabling Cyanide Tailing:

Product	Weight, per cent	Assay, oz./ton	Distribution of gold, per cent
Cut 1	13·4	0·119	39·8
Middling	16·0	0·04	16·0
Slime	70·6	0·025	44·2

The first cut (table concentrate) was reground for 30 minutes in a pulp to which 2 grammes lime was added. The pulp was filtered and transferred to a bottle and agitated in a cyanide solution, 1 pound KCN per ton, at a dilution of 2·4 : 1 for 48 hours.

Results of Cyanidation of Table Concentrate:

Product	Assay, Au oz./ton	Final solution, lb./ton		Reagents consumed, lb./ton		Re-covery, per cent
		KCN	CaO	KCN	CaO	
Tailing.............	0·08	0·9	0·45	1·20	18·92	32·7

Gold recovery by cyanidation of raw ore..................87·8 per cent

Recoverable gold in table concentrate.....................39·8 per cent of 12·2 per cent
= 4·86 per cent

Gold recovery by cyanidation table concentrate...........32·7 per cent of 4·86 per cent
= 1·49 per cent

Overall recovery.. = 87·8+1·49 = 89·29 per cent

SUMMARY

The results indicate that the gold is largely closely associated with the pyrite. Amalgamation is not satisfactory. Fine grinding, cyanidation, and tabling appear to offer the best method for recovery of the gold.

The ore should be ground to have 95 per cent pass through 200 mesh. Twenty-four hours' agitation in cyanide solution having 1 pound KCN per ton and a dilution of 2·5 : 1 gave 0·04 ounce per ton tailing and 87·8 per cent recovery of the gold. Tabling this cyanide tailing gave a pyrite concentrate of about 13 per cent by weight and a 0·025 ounce per ton tailing product. The table concentrate was reground and agitated in cyanide solution (1 pound KCN per ton) for 48 hours. An overall recovery of gold by this method gave 89·29 per cent. The average of the tailings from tabling and cyanidation tests was 0·035 ounce gold per ton.

Tabling appears to be more satisfactory than flotation for the concentration of the cyanidation tailing.

Ore Dressing and Metallurgical Investigation No. 530

COPPER-NICKEL ORE FROM THE CUNIPTAU MINES DEVELOPMENT CO., LTD., STRATHY TOWNSHIP, TEMAGAMI FOREST RESERVE, ONT.

Shipment. The shipment, consisting of 240 pounds of ore, was received on July 3, 1933. It was submitted by E. P. Muntz, Limited, Dundas, Ont.

Characteristics of the Ore. The ore is chiefly massive sulphides with a minor amount of gangue material. The gangue minerals were not determined, but are almost wholly represented by dark-coloured silicates.

The metallic minerals are chalcopyrite, sphalerite, pyrrhotite, pyrite, pentlandite, violarite, and millerite(?). These sulphides are mutually intimately associated in massive fine-grained aggregates. Pentlandite is the only prominent nickel mineral; this mineral has in some cases been altered to (or replaced by) violarite (supergene, $(NiFe)_3S_4$); millerite(?) is rarely present as small spikes in the pyrrhotite.

No mineral that could be identified as the bearer of the platinum group metals was observed in the polished sections.

Experimental Tests

The microscopic examination of the ore indicated that the association of the copper and nickel minerals was very intimate and that in order to free them the ore would have to be ground to a degree of fineness beyond the limits of economic operation. No attempt was, therefore, made to separate the copper and nickel.

The experimental work was confined to the production of a bulk concentrate containing the copper-nickel and precious metals, and the elimination of the gangue and barren sulphides in a tailing product.

FLOTATION

A series of flotation tests was made on the ore. A number of tests had to be run before the proper reagents and the amounts required could be determined. The ore was found to contain considerable acid-forming salts, which required neutralization with sodium carbonate before satisfactory flotation results could be obtained. The microscopic examination of the ore showed that the presence of these acid salts was due to a surface alteration by oxidization, indicated by the presence of the supergene mineral violarite.

It was found that by grinding the ore in water, filtering, and then using fresh water for flotation, that the amount of soda ash required could be reduced to 20 pounds per ton of ore (which is still an excessive amount).

The best result obtained is given in the following table. The reagents used to obtain these results were: soda ash, 20 pounds per ton; sodium xanthate, 0·3 pound per ton; and pine oil.

In making this test the ore was ground to 91·5 per cent minus 200 mesh.

The results of the test are shown below:—

Products	Weight, per cent	Assay					Distribution of metals, per cent				
		Oz./ton		Per cent							
		Au	Pt	Cu	Ni	Zn	Au	Pt	Cu	Ni	Zn
Feed................	100·00	0·055	0·625	7·14	2·88	5·30	100·0	100·0	100·0	100·0	100·0
Bulk concentrate......	61·25	0·05	0·95	11·50	3·87	7·28	54·8	91·5	98·7	82·4	84·1
Tailing..............	38·75	0·065	0·14	0·24	1·31	2·18	45·2	8·5	1·3	17·6	15·9

Ratio of concentration, 1 : 1·6

CONCLUSIONS

The sample received was oxidized, and this fact accounts for the rather poor results obtained.

Undoubtedly high recovery of all the metals would be obtained when fresh unoxidized ore is treated.

Ore Dressing and Metallurgical Investigation No. 531

GOLD ORE FROM ELBOW LAKE, CRANBERRY PORTAGE, MANITOBA

Shipment. A shipment of gold ore consisting of two lots, one a three-bag lot of 270 pounds, the other a one-bag lot of 80 pounds, was received on September 22, 1933. The samples were submitted by C. M. Bartram, 404-5 Bank of Hamilton Building, Toronto, Ont.

Characteristics. A microscopic examination was made on the three-bag lot. The gangue was found to consist chiefly of grey to white impure quartz with grey silicate minerals. Metallic minerals are disseminated erratically through this, and in the order of abundance are, pyrite, magnetite, chalcopyrite, pyrrhotite, and native gold. Pyrite is by far the most abundant metallic mineral. Native gold occurs as small grains in the pyrite, and in one instance, was observed in the quartz.

Experimental Tests

The shipment consisted of three bags of mine-run ore and one bag of quartz. Test work was carried out on the three-bag lot and comprised amalgamation and cyanidation tests.

Both lots were ground and sampled by standard methods and the feed from each assayed as follows:—

—	Three-bag lot, oz./ton	Quartz ore, oz./ton
Gold	0·155	1·075
Silver	0·21	0·08

AMALGAMATION

Test No. 1

A sample, 1,000 grammes, of −48-mesh ore was barrel-amalgamated with 7·5 c.c. mercury (10 per cent) in a 1 : 1 pulp for 1 hour.

Results:

Product	Assay, Au oz./ton	Recovery, per cent
Tailing	0·025	83·8

Screen Test of Tailing:

Mesh	Weight, per cent
+ 65	2·4
+100	12·05
+150	17·05
+200	15·4
−200	53·1
	100·00

Test No. 2

A sample, 1,000 grammes, of −100-mesh ore was amalgamated under the same conditions as Test No. 1.

Results:

Product	Assay, Au oz./ton	Recovery, per cent
Tailing	0·015	90·3

Screen Test of Tailing:

Mesh	Weight, per cent
+100	0
+150	5·4
+200	19·6
−200	75·0
	100·0

The amalgamation tests indicate that the bulk of the gold is free.

CYANIDATION

Test No. 3

Four 200-gramme lots of sized ore were agitated for 24 hours at a dilution of 3 : 1 in bottles in a cyanide solution of 1 pound KCN and 5 pounds CaO per ton.

Results:

Product	Assay, Au oz./ton	Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent
		KCN	CaO	KCN	CaO	
− 48	0·01	0·8	0·6	0·6	5·20	94·1
−100	0·01	0·8	0·42	0·6	5·74	94·1
−150	0·01	0·7	0·15	0·9	6·35	94·1
−200	0·005	0·7	0·27	0·9	6·19	96·7

SUMMARY

The cyanidation results show that the ore is readily treated by cyanidation, with good recovery, and giving a 0·005 ounce per ton tailing. The lime consumption averages around seven pounds per ton and the cyanide consumption less than a pound. Equally good results are obtained on −100-mesh ore as on 200-mesh ore.

The method of treatment recommended for this ore is the standard cyanide process, i.e. grinding in cyanide, agitation, and filtration.

Ore Dressing and Metallurgical Investigation No. 532

GOLD-BEARING COPPER ORE FROM GOGAMA, IN THE TOWNSHIP OF CHESTER, IN THE THREE DUCK AREA, SUDBURY MINING DISTRICT, ONT.

Shipment. A shipment consisting of 9 bags, approximate weight 920 pounds, was received September 11, 1933, from Mr. Fred Lawrence, No. 2 Kingston Road, Toronto, Ont.

The shipment consisted of two grades of ore, designated Lot No. 1 and Lot No. 2. The former weighed approximately 500 pounds, the latter 420 pounds. Each lot was ground to pass a 14-mesh screen and sampled by standard methods.

Characteristics:

Lot No. 1. A microscopic examination shows that the gangue is chiefly grey to white quartz with a very small amount of carbonate. The metallic minerals are chalcopyrite, pyrite, and small amounts of chalcocite, "limonite," pyrrhotite, and sphalerite, the two latter occurring only as traces. The existence of chalcocite and "limonite" indicate that the samples have undergone considerable surface oxidation.

The chalcopyrite occurs in granular masses, irregular stringers, and veins. Pyrite occurs in granular masses and disseminated grains.

Lot No. 2. This material is very similar to that of Lot No. 1. The gangue is coarsely granular, siliceous, and may contain a little feldspar. The metallic minerals are characteristically the same as in Lot No. 1, with the exception that no chalcocite was observed.

The feed samples assayed as follows:—

—	Lot No. 1	Lot No. 2
Copper	5·02 per cent	2·48 per cent
Iron	10·38 "	7·73 "
Gold	0·07 oz./ton	0·06 oz./ton
Silver	2·23 "	1·06 "

EXPERIMENTAL TESTS

Test work consisted in concentration of the gold and copper by flotation. The concentrate was analysed for the presence of platinum group metals.

Lot No. 1

Tests Nos. 3, 4, and 5

Samples, 1,000 grammes, of −14-mesh ore with 500 c.c. water, soda ash 4 pounds per ton and potassium ethyl xanthate 0·4 pound per ton were ground in a small pebble mill for 15 minutes. They were then floated using different oils as frothing reagents.

Test	Product	Weight, per cent	Assays				Distribution of metals, per cent			Flotation oil
			Au, oz./ton	Ag, oz./ton	Cu, per cent	Fe, per cent	Au	Ag	Cu	
3	Concentrate...	19·9	0·27	9·71	24·46	27·69	77·1	86·7	95·7	Shawinigan Chemical Co. No. 4, 5 drops.
	Tailing........	80·1	0·02	0·37	0·21	......	22·9	13·3	4·3	
		100·0								
4	Concentrate...	17·8	0·31	10·45	26·00	28·80	79·1	83·7	95·4	Shawinigan Chemical Co. No. 4A, 3 drops.
	Tailing........	82·2	0·02	0·44	0·27	......	20·9	16·3	4·6	
		100·0								
5	Concentrate...	19·3	0·278	10·13	24·72	28·70	76·9	87·6	95·9	Pine oil, 5 drops.
	Tailing........	80·7	0·02	0·34	0·25	......	23·1	12·4	4·1	
		100·0								

Ratio of Concentration:

Test No. 3 .... 5·02 : 1
Test No. 4 .... 5·62 : 1
Test No. 5 .... 5·18 : 1

Screen Tests on Tailings:

Test No. 3

Mesh	Weight, per cent
+ 48	0
+ 65	0·5
+100	4·4
+150	12·2
+200	10·8
−200	72·1
	100·0

Test No. 5

Mesh	Weight, per cent
+ 48	0
+ 65	0·1
+100	5·0
+150	12·3
+200	8·7
−200	73·9
	100·0

The results of Tests Nos. 3, 4, and 5 show a good recovery of copper and silver and a low gold tailing. The use of Shawinigan Chemical Co. No. 4A gave on these tests a higher grade concentrate than Oil No. 4 or pine oil.

Lot No. 2

Tests Nos. 6, 7, and 8

Test	Product	Weight, per cent	Assays				Distribution of metals, per cent			Flotation oil
			Au, oz./ton	Ag, oz./ton	Cu, per cent	Fe, per cent	Au	Ag	Cu	
6	Concentrate...	11·3	0·45	8·20	21·16	27·28	85·2	87·4	92·1	Pine oil, 6 drops.
	Tailing........	88·7	0·01	0·15	0·23	......	14·8	12·6	7·9	
7	Concentrate...	12·0	0·426	7·95	19·76	25·75	85·3	90·0	95·4	Shawinigan Chemical Co. No. 4, 4 drops.
	Tailing........	88·0	0·01	0·12	0·13	......	14·7	10·0	4·6	
8	Concentrate...	10·6	0·48	8·82	22·66	28·30	85·1	88·2	95·0	Shawinigan Chemical Co. No. 4A, 3 drops.
	Tailing........	89·4	0·01	0·14	0·14	......	14·9	11·8	5·0	

Ratio of Concentration:

Test No. 6..........	8·85 : 1
Test No. 7..........	8·34 : 1
Test No. 8..........	9·44 : 1

Screen Tests on Tailings:

Test No. 6

Mesh	Weight, per cent
+ 48...........	0
+ 65...........	0·8
+100...........	5·6
+150...........	11·7
+200...........	8·5
−200...........	73·4
	100·0

Test No. 7

Mesh	Weight, per cent
+ 48...........	0
+ 65...........	0·6
+100...........	4·1
+150...........	10·5
+200...........	8·3
−200...........	76·5
	100·0

Test No. 8

Mesh	Weight, per cent
+ 48...........	0
+ 65...........	0·6
+100...........	4·3
+150...........	10·2
+200...........	8·6
−200...........	76·3
	100·0

SUMMARY

Results of tests on the lower grade material show better recoveries of the gold, silver, and copper. The tailing carries only 0·01 ounce per ton in gold and the concentrates average 0·45 ounce per ton in gold against 0·29 ounce per ton on the higher grade material.

No difficulty was experienced in effecting a very satisfactory concentration of the copper, gold, and silver by flotation. The ore should be ground so that not less than 76 per cent will pass a 200-mesh screen.

A spectrographic analysis of a gold button from the assay of the concentrates showed no indication of platinum or platinum group metals.

Ore Dressing and Metallurgical Investigation No. 533

GOLD ORE FROM CHARTERED EXPLORERS, LIMITED, STURGEON LAKE, ONT.

Shipment. A shipment of 12 tons of gold ore was received September 13, 1933, from Chartered Explorers, Limited, 1006 Concourse Building, Toronto. The shipment was made from a property near Sturgeon lake, Ontario.

Sampling and Analysis. The shipment was very carefully sampled. The whole was crushed to $\frac{1}{4}$-inch and one-tenth cut out in a Vezin sampler, making Sample A. A duplicate sample was taken of the whole shipment. For this duplicate sample, the $\frac{1}{4}$-inch material was crushed in rolls to 14 mesh and one-tenth cut by the Vezin sampler for Sample B. Both Samples A and B were alternately crushed and sampled down by hand in a Jones sampler. The weights of the samples cut in the Jones were kept well within the limits generally specified for a rich and spotty ore.

Sample A		*Sample B*	
Gold	0·455 oz./ton	Gold	0·42 oz./ton
Copper	0·06 per cent	Copper	0·07 per cent

Average of A and B	
Gold	0·438 oz./ton
Copper	0·065 per cent

Characteristics of the Ore. The gangue consisted chiefly of quartz with considerable tourmaline. Native gold was observed in the ore, but the amount of sulphides present was relatively small.

EXPERIMENTAL TESTS

A number of small-scale batch tests were conducted on a part of the feed sample in order to determine approximately the degree of grinding necessary to free the gold and gold-bearing sulphides. These tests were made preliminary to a mill run on the whole shipment.

AMALGAMATION AND BLANKET CONCENTRATION

Test No. 1

The ore was ground to the following sizes and passed over a stationary amalgam plate and a corduroy blanket.

Mesh	Per cent
+ 48	6·00
− 48+ 65	17·6
− 65+100	18·15
−100+150	15·2
−150+200	6·65
−200	36·4
Total	100·00

The tailing assayed 0·05 ounce per ton, representing a recovery of 88·7 per cent of the gold.

Test No. 2

The ore was ground until 40 per cent passed 200 mesh. The recovery over the same set-up as used in Test No. 1 was 90·8 per cent.

Test No. 3

The ore was ground somewhat finer and passed over the amalgam plate and blanket.

Mesh	Per cent
+ 48	0·6
− 48+ 65	5·75
− 65+100	14·10
−100+150	19·15
−150+200	9·40
−200	51·00
Total	100·00

The tailing assayed 0·045, which represents an approximate recovery of 89 per cent.

A second series of amalgamation tests was made to determine the total amount of free gold produced by crushing the ore to various degrees of fineness.

Four lots of ore were ground dry and screened through 48-, 100-, 150- and 200-mesh screens. A sample of each of these was amalgamated with mercury for thirty minutes in a jar mill. The mercury was then separated from the ore and the tailing assayed for gold.

Summary of the results is given in the following table:—

Feed sample, Au 0·44 oz./ton.

Screen size	Tailing assay	Extraction, per cent
− 48 mesh	0·03	93·2
−100 "	0·025	94·3
−150 "	0·03	93·2
−200 "	0·025	94·3

LARGE-SCALE MILL TEST

The mill run was started using the following flow-sheet:—

The ore crushed to 14 mesh was fed to a 4-foot by 16-inch ball mill in closed circuit with a small Dorr classifier. The Dorr overflow was passed over a blanket table covered with corduroy blankets and the blanket tailing was floated in a 10-cell mechanical flotation cell.

During the first day's run an attempt was made to use the blanket table between the ball mill discharge and the classifier, in order to prevent the gold from accumulating in the grinding circuit. This was found impossible on account of the extra water required to prevent the blankets from banking-up with sands. The extra water needed to keep the pulp moving over the blankets diluted the classifier to such an extent that it was impossible to overflow any sand coarser than 100 mesh, which is much finer grinding than the preliminary tests indicated as necessary.

A brief summary of the results of the complete mill run follows. The assays given represent a sample taken from the start to the finish of the run.

Feed rate, lb./ton	Assays					Remarks
	Feed	Classifier overflow	Blanket tailing	Flotation concentrate	Flotation tailing	
381·7	0·44	0·155	0·04	2·20	0·01	Total amount of ore feed to mill was 17,931·0 lb.

Distribution of Gold, Per Cent:

Feed	Remaining in grinding circuit	Caught on blankets	Caught in flotation concentrate	Lost in flotation tailing
100	64·77	26·14	6·84	2·25

Screen Tests on Mill Products:

Mesh	Classifier overflow, per cent	Blanket tailing, per cent	Flotation concentrate, per cent	Flotation tailing, per cent
+ 28	0·25	0·40	....	0·25
− 28+ 35	2·10	2·40	0·30	2·25
− 35+ 48	6·00	6·05	1·40	6·30
− 48+ 65	9·65	9·65	3·85	10·10
− 65+100	12·95	13·20	5·05	13·85
−100+150	13·95	13·80	2·15	13·95
−150+200	13·75	13·75	4·85	15·70
−200	41·35	40·75	82·40	37·60
Total	100·00	100·00	100·00	100·00

The above tables show that 64·77 per cent of the gold accumulated in the grinding circuit; also that the average feed to the blanket table was only 0·155 ounce per ton and the tailing from the blanket table averaged 0·04 ounce per ton. The blankets, therefore, recovered approximately 75 per cent of the gold contained in the classifier overflow, or 26·14 per cent of the total gold in the ore. Flotation of the blanket tailing also recovered about 75 per cent of the gold in that product or, as shown in the table, 6·84 per cent of the total gold in the ore.

The grinding necessary to prevent undue loss of gold in the tailing is relatively coarse, averaging in the run only 41·35 per cent through 200 mesh with 2·35 per cent on 35 mesh. This grind gave an average loss in the tailing of only 2·25 per cent.

The following table gives the details of the mill run for each individual day's operation:—

Date	Feed, lb./hour	Density classifier overflow, per cent	Reagents to flotation circuit—lb./ton ore							Assays						
			To conditioner				To Cell No. 6			Daily samples—oz./ton						
			Soda ash	Xanthate	Barrett No. 4	Cresylic acid	Cresylic acid	No. 4A S.C.Ltd	Pine oil	Feed	Classifier overflow	Blanket tailing	Flotation concentrate	Flotation tailing	Table concentrate	Table tailing
Oct. 3..	400	34·0	1·12	0·26	0·075	....	0·03	....	......	0·345	0·06	0·05	1·03	0·015	....	0·01
Oct. 4..	370	32·0	3·4	0·32	0·09	....	0·09	....	......	0·525	0·12	0·05	2·64	0·01	....	0·01
Oct. 5..	370	33·0	......	0·32	0·09	....	....	0·06	......	0·42	0·115	0·07	1·67	0·03	....	0·01
Oct. 6..	405	34·0	......	0·30	0·09	....	....	0·06	......	0·385	0·12	0·05	2·21	0·035	0·04	0·0075
Oct. 7..	400	34·0	3·24	0·30	0·09	....	....	....	0·033	0·41	0·06	0·045	3·51	0·015	0·04	0·01
Oct. 10..	386	33·0	3·1	0·32	0·09	....	....	....	0·017	0·42	0·06	0·035	5·74	0·015	0·03	0·01
Oct. 11..	341	30·0	2·95	0·36	0·10	....	....	....	0·019	0·35	0·045	0·035	3·88	0·015	0·04	0·01

It should be noted that the assays of the daily sample do not represent a complete day's run, but only that of a few hours. They were taken as a guide for the following day's operation.

Particular attention is directed to the fact that at no time did the grinding circuit build up in gold to a point where the assay of the classifier overflow equalled the assay of the ore fed to the mill. In fact during the last two days' operation, when the feed was slightly reduced, the classifier overflow dropped from 0·12 ounce to as low as 0·045 ounce. Just how much ore would have been required to saturate the grinding circuit is difficult to estimate, but it is safe to assume that at least three times the amount of ore used would be required. Thus, if this flow-sheet were used at the mine, the mill would absorb gold for over a month after it was first started.

The above flow-sheet is not recommended for this ore. This mill run was made on request, with the expectation that the gold obtained in the products would give a check on the feed assays. At the end of the run the ball mill, classifier, elevator boot, and pumps, etc., were very carefully washed out, and the contents dried, weighed, and sampled. Despite the care taken, only 61·6 per cent of the gold fed to the mill could be accounted for in the various products.

The following table shows the distribution of the gold as found in these products:—

Products	Weight	Weight, per cent	Assay, oz./ton	Units	Distribution, per cent
Ore feed to mill	17,930	100·00	0·44	44·00	100·00
Clean-up of floor	610	3·40	0·65	2·210	8·2
Clean-out of ball mill	85	0·47	2·295	1·078	4·0
Clean-out of elevator boot	10	0·05	173·30	8·665	32·0
Clean-out of classifier	476	2·65	3·255	8·625	31·8
Clean-out of conditioning tank	195	1·09	0·38	0·414	1·5
Blanket concentrate	41	0·23	12·59	2·896	10·7
Flotation concentrate	185	1·03	2·20	2·267	8·4
Flotation tailing	16,329	91·08	0·01	0·911	3·4
Totals	17,931	100·00	0·27	27·066	100·0

It will be observed that the greater part of the gold accounted for as shown in the above table, was in the clean-up from the classifier and elevator boot, and is equivalent to 63·8 per cent of the total gold accounted for.

Undoubtedly the discrepancy between the total gold and the gold accounted for is due to fine gold lodging in cracks and corners of the elevator, classifier, and ball mill.

CYANIDE TESTS

A series of cyanide tests was conducted on part of the feed sample of the ore.

Four lots of ore were dry crushed respectively through 48, 100, 150, and 200 mesh. Each of these lots was cyanided separately for 24 hours, and a second series for 48 hours.

The strength of the cyanide solution used was 2 pounds per ton.

A summary of the results is given in the following table:—

Feed sample, Au 0·44 oz./ton

Screen size.	Period of agitation, hours	Tailing assay, Au oz./ton	Recovery, per cent	Reagents consumed, lb./ton	
				KCN	CaO
− 48 mesh	24	0·02	95·5	0·4	5·5
−100 "	24	0·01	97·7	0·7	6·2
−150 "	24	0·005	98·9	1·0	7·2
−200 "	24	0·005	98·9	1·3	8·2
− 48 "	48	0·005	98·9	1·0	5·8
−100 "	48	0·005	98·9	1·3	6·5
−150 "	48	0·005	98·9	1·6	7·4
−200 "	48	0·005	98·9	1·9	8·2

The results obtained from these cyanide tests show clearly the efficiency of the process on this ore.

CONCLUSIONS AND RECOMMENDATIONS

The small-scale amalgamation and blanket tests show that 85 to 90 per cent of the gold is freed as native gold by grinding to 40 per cent through a 200-mesh screen with approximately 1 per cent on 35 mesh.

Over 95 per cent of the gold can be extracted by cyanidation with the ore ground only to 48 mesh with approximately 55 per cent through 200 mesh, which is about as coarse as can be practically handled in a cyanide plant.

Two types of mill are, therefore, indicated as suitable for the treatment of this ore. The choice depends entirely on financial considerations. If a mill be built and operated during the development stages of the property, a stamp mill using both amalgamation plates and blankets is recommended. The alternative is a straight cyanide plant, using accessory apparatus such as traps to eliminate some of the coarse gold from the grinding circuit.

If the first type of mill, which is less expensive than the cyanide plant and can be more quickly constructed and put into operation, be chosen, it must be kept in mind that it will eventually become necessary to convert it into a cyanide mill or at least to add a cyanide annex.

Ore Dressing and Metallurgical Investigation No. 534

GOLD ORE FROM THE HOME GOLD MINING COMPANY, LIMITED, YALE MINING DIVISION, BRITISH COLUMBIA

Shipment. A shipment of 300 pounds of ore was submitted by the Home Gold Mining Company, Limited, 730 Standard Bank Building, Vancouver, B.C., from their property situated on the Divide, between the middle forks of the Ladner and Siwash creeks, Yale mining division. It was received at the Ore Dressing and Metallurgical Laboratories on September 5th, 1933.

Characteristics of the Ore:

In order to determine the character of the ore, six polished sections were prepared and examined microscopically.

Gangue. The gangue represented in the specimens studied consists chiefly of milky white quartz with fine stringers of impure carbonate. Locally the gangue is composed of a grey siliceous material, possibly chloritic in nature.

Sulphides. The sulphides in the ore are, in their order of abundance, pyrite, arsenopyrite, pyrrhotite, and chalcopyrite.

Pyrite and arsenopyrite are very abundant, and are disseminated throughout the gangue as medium- to coarse-grained, irregularly-formed crystals and grains. The size of these possibly varies usually from 2 to 3 millimetres in diameter Both minerals are much fractured and veined by gangue; they contain numerous inclusions of gangue, a considerable number of small irregular grains of pyrrhotite, occasional small grains of chalcopyrite, and extremely fine native gold.

Native Gold. A considerable number of extremely tiny rounded grains of native gold was observed in both the pyrite and the arsenopyrite. In most cases they were isolated, but occasionally the metal is associated with small grains of chalcopyrite. All the gold observed occurs within either the pyrite or the arsenopyrite, and is less than 6 microns in diameter; the smallest grains seen approximated to 0·1 micron in diameter, and were barely resolvable under the highest power oil-immersion objective available in this laboratory.

Purpose of the Experimental Tests. The purpose of the experimental tests was to determine the best method of extracting the gold.

An average assay of the ore was as follows:—

Gold	1·85 oz./ton
Arsenic	9·87 per cent
Iron	20·71 "
Sulphur	15·04 "
Copper	0·14 "

Experimental Tests

A series of small-scale tests was made on the ore consisting of amalgamation, flotation, flotation followed by table concentration, table concentration, cyanidation, and roasting followed by cyanidation.

By amalgamation with the ore ground dry to pass 35-, 48-, 65-mesh screens not more than 25 per cent recovery was obtained. No other amalgamation tests were made.

Details of the tests follow:—

AMALGAMATION

Tests Nos. 1 to 3

In this series of tests three representative samples of the ore, ground dry through 35-, 48-, and 65-mesh screens, were amalgamated with mercury for one hour in a jar mill. The amalgamation tailings were filtered, washed, and assayed for gold.

Summary:

Feed sample, Au 1·85 oz./ton

Test No.	Mesh	Tailing assay, Au, oz./ton	Recovery, per cent
1	−35	1·53	17·3
2	−48	1·46	21·1
3	−65	1·38	25·4

Screen tests on the material used in the foregoing tests showed the grinding to be as follows:—

All through	Weight, per cent +48	Weight, per cent −48+65	Weight, per cent −65+100	Weight, per cent −100+150	Weight, per cent −150+200	Weight, per cent −200	Total
35 mesh	1·70	16·55	21·00	13·25	6·00	41·50	100·0
48 mesh	....	2·60	22·45	15·80	6·00	53·15	100·0
65 mesh	....	....	7·4	13·2	6·60	72·8	100·0

FLOTATION

In Test No. 4 an attempt was made to float all the gold and have as large a ratio of concentration as possible. In Test No. 5 an attempt was made to float all the gold in a bulk concentrate.

Test No. 4

Charge to Ball Mill:

Ore .... 2,000 grammes −14-mesh ore
Water .... 1,000 c.c.
Na_2CO_3 .... 6 lb./ton
Sodium ethyl xanthate .... 0·2 "

Reagents to Cell:

Pine oil..0·05 lb./ton

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

Summary:

Feed to flotation, Au 1·85 oz./ton

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent	Ratio of concentration
Flotation concentrate	18·8	6·83	72·3	5·3 : 1
Flotation tailing	81·2	0·605	27·7	

A screen test on the flotation tailing showed the following:—

Mesh	Weight, per cent
− 48+ 65	0·05
− 65+100	1·15
−100+150	6·50
−150+200	5·70
−200	86·60

Test No. 5

Charge to Ball Mill:

Ore..2,000 grammes −14-mesh ore
Water..1,000 c.c.
Na_2CO_3..8·0 lb./ton
Aerofloat No. 25..0·07 "
Sodium ethyl xanthate..0·2 "

Reagents to Cell:

Pine oil..0·025 lb./ton

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

Summary:

Feed to flotation, Au 1·85 oz./ton

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent	Ratio of concentration
Flotation concentrate	41·4	3·975	93·4	2·4 : 1
Flotation tailing	58·6	0·20	6·6	....

A screen test on the flotation tailing showed the following:—

Mesh	Weight, per cent
− 65+100	0·1
−100+150	1·3
−150+200	2·1
−200	96·5

FLOTATION FOLLOWED BY TABLE CONCENTRATION

Test No. 6

The charge used in Test No. 5 was duplicated and the tailing from flotation was concentrated on the small Wilfley table.

Summary:

Feed to flotation, Au 1·85 oz./ton

Products	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent	Ratio of concentration
Flotation concentrate	47·11	3·51	81·68	
Flotation tailing	52·89	0·37	18·32	2·1 : 1
Table concentrate	11·95	1·01	11·25	
Table tailing	40·94	0·185	7·07	8·4 : 1

	Per cent
Recovery by flotation	81·68
Recovery by table concentration	11·25
Net recovery	92·93

Combined ratio of concentration, 1·7 : 1

TABLE CONCENTRATION

Test No. 7

In this test 2,000 grammes of the ore at −14 mesh was ground in a ball mill in a pulp dilution of 2 : 1 until 85 per cent −200 mesh. The pulp was concentrated on the small-scale Wilfley table. The products consisted of concentrate, middling, sand, and slime tailings.

Summary:

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent	Ratio of concentration
Table concentrate	22·3	4·17	45·8	4·5 : 1
Table middling	7·3	1·19	4·2	
Table sand tailing	30·2	1·17	17·4	
Table slime tailing	40·2	1·64	32·6	

Test No. 7 shows large loss of gold in tailing by this method of concentration.

CYANIDATION

Tests Nos. 8 to 11

In this series of tests samples of the ore at −14 mesh were ground dry through 48-mesh and 200-mesh screens. They were agitated in cyanide solution 1 pound KCN per ton for periods of 24 and 48 hours. The cyanide tailings were filtered, washed, and assayed for gold.

Summary:

Feed sample, Au 1·85 oz./ton

Test No.	Mesh	Period of agitation, hours	Tailing assay, Au oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
					KCN	CaO
8..................	−48	24	0·90	51·35	1·05	13·20
9..................	−48	48	0·98	47·03	1·35	14·40
10..................	−200	24	0·625	66·22	1·12	18·35
11..................	−200	48	0·65	64·86	1·41	19·10

It was found that the best recovery was obtained from the 24-hour tests.

GRINDING FOLLOWED BY CYANIDATION

Test No. 12

In this test 1,000 grammes of ore at −65 mesh was ground in a ball mill with 16 pounds lime per ton and dilution of 4 : 3. A screen test on the final tailing shows the grind to be 96 per cent −200 mesh.

The pulp was filtered, washed, and charged into a large Winchester and agitated in cyanide solution 1 pound KCN per ton for 24 hours, dilution 2 : 5. The cyanide tailing was filtered, washed, and assayed for gold.

Summary:

Feed sample, Au 1·85 oz./ton

Test No.	Period of agitation, hours	Tailing assay, Au oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
12..............................	24	0·665	64·1	1·00	21·25

HOT LIME TREATMENT FOLLOWED BY CYANIDATION

Test No. 13

A sample of −200-mesh ore was agitated for 8 hours in very hot water with 20 pounds lime per ton, dilution 1 : 3. The pulp was filtered, washed, and charged into a Winchester bottle and agitated with cyanide solution 1 pound KCN per ton for 48 hours, dilution 1 : 3. The cyanide tailing was filtered, washed, and assayed for gold.

Summary:

Feed sample, Au 1·85 oz./ton

Test No.	Period of agitation, hours	Tailing assay, Au oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
13..............................	48	0·59	68·1	1·12	23·5

ROASTING FOLLOWED BY CYANIDATION

Tests Nos. 14 to 15

In these tests two samples of ore at −200 mesh were roasted until all arsenic and sulphur were driven off. The roasted ore was washed in water, dilution 1 : 3, for one hour by agitation in Winchester bottles. The pulp was filtered and washed and agitated in cyanide solution, 1 pound KCN per ton for periods of 24 and 48 hours, dilution 1 : 3. The cyanide tailing was filtered, washed, and assayed for gold.

Summary:

Feed sample, Au 1·85 oz./ton

Test No.	Period of agitation, hours	Tailing assay, Au oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
14	24	0·24	87·0	0·25	23·5
15	48	0·205	88·9	0·65	18·9

CYANIDE FOULING TESTS

Tests Nos. 16 to 20

In these tests the same cyanide solution 1 pound KCN per ton was used on several charges of ore at −65 mesh. After each sample was cyanided the volume and strength of the solution was made up to the original volume and strength. The cyanide tailing was filtered, washed, and assayed for gold.

Summary:

Feed sample, Au 1·85 oz./ton.

Test No.	Period of agitation, hours	Tailing assay, Au oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
16	24	0·835	54·86	1·26	14·45
17	24	0·92	50·27	1·71	15·45
18	24	0·97	47·57	1·65	15·91
19	24	1·015	45·14	1·26	15·61
20	24	1·065	42·43	1·36	15·79

The extraction is progressively lowered, which is due to fouling of the solution.

SUMMARY AND CONCLUSIONS

The microscopic study of the ore has disclosed that gold occurs within the arsenopyrite crystals as extremely fine particles of native gold. Photomicrographs show these particles to be less than 6 microns in diameter, which is so fine that no commercial degree of crushing would expose them to the action of the cyanide. This conclusion is substantiated by the results of the experimental tests.

Amalgamation recovers a maximum of only 25 per cent. Straight cyanidation of the raw ore ground all through 200 mesh extracts a maximum of only 64·1 per cent, and after treatment with hot lime solution this extraction is raised to only 68·1 per cent.

There is left only one alternative, that is to roast the ore and then cyanide it. The experimental tests using this method indicate a minimum extraction of 88·9 per cent, which in practice would probably be increased to over 90 per cent.

A number of concentration tests were made, but owing to the large proportion of sulphides present in the sample received, the results could not be interpreted as commercial. The best result was obtained by grinding to 96·5 per cent through 200 mesh and concentrating by flotation. A flotation concentrate assaying about 4 ounces to the ton in gold was obtained with recovery of 93·4 per cent of the gold, but 41·4 per cent of the ore had to be floated as a concentrate; that is to say that for every 2·4 tons of ore mined, 1 ton of concentrate would have to be hauled from the mine to the smelter.

Attention is directed to the possibility that the average grade of ore, as it would be mined, would contain much less sulphide than the sample submitted for test purposes. If this is the case, the picture in regard to concentration of the ore might be entirely altered.

For example, the sample received contained over 50 per cent sulphide through which the gold is scattered in minute grains. Should the average of the ore as mined contain only 10 per cent sulphide, then a reasonable ratio of concentration would be obtained and the operation of a concentrator would show a profit. Considering the grade of ore represented by the sample submitted, which runs 1·85 ounces per ton in gold, it would undoubtedly be more profitable to ship the ore direct to the smelter than it would be to attempt to concentrate it in a mill.

Ore Dressing and Metallurgical Investigation No. 535

GOLD ORE FROM BIDGOOD KIRKLAND GOLD MINES, LIMITED, KIRKLAND LAKE, ONTARIO

Shipment. A shipment of 26 sacks of ore, net weight 2,000 pounds, was received September 29, 1933. The sample was submitted by Oscar F. Knutson, Manager, Bidgood Kirkland Gold Mines, Limited, Kirkland Lake, Ontario.

Characteristics of the Ore. The ore possesses a fine-textured, mottled greenish to brownish grey siliceous gangue with considerable white quartz and a small amount of carbonate as fine veinlets in the quartz.

The metallic minerals present are pyrite, chalcopyrite, sphalerite, native gold, and two unidentified minerals which may be tellurides.

Pyrite is the dominant metallic mineral and is disseminated in rather coarse irregular grains and well formed cubes; a small portion of this mineral is, however, quite fine-grained. Locally, the pyrite is so abundant that granular masses containing a considerable amount of gangue are formed.

Chalcopyrite is present in only very small amount, and as small irregular grains in the pyrite and gangue.

Sphalerite is exceedingly rare, and occurs as small grains in the gangue associated with pyrite, and as tiny grains in the pyrite.

An average assay of the ore was as follows:—

Gold	0·22 oz./ton
Silver	0·08 "

Experimental Tests

A series of small-scale tests was made on the ore in search of a practical method of treating it. The work included tests by amalgamation, cyanidation, flotation, and tabling. Maximum recovery obtainable by amalgamation was 45·5 per cent of the gold, and by straight cyanidation 91·0 per cent of the gold was extracted. By cyaniding the ore, tabling out the pyrite, regrinding, and recyaniding it, extraction can be increased to 94·1 per cent, leaving an average tailing assaying 0·013 ounce per ton in gold.

By straight flotation of the ore ground all through 35 mesh and 48 per cent through 200 mesh, 87·2 per cent of the gold was recovered in a concentrate assaying 2·04 ounce per ton in gold with a ratio of concentration of 9·5 : 1. When the ore was ground all through 100 mesh and 82 per cent through 200 mesh, recovery in the concentrate amounted to 86·5 per cent of the gold. The concentrate assayed 2·30 ounces per ton in gold and the ratio of concentration was 11·2 : 1.

Flotation of the cyanide tailings, although showing some recovery of the gold, did not produce results to compare with tabling.

Details of the tests follow:

AMALGAMATION AND CYANIDATION

Tests Nos. 1 and 2

Two samples of the ore were ground dry to pass through 150- and 200-mesh screens. A sample of each lot of finely ground ore was amalgamated with mercury for 30 minutes in a jar mill. The amalgamation tailings were sampled and assayed and a portion of each agitated in cyanide solution, 2·0 pounds KCN per ton, for 24 hours.

Summary:

Feed sample, Au 0·22 oz./ton.

Test No.	Mesh	Assay, Au oz./ton		Recovery, per cent		Reagents consumed, lb./ton	
		Amalgamation tailing	Cyanide tailing from amalgamation tailing	Amalgamation	Cyanidation	KCN	CaO
1	−150	0·12	0·035	45·5	38·6	0·45	6·7
2	−200	0·13	0·025	40·9	47·7	0·60	7·0

CYANIDATION

Tests Nos. 3 to 6

In this series of tests samples of the same two lots of ore ground dry that were used for Tests Nos. 1 and 2 were agitated in cyanide solution, 2·0 pounds KCN per ton, for periods of 24 and 48 hours. The cyanide tailings were filtered, washed, and assayed for gold.

Summary:

Feed sample, Au 0·22 oz./ton.

Test No.	Mesh	Period of agitation, hours	Tailing assay, Au oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
					KCN	CaO
3	−150	24	0·03	86·7	0·55	6·7
4	−200	24	0·02	90·8	0·55	6·8
5	−150	48	0·025	88·6	0·55	6·8
6	−200	48	0·025	88·6	0·55	7·2

CYANIDATION AND TABLING

Test No. 8

In this test the ore was ground 70·0 per cent through 200 mesh and agitated in cyanide solution for 48 hours. The cyanide tailing was sampled and assayed and the remainder passed over a small laboratory-size concentrating table. The sulphide concentrate thus produced was sampled and assayed and the remainder reground and recyanided.

Assays:

Feed sample	0·22	Au oz./ton
Cyanide tailing	0·025	"
Table concentrate from cyanide tailing	0·13	"
Table tailing	0·01	"
Cyanide tailing from table concentrate:		
24-hour agitation	0·05	"
48-hour agitation	0·04	"
Average tailing (cal.)	0·013	"
Net extraction	94·1	per cent.

Results:

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent	Total gold
Table concentrate	10·7	0·13	60·9	6·4
Table tailing	89·3	0·01	39·1	4·1
Cyanide tailing (cal.)	100·0	0·023	100·0	

CYANIDATION AND FLOTATION

Test No. 9

In this test the ore was ground practically all through 200 mesh and agitated in cyanide solution, 2·0 pounds KCN per ton, for 48 hours. The cyanide tailing was then filtered, washed with water, and floated. The flotation concentrate and tailing were assayed for gold and the cyanide tailing assay was calculated from them.

Summary:

Feed sample, Au 0·22 oz./ton.

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent	Reagents consumed, lb./ton	
				KCN	CaO
Flotation concentrate	11·0	0·08	33·5		
Flotation tailing	89·0	0·02	66·5		
Cyanide tailing (cal.)	100·0	0·027	100·0	0·40	6·50

Extraction by cyanidation:

(0·22−0·027) ÷ 0·22 = 87·7 per cent

Recovery in flotation concentrate:

(100·0−87·7) 33·5 = 4·1 per cent

FLOTATION

Test No. 7

In this test the ore at −14 mesh was ground in a ball mill for 20 minutes and then floated.

Charge to Ball Mill:

Ore	2,000 grammes
Water	1,500 c.c.
Na_2CO_3	5 lb./ton

Reagents to Cell:

Potassium amyl xanthate	0·10 lb./ton
Pine oil	0·05 "

Summary:

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Concentrate	10·5	2·04	87·2
Tailing	89·5	0·035	12·8
Feed (cal.)	100·0	0·246	100·0

Ratio of concentration, 9·5 : 1.

A screen test of the flotation tailing showed the grinding to be as follows:—

Mesh	Weight, per cent	Cumulative weight, per cent
+ 48	0·6	0·6
− 48+ 65	4·6	5·2
− 65+100	16·5	21·7
−100+150	15·1	36·8
−150+200	15·3	52·1
−200	47·9	100·0
Total	100·0	

FLOTATION

Test No. 13

In this test the ore at −14 mesh was ground for 40 minutes in a ball mill and then floated.

Charge to Ball Mill:

Ore	2,000 grammes
Water	1,500 c.c.
Na_2CO_3	5 lb./ton
Aerofloat No. 25	0·07 "

Reagents to Cell:

Potassium amyl xanthate	0·10 lb./ton
Pine oil	0·05 "

Summary:

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Concentrate	8·9	2·30	86·5
Tailing	91·1	0·035	13·5
Feed (cal.)	100·0	0·236	100·0

Ratio of concentration, 11·2 : 1.

GRINDING TESTS ON THE ORE

Three samples of the ore were ground in jar mills for periods of 25, 30, and 40 minutes respectively. In each case the dried pulp was screened to determine the grinding. The results may be tabulated as follows:—

Period of grinding, minutes	Weight, per cent +48	Weight, per cent −48+65	Weight, per cent −65+100	Weight, per cent −100+150	Weight, per cent −150+200	Weight, per cent −200	Total
25	0·1	1·0	8·2	14·5	20·1	56·1	100·0
30	Nil	0·1	2·1	9·2	19·5	69·1	100·0
40	Nil	Nil	0·4	3·9	13·7	82·0	100·0

CONCLUSIONS

The work carried out on this ore seems to point to cyanidation with table concentration, regrinding, and re-agitation of the pyrite as the most practical method. This is standard practice at the Hollinger Consolidated Gold Mines, Ltd.

Referring to Test No. 8 it can be seen that 10·7 per cent of the weight of the cyanide tailing in the form of a pyrite concentrate contains 60·9 per cent of its gold. The saving in grinding effected by so concentrating is obvious. At the same time the table tailing is low enough in gold to be discarded.

Flotation of the ore is not to be recommended because the tailing loss is too high, and a comparison of Tests Nos. 7 and 13 shows that this cannot be reduced by fine grinding. Furthermore, flotation would probably have to be done in a soda ash circuit, and if the concentrate or tailing were to be cyanided they would first have to be filtered and washed to clear out the soda ash and then reconditioned with lime. Cyanidation with table concentration offers the further advantage of grinding in cyanide solution.

Ore Dressing and Metallurgical Investigation No. 536

GOLD ORE FROM ALCONA GOLD MINES, LIMITED, ALCONA, ONTARIO

Shipment. A shipment of five sacks of ore, net weight 500 pounds, was received September 7, 1933. The sample was submitted by Chas. R. Richardson, Manager, Alcona Gold Mines, Limited, on instructions from D. G. H. Wright, Consulting Engineer, 53 King St. West, Toronto.

Characteristics of the Ore. The gangue is grey to white translucent quartz, locally stained yellow by iron. The metallic minerals observed include pyrite, chalcopyrite, sphalerite, and galena, but no native gold was seen. In addition the presence of another mineral was noted, here termed Mineral X.

Test on Mineral X failed to give sufficient information for its identification but it corresponds most closely to keeleyite ($PbS.Sb_2S_3$). The mineral occurs as tiny rounded dots in the galena.

Pyrite is disseminated as rather coarse grains in the quartz. Sphalerite, chalcopyrite, and galena form granular aggregates and irregular stringers in the quartz and are usually rather intimately associated.

An average analysis of the shipment was as follows:—

Gold	1·20 oz./ton
Silver	4·14 "
Copper	0·27 per cent
Lead	1·71 "
Zinc	0·39 "
Iron	4·36 "
Sulphur	3·16 "

Experimental Tests

A series of small-scale tests was made on this ore for the purpose of finding out how the gold could be recovered from it. The work included tests by amalgamation, cyanidation, flotation, and tabling. A recovery ranging from 18 to 29 per cent of the gold was obtained by amalgamation of the ore ground dry to various sizes, the highest recovery being obtained with the coarsest size. Cyanidation of the amalgamation tailing gave an additional recovery ranging from 58 to 70 per cent, in this case the highest recovery being obtained with the finest size.

By straight cyanidation of the ore for periods of 24 and 48 hours, recovery ranging from 23 per cent with the coarsest size for 24 hours to over 88 per cent with the finest size for 48 hours was obtained.

By flotation 94·3 per cent of the gold was recovered in a flotation concentrate amounting to 14·6 per cent of the weight of the feed. In this case the ore was ground 99 per cent through 200 mesh. By grinding a little coarser, 96 per cent through 200 mesh, 89·7 per cent of the gold was obtained in a concentrate amounting to 10·6 per cent of the weight of the feed. Cyanidation tests on these flotation tailings gave very encouraging results, a tailing assaying 0·01 ounce per ton in gold being produced.

By tabling a sample of the ore, 63 per cent of the gold was recovered in a sulphide concentrate representing 16·5 per cent of the weight of the feed. After amalgamation, regrinding and cyanidation, 97 per cent of the gold in the table concentrate was recovered; and 87·7 per cent of the gold in the table tailing was recovered by straight cyanidation. These operations resulted in a net recovery of 93·6 per cent of the gold in the ore. However, when the ore was cyanided first and then tabled the table tailing assayed too high to be discarded.

Chemical analysis showed the pregnant solutions to be very strongly reducing owing to the formation of thiocyanates when the ore was cyanided directly. By agitating for 24 hours and then filtering and repulping in fresh solution the cyanide tailing was reduced to 0·07 ounce per ton in gold representing a recovery of 94·2 per cent.

By grinding the ore 99 per cent through 200 mesh and floating and then blanketing the flotation tailing 94·6 per cent of the gold was recovered in the flotation concentrate and 2·5 per cent was recovered in the blanket concentrate. The remainder, 2·9 per cent, was lost in the blanket tailing.

Details of the tests follow.

AMALGAMATION AND CYANIDATION

Tests Nos. 1 to 4

In this series of tests samples of the ore ground dry through 48-, 100-, 150-, and 200-mesh screens were amalgamated with mercury for 30 minutes in a jar mill. The amalgamation tailings were sampled for assay and the remainder agitated for 24 hours in cyanide solution. The cyanide tailings were then assayed for gold.

Summary:

Feed sample, Au 1·20 oz./ton.

Screen size	Assay, Au oz./ton		Extraction, per cent			Reagents consumed, lb./ton	
	Amalgamation tailings	Cyanide tailings	Amalgamation	Cyanidation	Total	KCN	CaO
−48 mesh	0·845	0·15	29·6	57·9	87·5	5·3	5·25
−100 mesh	0·86	0·16	28·3	58·3	86·6	7·3	5·25
−150 mesh	0·88	0·14	26·7	61·7	88·4	7·35	6·8
−200 mesh	0·98	0·145	18·3	69·6	87·9	9·3	7·5

CYANIDATION

Tests Nos. 5 to 12

In this series of tests samples of the same four lots of dry-ground ore that were used for Tests Nos. 1 to 4 were agitated in cyanide solution, 2·0 pounds KCN per ton, for periods of 24 and 48 hours. The cyanide tailings were filtered, washed, and assayed for gold.

Summary:

Feed sample, Au 1·20 oz./ton.

Screen size	Period of agitation, hours	Tailing assay, Au oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
− 48 mesh	24	0·92	23·3	3·0	5·0
−100 mesh	24	0·70	41·7	3·7	5·25
−150 mesh	24	0·61	49·2	4·1	6·5
−200 mesh	24	0·275	77·1	6·0	7·1
− 48 mesh	48	0·575	52·1	4·4	6·25
−100 mesh	48	0·40	66·7	4·9	6·25
−150 mesh	48	0·44	63·3	5·25	8·1
−200 mesh	48	0·14	88·3	7·6	8·5

FLOTATION

Test No. 13

In this test 2,000 grammes of the ore at −14 mesh was ground for 40 minutes in a ball mill and then floated.

Charge to Ball Mill:

Ore	2,000 grammes
Water	1,500 c.c.
Na_2CO_3	10·0 lb./ton
Aerofloat No. 25	0·07 "

Reagents to Cell:

Potassium amyl xanthate	0·10 lb/ton
Pine oil	0·05 "

The concentrate and tailing were filtered, washed, and assayed for gold, silver, copper, lead, and zinc.

Summary:

Product	Weight, per cent	Assays					Distribution of metals, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Concentrate	10·6	9·94	36·68	2·76	15·07	2·40	89·7	92·7	99·0	89·5	74·0
Tailing	89·4	0·135	0·34	trace	0·21	0·10	10·3	7·3	1·0	10·5	26·0
Feed (cal.)	100·0	1·174	4·19	0·30	1·79	0·34	100·0	100·0	100·0	100·0	100·0

Further determinations were made on this concentrate to ascertain its desirability as smelter feed. The results are as follows:—

	Per cent
Iron	22·75
Arsenic	Trace< 0·05
Sulphur	27·22
Silica	24·90
Lime	0·35
Magnesia	0·21

A screen test of the flotation tailing showed the grinding to be as follows:–

Mesh	Weight, per cent	Cumulative weight, per cent
+100	0·05	0·05
−100+150	1·40	1·45
−150+200	2·50	3·95
−200	96·05	100·00
Total	100·00	

FLOTATION

Test No. 14

This test was carried out in the same way as Test No. 13 except that the ore was ground for one hour instead of for 40 minutes and one drop of Tarol No. 1 was added to the cell to make the froth more consistent.

Summary:

Product	Weight, per cent	Assays					Distribution of metals, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Concentrate	14·6	7·28	26·38	1·90	10·96	2·40	94·3	95·1	99·0	91·2	99·0
Tailing	85·4	0·075	0·23	trace	0·18	trace	5·7	4·9	1·0	8·8	1·0
Feed (cal.)	100·0	1·127	4·05	0·29	1·75	0·36	100·0	100·0	100·0	100·0	100·0

A screen test of the flotation tailing showed the grinding to be as follows:—

Mesh	Weight, per cent	Cumulative weight, per cent
+150	0·20	0·2
−150+200	0·80	1·0
−200	99·00	100·0
Total	100·0	

Samples of the flotation tailings from Tests Nos. 13 and 14 were agitated in cyanide solution, 2·0 pounds KCN per ton, for 24 hours, and in each case a cyanide tailing assaying 0·015 ounce per ton in gold was produced. This brings the total gold recovery in these two tests up to 98·75 per cent.

FLOTATION AND CYANIDATION

Test No. 20

Realizing that flotation in a soda ash circuit would be undesirable if the tailing is to be cyanided some flotation tests were made using neutral and lime circuits. In this test the ore was ground in water for 40 minutes and then floated. A sample of the flotation tailing was cyanided for 24 hours.

Charge to Ball Mill:

Ore	2,000	grammes
Water	1,500	c.c.
Aerofloat No. 25	0·07	lb./ton

Reagents to Cell:

Potassium amyl xanthate	0·10	lb./ton
Pine oil	0·40	"

Summary:

Product	Weight, per cent	Assays					Distribution of metals, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb. per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Concentrate	10·5	9·10	31·58	2·80	12·57	2·86	90·7	90·7	84·6	94·2	99·0
Tailing	89·5	0·11	0·38	0·06	0·09	trace	9·3	9·3	15·4	5·8	1·0
Feed (cal.)	100·0	1·05	3·66	3·48	1·40	0·31	100·0	100·0	100·0	100·0	100·0
Tailing cyanided	89·5	0·01									

Test No. 22

In this test the ore was ground for 40 minutes and then floated. A sample of the flotation tailing was cyanided for 24 hours.

Charge to Ball Mill:

Ore	2,000	grammes
Water	1,500	c.c.
Barrett No. 4	0·07	lb./ton
Potassium amyl xanthate	0·10	"

Reagents to Cell:

Pine oil	0·05	lb./ton

Summary:

Product	Weight, per cent	Assays					Distribution of metals, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Concentrate........	6·9	12·67	43·76	3·40	16·25	3·12	89·5	89·9	83·4	86·4	95·6
Tailing............	93·1	0·11	0·37	0·05	0·19	trace	10·5	10·2	16·6	13·6	4·4
Feed (cal.).........	100·0	0·98	3·36	0·28	1·30	0·22	100·0	100·0	100·0	100·0	100·0
Tailing cyanided....	93·1	0·01									

Test No. 24

In this test the ore was ground in a ball mill for 40 minutes with lime and potassium amyl xanthate. A sample of the flotation tailing was cyanided for 24 hours.

Charge to Ball Mill:

Ore...	2,000	grammes
Water...	1,500	c.c.
Lime..	0·50	lb./ton
Potassium amyl xanthate................................	0·10	"

Reagents to Cell:

Pine oil..	0·10	lb./ton

Summary:

Product	Weight, per cent	Assays					Distribution of metals, per cent				
		Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Au	Ag	Cu	Pb	Zn
Concentrate........	7·2	12·50	43·70	3·30	16·76	3·22	90·7	89·0	96·2	91·6	71·4
Tailing............	92·8	0·10	0·40	trace	0·12	0·10	9·3	11·0	3·8	8·4	28·6
Feed (cal.).........	100·0	0·99	3·54	0·25	1·32	0·32	100·0	100·0	100·0	100·0	100·0
Tailing cyanided...	92·8	0·01									

TABLING AND CYANIDATION

Test No. 15

In this test the ore at −14 mesh was ground in a ball mill for 30 minutes. The pulp was then passed over a small laboratory-size table giving two products, a sulphide concentrate and a tailing. These two products were sampled and assayed. The table tailing was then agitated in cyanide solution, 1·0 pound KCN per ton, for periods of 24 and 48 hours and the cyanide tailing was assayed for gold and silver. The table concentrate was amalgamated with mercury in a jar mill for 30 minutes, then reground for 25 minutes and agitated in cyanide solution, 5·0 pounds KCN per ton, for periods of 24 and 48 hours. These cyanide tailings were also assayed for gold and silver.

Summary:

Product	Weight, per cent	Assay		Distribution of precious metals, per cent	
		Au, oz./ton	Ag, oz./ton	Gold	Silver
Table concentrate	16·56	4·54	11·92	63·0	51·0
Table tailing	83·44	0·53	2·27	37·0	49·0
Feed (cal.)	100·0	1·194	3·87	100·0	100·0

The amalgamation and cyanidation tests made on the products gave the following results:—

Results of Cyanidation of Table Tailing:

Product	Assay, oz./ton		Content, per cent total		Extraction, per cent		Extraction, per cent total		Reagents consumed, lb./ton	
	Au	Ag	Au	Ag	Au	Ag	Au	Ag	KCN	CaO
Table tailing	0·53	2·27	37·0	49·0						
Table tailing cyanided: 24 hours	0·145	0·59	......	......	72·6	74·0	26·9	36·3	2·4	4·8
Table tailing cyanided: 48 hours	0·065	0·44	......	......	87·7	80·6	32·5	39·5	2·77	5·0

Results of Amalgamation and Cyanidation of Table Concentrate:

Product	Assay, oz./ton		Content, per cent total		Extraction, per cent		Extraction, per cent total		Reagents consumed, lb./ton	
	Au	Ag	Au	Ag	Au	Ag	Au	Ag	KCN	CaO
Table concentrate	4·54	11·92	63·0	51·0						
Table concentrate amalgamated	3·01	11·02	......	......	33·3	7·6	21·0	3·9		
Amalgamation tailing cyanided, 24 hours	0·215	4·05	......	......	61·9	58·4	39·0	29·8	7·75	8·8
Amalgamation tailing cyanided, 48 hours	0·135	3·40	......	......	63·7	63·9	40·1	32·6	8·7	9·1

Total extraction from ore by amalgamating table concentrate and cyaniding the amalgamation tailing along with the table tailing for 24 hours.

Gold, 26·9+21·0+39·0 .... = 86·9 per cent
Silver, 36·3+3·9+29·8 .... = 70·0 "

Average tailing assays calculated:—

Gold .... 0·157 oz./ton
Silver .... 1·16 "

Total extraction by amalgamation as above, but cyaniding both products for 48 hours:—

Gold, 32·5+21·0+40·1 .... = 93·6 per cent
Silver, 39·5+3·9+32·6 .... = 76·0 "

Average tailing assays calculated:—

Gold .... 0·077 oz./ton
Silver .... 0·93 "

CYANIDATION AND TABLING

Test No. 16

In this test the ore, after 30 minutes' grinding, was agitated in cyanide solution, 1 pound KCN per ton, for 48 hours. The cyanide tailing was then passed over a small concentrating table. The table concentrate and tailing were assayed for gold and silver and a sample of the table concentrate reground, amalgamated, and cyanided.

Summary:

Product	Weight, per cent	Assays, oz./ton		Distribution of precious metals, per cent	
		Au	Ag	Au	Ag
Table concentrate	13·0	5·20	10·44	56·2	32·7
Table tailing	87·0	0·19	0·87	13·8	18·3
Cyanide tailing (cal.)	100·0	0·84	2·11	70·0	51·0
Feed	100·0	1·20	4·14	100·0	100·0

After regrinding, amalgamation and cyanidation, the precious metal content of the table concentrate was reduced to 0·105 ounce per ton in gold and 1·57 ounces per ton in silver.

This represents a recovery of 96·0 per cent of the gold and 85·0 per cent of the silver contained in the table concentrate.

Therefore, the net recoveries are:—

(a) By Cyanidation of the Ore:

		Per cent
Gold	100·0 − 70·0	= 30·0
Silver	100·0 − 51·0	= 49·0

(b) By Amalgamation and Cyanidation of the Table Concentrate:

Gold	96·0 × 56·2	= 54·0
Silver	85·0 × 32·7	= 27·8

Total recovery:

Gold	54·0 + 30·0	= 84·0
Silver	27·8 + 49·0	= 76·8

Average tailing assays calculated:

Gold	0·179 oz./ton
Silver	0·961 "

In this test the ore was treated with a 1·0 pound per ton KCN solution, but the use of a stronger solution would have resulted in better extraction with consequent lower grade feed to the table. (*See* Tests Nos. 5 to 12.) Under these circumstances it might be possible to produce a table tailing low enough to discard.

ANALYSIS OF PREGNANT SOLUTION

Test No. 17

A sample of ore all ground through 200 mesh was agitated in cyanide solution, 2·0 pounds KCN per ton, for 48 hours. The pregnant solution was then filtered off and the following determinations made on it:—

	Grammes per litre
Cu	0·1388
KCNS	0·972
Fe	0·0091
Reducing power	692 c.c. $\frac{N}{10}$ $KMNO_4$ per litre.

The strongly reducing condition of the solution is no doubt in part responsible for the poor extraction and if it were used over and over again on fresh ore extraction would be greatly interfered with.

CYANIDATION

Test No. 18

In this test a sample of the ore all ground through 200 mesh was agitated in cyanide solution, 2·0 pounds per ton KCN, for 24 hours. The tailing was then filtered, washed, and repulped in fresh cyanide solution of the same strength and agitated for another 24 hours. The final tailing was filtered, washed, and assayed for gold.

The tailing assayed:—

Gold	0·07 oz./ton
Silver	0·83 "

94·2 per cent of the gold and 80·0 per cent of the silver being extracted.

FLOTATION AND BLANKETING

Test No. 19

In this test the ore was ground for one hour in a ball mill and then floated. The flotation tailing was then passed over a corduroy blanket. The products were assayed for gold and silver.

Summary:

Product	Weight, per cent	Assays		Distribution of precious metals, per cent	
		Au oz./ton	Ag oz./ton	Au	Ag
Flotation concentrate	15·9	6·80	24·46	94·6	95·1
Blanket concentrate	8·9	0·32	0·56	2·5	1·2
Blanket tailing	75·2	0·045	0·20	2·9	3·7
Feed (cal.)	100·0	1·14	4·09	100·0	100·0

CONCLUSIONS

Results of the test work show conclusively that this ore cannot be successfully treated by an all-cyanide process or by cyanidation following amalgamation. High tailing assays and the strongly reducing condition of the pregnant solution, which will probably send the tailing higher in the next cycle, render this process impractical.

Concentration and regrinding of the sulphides in an all-cyanide flow-sheet improves extraction to some extent but not enough to produce a good tailing. (*See* Tests Nos. 15 and 16.)

It does, however, seem good practice to grind the ore 95 per cent through 200 mesh and take off a high-grade flotation concentrate for sale to a smelter after it has been barrel-amalgamated, and cyanide the flotation tailing. As the ore can be floated in neutral or lime pulp no reconditioning of the flotation tailing will be necessary before cyanidation. (*See* Tests Nos. 20, 22, and 24.) A cyanide tailing assaying Au 0·01 ounce per ton can be produced from the flotation tailing in 24 hours with cyanide consumption reduced to the low figure of 0·40 pound KCN per ton of tailing.

A comparison of results obtained by different grindings may be obtained from Tests Nos. 13 and 14. Considering these two tests, the coarser grinding of Test No. 13 is to be recommended because the concentrate produced is higher grade, the ratio of concentration is better, and more of the gold will be recovered as bullion at the mine. Still coarser grinding might prove more desirable depending on the grade of concentrate wanted and the ease with which the flotation tailing can be cyanided. This information, if required, will call for further experimental work on the ore.

Ore Dressing and Metallurgical Investigation No. 537

COPPER ORE FROM VENTURES, LIMITED, OPEMISKA LAKE, QUEBEC

Shipment. A shipment consisting of six sacks of ore, numbered 1, 4, 5, 6, 7, and 8, total weight approximately 150 pounds, was received October 10, 1933, from the Haileybury Assay Office. This was shipped on instructions from M. B. Huston for Ventures, Limited, 100 Adelaide Street West, Toronto, and came from that company's holdings at Lake Opemiska, Quebec.

Characteristics of the Ore. The material had been crushed to about ¼-inch and consisted of sulphides of copper and iron in a siliceous gangue. The following analyses were furnished by the Haileybury Assay Office:—

Sample No.	Gold, oz./ton	Silver, oz./ton	Copper, per cent
1	0·07	1·30	11·08
4	0·30	0·80	8·28
5	0·17	1·70	14·00
6	0·02	trace	1·72
7	0·09	1·50	9·56
8	0·13	1·90	7·40

Experimental Tests

One half of each sample was reserved for further investigations. The remainders of Samples Nos. 1, 4, 5, 7, and 8 were mixed, crushed to pass 10 mesh and sampled. This mixture on analysis was found to contain 10·66 per cent copper, and 0·125 ounce gold per ton.

After sampling, the mixture was passed over a quarter-deck Buchart table and a concentrate roughed off. The tailing was reserved for flotation.

Table concentration recovered 69·5 per cent of the copper and 72·1 per cent of the gold, a ratio of concentration of 2·13 : 1. This product contained 15·80 per cent copper; 0·22 ounce gold per ton.

Flotation of the reground table tailing recovered 94 per cent of the copper and 87·0 per cent of the gold, or a total recovery by table concentration and flotation of 98·2 per cent of the copper and 96·4 per cent of the gold.

The flotation concentrate assayed 31·50 per cent copper; 0·36 ounce gold per ton.

TABLING

Product	Weight, per cent	Assay		Distribution of metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Feed	100·0	10·66	0·125	100·0	100·0
Table concentrate	46·9	15·80	0·22	69·5	72·1
Table tailing	53·1	6·13	0·075	30·5	27·9

Ratio of concentration is 2·13 : 1.

The table concentrate had the following analysis:—

Copper	15·8 per cent
Gold	0·22 oz./ton
Iron	30·96 per cent
Sulphur	18·4 "
Silica	20·4 "

FLOTATION OF TABLE TAILING

The tailing from the table was ground wet to pass 78 per cent through 200 mesh, 10 pounds soda ash was added to the ball mill and a concentrate removed following the addition of 0·10 pound sodium xanthate and 0·12 pound pine oil per ton. The concentrate was cleaned twice.

Product	Weight, per cent	Assay		Distribution of metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Feed	100·0	6·13	0·075	100·0	100·0
Flotation concentrate	18·3	31·50	0·36	94·0	87·0
Flotation middling	2·2	12·02	0·085	4·3	2·5
Flotation tailing	79·5	0·13	0·01	1·7	10·5

Ratio of concentration is 5·46 : 1.

The flotation concentrate had the following analysis:—

Copper	31·50 per cent
Gold	0·36 oz./ton
Iron	30·65 per cent
Sulphur	32·6 "
Silica	2·3 "

From each 100 tons of feed to the table there is recovered 46·9 tons of table concentrate assaying 15·8 per cent copper and 0·22 ounce gold per ton and 9·1 tons of flotation concentrate assaying 31·5 per cent copper and 0·36 ounce gold. This gives 56 tons of mixed concentrates assaying—

Copper	18·35 per cent
Gold	0·24 oz./ton
Iron	30·9 per cent
Sulphur	20·74 "
Silica	17·49 "

with a total recovery of 98·2 per cent of the copper and 96·4 per cent of the gold, not taking into consideration the metals in the flotation middling.

CONCENTRATION OF SAMPLE No. 6

This low-grade sample was ground wet with 10 pounds soda ash per ton to pass 78 per cent minus 200 mesh and floated with 0·10 pound sodium xanthate and 0·12 pound pine oil per ton. The concentrate was cleaned once.

Product	Weight, per cent	Assay		Distribution of metals, per cent	
		Cu, per cent	Au, oz./ton	Cu	Au
Feed	100·0	1·72	0·02	100·0	100·0
Flotation concentrate	7·3	22·12	0·45	91·7	73·6
Flotation middling	1·1	2·47	0·24	1·5	5·9
Flotation tailing	91·6	0·13	0·01	6·8	20·5

Each 100 tons of ore milled produces 7·3 tons of concentrate assaying:

Copper	22·12 per cent
Gold	0·45 oz./ton
Iron	33·7 per cent
Sulphur	37·45 "
Silica	2·29 "

The concentrate produced in these tests was examined for magnetic content.

A sample of the table concentrate from Samples Nos. 1, 4, 5, 7, and 8 was found to contain 23·4 per cent magnetics. Chemical analysis showed this to contain 72·85 per cent magnetite and 15·29 per cent pyrrhotite. The table concentrate, therefore, contains 17·0 per cent magnetite, 3·6 per cent pyrrhotite.

The flotation concentrate from the table tailing contained only 0·6 per cent magnetic minerals, and that from Sample No. 6, 4 per cent.

CONCENTRATION OF COMPOSITE SAMPLE FROM ALL LOTS

Previous tests were made by roughing off a large amount of the mineral on tables followed by flotation of the table tailing. In this test, the entire ore was floated.

Samples Nos. 1, 4, 5, 6, 7, and 8 were mixed and assayed. The mixture was found to contain:—

Copper	9·14 per cent
Gold	0·09 oz./ton
Silver	1·22 "

A sample of the ore was ground wet to pass 78 per cent through 200 mesh, 10 pounds soda ash per ton was added to the grinding mill. A concentrate was removed following the addition to the flotation machine of 0·07 pound sodium xanthate and 0·10 pound pine oil per ton. The concentrate was cleaned once.

Results:

Product	Weight, per cent	Assay			Distribution of metals, per cent		
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
Flotation concentrate	30·7	29·48	0·36	3·26	98·3	89·4	82·3
Flotation middling	2·9	1·30	0·225	0·79	0·4	5·3	1·9
Flotation tailing	66·4	0·18	0·01	0·29	1·3	5·3	15·8

These results show that 98·7 per cent of the copper, 94·7 per cent of the gold and 84·2 per cent of the silver is floated in the rougher concentrate, which has an analysis of 27·05 per cent copper, 0·35 ounce gold and 3·05 ounces silver per ton.

Samples of the flotation products were passed through a magnetic separator and the magnetic products analysed. Magnetite and pyrrhotite were then calculated.

Product	Magnetite, per cent	Pyrrhotite, per cent
Concentrate	0·6	1·7
Middling	8·7	4·7
Tailing	15·4	1·0

These results indicate that flotation eliminates practically all magnetite from the concentrate. Previous tests, in which a rough concentration was made by jigs or tables, showed the product of concentration to contain 17·0 per cent magnetite.

Ore Dressing and Metallurgical Investigation No. 538

GOLD ORE FROM THE GUNVILLE CLAIMS, MANITOBA

Shipment. A shipment of 10 bags of gold ore weighing 500 pounds was received October 10, 1933, from G. H. Clare & Company, Saskatoon, Sask., said to come from the Gunville claims, near Sherridon, Manitoba.

Characteristics of the Ore. Polished sections of the ore were examined microscopically and found to consist of a gangue essentially of quartz and carbonate, through which sulphides and native gold are rather sparsely distributed. The metallic minerals identified were pyrite, chalcopyrite, sphalerite, pyrrhotite, arsenopyrite, and native gold.

A large percentage of the gold occurs in the pyrite, which is commonly coarse-textured.

EXPERIMENTAL TESTS

After crushing, grinding, and sampling, analysis showed the shipment to contain 0·155 ounce gold and 0·02 ounce silver per ton.

The test work included amalgamation, flotation, blanket concentration, and cyanidation.

Approximately 84 per cent of the gold is free at minus 100-mesh grinding and was recovered by jar-amalgamation.

Cyanidation recovered 96·8 per cent of the gold from minus 48-mesh material. No increase in extraction was obtained by finer grinding.

Flotation of minus 200-mesh material recovers 86 per cent of the gold.

AMALGAMATION AND CYANIDATION

Test No. 1

A sample of the ore ground to pass 48 mesh with 40 per cent minus 200 mesh, was jar-amalgamated. The tailing from this operation was cyanided 48 hours with a 1 pound KCN solution, 1 : 3 dilution; 5 pounds lime added for protective alkalinity.

Results:

Feed	0·155 oz./ton
Amalgamation tailing	0·04 "
Recovery	74·2 per cent
24-hour cyanide tailing	0·005 oz./ton
48-hour cyanide tailing	0·005 "
Total recovery	96·8 per cent

Test No. 2

A sample ground to pass 100 mesh was treated in the same manner.

Results:

Feed	0·155 oz./ton
Amalgamation tailing	0·025 "
Recovery	83·9 per cent
24-hour cyanide tailing	0·005 oz./ton
48-hour cyanide tailing	0·005 "
Total recovery	96·8 per cent

CYANIDATION

Test No. 3

Samples of the ore were ground to different sizes and cyanided 1 : 3 dilution for 48 hours with a 1 pound KCN solution. Sufficient lime to maintain protective alkalinity was added.

Results:

Mesh	Agitation, hours	Feed, Au oz./ton	Tailing, Au oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
					KCN	CaO
48	24	0·155	0·005	96·8	0·3	3·6
48	48	0·155	0·005	96·8	0·3	3·8
100	24	0·155	0·005	96·8	0·3	4·5
100	48	0·155	0·005	96·8	0·3	4·5
150	24	0·155	0·005	96·8	0·3	6·8
150	48	0·155	0·005	96·8	0·3	6·8
200	24	0·155	0·005	96·8	1·5	7·8
200	48	0·155	0·005	96·8	1·8	9·7

The gold in the ore is readily soluble in cyanide solutions, maximum extraction being obtained from 48-mesh material within 24 hours.

FLOTATION AND BLANKET CONCENTRATION

Test No. 4

A sample of the ore was ground in water together with 5 pounds soda ash and 0·07 pound Aerofloat No. 25 per ton until 71 per cent passed 200 mesh. Following the conditioning of the pulp with 0·10 pound sodium xanthate and 0·06 pound pine oil, a concentrate was removed by flotation. After this, the flotation tailing was passed over a corduroy blanket and a second concentrate removed.

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Feed (cal.)	100·00	0·15	100·0
Flotation concentrate	2·91	4·11	79·1
Flotation tailing	....	0·03	....
Blanket concentrate	6·67	0·135	6·0
Blanket tailing	90·42	0·025	14·9

The results show that 79·1 per cent of the gold is recovered in a flotation concentrate—ratio of concentration 34·4 : 1, assaying 4·11 ounces gold per ton. Blankets remove 6 per cent of the gold in a product a little lower in value than the feed. The procedure leaves a tailing containing 0·025 ounce gold.

Test No. 5

A sample was ground to pass 90 per cent through 200 mesh and floated as in Test No. 4.

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Feed (cal.)	100·00	0·17	100·0
Concentrate	3·96	3·80	86·2
Tailing	96·04	0·025	13·8

Flotation of the finely ground ore recovers 86·2 per cent of the gold in a product assaying 3·80 ounces per ton, with a ratio of concentration of 25·3 : 1.

Test No. 6

A sample ground 46·5 per cent minus 200 mesh was passed over a corduroy blanket and a concentrate removed.

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Feed	100·00	0·14	100·00
Blanket concentrate	7·36	1·40	73·5
Blanket tailing	92·64	0·04	26·5

SUMMARY AND CONCLUSIONS

The shipment is of low value, containing but 0·155 ounce per ton in gold.

Flotation results in a recovery of 86·2 per cent of the gold from ore ground 90 per cent minus 200 mesh, with a ratio of concentration of 25·3 : 1.

Cyanidation of the ore ground to pass 48 mesh recovers 96·8 per cent of the gold within 24 hours. Finer grinding or longer agitation does not increase the recovery.

A combination of flotation and blanket concentration does not give maximum recovery.

As the gold in the ore is readily soluble in cyanide solution, flotation and blanket concentrates would be amenable to cyanidation also.

The process indicated by these tests for this character of ore is straight cyanidation of approximately 48-mesh material. This recovers 0·02 ounce a ton more than any other process investigated.

Ore Dressing and Metallurgical Investigation No. 539

GOLD ORE FROM MERIDIAN MINING COMPANY, LIMITED, CAMBORNE, BRITISH COLUMBIA

Shipment. A shipment of 600 pounds of ore was received October 11, 1933, from the Meridian Mining Company, Limited. These samples were taken from the Eva and Criterion veins of the company's property near Camborne.

Character of Ore. Four polished sections were made from selected specimens and were examined microscopically.

The gangue is milky white quartz and very fine-textured, greenish grey, schistose, siliceous rock, which is probably somewhat chloritic in nature.

The metallic minerals present in the polished sections are pyrite, galena, chalcopyrite, and sphalerite. The pyrite is sparingly disseminated in the greenish grey gangue in the form of imperfect crystals. Galena, chalcopyrite, and sphalerite occur in very small amount as small irregular masses, stringers, and discontinuous veinlets in the quartz gangue. The latter three minerals are usually intimately associated. Some coarse-grained native gold was also observed in the hand specimens, but none happened to be present in the polished sections made of the ore.

Sampling and Analysis. The whole shipment was crushed and sampled. The analysis of the sample cut from the shipment was as follows:—

Gold	0·66 oz./ton	Zinc	0·28 per cent
Silver	0·42 "	Silica	89·2 "
Lead	0·26 per cent	Iron	2·5 "
Copper	0·01 "	Sulphur	0·76 "

Arsenic—trace

EXPERIMENTAL TESTS

A series of flotation tests was run, but owing to the presence of coarse free gold a low tailing could not be obtained and it was found necessary to pass the flotation tailing over a corduroy blanket in order to recover the coarse gold.

Straight cyanidation and plate amalgamation tests were also made.

FLOTATION

Five flotation tests were made on ore ground through various screen sizes. The following reagents were used in Tests Nos. 1, 4, 5:—

	Lb./ton
Soda ash	3·0
Aerofloat No. 25	0·05
Amyl xanthate	0·10
Pine oil	0·02

In Test No. 2 a second concentrate was made by the addition of ½ pound copper sulphate per ton. In Test No. 4, in addition to the reagents given above, ½ pound copper sulphate per ton was used.

The results of these tests are summarized in the following table, which is followed by tables giving the results of screen tests to show the size to which the ore was ground for each test.

Test No.	Product	Weight, per cent	Assays, Au, oz./ton	Distribution of gold, per cent
1	Concentrate	5·5	10·98	86·00
	Tailing	94·5	0·105	14·00
	Total	100·0	0·70	100·00
2	Concentrate No. 1	3·9	16·46	89·1
	Concentrate No. 2	3·5	0·39	1·9
	Tailing	92·6	0·07	9·0
	Total	100·0	0·70	100·0
3	Concentrate	4·1	14·3	75·3
	Tailing	95·9	0·20	24·7
	Total	100·0	0·77	100·0
4	Concentrate	4·8	13·02	93·6
	Tailing	95·2	0·045	6·4
	Total	100·0	0·67	100·0
5	Concentrate	7·3	9·24	91·2
	Tailing	92·7	0·07	8·8
	Total	100·0	0·73	100·0

Screen Tests:

Test No. 1

Mesh	Weight, per cent
+35	1·25
− 35+ 48	5·25
− 48+ 65	13·55
− 65+100	19·15
−100+150	15·70
−150+200	12·45
−200	32·65
Total	100·00

Test No. 2

Mesh	Weight, per cent
+ 48	1·35
− 48+ 65	6·75
− 65+100	15·80
−100+150	18·60
−150+200	16·60
−200	40·90
Total	100·00

Test No. 3

Mesh	Weight, per cent
+ 48	0·2
− 48+ 65	1·5
− 65+100	8·4
−100+150	15·5
−150+200	20·4
−200	54·0
Total	100·0

Test No. 4

Mesh	Weight, per cent
+ 65	0·5
− 65+100	5·0
−100+150	12·6
−150+200	19·9
−200	62·0
Total	100·0

Test No. 5

Mesh	Weight, per cent
+ 65	0·1
− 65+100	2·2
−100+150	8·4
−150+200	16·9
−200	72·4
Total	100·0

It is apparent from the results of these tests that it is not necessary to grind finer than 60 per cent through 200 mesh, provided only one or two per cent remains on 65 mesh.

A detailed analysis was made of the concentrate from Test No. 4. This concentrate would represent approximately the grade of product that would be available for shipment to a smelter.

Gold	13·02 oz./ton
Silver	6·92 "
Lead	4·94 per cent
Zinc	2·10 "
Iron	12·05 "
Sulphur	11·74 "
Silica	41·03 "

The ratio of concentration would be about 21 : 1.

BLANKET CONCENTRATION OF FLOTATION TAILING

The tailings from flotation Tests Nos. 2, 3, 4, and 5 were combined and sampled. The combined sample was found to assay

Gold .... 0·075 oz./ton

This was repulped with water and run over a stationary plate covered with corduroy blanket cloth.

The recovery on the blanket was 57·4 per cent.

FLOTATION AND BLANKET CONCENTRATION COMBINED

Taking the average recovery obtained from all five flotation tests, which is 87·3 per cent, the additional recovery of 57·4 per cent of the gold remaining in the flotation tailing brings the total recovery up to 95 per cent.

The total ratio of concentration for shipping purposes can be taken at 20 : 1, and the average analysis of the concentrate for this grade of ore would be as shown for Test No. 4.

AMALGAMATION ON STATIONARY PLATES

Two lots of ore were ground to approximately 50 per cent and 60 per cent respectively through 200 mesh in a small wet grinding ball mill and passed over a small stationary amalgam plate.

Test No. 1:

Amalgamation tailing	0·25 oz./ton
Recovery on plate	62·2 per cent

Test No. 2:

Amalgamation tailing	0·20 oz./ton
Recovery on plate	69·7 per cent

CYANIDATION

Three lots of ore were ground respectively to approximately 50 per cent, 60 per cent, and 70 per cent through 200 mesh, and cyanided.

Results:

Test No.	Assay of tailing, Au oz./ton	Extraction, per cent	Reagents consumed, lb./ton ore	
			KCN	CaO
1	0·01	98·5	0·37	3·0
2	0·01	98·5	0·37	3·0
3	0·01	98·5	0·72	3·0

The total time of treatment was 48 hours' agitation in an 0·05 per cent KCN solution.

The results of these cyanide tests indicate that the ore is amenable to treatment by cyanidation. The extraction is high and the cyanide and lime consumption are low.

CONCLUSIONS

The experimental work shows that either flotation or straight cyanidation will give high recovery from this ore. In the test work cyanidation gave slightly higher recovery, but in actual practice the recovery by either method would likely prove to be about the same.

The choice of method depends more on local conditions and financial considerations, such as freight and smelter rates, and capital cost of plant, rather than on metallurgical results.

If a cyanide plant is erected, grinding in cyanide solution to approximately 60 per cent through 200 mesh is recommended, followed by at least 30 hours' agitation. The use of filters in place of counter-current decantation is also suggested. If a flotation mill be decided on, provision should be made to grind to 60 per cent through 200 mesh. The tailing from the flotation machines should be passed over blankets in order to save any coarse gold not recovered by flotation. The use of a unit cell between the ball mill and classifier is strongly recommended.

Ore Dressing and Metallurgical Investigation No. 540

GOLD ORE FROM MIKADO MINE AT BAG BAY, SHOAL LAKE, LAKE OF THE WOODS, ONTARIO

Shipment. A shipment of three sacks of ore, net weight approximately 200 pounds, was received November 2, 1933. The sample was submitted by I. A. Lindsley, Vice-President, Kenora Prospectors and Miners, Limited, Box 581, Kenora, Ontario.

Characteristics of the Ore. The gangue consists essentially of light grey quartz which is traversed by small irregular stringers of grey carbonate. Inclusions of light green (probably chloritic) altered country rock are present locally.

The metallic minerals occur in very irregular and discontinuous stringers in the quartz, and are in some cases associated with the carbonate. They comprise pyrite, chalcopyrite, pyrrhotite, sphalerite, tetradymite, and native gold.

An average analysis of the ore was as follows:—

Gold	0·78 oz./ton
Silver	0·44 "

Experimental Tests

A series of small-scale experimental tests was carried out on this ore along the same lines as the work done on shipments of ore from the Cedar Island property received last year. The object was to find out whether or not this ore would respond to any or all of the processes by which the Cedar Island ore was treated and so make possible the treatment of both ores simultaneously in one mill.

The work included tests by amalgamation, cyanidation, flotation, blanketing, and hydraulic classification. In the hydraulic classifier 58·1 per cent of the gold settled out with 9·1 per cent of the weight of the ore. By amalgamation and flotation of the amalgamation tailing 96·9 per cent of the gold was recovered. In this case 86·5 per cent of the gold was recovered by amalgamation. By straight cyanidation recoveries ranging from 95·5 per cent of the gold when ore ground dry through 48 mesh was agitated for 24 hours to 98·7 per cent of the gold when ore ground dry through 200 mesh was agitated for 48 hours, were obtained. Flotation followed by blanketing produced a flotation concentrate containing 83·5 per cent of the gold and a blanket concentrate containing 9·9 per cent of it with a combined ratio of concentration of 8·5 : 1. Blanketing of the ore placed 78·6 per cent of the gold in a blanket concentrate amounting to 5·5 per cent of the weight of the ore and assaying 11·66 ounces per ton in gold. Details of the tests follow.

HYDRAULIC CLASSIFICATION

Test No. 1

In this test the ore at −14 mesh was ground in a ball mill for 15 minutes and then put through a hydraulic classifier where the gold and heavy minerals were allowed to settle against a slowly rising current of water. The two products were assayed for gold.

Summary:

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Classifier oversize	9·1	5·195	58·1
Classifier overflow	90·9	0·375	41·9
Feed (cal.)	100·0	0·814	100·0

AMALGAMATION AND FLOTATION

Test No. 2

In this test the ore at −14 mesh was ground in a ball mill for 15 minutes and the pulp amalgamated with mercury in a jar mill for 30 minutes. The amalgamation tailing was then floated with the following reagents:—

Na_2CO_3	4·0 lb./ton
Potassium amyl xanthate	0·10 "
Pine oil	0·15 "

The flotation concentrate and tailing were assayed for gold and the amalgamation tailing assay calculated from them.

Summary:

Feed sample, Au 0·78 oz./ton

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Flotation concentrate	3·4	3·45	68·8
Flotation tailing	96·6	0·055	31·2
Amalgamation tailing (cal.)	100·0	0·170	100·0

Recovery by amalgamation (0·78−0·170)÷0·78	78·2 per cent
Recovery in flotation concentrate (100·0−78·2)×68·8	15·0 "
Net recovery	93·2 "
Loss in flotation tailing	6·8 "

A screen test on the flotation tailing showed the grinding to be as follows:—

Mesh	Weight, per cent	Cumulative weight, per cent
+ 48	0·4	0·4
− 48+ 65	3·0	3·4
− 65+100	12·1	15·5
−100+150	18·8	34·3
−150+200	19·7	54·0
−200	46·0	100·0
Total	100·0	

Test No. 13

This test is similar to Test No. 2 in all respects but grinding. The ore at − 14 mesh was ground in a ball mill for 20 minutes instead of 15 minutes. The reagent combination was exactly the same.

Product	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
Flotation concentrate	4·7	1·88	77·3
Flotation tailing	95·3	0·025	22·7
Amalgamation tailing (cal.)	100·0	0·105	100·0

Recovery by amalgamation (0·78−0·105)÷0·78 ... =86·5 per cent
Recovery in flotation concentrate (100·0−86·5)×77·3 ... =10·4 "

Net recovery ... 96·9 "
Loss in flotation tailing ... 3·1 "

A screen test on the flotation tailing showed the grinding to be as follows:—

Mesh	Weight, per cent	Cumulative weight, per cent
+100	0·6	0·6
−100+150	5·7	6·3
−150+200	23·0	29·3
−200	70·7	100·0
Total	100·0	

CYANIDATION

Tests Nos. 3 to 10

In this series of tests four lots of the ore were ground dry all through the following screen sizes: 48, 100, 150, and 200 mesh. Samples of each lot were agitated in cyanide solution, 2 pounds per ton KCN, for periods of 24 and 48 hours. The cyanide tailings were assayed for gold.

Summary:

Test No.	Mesh	Period of agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
					KCN	CaO
3	− 48	24	0·035	95·5	0·60	3·7
4	−100	24	0·025	96·8	0·60	4·0
5	−150	24	0·020	97·4	0·90	4·0
6	−200	24	0·020	97·4	0·90	4·0
7	− 48	48	0·020	97·4	0·60	3·8
8	−100	48	0·020	97·4	0·60	4·1
9	−150	48	0·015	98·1	0·90	4·1
10	−200	48	0·010	98·7	0·90	4·1

AMALGAMATION AND CYANIDATION

Tests Nos. 11 and 12

In these tests samples of the ore ground dry through 48- and 100-mesh screens were amalgamated with mercury in a jar mill for 30 minutes. The amalgamation tailings were sampled and assayed for gold and portions of each agitated in cyanide solution, 2·0 pounds KCN per ton for 24 hours. The cyanide tailings were also assayed for gold.

Summary:

Test No.	Mesh	Amalgamation tailing assay, Au oz./ton	Cyanide tailing assay, Au oz./ton	Extraction by amalgamation, per cent	Extraction by cyanidation, per cent	Reagents consumed, lb./ton	
						KCN	CaO
11	− 48	0·185	0·03	76·3	19·9	0·6	3·8
12	−100	0·24	0·025	69·2	21·6	0·6	4·0

FLOTATION AND BLANKETING

Test No. 14

In this test the ore at −14 mesh was ground in a ball mill for 30 minutes and then floated. The flotation tailing was then passed over corduroy blankets. The flotation concentrate, the blanket concentrate, and the blanket tailing were assayed for gold.

Charge to Ball Mill:

Ore	1,000	grammes
Water	750	c.c.
Na_2CO_3	4	lb./ton
Aerofloat No. 25	0·07	

Reagents to Cell:

Potassium amyl xanthate	0·10 lb./ton
Pine oil	0·10 "

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Flotation concentrate	7·7	9·38	83·5
Blanket concentrate	3·7	2·31	9·9
Blanket tailing	88·6	0·065	6·6
Feed (cal.)	100·0	0·87	100·0

BLANKETING

Test No. 15

In this test 1,000 grammes of the ore at −14 mesh was ground in a ball mill for 30 minutes without the addition of any reagents. The pulp was then passed over a corduroy blanket and the concentrate and tailing were assayed for gold.

Summary:

Product	Weight, per cent	Assay, Au, oz./ton	Distribution of gold, per cent
Blanket concentrate	5·5	11·66	78·6
Blanket tailing	94·5	0·185	21·4
Feed (cal.)	100·0	0·82	100·0

CONCLUSIONS

This ore is not so readily reduced by grinding as is the Cedar Island ore. Apart from this, however, the response to various methods of treatment is approximately the same. Maximum recovery by amalgamation was slightly higher at 86·5 per cent of the gold (*see* Test No. 13), whereas maximum recovery obtained by flotation was 83·5 per cent of the gold (*see* Test No. 14). The amalgamation test here referred to was of the barrel-amalgamation type and although it cannot be said with certainty that these results would be duplicated using amalgamation plates, the results are nevertheless useful for purposes of comparison.

Recovery by cyanidation ranging from 95·5 to 98·7 per cent of the gold is just about on a par with that obtained on Cedar Island ore, and at the same time the reagent consumption is somewhat lower. The hydraulic classifier, which gives a fair indication of results to be expected from the use of a hydraulic trap in practice, gave an oversize product containing 58·1 per cent of the gold and amounting to 9·1 per cent of the weight of the ore. This also corresponds closely to the results obtained from similar tests on Cedar Island ore.

There seems, therefore, to be no reason why these two ores should not be treated in the same mill, but by reason of the difference in toughness of the two ores it will be necessary to maintain a constant proportion of each of them in the mill feed in order not to upset the grinding circuit.

Ore Dressing and Metallurgical Investigation No. 541

GOLD ORE FROM THE MANLEY MINE, NEAR DUPUY, QUEBEC

Shipment. A shipment of gold-bearing quartz ore from the Manley mine, near Dupuy, Quebec, was received on November 17 and consisted of 10 bags having an approximate weight of 700 pounds. The shipment was submitted by Dr. T. L. Gledhill, Consulting Geologist, Room 505, 67 Yonge Street, Toronto, Ontario.

Characteristics. The gangue consists of milk-white vein quartz. A very small amount of carbonate accompanies the sulphides in the stringers. The sulphides are intimately admixed and occur in irregular and discontinuous stringers in the quartz. They comprise pyrite, chalcopyrite, sphalerite, galena, and covellite. The pyrite has been considerably shattered, veined, and actively replaced by sphalerite, chalcopyrite, and galena. The native gold is intimately associated with pyrite, sphalerite, and galena, and to a lesser extent with chalcopyrite. Microscopic measurements show the approximate grain size of the gold to be rather coarse.

No minerals resembling tellurides of gold were seen and it is thought that most, if not all, of the metal occurs in the native state.

Experimental Tests

The ore was ground and sampled by standard methods and a feed sample assayed as follows:—

Gold	1·44 oz./ton
Silver	2·92 "
Tellurium	0·01 per cent
Tungsten	0·22 "
Lead	0·22 "
Copper	0·23 "
Zinc	0·05 "

Standard cyanidation, amalgamation, flotation, and blanket concentration tests were carried out.

CYANIDATION

Tests Nos. 1 and 2

These were cyanidation tests on different sizes of ore. Samples of 200 grammes were agitated in a cyanide solution of 1 pound KCN per ton and 5 pounds CaO per ton, with a pulp ratio of 3 : 1.

Results of Test No. 1—24-hour agitation:

Product	Assays, oz./ton		Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent	
	Au	Ag	KCN	CaO	KCN	CaO	Au	Ag
− 48	0·55	1·06	1·10	0·72	1·95	5·34	61·8	63·7
−100	0·375	0·94	0·85	0·57	2·70	5·79	73·9	67·7
−150	0·21	0·81	0·60	0·47	3·45	6·09	85·4	72·2
−200	0·19	1·06	0·50	0·42	3·75	6·24	86·8	63·7

Screen Tests on Cyanide Tailings Test No. 1:

−48 *Mesh*

Mesh	Weight, per cent
+ 65	12·6
+100	20·4
+150	18·0
+200	15·0
−200	34·0

−100 *Mesh*

Mesh	Weight, per cent
+150	18·9
+200	29·2
−200	51·9

−150 *Mesh*

Mesh	Weight, per cent
+200	17·2
−200	82·8

Results of Test No. 2—48-hour agitation:

Product	Assays, oz./ton		Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent	
	Au	Ag	KCN	CaO	KCN	CaO	Au	Ag
− 48	0·445	0·94	1·7	0·75	2·4	5·25	69·1	67·8
−100	0·105	0·69	1·3	0·55	3·6	5·85	92·7	76·3
−150	0·075	0·63	1·0	0·40	4·5	6·30	94·8	78·4
−200	0·095	0·78	0·8	0·35	5·1	6·55	93·4	73·2

The results of straight cyanidation show fairly high gold content in the tailing. The assays also indicate that the time of agitation is a determining factor in the gold extraction. This is to be expected from the nature of the gold grains in the ore.

AMALGAMATION AND CYANIDATION

Amalgamation followed by cyanidation of the amalgamation tailings was carried out on two samples of ore, −48 mesh and −100 mesh. Amalgamation consisted of barrel-amalgamating 1,000 grammes of ore, 1,000 c.c. water, 7·5 c.c. (approx. 100 grammes) of mercury for 1 hour. The amalgam was separated in a hydraulic classifier and the tailing filtered and two 200-gramme samples were cyanided in a KCN solution of 1 pound per ton with 5 pounds CaO per ton at a pulp dilution of 3 : 1 for 24 hours and 48 hours respectively. A screen analysis was made on 500 grammes.

Test No. 3

Amalgamation −48 mesh:

Weight, grammes	Product	Assay, Au, oz./ton	Recovery, per cent
1,000	Tailing	0·58	59·7

Cyanidation of Amalgamation Tailing:

Time, hours	Product	Assay, oz./ton		Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent
		Au	Ag	KCN	CaO	KCN	CaO	Au
24	−48	0·105	1·00	1·0	0·4	2·25	3·8	81·9
48	−48	0·075	0·80	0·9	0·5	2·55	3·5	87·0

Test No. 4

Amalgamation —100 mesh:

Weight, grammes	Product	Assay, Au, oz./ton	Recovery, per cent
1,000	Tailing	0·71	50·7

Time, hours	Product	Assay, oz./ton		Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent
		Au	Ag	KCN	CaO	KCN	CaO	Au
24	−100	0·085	0·96	0·7	0·3	3·15	4·1	85·3
48	−100	0·075	0·95	0·4	0·2	4·05	4·4	87·0

Test No. 3

Gold recovered by amalgamation = 59·7 per cent
Gold recovered by cyanidation=87 per cent of 40·3 per cent = 35·1 "

Overall recovery = 94·8 "

Test No. 4

Gold recovered by amalgamation = 50·7 per cent
Gold recovered by cyanidation=87 per cent of 49·3 per cent = 42·9 "

Overall recovery = 93·6 "

Screen Analysis Amalgamation Tailing:—

Test No. 3 —48 mesh

Mesh	Weight, per cent	Assay, Au oz./ton	Distribution of gold, per cent
+ 65	14·0	0·625	15·1
+100	24·5	0·480	20·2
+150	17·3	0·435	12·9
+200	16·9	0·470	13·7
−200	27·3	0·810	38·1
	100·0		100·0

Test No. 4 —100 mesh

Mesh	Weight, per cent
+150	13·1
+200	37·6
−200	49·3
	100·0

Gold recovery by amalgamation is low and the overall recovery after cyanidation of the tailing is no higher than by straight cyanidation. The screen analysis would indicate that the greatest amount of gold is tied up in the −200-mesh ore.

FLOTATION

Test No. 5

1,000 grammes −14-mesh ore.
2·0 lb./ton soda ash.
0·8 lb./ton potassium xanthate.
500 c.c. water.
Ground for 15 minutes.

Additions to cell:

Aerofloat No. 25..... 0·28 lb./ton

Product	Weight, per cent	Assay, oz./ton		Distribution of metals, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Concentrate	4·7	23·73	56·82	78·0	86·4	21·2 : 1
Tailing	95·3	0·33	0·44	22·0	13·6	

Test No. 6

1,000 grammes −14-mesh ore.
4·0 lb./ton soda ash.
0·8 lb./ton potassium xanthate.
500 c.c. water.
Ground for 15 minutes.

Additions to cell:

Aerofloat No. 25....... 0·28 lb./ton
Pine oil................ 0·05 lb./ton

Product	Weight, per cent	Assay, oz./ton		Distribution of metals, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Concentrate	5·2	20·28	48·88	82·2	86·0	19·2 : 1
Tailing	94·8	0·24	0·44	17·8	14·0	

Test No. 7

1,000 grammes −14-mesh ore.
2·0 lb./ton soda ash.
0·8 lb./ton potassium xanthate.
500 c.c. water.
Ground for 15 minutes.

Additions to cell:

Sodium Aerofloat....... 0·10 lb./ton
Pine oil................. 0·10 "

Product	Weight, per cent	Assay, oz./ton		Distribution of metals, per cent		Ratio of concentration
		Au	Ag	Au	Ag	
Concentrate	5·7	21·73	47·58	85·0	75·6	17·5 : 1
Tailing	94·3	0·23	0·36	15·0	24·4	

Screen Tests Flotation Tailings:

Test No. 5 Mesh	Weight, per cent	Test No. 6 Mesh	Weight, per cent	Test No. 7 Mesh	Weight, per cent
+ 48	1·5	+ 48	0·4	+ 48	0·2
+ 65	4·7	+ 65	3·6	+ 65	2·2
+100	19·1	+100	14·4	+100	11·7
+150	19·7	+150	20·7	+150	20·8
+200	18·0	+200	19·6	+200	20·4
−200	37·1	−200	41·3	−200	44·7

Flotation results show a very satisfactory concentration of gold. A test was made on the flotation tailing from Test No. 7 by cyanidation. Two 200-gramme lots were run, one for 24 hours and one for 48 hours. A cyanide solution of 1 pound KCN per ton and 5 pounds CaO per ton was used. Prior to cyanidation the tailing was ground for 30 minutes.

CYANIDATION FLOTATION TAILING

Test No. 8

Results:

Time, hours	Product	Assay, oz./ton		Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent
		Au	Ag	KCN	CaO	KCN	CaO	Au
24	Tailing	0·01	Trace	0·9	0·5	1·47	4·50	95·6
48	"	0·015	"	0·9	0·3	1·47	4·10	93·4

Screen Test of Cyanide Tailing:

Mesh	Weight, per cent,
+150	0·4
+200	3·1
−200	96·5
	100·0

	Per cent
Recoverable gold in flotation concentrate	85·0
Gold recovered by cyanidation of flotation tailing=95·6 per cent of 15·0 per cent	14·34
Overall recovery	99·34

BLANKET CONCENTRATION

Test No. 9

A sample, 1,000 grammes, of −14-mesh ore was ground for 15 minutes and the pulp was then run over a small corduroy blanket. A 200-gramme sample of the tailing was agitated in a 1 pound KCN per ton cyanide solution for 24 hours.

Blanket Concentration:

Product	Weight, per cent	Assays, oz./ton		Distribution of metals, per cent	
		Au	Ag	Au	Ag
Concentrate	7·6	10·375	11·74	59·5	30·4
Tailing	92·4	0·58	2·21	40·5	69·6

Cyanidation of Blanket Tailing:

Product	Assay, oz./ton		Final solution, lb./ton		Consumption, lb./ton		Recovery, per cent	
	Au	Ag	KCN	CaO	KCN	CaO	Au	Ag
Tailing.....	0·095	0·575	0·6	0·2	1·20	4·40	83·6	78·9

Screen Test Blanket Tailing:

Mesh	Weight, per cent
+ 48	0·8
+ 65	5·6
+100	16·7
+150	20·6
+200	16·4
−200	39·9
	100·0

	Per cent
Recoverable gold by blanket concentration	59·5
Gold recovered by cyanidation of blanket tailing, 83·6 per cent of 40·5 per cent	33·86
Overall recovery of gold	93·36
Recoverable silver by blanket concentration	30·4
Silver recovered by cyanidation of blanket tailing, 73·9 per cent of 69·6 per cent	54·43
Overall recovery of silver	84·83

SUMMARY OF RESULTS

Straight cyanidation of the ore gives a recovery of 94·8 per cent on −150-mesh ore agitated for 48 hours. Despite the good recovery a high tailing (0·075 ounce of gold per ton) is made. The grain size of the gold may account for the slower action of the cyanide, resulting in a high tailing.

Amalgamation does not appear to offer a satisfactory method of recovery. By barrel amalgamation only 59 per cent of the gold was recovered; and by cyanidation of the tailing an overall recovery of 94·8 per cent was obtained. An 0·075 ounce per ton tailing was made.

A good concentration of gold is possible by flotation. In Test No. 7, 85 per cent of the gold is recovered in the concentrate at a concentration ratio of 17·5 : 1. Cyanidation of the flotation tailing gives a 0·01 ounce per ton tailing and indicates an overall recovery of 99·34 per cent.

Blanket concentration followed by cyanidation of the tailing gives a possible overall recovery of 93·36 per cent of the gold. The cyanide tailing is higher in this lot than in straight cyanidation of the ore and consequently blanket concentration offers no advantages over straight cyanidation.

Ore Dressing and Metallurgical Investigation No. 542

FLOTATION CONCENTRATE FROM VIDETTE GOLD MINES, LIMITED, AT VIDETTE LAKE, B.C.

Shipment. A shipment of approximately 100 pounds of flotation concentrate was received October 5, 1933. The sample was submitted by Gordon F. Dickson, Managing Director, Vidette Gold Mines, Limited, Pacific Building, Vancouver, B.C.

Characteristics. The sample submitted was a sulphide concentrate carrying copper, gold, and silver.

An average analysis of the sample was as follows:—

Gold	11·875	oz./ton
Silver	26·27	"
Copper	1·68	per cent
Lead	0·05	"
Iron	25·06	"
Silica	23·33	"
Sulphur	25·63	"
Arsenic	Nil	

Experimental Tests

The object of the work done on this shipment was to see if anything could be done to improve extraction of the gold by cyanidation. The work was, therefore, limited to cyanidation tests, the details of which follow. The maximum recovery obtained was 94·2 per cent of the gold with a tailing assaying 0·69 ounce per ton in gold.

Tests Nos. 1 and 2

In these two tests samples of the concentrate were agitated without grinding in cyanide solution, 5 pound KCN per ton, for periods of 24 and 48 hours. The tailings were assayed for gold.

Summary:

Feed sample, Au 11·875 oz./ton.

Test No.	Period of agitation, hours	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
1	24	2·09	82·4	13·9	13·8
2	48	1·37	88·5	20·3	15·0

Tests Nos. 3 to 6

In this series of tests samples of the concentrate were ground for varying periods of time in a ball mill and then agitated in cyanide solution, 10 pounds KCN per ton for 48 hours. The tailings were assayed for gold.

Summary:

Feed sample, Au 11·875 oz./ton.

Test No.	Period of grinding, minutes	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton	
				KCN	CaO
3	20	0·825	93·1	23·0	19·8
4	30	0·83	93·0	25·0	19·9
5	40	0·725	93·9	26·2	19·9
6	50	0·775	93·5	30·5	20·3

Screen tests on the cyanide tailings from Tests Nos. 3 to 6 showed the grinding to be as follows:—

Test No.	Weight, per cent +65	Weight, per cent −65+100	Weight, per cent −100+150	Weight, per cent −150+200	Weight, per cent −200	Total
3	0·05	0·10	0·35	2·0	97·5	100·0
4	0·0	0·15	0·25	2·0	97·6	100·0
5	0·0	0·10	0·20	1·8	97·9	100·0
6	0·0	0·10	0·30	1·0	98·6	100·0

A further screen test on the cyanide tailing from Test No. 4 showed it to be 97 per cent through 325 mesh.

Test No. 7

In this test 1,000 grammes of the concentrate was amalgamated with mercury for 30 minutes in a jar mill. The amalgamation tailing was sampled and assayed for gold and a portion reground and agitated in cyanide solution, 10 pounds KCN per ton, for 48 hours. The cyanide tailing was also assayed for gold.

Summary:

Feed sample	11·875 oz./ton
Amalgamation tailing	11·750 "
Cyanide tailing from amalgamation tailing	0·71 "
Total extraction	94·0 per cent

Reagents Consumed:

KCN	36·3 lb./ton
CaO	19·0 "

Test No. 8

In this test a sample of the concentrate was ground for 20 minutes in a jar mill and then agitated in cyanide solution 10 pounds KCN per ton, for 24 hours. The pulp was then filtered, the tailing washed and repulped in fresh cyanide solution 10 pounds KCN per ton, and agitated for another 24 hours. The final tailing was then filtered, washed, and assayed for gold.

Summary:

Feed sample ....11·875 oz./ton
Final cyanide tailing... 0·69 "
Extraction..94·2 per cent

Reagents Consumed:

KCN...33·8 lb./ton
CaO...21·0 "

Tests Nos. 9 and 9A

The concentrate used for these tests was given a partial roast in a gas-fired muffle furnace. The loss in weight was 14 per cent and the calcine contained a considerable quantity of soluble copper and iron sulphates. The calcine assayed:—

Au..13·70 oz./ton
Ag..30·23 "

In Test No. 9 a sample of the calcine was washed and agitated in cyanide solution, 10 pounds KCN per ton, for 48 hours. In Test No. 9A the sample of calcine was ground in a jar mill for 30 minutes, then washed and agitated in cyanide solution, 10 pounds KCN per ton, for 48 hours. In each case the tailing was filtered, washed, and assayed for gold.

Summary:

Feed sample: Au 13·70 oz./ton.

Test No.	Tailing assay, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton calcine	
			KCN	CaO
9	1·95	85·8	20·2	19·0
9A	0·98	92·1	89·0	33·5

Test No. 10

In this test a sample of the concentrate was agitated in cyanide solution, 10 pounds KCN per ton, for 24 hours. It was then filtered, washed, and reground for 30 minutes. The reground tailing was then repulped in fresh cyanide solution, 10 pounds KCN per ton, and agitated for 48 hours. The final tailing was assayed for gold.

Summary:

Feed sample...11·875 oz./ton
Cyanide tailing... 0·75 "
Extraction..93·7 "

Reagents Consumed:

KCN...62·3 lb./ton
CaO...16·6 "

Test No. 11

In this test the concentrate was ground in a ball mill for 25 minutes and then agitated in cyanide solution, 10 pounds KCN per ton, for 48 hours. Lead acetate was added to the pulp in the proportion of 2 pounds per ton concentrate. The tailing was assayed for gold.

Summary:

Feed sample	11·875 oz./ton
Cyanide tailing	0·77 "
Extraction	93·6 per cent

Reagents Consumed:

KCN	49·9 lb./ton
CaO	14·5 "

CONCLUSIONS

It has not been found possible to produce a cyanide tailing lower than 0·69 ounce per ton in gold and that with a high reagent consumption. The copper in the concentrate no doubt accounts for a large part of the cyanide consumption. Not knowing the ratio of concentration by which this concentrate was produced it is impossible to calculate the results back to terms of the original ore. Treatment of the concentrate by smelting would be much more efficient from a purely metallurgical point of view, but, without detailed information as to freight rates and smelter charges, it would be difficult to say which process would give the greater net return to the owners.

Ore Dressing and Metallurgical Investigation No. 543

GOLD ORE FROM SULLIVAN GOLD MINES, LIMITED, DUBUISSON TOWNSHIP, ABITIBI COUNTY, QUEBEC

Shipment. A shipment consisting of six bags of ore, approximate weight 704 pounds, was received on October 5, 1933. It was submitted by Harry A. Kee, 100 Russel Road, Toronto, Ont., for the Sullivan Gold Mines, Limited, Dubuisson township, Que.

Characteristics. 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 through 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

The ore was crushed and sampled by standard methods and the feed assayed with the following result:—

Gold..0·785 oz./ton

Test work consisted of blanket concentration tests on different sizes of ore, followed by cyanidation of the blanket tailings.

Cyanidation tests on the raw ore were also carried out.

BLANKET CONCENTRATION TESTS

Tests Nos. 1, 2, 3, and 4

Samples of 1,000 grammes were ground to pass 48-, 100-, 150-, and 200-mesh screen. The ore was fed into a mixing-box with water from which the pulp passed over a table covered with corduroy blanket cloth. The slope of the table was $2\frac{1}{2}$ inches to the foot. The results of four tests are shown below:—

Test	Size	Product	Weight, per cent	Assays, Au, oz./ton	Distribution of gold, per cent	Ratio of concentration
1........	− 48	Concentrate.................... Tailing........................	14·1 85·9 100·0	4·37 0·19	79·0 21·0	7·1 : 1
2........	−100	Concentrate.................... Tailing........................	15·9 84·1 100·0	4·58 0·25	77·2 22·8	6·28 : 1
3........	−150	Concentrate.................... Tailing........................	10·4 89·6 100·0	5·14 0·26	69·6 30·4	9·52 : 1
4........	−200	Concentrate.................... Tailing........................	14·5 85·5 100·0	6·16 0·39	72·8 27·2	6·9 : 1

Screen Tests on Blanket Tailing:

−48 mesh

Mesh	Weight, per cent
+ 65...........	1·1
+100...........	12·3
+150...........	17·0
+200...........	17·8
−200...........	51·8
	100·0

−100 mesh

Mesh	Weight, per cent
+150...........	0·3
+200...........	22·9
−200...........	76·8
	100·0

−150 mesh

Mesh	Weight, per cent
+150...........	0·5
+200...........	8·0
−200...........	91·5
	100·0

Samples, 200 grammes, of the tailings from the blanket tests were agitated in bottles in cyanide solution, 1 pound KCN per ton, and 5 pounds CaO per ton for 24 hours at a pulp ratio of 3 : 1.

CYANIDATION TESTS

Tests Nos. 1A, 2A, 3A, and 4A

Test	Product	Assays, Au, oz./ton	Final solutions, lb./ton		Reagents consumed, lb./ton		Recovery, per cent
			KCN	CaO	KCN	CaO	
1A	−48 tailing.............	0·015	1·0	0·6	0·1	5·20	92·1
2A	−100 tailing............	0·02	1·0	0·6	0·1	5·20	92·0
3A	−150 tailing............	0·015	1·0	0·35	0·1	5·95	94·2
4A	−200 tailing............	0·025	1·0	0·45	0·1	5·65	93·5

The results indicate better gold recovery and lower tailing on the −48-mesh ore.

The capitulation of these results is as follows:—

	Per cent
—48-mesh ore—	
Recoverable gold in blanket concentrate	79·0
Gold recovery by cyanidation of blanket tailing, 92·1 per cent of 21 per cent	19·34
Possible overall recovery	98·34
—100-mesh ore—	
Recoverable gold in blanket concentrate	77·2
Gold recovery by cyanidation of blanket tailing, 92·0 per cent of 22·8 per cent	20·77
Possible overall recovery	97·97
—150-mesh ore—	
Recoverable gold in blanket concentrate	69·6
Gold recovery by cyanidation of blanket tailing, 94·2 per cent of 30·4 per cent	28·63
Possible overall recovery	98·23
—200-mesh ore—	
Recoverable gold in blanket concentrate	72·8
Gold recovery by cyanidation of blanket tailing, 93·5 per cent of 27·2 per cent	25·43
Possible overall recovery	98·23

CYANIDATION TESTS ON RAW ORE

Tests Nos. 5 and 6

Samples of 200 grammes of the ore ground to −48, −100, −150, and −200 mesh were agitated in bottles in solution of 1 pound KCN per ton with 5 pounds CaO per ton at a pulp ratio of 3 : 1 for 24 hours and 48 hours.

Test No. 5—24 hours:

Product	Assays, Au, oz./ton	Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent
		KCN	CaO	KCN	CaO	
− 48	0·01	0·95	0·6	0·15	5·20	98·7
−100	0·01	0·90	0·475	0·30	5·58	98·7
−150	0·01	0·90	0·325	0·30	6·03	98·7
−200	0·015	0·80	0·375	0·60	5·88	98·0

Test No. 6—48 hours:

Product	Assays, Au, oz./ton	Final solution, lb./ton		Reagents consumed, lb./ton		Recovery, per cent
		KCN	CaO	KCN	CaO	
− 48	0·015	0·8	0·55	0·6	5·35	98·0
−100	0·015	0·9	0·45	0·3	5·65	98·0
−150	0·015	0·9	0·30	0·3	6·10	98·0
−200	0·02	0·9	0·35	0·3	5·95	97·4

CONCLUSIONS

The results of the blanket tests show the best concentration is effected on −48-mesh ore, with a 0·015 ounce per ton tailing from cyanidation of the blanket tailing and a possible overall gold recovery of 98·34 per cent.

By straight cyanidation of the raw ore an extraction of 98·7 per cent of the gold was obtained on the −48-mesh ore with a 0·01 ounce per ton tailing, time of agitation being 24 hours.

Agitation for 48 hours yielded a slightly higher tailing which would indicate a slight re-precipitation of the gold. This condition is apparent also on the finely ground product.

Results of the tests indicate that straight cyanidation of the ore ground to −48-mesh with 24 hours' agitation in cyanide of 1 pound KCN per ton concentration offers a better gold recovery than preliminary concentration of the ore followed by cyanidation.

The lime consumption averages about 5·5 pounds per ton, and the cyanide consumption is very low.

Ore Dressing and Metallurgical Investigation No. 544

CLEANING OF BROKEN GLASS

Shipment. A shipment of five bags of broken glass, shipping weight 670 pounds, was received on May 11, 1933, from the Consumers' Glass Company, Montreal, Que.

Purpose of Experimental Tests. The glass was sent in at the request of H. J. Emery, General Manager of the Canadian Kaolin Silica Products, Limited, of Montreal, who requested that tests be made to see if the iron content of the glass could be materially reduced by a fairly simple process. Mr. Emery pointed out that about 15 per cent of the furnace charge in glass-making is broken glass, called cullet, and that if the iron in the cullet could be reduced it would result in an improvement in the colour of the glass produced by the furnace.

Characteristics of the Glass. The glass consists mostly of broken bottles and contains as impurities iron bottle caps, corks, wood, roots, paper, cinders, coal, sand, and rock.

EXPERIMENTAL TESTS

Test No. 1

One bag of broken glass was washed on an 0·525-inch opening screen. The oversize was hand-picked into clean glass and a discard consisting of bottle caps, corks, cinders, wood, and roots. The undersize, after decanting the slime, was dried and screened on a 48-mesh screen to remove fine sand, crushed to 10 mesh and tabled on a large Wilfley table making a concentrate of cinders and glass, a middling of glass, and a tailing of coal and glass. The table products were dried and run over an Ullrich high-power magnetic machine making a magnetic and a non-magnetic product in each case. The products were:—

Product	Pounds	Per cent Fe_2O_3	Pounds Fe_2O_3
+0·525-inch glass	83·25	0·23	0·1915
+0·525-inch discard	1·50		
−48 sand	0·69	8·42	0·0581
Table concentrate	1·75	11·31	0·1979
Table middling	22·87	1·02	0·2333
Table tailing	3·37	1·17	0·0394
Magnetic from table concentrate	0·25	50·96	0·1274
" " " middling	0·69	19·17	0·1323
" " " tailing	0·25	14·32	0·0358
Non-magnetic from table concentrate	1·19	0·38	0·0045
" " " middling	22·19	0·14	0·0311
" " " tailing	3·00	0·28	0·0084

This test shows that washing on the 0·525-inch opening screen and hand-picking gives about two-thirds of the feed as clean glass analysing 0·23 per cent Fe_2O_3.

Screening the dried minus 0·525-inch glass on 48 mesh removes a very small amount of sand high in iron. If this screening had not been done, quite likely the table and magnetic machine would have removed most of the iron-bearing portion of the minus 48-mesh sand.

The table feed ran about 1·68 per cent Fe_2O_3, and the glass from the table ran 1·02 per cent Fe_2O_3. This shows that the table is not very good in removing iron.

The magnetic machine did very well in reducing the iron content of the glass fed to it. The combined non-magnetics would be 0·17 per cent Fe_2O_3, which is about the result that would be obtained if the tabling had been omitted.

If the magnetic treatment had been omitted there would have been recovered 83·25 pounds of glass at 0·23 per cent Fe_2O_3 and 22·87 pounds at 1·02 per cent Fe_2O_3, or in all 106·12 pounds at 0·40 per cent Fe_2O_3.

If the tabling had been omitted there would have been recovered 83·25 pounds of glass at 0·23 per cent Fe_2O_3 and 26·38 pounds at 0·17 per cent Fe_2O_3, which would equal 109·63 pounds at 0·21 per cent Fe_2O_3.

The glass used in the test was not weighed as it was wet. The weight of dry material in it would be about 125 pounds.

Test No. 2

One bag of broken glass, 151 pounds net weight, was placed in the Akins classifier, which was used as a washer. The large pieces of glass blocked the screw flight. The glass was washed on an 0·525-inch opening screen. Wood and cork were removed from the plus 0·525-inch by dropping it into a tub kept overflowing by means of a hose. As some fines were still included in the plus 0·525 material, it was washed in a 2- by 3-foot mill for five minutes and screened again on the 0·525-inch screen, the undersize being put with the first minus 0·525 inch.

The combined minus 0·525-inch was then washed in the Akins. As there was not enough of it to discharge it was washed in the 2- by 3-foot mill for 30 minutes with water running through the mill.

All water used in the test went to a Dorr thickener.

The plus 0·525-inch when dried weighed 98 pounds. It was crushed to pass 8 mesh and run over the Ullrich high-power magnetic machine.

The minus 0·525-inch when dried weighed 49·5 pounds. It was screened on 48 mesh. The plus 48 mesh was crushed to pass 8 mesh and run over the Ullrich. The feed to the Ullrich was 45·25 pounds.

The minus 48 mesh, 0·15 pound, was run over the Ullrich.

In crushing the plus and minus 0·525-inch to 8 mesh a little iron was obtained on the 8-mesh screen in each case.

The following table gives the results:—

Product	Pounds	Per cent Fe_2O_3	Pounds Fe_2O_3
− 8 non-magnetic from +0·525 inch	97·25	0·16	0·1556
− 8 non-magnetic from −0·525 inch	44·12	0·16	0·0706
−48 non-magnetic	0·13	0·30	0·0004
− 8 magnetic from +0·525 inch	1·05	28·10	0·2951
− 8 magnetic from −0·525 inch	0·79	17·35	0·1371
−48 magnetic	0·02	50·77	0·0102
+ 8 iron from crushing +0·525 inch	0·06	142·93	0·0858
+ 8 iron from crushing −0·525 inch	0·03	142·93	0·0429
Fine from Dorr thickener	6·69	4·08	0·2730
Wood and cork	0·05	0·00	0·0000
Total	150·19	0·71	1·0707

The results of this test can be regarded as showing what would be obtained in practice by washing in an Akins built heavy and powerful enough so as not to become blocked by the large pieces of glass, drying the washed glass, screening on 48 mesh, crushing the plus 48 mesh to 8 mesh, and running the minus 8 mesh over a strong magnetic machine.

The test gave 141·37 pounds of glass analysing 0·16 per cent Fe_2O_3.

Had the glass not been screened on 48 mesh there would have been obtained 141·5 pounds of glass analysing 0·16 per cent Fe_2O_3. This shows that it is not worth while screening on 48 mesh.

If the glass had just been washed there would have been recovered 143·45 pounds analysing 0·56 per cent Fe_2O_3.

Test No. 3

One bag of broken glass, 113 pounds net, was crushed to pass a 0·525-inch opening screen. Some iron and wood was left on the screen. The minus 0·525-inch weighed 112 pounds 3 ounces. It was fed to the Akins classifier in 3 minutes and at the end of another 6 minutes very little glass was being discharged by the screw flight. The glass that was not discharged from the Akins was washed for 20 minutes in the 2- by 3-foot mill with water running through. The mill door was then taken off and washed down and the mill run for 5 more minutes. This was done to wash any glass that had stuck in the crack around the door. All water used in the test went to a Dorr thickener.

Results:

Product	Pounds	Per cent Fe_2O_3	Pounds Fe_2O_3
+0·525-inch iron	0·02	142·93	0·0286
+0·525-inch wood	0·02	0·00	0·0000
Glass from Akins	55·00	0·34	0·1870
Glass from mill	51·25	0·45	0·2306
Fine from thickener	5·37	2·33	0·1251
Total	111·66	0·51	0·5713

Washing 113 pounds of glass after crushing it to 0·525 inch gave 106·25 pounds of glass analysing 0·39 per cent Fe_2O_3. The washed glass after sampling, 54 pounds from the Akins and 50 pounds 12 ounces from the mill, was screened on 20-, 28-, and 48-mesh screens.

Results:

+ 20	99 lb.	0·37 per cent Fe_2O_3
−20 + 28	2 lb. 7 oz.	0·69 " "
−28 + 48	2 lb.	1·15 " "
−48	10 oz.	2·92 " "
	104 lb. 1 oz.	

The above sizes after sampling were run over the Ullrich machine.

Product	Pounds	Per cent Fe_2O_3	Pounds Fe_2O_3
+ 20 magnetic	0·75		
−20 + 28 "	0·04		
−28 + 48 "	0·05		
− 48 "	0·02		
+ 20 non-magnetic	96·12	0·22	0·2115
−20 + 28 "	1·83	0·14	0·0026
−28 + 48 "	1·66	0·16	0·0027
− 48 "	0·42	0·20	0·0008
Total	100·89		

Out of 113 pounds of glass there was recovered by crushing to 0·525 inch, washing and magnetic separation, 100·03 pounds of glass analysing 0·22 per cent Fe_2O_3. If no samples had been taken of the washed glass slightly more would have been recovered.

Half-inch glass is a little too large for the Ullrich machine to handle. This resulted in the coarsest non-magnetic running 0·22 per cent Fe_2O_3. Better results would be obtained if the feed to the Ullrich were smaller.

Test No. 4

One bag of broken glass, net weight 139 pounds, was crushed to pass a 4-mesh screen. A small amount of paper, wood, and iron was left on the screen. The minus 4 mesh was washed in the Akins, being fed in six minutes. The glass that did not discharge from the Akins was washed for 45 minutes in the 2- by 3-foot mill with water running through. All water used in the test went to a Dorr thickener.

Product	Pounds	Per cent Fe_2O_3	Pounds Fe_2O_3
+4 wood and paper	0·05	0·00	0·0000
+4 iron	0·08	142·93	0·1143
Glass from Akins	53·12	0·47	0·2497
Glass from mill	63·44	0·47	0·2982
Fine from thickener	10·94	1·60	0·1750
Total	127·63	0·66	0·8372

Crushing 139 pounds of glass to 4 mesh and washing it gave 116·56 pounds of glass analysing 0·47 per cent Fe_2O_3.

Fifty-two pounds eight ounces of glass from the Akins and 62 pounds 7 ounces of glass from the mill were mixed and screened on 20-, 28-, and 48-mesh screens.

− 4 + 20	94 lb. 2 oz.
−20 + 28	9 lb. 1 oz.
−28 + 48	8 lb. 5 oz.
−48	3 lb. 1 oz.
	114 lb. 9 oz.

The above sizes were run over the Ullrich magnetic machine, giving—

Product	Pounds	Per cent Fe_2O_3	Pounds Fe_2O_3
− 4 + 20 magnetic	0·62		
−20 + 28 "	0·06		
−28 + 48 "	0·12		
−48 "	0·04		
− 4 + 20 non-magnetic	93·44	0·16	0·1495
−20 + 28 "	9·00	0·12	0·0108
−28 + 48 "	8·19	0·17	0·0139
−48 "	3·06	0·26	0·0080
Total	114·53		

Out of 139 pounds of glass there was recovered 113·69 pounds analysing 0·16 per cent Fe_2O_3 by crushing to 4 mesh, washing, and magnetic separation. If no samples had been taken from the washed glass a slightly larger amount of glass would have been recovered.

A small sample of the −4 +20 non-magnetic was hand-picked into clean glass and dark product consisting mostly of coal.

Clean glass	158·7 grammes	0·15 per cent Fe_2O_3
Dark product	0·79 "	2·91 "
Total	159·49 "	0·16 "

This hand-picking showed that −4 +20 non-magnetic was very clean.

FLOTATION

One-eighth of a bag of glass, 17 pounds 13 ounces, was taken in such a way as to give a representative sample and crushed to 20 mesh. From the crushing there resulted 2·95 grammes of +20-mesh iron, 0·37 grammes of +20-mesh wood and cork, and 17 pounds 10 ounces of −20-mesh glass. Two flotation tests were made on the −20 mesh.

Test No. 1

A lot of 500 grammes of −20-mesh glass was ground for 5 minutes in a small porcelain jar half full of pebbles with 1,000 c.c. of water and 3 drops of a mixture of 15 per cent pine oil, 75 per cent coal-tar creosote and 10 per cent coal tar. The ground charge was mixed for one minute in a small

Ruth flotation machine with 3 drops of pine oil and then floated for five minutes. Only coal floated. The glass from which the coal had been floated was washed by stirring in a pail and decanting several times until all fines had been removed. The washed glass was dried and run over the Ullrich magnetic machine to remove iron, cinders, etc.

Results:

Product	Grammes	Per cent Fe_2O_3
Coal from flotation	25·7	
Fine from washing	110·5	
Magnetic	5·6	
Non-magnetic	337·5	0·23
Total	479·3	

Test No. 2

Another lot of −20-mesh glass was floated as in the first flotation test except that the glass was ground with one-quarter gramme of soda ash and floated with 3 drops of pine oil and 3 drops of oleic acid for 3 minutes and then floated with the same amount and kind of reagents for 5 minutes. Only the coal floated. The glass was washed and run over the magnetic machine as in the first test.

Product	Grammes	Per cent Fe_2O_3
Coal from flotation	17·2	
Fine from washing	111·7	
Magnetic	6·8	
Non-magnetic	351·4	0·17
Total	487·1	

The flotation tests show that only the coal floats and that there is a large loss of glass in the fines from the washing.

SUMMARY AND CONCLUSIONS

Test No. 1 shows that by screening the glass on ½-inch, washing and picking the +½-inch, crushing the −½-inch to 10 mesh, washing and drying it and running it over a magnetic separator, the cleaned +½-inch glass was 0·23 per cent Fe_2O_3 and the cleaned −½-inch glass, 0·17 per cent Fe_2O_3, or together 0·21 per cent Fe_2O_3. It is possible that a more thorough washing of the +½-inch glass than was given with a hose would remove all the slime and a lower result than 0·23 per cent Fe_2O_3 would be obtained. It would seem that the hand-picked glass if washed properly should be as low or lower than the product from the magnetic separation.

Test No. 2 indicates that by crushing the glass to 8 mesh, washing it, drying the washed glass and separating with a magnetic machine, a clean glass product 0·16 per cent Fe_2O_3 would be obtained.

Test No. 3 shows that treating the glass crushed to ½-inch by washing, drying, and magnetic separation gives a cleaned glass product 0·22 per cent Fe_2O_3.

Test No. 4 gave a clean glass product 0·16 per cent Fe_2O_3 by crushing to 4 mesh, washing, and magnetic separation.

The flotation tests gave a clean glass product of 0·23 and 0·17 per cent Fe_2O_3 but most of the cleaning was due to washing and magnetic separation.

The best method found to clean the glass, and that recommended, is to crush to 4 mesh, wash in a screw-flight washer, dry the washed glass and remove iron, cinders, etc., from it by a high-power magnetic machine.

If the glass could be fed damp to the furnace, then the magnetic separation could be made after washing and simply draining the glass. This would dispense with the drying treatment, but it would be necessary to use a magnetic machine capable of operating wet.

Ore Dressing and Metallurgical Investigation No. 545

GARNET ROCK FROM LOT 25, RANGE B, JOLY TOWNSHIP, LABELLE COUNTY, QUEBEC

Shipment. A shipment of eight bags of garnet rock was received (5 bags, gross weight 304 pounds, on October 11, 1933, and 3 bags, gross weight 191 pounds, on October 17, 1933) from Labelle, Quebec. The shipment which was sent in by Eugene McNicoll, 354 Ste. Catherine Street East, Montreal, was taken from the E. ½ lot 25, range B, Joly township, Labelle county, Quebec.

Purpose of Experimental Tests. Mr. McNicoll wished a test made using water concentration and another test using magnetic separation, in order that the most suitable method of producing a garnet concentrate might be obtained.

Characteristics of the Garnet Rock. The shipment consisted of garnet varying in size from ⅛ to 1 inch in white feldspar. A small amount of brown mica was also present.

Experimental Tests

After selecting two specimens from the five bags that were received October 11, 1933, the remaining 292 pounds 10 ounces was crushed to 8 mesh by means of a small jaw crusher, set of rolls, and hand screen. The loss in crushing was 14 ounces.

Test No. 1

A lot, 243 pounds 9 ounces, of 8 mesh was screened on a Hummer vibrating screen into the following sizes:—

Size	Lb.	Oz.
− 8+10	76	8
−10+14	32	2
−14+20	45	0
−20+26	15	14
−26+35	20	9
−35+42	7	10
−42+60	11	5
−60	28	13
	237	13

The loss in screening was 5 pounds 12 ounces.

The first two sizes, −8+10 mesh and −10+14 mesh, were treated in a James two-compartment jig. As the rate of feed was rather slow, all the concentrate was taken from the first compartment. The jig beds were cleaned up by the use of a small Harz jig.

The remaining sizes were tabled on a large Wilfley table, the middling being re-run a few times to give similar results to a table having a middling return. The products from jigging and tabling were:—

Size	Concentrate		Tailing	
	lb.	oz.	lb.	oz.
− 8+10	17	0	58	6
−10+14	8	14	22	14
−14+20	15	10	29	2
−20+26	5	12	10	2
−26+35	8	0	12	12
−35+42	3	4	5	10
−42+60	3	15	7	4
−60	5	5	15	15
Total	67	12	162	1

The loss in jigging and tabling was 8 pounds.

The jig and table concentrates were run over an Ullrich high-power magnetic separator, first using a low current so as to remove any highly magnetic material and then with a stronger current to lift the garnet and leave non-magnetic gangue minerals. The products were:—

Size	Strongly magnetic		Garnet product		Non-magnetic	
	lb.	oz.	lb.	oz.	lb.	oz.
− 8+10	0	0	15	15	1	1
−10+14	0	0	8	6	0	7
−14+20	0	0	14	12	0	14
−20+26	0	0	5	8	0	5
−26+35	0	0	7	6	0	10
−35+42	0	0	3	0	0	3
−42+60	0	0	3	11	0	7
−60	0	2	4	9	0	12
Total	0	2	63	3	4	11

No loss of material occurred during the magnetic separation. The garnet product recovered was equal to 25·86 per cent of the feed.

The concentrate appeared to be good.

Test No. 2

A lot, 48 pounds 3 ounces, of −8-mesh material was screened on 20-, 35-, and 60-mesh screens and the sizes run over the Ullrich machine, first with a low current to separate out any strongly magnetic material, and then with a stronger current to lift the garnet. The concentrate from this test was not so good as that in Test No. 1, as the mica was lifted by the magnetic machine with the garnet.

SUMMARY AND CONCLUSIONS

In jigging a small quantity of material, the results are never so good as in commercial work because to determine the recovery the bed has to be separated as well as possible and this results in the concentrate being poorer. It is believed that in a regular plant jigging would give a concentrate that would not need to be treated magnetically.

Garnet concentrate made on a table always contains a little sand and it would be necessary to use a magnetic separator to clean it.

The garnet rock contains almost no strongly magnetic material, so a low-power magnetic machine would not be needed.

The sizing in the tests described is rather close; it might be possible to get good results with less screens. The minus 60-mesh concentrate is not good; in practice it would not be necessary to use a screen so fine.

Good garnet concentrate can be recovered from the rock submitted. The recovery would be about 25 per cent of the feed.

The method of treatment recommended should follow the lines indicated in Test No. 1. The rock should be crushed to 8 mesh, screened into various sizes, the sizes larger than 14 mesh should be jigged, the minus 14-mesh material tabled, the jig and table concentrates dried, and the table concentrate run over a high-power magnetic machine.

Before proceeding with the development of the property from which the garnet rock came, it is imperative that the garnet concentrate be tested as to its suitability for abrasive papers. Also it should be ascertained whether the garnet-paper makers will accept garnet concentrate as fine as 8 mesh, as at present their garnet comes to them in much larger sizes.

Ore Dressing and Metallurgical Investigation No. 546

GARNET ROCK FROM LOT 10, RANGE I, JOLY TOWNSHIP, LABELLE COUNTY, QUEBEC

Shipment. A shipment of two bags of garnet rock, gross weight 241 pounds, was received on October 20, 1933, from Labelle, Quebec. The shipment, which was sent in by the Montreal Garnet Products Regd., 2000 McGill College Avenue, Montreal, Quebec, was taken from lot 10, range I, Joly township, Labelle county, Quebec.

Purpose of Experimental Tests. Tests were made to determine the quantity and quality of the garnet concentrate that could be recovered.

Characteristics of the Garnet Rock. The shipment consisted of small garnets ranging in size up to $\frac{1}{8}$ inch in a gangue of very fine-grained quartz, feldspar, and sillimanite. A fair amount of graphite and a small quantity of brown mica are present.

Experimental Tests

Two pieces of rock were selected as specimens, the remainder, 236½ pounds, was crushed to 8 mesh and sized.

Size	Lb.	Oz.
− 8+10	62	0
−10+14	19	2
−14+20	28	15
−20+26	20	4
−26+35	24	6
−35+42	11	3
−42+60	13	13
−60	52	3
Total	231	14

The first two sizes were jigged in a two-compartment James jig. As the feed to the jig was slow all the concentrate was taken from the first jig. There was only enough of the second size to fill one jig, so only a first jig concentrate and bed were made. The jig beds were cleaned up on a small Harz jig. The remaining sizes were tabled on a Wilfley table making a concentrate, middling, and tailing. The middlings were re-run twice to give similar results to a table with a middling return.

The concentrates from jigging and tabling were run over an Ullrich magnetic machine; first, with a weak current to lift any highly magnetic material, and then with a strong current to lift the garnet and leave non-magnetic gangue. The low current lifted practically nothing except from the −60-mesh size, from which it raised a small amount.

In running the three coarser sizes different strengths of current were used to lift the garnet, resulting in different grades of product in each size. The better grades were lifted with the lower currents. With a stronger current some middling was lifted.

The products obtained are as follows:—

Jig and Table Tailing

Size	Lb.	Oz.
−8+10	31	12
−10+14	9	12
−14+20	15	2
−20+26	10	13
−26+35	13	13
−35+42	7	0
−42+60	9	1
−60	35	6
Total	132	11

Products from Magnetic Separation:

Size	Strongly magnetic		Garnet product		Non-magnetic	
	lb.	oz.	lb.	oz.	lb.	oz.
−8+10	0	0	26	10	0	14
−10+14	0	0	8	8	0	6
−14+20	0	0	12	14	0	15
−20+26	0	0	8	15	0	11
−26+35	0	0	9	12	1	3
−35+42	0	0	3	15	0	9
−42+60	0	0	4	2	0	10
−60	0	1	5	15	1	15
Total	0	1	80	11	7	3

A current of ½ ampere was used to lift any strongly magnetic. Different strengths of current varying from 3 to 5 amperes were used to lift the garnet. In the −8+10-mesh 3, 3½, and 4 amperes were used, in the −10+14-mesh 3 and 4 amperes, in the −14+20 mesh 4 and 5 amperes, and in all the other sizes 5 amperes only. The following table gives the details of the various garnet products:—

Size	Current amperes	Lb.	Oz.	Grade
−8+10	3	13	0	Fair.
−8+10	3½	10	4	Poor.
−8+10	4	3	6	Very poor.
−10+14	3	4	8	Good.
−10+14	4	4	0	Fair.
−14+20	4	11	15	Good.
−14+20	5	0	15	Fair.
−20+26	5	8	15	Good.
−26+35	5	9	12	Good.
−35+42	5	3	15	Good.
−42+60	5	4	2	Fair.
−60	5	5	15	Fair.
Total	....	80	11	

The garnet is purple, which is not the colour of the garnet generally used for making garnet paper.

A recovery of 80 pounds 11 ounces of garnet concentrate was made from 236½ pounds of material, which is a recovery of 34·12 per cent. All the concentrates are not good, as the above table shows.

SUMMARY AND CONCLUSIONS

The $-8+10$ mesh did not produce any good garnet concentrate and the $-10+14$ mesh gave only a small amount. The sizes from 14 to 42 mesh yielded a fair proportion. This shows that the garnet rock should be crushed to 14 mesh.

The concentrates from the $-42+60$ and -60 mesh were only fair in grade and very fine. Possibly it would be as well to concentrate these sizes with the size immediately larger.

The sizing in the test work was very close. Good results might be secured by using fewer screens.

As the garnet rock is low in strongly magnetic material, a low-power magnetic machine would not be required.

Garnet concentrate made on a table always contains some gangue, therefore it should be cleaned with a high-power magnetic machine.

The method of concentration indicated by the test work is to crush to 14 mesh, size on 20, 28, and 35 mesh, table each size, dry the concentrate and run it through a high-power magnetic separator.

The recovery of good concentrate by the above method would be about 30 per cent.

However, before attempting to develop the deposit, it is imperative that the concentrate be tested to determine its suitability for making abrasive paper, and it should be ascertained, also, whether the garnet-paper makers will take garnet concentrates as fine as 14 mesh and of a purple colour. At present most makers purchase only material from 14 mesh to ½ inch and of a red colour.

Ore Dressing and Metallurgical Investigation No. 547

BARITE FROM LANGMUIR TOWNSHIP, TIMISKAMING DISTRICT, ONTARIO

Shipment. Two shipments of barite from the Premier Langmuir mine, 35 miles due south of Connaught, on the Night Hawk river, were submitted by A. D. F. McIntosh, President of the Canada Night Hawk Mines, Limited, 333 Lake Avenue, Rochester, N.Y. The first shipment called Lot No. 1, which was received June 6, 1932, consisted of four bags weighing 398 pounds gross. The second shipment called Lot No. 2, which was received August 16, 1932, consisted of five bags, gross weight 425 pounds.

Purpose of Experimental Tests. Tests were desired to discover if it would be possible to prepare a high-grade barite product from this material.

Characteristics of the Barite. The shipment consisted of white crystalline barite containing a little galena, calcite, and quartz.

Sampling and Analysis. After each lot was crushed a sample for analysis was cut out by a Jones riffle. These samples gave:—

—	Lot No. 1	Lot No. 2
	per cent	per cent
$BaSO_4$	86·76	84·72
$SrSO_4$	2·06	
$CaCO_3$	5·07	
$MgCO_3$	0·15	
SiO_2	4·99	
PbS	0·10	
$Fe_2O_3+Al_2O_3$	0·12	

Experimental Tests

Lot No. 1

The net weight of Lot No. 1 was 394 pounds. After picking out 3¼ pounds of specimens and 3 pounds 6 ounces of quartz and rock the balance was crushed twice in a small jaw crusher and once in a small set of rolls. A 1½-pound sample was then cut out. The sample ran 86·76 per cent $BaSO_4$ and 0·09 per cent Pb. The 386 pounds left after sampling was crushed to 10 mesh by means of the rolls and a hand screen, after which it weighed 381 pounds. The minus 10 mesh was separated into three sizes in a Richards launder classifier and the three sizes were tabled making in each size a barite product and a tailing. In tabling no line is present between barite and tailing except in the finest size where a faint line can sometimes be seen when looking at an angle. The method of cutting used was to take about a third for the tailing. The barite products were re-run to remove galena.

As screen tests on the sizes from the classifier showed the classifier did not do good work it was thought the table products would not be very good so they were all mixed and screened on 28 and 65 mesh by hand. The three sizes were tabled and the barite concentrates from the two finer sizes were re-tabled to remove galena. There was almost no galena in the coarsest size. The barite products from this work analyse as follows:—

Mesh	Per cent
−10+28	92·90 $BaSO_4$
−28+65	93·95 "
−65	97·23 "

The minus 65-mesh product is fairly good but the other two sizes are too low.

Lot No. 2

Tests Nos. 1 and 2

The net weight of Lot No. 2 was 420 pounds. After picking out 5 pounds 3 ounces of specimens the remainder was screened on a ½-inch screen as there was some dirt in the fine. This left 401 pounds 9 ounces, which was crushed to 10 mesh. The minus 10 mesh weighed 399 pounds 3 ounces. A sample, 1 pound 7 ounces, was cut out for analysis and gave 84·72 per cent $BaSO_4$. The remainder of the minus 10 mesh was cut into two parts by a Jones riffle. One part, 197 pounds, was screened on 26 and 60 mesh, the three sizes were tabled and the barite concentrates from the two finer sizes were re-run to remove galena. The other part of the minus 10 mesh was classified into three sizes, each of the sizes tabled, and the barite products re-tabled to remove galena. The concentrates from this work analysed as follows:—

Test No.	Size	Per cent
1	−10+26	89·80 $BaSO_4$
1	−26+60	94·94 "
1	−60	96·60 "
2	Coarse size	93·94 "
2	Medium size	90·08 "
2	Fine size	95·24 "

These tests show that screening before tabling gives better results than classifying.

Tests Nos. 3, 4, and 5

In Test No. 3 the products from Tests Nos. 1 and 2 were mixed, screened on 26 and 60 mesh and the three sizes tabled on a Wilfley table fitted with a side cutter that separated the concentrate from the tailing.

A lead product was cut out at the end of the table. About one-fourth of the width of material coming across the table was cut out for the tailing. Test No. 4 was made on the mixed products of Test No. 3 and was the same except that one-half was cut out for the tailing. Test No. 5, on the mixed products of Test No. 4, was the same as Tests Nos. 3 and 4 except that one-third was cut out for tailing. The barite concentrates from these tests were as follows:—

Test No.	Mesh	Per cent
3	−10+26	85·68 $BaSO_4$
3	−26+60	91·96 "
3	−60	95·74 "
4	−10+26	88·16 "
4	−26+60	94·34 "
4	−60	97·86 "
5	−10+26	88·00 "
5	−26+60	94·00 "
5	−60	96·52 "

These tests showed that high-grade barite products could not be obtained in the −10+26-mesh size, that products of about 94 per cent $BaSO_4$ could be secured from the −26+60-mesh size, and products of about 97 per cent $BaSO_4$ from the −60-mesh size.

Tests Nos. 6 and 7

For Test No. 6 the products of Test No. 5 were mixed, crushed to 20 mesh and screened on 42 mesh. Both sizes were tabled on a Wilfley table making a lead product, a barite concentrate, and a tailing. For the tailing about one-third of the width of the material going across the table was cut out. In Test No. 7 the products from Test No. 6 were mixed in the two sizes and tabled as in Test No. 6 except that one-fourth was cut out for the tailing. The barite concentrates from this work analysed as follows:

Test No.	Mesh	Per cent
6	−20+42	92·48 $BaSO_4$
6	−42	97·44 "
7	−20+42	90·56 "
7	−42	97·90 "

Tests Nos. 6 and 7 show that a product about 92 per cent $BaSO_4$ can be made in the −20+42-mesh size, and one of about 97·5 per cent in the −42-mesh size.

Tests Nos. 8, 9, and 10

These three tests were made to see what results could be obtained by closer sizing. For Test No. 8 the products of Test No. 7 were mixed, screened on 26-, 35-, 42-, and 60-mesh screens, and each size tabled making a lead product, a barite concentrate, and a tailing. No lead product was made in the −20+26 mesh. Tests Nos. 9 and 10 were similar to Test No. 8 and they were made on the mixed products of Tests Nos. 8 and 9 respectively.

In Test No. 8, one-fourth of the width of material coming across the table was cut out for tailing. In Test No. 9, one-third was cut out in all sizes except the −60 mesh from which one-fifth was cut. In Test No. 10, one-half was the cut in all sizes except the −60 mesh from which one-sixth was cut. The results from these tests were as follows:—

Size of concentrates	Concentrate Test No. 8, per cent $BaSO_4$	Concentrate Test No. 9, per cent $BaSO_4$	Concentrate Test No. 10, per cent $BaSO_4$
−20+26	90·06	95·40	94·30
−26+35	91·10	94·14	94·82
−35+42	93·62	96·54	96·34
−42+60	95·30	97·48	96·52
−60	97·14	98·74	96·58

The above table shows the results that can be obtained by close sizing. They are slightly better. It will be noticed that only the −60 mesh and −42+60 mesh give high-grade products.

Test No. 11

Test No. 11 was made to determine what kinds of products would result from still finer crushing. The products from Test No.10 were mixed, all crushed to pass 42 mesh and screened on 60 mesh. The two sizes were tabled making a lead product, a barite concentrate, and a tailing in each size. In the −42+60 mesh one-third of the width of material on the table was cut out for the tailing, and in the −60 mesh one-fourth. The results of this test were as follows:—

—	Mesh	Weight, per cent
Concentrate..	−42+60	96·88 $BaSO_4$
"	−60	97·48 "

The test showed that crushing to 42 mesh gave better results than crushing to 20 mesh. The concentrates produced are fairly high-grade. The 60-mesh concentrate is better than the −42+60-mesh concentrate.

FLOTATION

Four small flotation tests were made on the minus ½-inch material screened out of Lot No. 2. The best concentrate obtained was 93·32 per cent $BaSO_4$. The difficulty with flotation for the barite is that the calcite present floats and lowers the grade of the concentrate.

SUMMARY AND CONCLUSIONS

The tests conducted show that an improvement is secured in the barite table concentrate by finer crushing, closer sizing, and using screens instead of a classifier.

The finer the size tabled the better the grade of concentrate and the poorer the recovery. The following table gives the grade of concentrate and recovery that was obtained by treating the different sizes:—

Size	Barite, per cent	Recovery, per cent
−20+26	95	87
−26+35	95	80
−35+42	97	80
−42+60	97	80
−60	98	70

When some of the barite concentrates were ground fine, it was found that although they were white their colour was not so good as a sample of barite at present on the market.

Just what could be done with the barite commercially depends on what grade of barite could be sold. If a 97 per cent grade were high enough, the barite could be crushed to 35 mesh, screened on 42 and 60, and each of the sizes tabled separately. If a 98 per cent grade were required, it could be made by crushing to 60 mesh and tabling, but this would not be commercial as the capacity of a table on 60-mesh material is very small.

Ore Dressing and Metallurgical Investigation No. 548

DIATOMITE FROM BURNABY LAKE, NEW WESTMINSTER MINING DIVISION, B.C.

Shipment. A shipment of 37 bags of diatomite, weight 1,620 pounds, was received December 16, 1932, from Coast Quarries, Limited, Vancouver, B.C. The diatomite came from Burnaby Lake, New Westminster, 7 miles from Vancouver, and had been sent in at the request of V. L. Eardley-Wilmot of the Mineral Resources Division of the Mines Branch.

Purpose of Experimental Tests. Mr. Eardley-Wilmot wished tests made to determine if any products suitable for commercial uses could be prepared from the diatomite.

Arrangements for Experimental Tests. Mr. Eardley-Wilmot was present during some of the test work, and examined all products to determine their commercial significance.

Characteristics of the Diatomite. The crude diatomite is of a greyish earthy appearance, and contains some roots and fine sand; about 25 per cent moisture is present.

Sampling and Analysis. No general sample was taken of the whole shipment, but samples of the diatomite used in the different tests were run for moisture, and the dried material for loss on ignition. The results were:—

Sample	Moisture, per cent	Loss on ignition, per cent
Feed Test No. 2	29·78	30·00
" " 3	19·74	27·61
" " 4	26·89	27·03
" " 5	23·24	24·75
" " 8	8·91	36·88

Experimental Tests

The experimental work consisted of a series of tests, first with cyclones and second with centrifugal air separators. Some work was done using water classification, but it is doubtful if this method would have any commercial application.

CYCLONE TESTS

In Tests Nos. 1 to 4 inclusive the diatomite was calcined in a rotary electric furnace. The product was ground in a Raymond No. 0000 automatic pulverizer and the ground material treated in cyclones. Only the details of Test No. 4 will be given as the first three tests were of a preliminary nature.

Test No. 4

Fifty pounds of crude diatomite was put through a ½-inch screen by hand and a 6-ounce sample was cut out. Previous tests had shown that if the material were not put through the ½-inch screen the larger pieces were likely to contain uncalcined cores. The diatomite was calcined 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 is insulated. The furnace is equipped with an automatic electric heat control and with devices for slowly revolving the cylinder and for tilting the furnace.

The diatomite was charged into the furnace, which had been preheated to 500° F. The automatic control was set to maintain the temperature at 1,200° F. In 3½ hours the diatomite was completely calcined. When removed from the furnace and cooled it weighed 20 pounds 7 ounces.

The calcined diatomite was ground in the Raymond pulverizer and fed to a fan that exhausted through 2-foot, 3-foot, and 4-foot collectors in series. The discharge from the 4-foot collector went outside.

After this first run, the 2-foot and 3-foot collectors were cleaned out and the contents of the 2-foot collector again fed into the series of collectors. After this second run the same operation was repeated for the third run. The products from this test were:—

Product	Lb.	Oz.	Per cent −325 mesh
Run 1, 3-ft. collector	3	13	99·50
" 2, 3-ft. "	0	12	99·91
" 3, 3-ft. "	0	6	99·48
" 3, 4-ft. "	3	11	99·95
" 3, 2-ft. "	4	12	85·64
Total	13	6	

The loss in grinding and separating was 7 pounds 1 ounce, and took place mostly in the discharge from the 4-foot collector.

Samples of the products were sent to Mr. Eardley-Wilmot, who reported that although they were much like those obtained by use of a centrifugal air separator, they were not quite so good.

AIR SEPARATOR TESTS

Tests Nos. 5 and 6 were air separator tests. Only the details of Test No. 6 will be given.

Test No. 6

Fifty pounds of the crude diatomite was calcined in the electric furnace and gave 29 pounds of calcined product, which was ground in a Raymond pulverizer. The Raymond discharge went direct to a 30-inch Gayco centrifugal air separator. The oversize of the separator after each run was re-run through the separator until four runs in all had been made. The products from this work were:—

Product	Lb.	Oz.	Per cent −325 mesh
Run 1, Fine	15	14	99·80
" 2, Fine	6	15	99·82
" 3, Fine	2	7	98·83
" 4, Fine	0	14	98·72
" 4, Oversize	0	13	67·73
" 4, Clean-up	0	8	92·41
Total	27	7	

The loss in grinding and separating was 1 pound 9 ounces.

Samples of the different products were sent to Mr. Eardley-Wilmot for examination. He reported that the fine from Run No. 1 was the best product and, although it was not suitable for sugar filtration, it could be used in concrete, insulation products such as bricks, and as a general filler.

SUMMARY AND CONCLUSIONS

The air separator gives better results than the cyclones as the loss is smaller, the recovery is greater, and the fine product is better.

No high-grade diatomite products suitable for sugar filtration can be made from the material as shipped, but products for other uses such as concrete, insulating, and filling can be obtained.

The best method of treating the diatomite would be to break it up to about $\frac{1}{2}$-inch size, calcine, grind so as to free the diatoms and separate with cyclones or air separators.

Cyclones are cheaper than air separators, but the air separators give better results. If the exhaust air from the cyclones were put through a very large settling-chamber the loss of diatomite would be greatly reduced.

Ore Dressing and Metallurgical Investigation No. 549

THE PRODUCTION OF SPONGE IRON FROM MOOSE MOUNTAIN CONCENTRATE

Shipment. A shipment of 300 pounds of minus 300-mesh concentrate was received on August 5, 1933. This shipment was made from Sellwood, Ont., at the request of W. Rowland Cox, 120 Broadway, New York.

Object of Investigation. To produce sponge iron from high-grade, finely divided Moose Mountain concentrate, and to determine the suitability of the sponge iron so produced for direct conversion into wrought iron and steel.

Physical and Chemical Characteristics of Shipment. In the concentration of Moose Mountain ore by wet magnetic separation, the crude ore is finely ground so that 90 per cent passes through a 200-mesh screen. This minus 200-mesh material averages about 69 per cent iron, 0·01 per cent phosphorus, and 4·5 per cent silica. Approximately 20 per cent of this average concentrate will pass through a 300-mesh screen and it is this minus 300-mesh material that was submitted for our tests.

A certificate of analysis by John H. Banks of the John H. Banks Analytical and Research Laboratories, New York City, accompanied the shipment. This analysis and one made in our own laboratories from a sample cut from this 300-pound shipment are tabulated below:—

John H. Banks Laboratories:

Dried at 212°F.

	Per cent
Iron	71·52
Silica	1·09
Phosphorus	0·002
Sulphur	....

Mines Branch Laboratories:

Dried at 212°F.

	Per cent
Iron	71·63
Silica	1·12
Phosphorus	0·008
Sulphur	0·005

The results of the two laboratories agree satisfactorily and show the material to be an exceptionally high-grade product.

SINTERING

Agglomeration of this finely divided material by sintering was deemed advisable for two reasons:—

(1) In our rotary retort in which our low-temperature reductions are carried out, good reductions are difficult to obtain with very fine material, especially when a gaseous reducing agent is used.

(2) The briquetting of finely divided sponge iron is difficult, if not impossible.

In none of the gas reduction processes that have come to our attention is finely divided material used. In the Norsk-Staal process, as used in Bochum, Germany, the finely divided concentrate is briquetted before being charged into the metallizing furnace, and in the Wiberg process either lump ore or sintered concentrate is charged into the stack type furnace.

The sintering of finely divided material is rather difficult and calls for special consideration, because, ordinarily, the tendency of such material to form an impervious bed and to be drawn through the grates militates against the obtaining of a good sinter. It is the opinion of some that ordinary hard sinter is difficult to reduce and that a "soft" sinter is much more easily reduced and is, therefore, technically and economically more desirable. This is quite plausible and it may well be that for a given sponge iron process there is much to be gained by modifying the sintering practice to produce a sinter no harder than the process demands. As some processes, such as the Wiberg, apparently demand a sinter strong enough to withstand the weight and movement of the column of material in a stack, it seems clear that a well sintered material is sometimes not desirable. In this work no attempt was made to produce anything but a definitely well sintered material, especially in view of the fact that it is our experience that there is no difficulty in obtaining a satisfactory reduction from such material under the conditions obtaining in these laboratories.

On account of the small amount of material available and the consequent necessity of minimizing losses, the concentrate was sintered in batches of 50 pounds. The first batch was made up of 50 pounds of concentrate, 2·5 pounds coke (−20 mesh) and $6\frac{3}{4}$ pounds of water. This was mixed in a tumbling-barrel and an attempt was made to "pelletize" the mixture without much success. This batch sintered poorly and there was a large loss (16 pounds) through the grates.

In each of the subsequent batches a layer of coarse sinter was placed on the grate to act as a bed for the charge and a quantity of "returns," consisting of fine sinter and semi-sintered material from the preceding sinter batch, was included in the mix which was made up as follows:—

—	Weight, pounds	Weight, per cent
Concentrate	35·0	61·2
"Returns"	14·5	25·3
Coke	2·0	3·7
Water	5·70	9·8
		100·0

The batches made from the above mix "pelletized" in the mixer quite satisfactorily and yielded a well sintered product and a high recovery. A total of 295 pounds of concentrate was sintered and a total of 276 pounds of sinter recovered. Of the total loss of 19 pounds, 16 was lost in the first experiment so that only 3 pounds was lost through the grates, once the use of "returns" in the mix and of coarse sinter on the grates was begun.

It is not suggested that as good or better results cannot be obtained from a mix using less "returns" or that the type of sinter obtained is best suited for our purpose. The sinter obtained was well fused, with large pores and was quite friable. The analysis of a sample believed to be representative is as follows:—

	Per cent
Iron	71·05
Insoluble	1·47
Phosphorus	0·015
Sulphur	0·010

METALLIZING

Reduction by Means of Charcoal. The most convenient apparatus available in these laboratories for the metallization of iron ores and concentrates on a moderate scale is an electrically heated rotary retort. This heat-resisting alloy retort, which has a diameter of 14¾ inches and effective loading length of 48 inches, forms the heating chamber of a 60 k.w. furnace.

In this work, a batch consisting of 60 pounds of sinter, crushed to ¼ inch, and 20 pounds of 3-mesh charcoal was charged into the retort. The retort and its contents were then brought up to a temperature of 1,800° F. and held at that temperature for about 7 hours, or until the flame produced at the vent pipe of the retort by the burning of the gas resulting from the reaction between the carbon of the charcoal and the oxygen of the sinter became very low. The reduced material was then discharged into a special container provided with a tightly fitting cover. When cool, the retort product was given two passes over a drum-type, dry magnetic separator in order to separate the charcoal ash and excess charcoal from the sponge iron.

Three batches, or a total of 180 pounds of sinter, were metallized in this manner and a total of 126 pounds of sponge iron recovered therefrom. The chemical analysis of this charcoal-reduced sponge iron is as follows:—

	Per cent
Total iron	95·20
Metallic iron	94·50
Insoluble	2·89
Phosphorus	0·020
Sulphur	0·030
Carbon	0·24

Reduction by Means of City Gas. As the sponsor of these tests was interested in gas reduction, it was decided to reduce a charge of sinter by means of city gas. City gas is not nearly so satisfactory a reducing gas as is the carbon monoxide-hydrogen mixture used in the gas reduction processes under commercial development. However, it has proved satisfactory and convenient in experimental work and has been successfully used for reduction work in these laboratories for some time.

This gas reduction was carried out in the same rotary retort that was used for the charcoal reductions above described, the retort being so constructed that the city gas can be introduced continuously through a pipe located axially in the back of the retort, and turned at the vent pipe in the plug or door. In carrying out this reduction the remainder of the sinter, amounting to 88 pounds, was crushed to ¼ inch and charged into the retort, which was then brought up to a temperature of 1,800° F., held at this temperature for 11 hours, and the content then discharged into the special container. The chemical analysis of this gas-reduced sponge iron is given below:—

	Per cent
Total iron	95·20
Metallic iron	94·60
Insoluble	2·31
Phosphorus	0·015
Sulphur	0·02
Carbon	1·99

The carbon content of this gas-reduced material is very high, being much higher than that of the charcoal-reduced material. This high carbon content is by no means a necessary characteristic of gas-reduced material, for the present investigators have often produced well reduced sponge iron containing only a few points of carbon by the use of city gas, but it must be attributed to carburization produced by the prolonged exposure of the reduced iron to the action of carburizing gas. In other words, the time the material was held at temperature was longer than necessary. In a continuous process operating commercially the production of a sponge iron low in carbon would be ensured by means of rate of charging and discharging. However, a high carbon content is probably desirable in sponge iron destined for conversion into steel in open-hearth furnaces.

BRIQUETTING

Sponge iron as discharged from the metallizing furnace is in the loose and powdery form, and on account of its low density and susceptibility to excessive oxidation during melting, is physically unsuitable for melting in steel-making furnaces. Briquetting is, therefore, desirable, if not actually necessary.

In these experiments approximately 125 pounds of charcoal-reduced material and 60 pounds of gas-reduced sponge were briquetted in a 70-ton Southwark briquetting machine. Both lots briquetted very satisfactorily. The briquettes were 2 inches in diameter, averaged about 1¼ inches in length, and were about two-thirds as dense as solid steel.

STEEL- AND IRON-MAKING TESTS

Conversion into Steel. No melting tests were carried out on these high-grade briquettes as it was felt that the work on the lower grade material produced from screenings from briquetted concentrate, already reported, demonstrated clearly that such material is capable of being converted into quality steel by simple melting and recarburizing.

Conversion into "Wrought Iron." The fact that sponge iron briquettes are composed of substantially carbonless iron plus a certain amount of non-metallic (gangue) material that is probably capable of being converted into slag by heating the briquette to a white heat, has led to the suggestion that the equivalent of commercial wrought iron could be produced from these high-grade briquettes by heating them to a welding temperature, and giving the highly heated agglomerated mass the series of treatments ordinarily given the product of the puddling furnace in the manufacture of commercial wrought iron.

These high-grade sponge iron briquettes contain about 5 per cent of non-metallic material that conceivably may be converted into slag at a white heat. That this amount of slag is probably not in excess of that contained in the puddle ball as it is removed from the puddling furnace may be inferred from a statement by Bradley-Stoughton in his book "The Metallurgy of Iron and Steel," to the effect that even after a large amount of slag has been squeezed out in the squeezing and rolling operations, the muck bar usually still contains one to two per cent of slag.

It seems probable that in order to get an adequate expulsion of slag from the iron in the squeezing and rolling operations, the contained slag must be quite fluid at the operating temperature and that the latter must be very high. The fluidity of a slag at a given temperature is a function of its composition and it is a question if the composition of the residual gangue in these sponge iron briquettes is similar to that of the slag contained in wrought iron. It is, therefore, not certain that the fluidity of the slag formed by the non-metallic matter in these briquettes at the working temperature of, say, 2,400° F., will permit its expulsion to the required amount during mechanical hot working.

It is hardly possible in these laboratories to duplicate or even approximate to the squeezing and rolling operations by means of which this slag elimination is effected. However, in order to get some idea of the possibility of converting this high-grade Moose Mountain sponge iron directly into wrought iron, a few small-scale experiments were carried out.

In these experiments two charcoal-reduced briquettes were placed in a "globar" furnace, one on top of the other so as to form a cylinder 2 inches in diameter and approximately $2\frac{1}{2}$ inches high. They were then heated to about 2,400° F. and hammered by hand to a section approximately $\frac{1}{2}$ inch by $1\frac{1}{2}$ inches. Four of such sections were then placed together to form a pile about 2 inches by $1\frac{1}{2}$ inches in section, and, after repeated heatings, gradually reduced to a section of about 1 inch square. No difficulty was experienced in welding the different pieces together, but, as the following data will demonstrate, there was but little expulsion of slag. In addition to this bar, which was made from eight briquettes, a similar but shorter one was made from four briquettes.

CHEMICAL AND PHYSICAL ANALYSIS

Chemical Analysis

Drillings taken from the end of the longer of these two bars gave the following results:—

	Per cent		Per cent
Carbon	0·02	Total iron	96·08
Manganese	0·01	Silica	1·58
Phosphorus	0·019		
Sulphur	0·009		

The carbon, manganese, phosphorus, and sulphur contents of this material are in line with what is characteristic of good wrought iron, but the comparatively low iron content and the high silica both point to a high slag content. Since the iron content of the wrought material is higher than that of the briquettes from which it was forged, some elimination of slag is indicated but it still contains 2 to 3 per cent too much slag.

Microstructure

A photomicrograph of a longitudinal section taken from the end of the longer bar, magnification 100 diameters, confirms the results of the chemical analysis in that it shows an excessive amount of slag in a ferritic matrix. The microstructure of the slag is, in general, similar to that of the slag contained in commercial wrought iron, but there is too much of it.

Tensile Properties

The shorter of the two forged bars was machined into a standard A.S.T.M. 0·505 by 2·0 inches tensile test bar and tested on an Amsler tensile and compression machine. The results of this test are as follows:—

Tensile strength	39,000 lb./sq. in.
Yield point	34,000 " (?)
Elongation (in 2 inches)	8·5 per cent
Reduction in area	8·9 "

The value obtained for the yield point seems very high, but, as it is the stress at which a pronounced recession of the indicating pointer took place during loading, it is recorded. In hydraulic machines of the Amsler type this recession of the pointer corresponds to the drop of the beam in the lever type of machine. It will also be noted that although the strength of the iron is fair, the ductility and toughness, as measured by elongation and reduction in area, are very low. This is attributed to the excessive slag content of the iron.

That additional hot work, with the (probable) accompaniment of further expulsion of slag, will result in a decided improvement in mechanical properties was demonstrated by the following experiment. The two fractured pieces resulting from the above-mentioned tensile test, together with the two fractured pieces resulting from the cold bend test referred to later, were hammer-welded together into a small bar under conditions calculated to expel some of this excess slag. From this bar another standard A.S.T.M. tensile test bar was machined and tested as before. The results of this test are:—

Tensile strength	43,500 lb./sq. in.
Yield point	37,500 "
Elongation (in 2 inches)	20·0 per cent
Reduction in area	18·8 "

From these figures it is clear that the re-working of the iron has resulted in a decided improvement in its properties, and the microstructure, together with the fact that the iron content of the re-worked bar is 97·90 per cent as compared with 96·08 per cent before, indicates that this improvement is due in part at least to the further elimination of the excess slag.

Cold Bend Test. The longer of the two original forged bars was machined into a round-cornered flat, $\frac{3}{4}$ inch by $\frac{3}{8}$ inch by 7 inches long, and an attempt was made to bend it over a 1-inch round bar. The test specimen broke after bending about 30 degrees. This lack of ductility confirms the result of the first tensile test and is, no doubt, due to the excessive slag content of the iron.

SUMMARY AND CONCLUSIONS

This investigation has demonstrated that the production of an exceptionally high-grade sponge iron from minus 300-mesh Moose Mountain concentrate is technically simple and has indicated the general process by which this may be accomplished.

That this high-grade sponge iron is a very satisfactory base for the manufacture of quality steel, is clearly indicated from the results of the work and those of a previously reported investigation on the production of sponge iron from Moose Mountain concentrate.

Data have also been presented that support the belief that a product similar in microstructure and properties to commercial wrought iron may be produced directly from this high-grade sponge iron by mechanical work at high temperatures. The attainment of this objective appears to depend largely upon the ability of the squeezers and other mechanical devices used in the industry to squeeze out to the necessary extent the fused gangue or slag contained in the white hot sponge iron. Should larger scale, practical tests indicate that this contained slag does not possess the degree of fluidity necessary to make its expulsion commercially feasible, it seems probable that this could be corrected by the addition of small amounts of a suitable flux, such as manganese oxide or soda ash, to the sinter mix before sintering.

Ore Dressing and Metallurgical Investigation No. 550

A. HEALTH HAZARDS IN THE PRODUCTION AND HANDLING OF RADIUM

B. PRECAUTIONS TAKEN DURING A LABORATORY INVESTIGATION OF THE TREATMENT OF RADIUM-BEARING ORES

INTRODUCTION

The dangers incident to handling or working with high-grade radium compounds, such as radium salts and needles, are well recognized, and precautions of a very definite nature are prescribed to protect those who work with such materials.[1] In the case, however, of material having lower radioactivity, such as ores and concentrates, the dangers are less appreciated and consequently the methods of protection to be adopted are not so clearly defined.

The ensuing report is divided into two main sections: the first (Part A) dealing with a brief discussion on the subject of hazards, and the second (Part B) constituting a report of the precautionary measures taken during an investigation of the treatment of radium-bearing ores.

The preparation of the first section necessitated a careful study of the literature dealing with radium poisoning and the effects of radioactive compounds upon the human system.

PART A

THE BIOLOGICAL EFFECT OF RADIATIONS

The dangers of exposure to radium and other radioactive material consist essentially of:—

(a) Injuries to the superficial tissues.

(b) Derangement of the internal organs and changes in the blood.

Radioactivity is that property possessed by certain elements in which spontaneous atomic disintegration occurs accompanied by the emission of electrically charged particles or rays. These radiations given off by radioactive bodies have peculiar physical properties, two of which, penetration of solid matter and ionizing power, apply to the subject under discussion. The radiations consist of three types, known as alpha and beta particles and gamma rays. The first is weakly penetrative, while the last two will penetrate considerable thicknesses of solid material.

The disintegration product of radium is a gas, radium emanation more usually referred to as radon, which in turn breaks down into a series of radioactive elements.[2]

The action of radium on the body tissue is subject to a definite biological law known as the Law of Arndt-Schultz.[3]

"The action of X-rays and radium on the living cell is excitation with a feeble dose, inhibitance with a moderate dose, and destruction with a strong dose."

The application of this law in the effect of radium upon the blood-producing centres shows that "small doses increase temporarily red and white cell production. After a shorter or longer time, small or larger doses cause partial or almost total destruction of leucocytes and a diminution in the erythrocytes . . . . "[4]

The external action of the radiations on the body tissue is due to beta particles which penetrate the skin and result in severe burns which are not only difficult to heal but in extreme cases often produce a cancerous condition. Gamma rays, the most penetrative of the radiations, affect the internal organs, blood-producing and reproductive systems. The penetrative power of alpha particles is very weak despite the fact that the alpha particle is one of the most destructive agents known to science. Their action on superficial tissue is negligible, but it is possible that their action upon the delicate tissue of the lungs might eventually lead to a weakening of the lungs or the production of some unknown disorders.[5] In the case of radioactive deposits in the bone structure, this aspect of alpha ray action is further apparent when we consider that 92 per cent of the radiation is from alpha particles. This atomic bombardment will thus take place in the bone structure or living tissue with possible disruptive results.

Radioactive compounds may be introduced into the system in several ways. They may be swallowed or inhaled in the form of dust, or inhaled as the gas radium emanation (radon). To prevent the entrance of active material into the body is imperative. A very small amount entering the body over a period of years may prove as dangerous as a larger amount in a shorter time. Radium is not eliminated from the system to the same extent as is the case with most other poisons, but a certain proportion will displace the calcium in the bone structure thus becoming a fixed source of radioactivity in the body itself. The results are not immediately apparent. Four or five years may elapse before the effects of this deposited radium will result in necrosis of the bone, anaemia, or development of cancer.

The quantity of radium ingested and fixed in the human system necessary to produce fatal results is estimated at 10 micrograms (0·010 milligram)[6]. Consideration of the very minute quantity which is dangerous must be realized in dealing with the problem of precautions.

As radium is the disintegration product of a series of elements originating in the element uranium, it follows that any mineral containing uranium will also contain radium in a definite ratio of about 1 in 3,000,000.[7]

In an article by Martland[8] the Schneeburg and Joachimsthal lung cancers are discussed at length. For several centuries the undergound workers in the cobalt mines of the Schneeburg district, Saxony, have been subject to a pulmonary disease resulting in death in middle life. An official investigation[9] extending over a period of three and a quarter years resulted in establishing a diagnosis of carcinoma of the lungs in 62 per cent of the deaths among miners. During the same period 362 persons from the same district, but not working in the mines, were subjected to examination but showed no indication of malignancy in the lung.

Numerous theories have been advanced as to the cause of these carcinomata. Apart from rock dust, dampness, and volatile arsenical compounds, the mine air was distinctly radioactive, having a radon content as high as 50 Maché units, and the ore also contained uranium.

Martland supports the theory that the effect may be entirely due to the inhalation of emanation.

Similar conditions among the workers in the pitchblende mines of Joachimsthal are making their appearance.

So high has the incidence of malignancy of the lungs among these miners become that the Medical Section of the League of Nations appointed a sub-commission to study cancer of the lungs among the miners of Jachymov (Joachimsthal), Czechoslovakia.[10]

Apparently opinions differ as to the relative dangers which are incurred from a continued exposure to an atmosphere containing radioactive substances either as radium emanation (radon) or as dust; nevertheless, sufficient evidence has been cited to warrant the taking of every possible precaution.

PRECAUTIONS AND PROTECTION

Having established the conditions under which the body is subject to danger, it is possible to study the necessary precautions for protection. Briefly, protective measures must be adopted to deal with:—

(1) External effects of beta rays.

(2) Internal action of gamma rays.

(3) Effects of radon on the lungs and blood-producing centres.

(4) Dangers from inhalation of radioactive dust.

Regulations governing the protective measures for radium workers in hospitals are found in "International Recommendations for X-Ray and Radium Protection".[11] In general these recommendations apply to all those who work with radium or radioactive material. Protection of the hands from beta rays is secured by distance. Long-handled forceps or boxes should be used in carrying radium or its concentrates from place to place. Speed in handling is also essential. For gamma ray protection dependence is placed on lead screens and all operations, such as filling containers and needles, should be done behind lead of at least one inch in thickness.

In pitchblende treatment and all operations prior to the refining of radium, the rules for protection are not so well defined as those laid down for handling the refined salt.

At this early stage of operation, the dangers are from emanation and dust. Ventilation is a primary consideration, and forced draught fans in all rooms are necessary. Good natural ventilation should not be overlooked. Personal cleanliness among workers should be emphasized. Hands should be washed frequently, taking particular care to brush out under the finger nails. Overalls worn in the laboratory should be left there after work and frequently washed. Any clothing subject to become coated with ore dust or radioactive material should never be worn away from the laboratory.

Rooms containing pitchblende ore should be equipped with ventilating fans to draw off the active gases in the atmosphere.

Grinding operations should preferably be carried out wet in order to reduce to a minimum the tendency for dust dissemination.

As a check on the efficiency of the ventilation system, tests on the radioactivity of the room air should be carried out at frequent periods.

In the case of workers, electroscopic tests on their expired air[12] should be made at least once a week and a blood test carried out monthly.

By adopting the above measures it is possible to discover any weakness in the protective scheme and make the necessary adjustments. The worker showing high radioactivity may be removed temporarily from his work before any serious effects are apparent in his general health. In respect to this, Dr. S. C. Lind states that there appear to be great individual differences in susceptibility to radiation. He considers it would be good practice to change crystallizers and others having intimate contact with radium several times a year by means of rotation. Three months of continuous crystallizing ought to be a maximum, one or two would be better. The same applies to one doing gamma-ray measurements.[13]

It might appear that the taking of such elaborate precautions is overemphasized, but, when the insidious and deadly nature of radium is considered, too much care can not be taken. Only by observing such regulations can the hazards be reduced to a safe minimum.

PART B

In 1931, an investigation on the recovery of radium from the pitchblende ores of the Great Bear Lake district of Canada was undertaken in the laboratories of the Mines Branch at Ottawa.[14] The hazards involved in working with radioactive materials were duly considered, and an investigation of these hazards and the precautions necessary to meet them was made during the period in which the investigation on the radium ore was being carried out. It was assumed at the beginning that working with the Great Bear Lake pitchblende might entail hazards that were not present in the case of low-grade radium ores which had been treated previously in other parts of the world, and subsequent results have proved this assumption to be well founded. During the writer's visit to the radium refining laboratory of the University of Missouri, the methods employed there for examining the workers and the general precautionary measures in force were carefully studied. These methods of examining workers were adopted as part of the protective scheme in the Mines Branch Laboratories.

The Department of Pensions and National Health extended the fullest possible co-operation to the Department of Mines in the examination of the radium workers.

EXPIRED AIR TEST

This test offers the most satisfactory method for checking the radioactive condition due to the inhalation of radon or ingestion of radium. In the Mines Branch Laboratories tests were made at least once a week. By this test it was possible to detect the presence of radon in the lungs, and if present the worker was removed from the laboratory for a day or two to free the lungs of radioactivity.

The equipment used consisted of an electroscope of the Lind type with several ionization chambers of 2·7 litres' capacity. The subject exhales air regularly, with deep breaths, through drying bottles containing calcium chloride and soda lime, into the chamber. Exhalation is continued for three minutes, after which the taps on the chamber are closed and it is set aside for three hours to allow any radon present to come into equilibrium with its decomposition products, after which a measurement is made and the radon calculated in curies per litre.

Certain conditions must be observed in carrying out these tests.

As the air entering the chamber should be dry and free from carbon dioxide, suitable drying agents and soda lime or other absorbent of carbon dioxide are used. A low natural leak on the electroscope is desirable. The method of blowing air through the chamber is important. The exhaled air should be drawn from the base of the lungs by a normal respiratory action, as radon, one of the heaviest known gases, will tend to accumulate in the base of the lungs.

If the radon is due to the inhalation of gas, its elimination is accelerated by exercise in the open air. If it is caused by radium ingested in the system, it will be impossible to remove it. The test, therefore, offers an excellent method of determining whether a worker has taken radium element into his system or only taken the gas into his lungs.

The disintegration products from radon will form an active deposit on the inside tissue of the body (lungs and digestive system). Several of these products have very high alpha ray activity and an appreciable span of life. They will be eliminated by normal processes, but during their sojourn in the body they constitute a source of radioactivity.

Table I shows the record of tests made on the workers in the Mines Branch Laboratories during the period of large-scale work on the Great Bear Lake pitchblende.

TABLE I

Record of Expired Air Tests on Radium Workers

—	10^{-11} curies of radon per litre		
	Subject R.J.T.	Subject W.R.McC.	Subject M.R.K.
27/ 2/32	0·667	0	....
7/ 3/32	0	....	2·74
11/ 3/32	0·164	0	0
18/ 3/32	0	....	0·07
8/ 4/32	....	....	0
22/ 4/32	216·24	....	....
25/ 4/32	59·4	14·7	2·94
27/ 4/32	....	12·59	....
29/ 4/32	0·026	0·93	0·76
2/ 5/32	....	0	....
6/ 5/32	0	0	0·54
13/ 5/32	0·047	0	....
20/ 5/32	1·87	36·68	1·34
26/ 5/32	....	2·32	0
27/ 5/32	0	0	0
2/ 6/32	0·187	0	0
10/ 6/32	0·98	0	0
17/ 6/32	0	0	0
24/ 6/32	0	0	0
15/ 7/32	0	0	2·26
22/ 7/32	1·14	0	1·32
29/ 7/32	7·48	4·49	0
2/ 8/32	46·8	0	....
3/ 8/32	15·26	0	....
4/ 8/32	12·22	....	....
5/ 8/32	7·26	0	0
8/ 8/32	0	....	....
12/ 8/32	5·22	10·65	0
15/ 8/32	....	9·33	....
16/ 8/32	....	4·87	....
17/ 8/32	....	2·26	....
19/ 8/32	....	1·42	5·37
2/ 9/32	5·57	0	0
9/ 9/32	0	1·94	0
16/ 9/32	1·45	0	0
19/ 9/32	0·36	....	....
23/ 9/32	....	2·29	0
30/ 9/32	....	0·46	0
21/10/32	0	....	0
28/10/32	0	0	0
4/11/32	16·5	....	0
7/11/32	0	....	....
25/11/32	0	0	0
2/12/32	0·33	0	0
9/12/32	0	....	0·14
16/12/32	0·05	0	0

The practice was to make the tests on Fridays and if the subject showed radioactivity he remained away from the laboratory until Monday. Usually this was sufficient time to free the lungs of radon. A study of the table shows some interesting facts relative to the elimination of radon. Subject R.J.T. showed $46 \cdot 8 \times 10^{-11}$ curies per litre on August 2nd. Tests were made daily for the three days following and then on the 8th when the results were zero. This shows the time factor that is involved in freeing the lungs from emanation. On August 12th subject W.R.McC. had a reading of $10 \cdot 65 \times 10^{-11}$ curies per litre. After being away from the laboratory during Saturday and Sunday a reading on Monday showed $9 \cdot 33 \times 10^{-11}$ curies per litre. On Thursday the radon had fallen to $1 \cdot 42 \times 10^{-11}$ curies per litre. No definite rate of elimination was obtained from the tests as there are many factors which enter into this question. Personal characteristics of the subject would appear to be a factor. The way a person breathes and his activity in the fresh air would determine very largely the rate at which he rids his lungs of radon. Sufficient data were obtained, however, to prove that a time factor is involved. The gradual decomposition of the element itself will ultimately take place. But it is desirable to expel the radon as quickly as possible so that a minimum of the active decay products is deposited on the lung tissue.

One test was carried out on a worker who had been exposed to dust from grinding pitchblende ore. A dust mask was worn during the work. A reading of $2 \cdot 84 \times 10^{-11}$ curies per litre was recorded. No further tests were made, as grinding operations were discontinued. It is not possible to state whether radon shown by the test was due to inhaled gas or to dust particles. The former was probably the case as the atmosphere around a quantity of pitchblende would have a very definite radon content.

During the first seven months the tests were carried out in the Radium Measuring Laboratory of the Mines Branch by the writer. The later tests were carried out under the same conditions in the Food and Drugs Laboratory of the Department of Pensions and National Health by R. D. Whitmore, under the direction of H. M. Lancaster, Chief Dominion Analyst.

BLOOD EXAMINATION

One of the first effects to be looked for in a person exposed to radium is a disturbance in hematogenesis. Soon after work had been commenced on a large scale, discussions took place with officers of the Dominion Department of Pensions and National Health and arrangements were made for a monthly blood count on the radium workers. The physiological tests were made by B. W. Culyer under the direction of Dr. N. MacL. Harris, Chief of the Laboratory of Hygiene.

Tests were started in April, 1932, and continued for fourteen months. The differential count was carried out by the regulation method, blood being taken from the lobe of the ear; the normal limits of Emerson were regarded as standard. Tests were continued for four months after the workers had ceased working with radium-bearing material.

The normal value for hemoglobin is 90 to 100 per cent; for erythrocytes (red blood cells), $4 \cdot 5$ to $5 \cdot 5$ millions per cubic millimetre; for leucocytes (white blood cells), 6,000 to 8,000; for polymorphonuclear leucocytes, 63 to 70 per cent; eosinophils, 2 per cent; basophils, $0 \cdot 5$ per cent; lymphocytes, 20 to 30 per cent; large mononuclear leucocytes, 10 per cent.

The results of blood counts on three workers are given in Table II.

TABLE II

Date	Hemoglobin, per cent	Erythrocytes, millions per cu. mm.	Leucocytes, per cu. mm.	Differential Leucocyte Count: Polymorphonuclears, per cent	Eosinophils, per cent	Basophils, per cent	Lymphocytes, per cent	Large mononuclears, per cent
Subject—R.J.T.								
27/ 4/32	90+	4,520	7,200	62·5	2·5	1·0	28·0	5·5
27/ 5/32	95	4,170	8,600	66·66	2·33	0·33	23·33	7·33
27/ 6/32	95	4,600	7,400	42·0	0·66	1·0	44·33	12·0
4/ 8/32	90+	4,500	8,000	63·0	0·66	0·66	22·66	13·0
1/ 9/32	90	4,750	8,700	54·75	0·75	0·75	38·0	5·75
27/10/32	90	4,600	9,000	54·0	1·25	0·5	35·75	8·5
1/12/32	90	4,500	7,900	44·0	0·0	0·33	51·66	4·0
23/12/32	90	4,600	8,000	46·5	0·5	0·5	47·5	5·0
4/ 1/33	90	4,600	8,500	52·5	2·0	1·0	40·0	4·5
1/ 2/33	90	4,410	8,400	40·0	3·0	0·0	50·0	6·5
1/ 3/33	90	4,500	8,400	59·5	2·5	0·0	32·5	5·5
3/ 4/33	90	4,600	8,000	63·0	1·0	0·5	31·5	4·0
2/ 5/33	90	4,500	8,200	57·0	1·0	0·5	34·5	7·0
6/ 6/33	90	4,600	8,000	66·5	3·5	0·5	16·5	13·0
Subject—W.R.McC.								
27/ 4/32	95	4,800	7,600	63·33	3·66	0·33	21·33	11·33
27/ 5/32	95	4,490	7,800	62·00	2·0	0·0	26·0	10·0
27/ 6/32	90+	4,800	8,000	73·33	0·33	1·33	21·0	4·0
4/ 8/32	95+	5,000	7,800	69·5	0·5	....	25·0	5·0
1/ 9/32	90+	4,700	8,950	58·33	1·66	0·33	32·33	7·33
27/ 9/32	90	4,500	8,750	49·0	1·5	1·5	29·5	18·5
27/10/32	90	4,650	8,000	51·66	0·66	1·0	37·33	9·33
1/12/32	90	4,700	7,200	54·33	1·66	1·0	33·0	10·0
4/ 1/33	90	4,600	7,150	58·0	2·0	1·0	33·5	5·5
1/ 2/33	90	4,780	7,200	60·0	1·0	0·5	31·0	7·5
1/ 3/33	90	4,740	7,350	50·0	2·5	1·5	38·5	7·5
3/ 4/33	90	4,600	7,200	55·0	2·5	1·5	33·5	7·5
2/ 5/33	85+	4,200	7,200	55·0	0·0	1·5	37·5	5·5
6/ 6/33	90	4,350	7,400	70·0	0·0	0·33	18·33	11·33
Subject—M.R.K.								
27/ 4/32	98	4,200	8,500	65·0	1·5	0·5	25·0	8·0
27/ 5/32	95+	3,670	9,950	64·33	1·66	1·33	23·0	9·66
27/ 6/32	90+	3,950	8,350	61·0	0·66	2·0	21·0	15·33
4/ 8/32	90	4,100	8,750	62·0	1·0	1·0	28·0	8·0
1/ 9/32	90	4,200	8,200	60·5	1·0	1·5	27·5	9·5
27/ 9/32	90	4,400	8,400	62·0	1·5	0·0	23·0	13·5
27/10/32	90	4,400	9,000	70·33	0·0	1·0	23·33	5·33
1/12/32	90	4,500	8,650	59·0	1·0	1·0	29·0	10·0
4/ 1/33	90	4,600	8,500	65·0	1·0	1·0	26·5	6·5
1/ 2/33	90	4,400	8,200	58·5	0·5	1·5	33·0	6·5
1/ 3/33	85	4,500	7,800	60·5	1·5	1·5	29·5	7·0
3/ 4/33	90	4,400	7,850	63·5	0·5	0·5	32·5	3·0
2/ 5/33	90	4,200	7,400	64·0	0·0	0·5	26·0	9·5
6/ 6/33	90	4,400	7,500	60·0	1·0	2·0	29·5	7·5

The fifth blood test on subject R.J.T. showed an increase in lymphocytes, a decrease in polymorphonuclears, and a slight increase of white cells. This abnormal condition continued for a period of six months. Work on the pitchblende was suspended in February and subsequent blood tests on subject R.J.T. showed a gradual return to the normal limits.

Similar reactions were noted in the blood count on subject W.R.McC., although the variation from the normal limit was not so marked as in the case of R.J.T.

It will be seen by examining Table I on expired air tests that both subjects showed the presence of radioactivity almost continually during the period from February to December. It was not, however, until September that an irregularity in the blood count was observed. The blood count of M.R.K. remained practically normal throughout.

The results of these tests conform to what one might expect from the biological laws governing the action of radioactive substances on the human system. The continued action of the small doses of radon eventually affected the blood-producing system, as evidenced by the blood count.

The abnormal blood condition was not in these cases a serious one, and absence from contact with radioactive materials enabled the blood-producing system to assume a normal condition inside of a couple of months.

But the results of these tests are extremely interesting, however, as they show that a hazard may exist in the breathing of air containing even small amounts of radon. The precautions taken made it possible to detect the first symptoms and steps were taken to check any further effects on the subject by removing him from all contact with radioactive material until such time as his blood count was again normal.

It is essential that all workers should be subjected to a medical and blood examination prior to starting work with radium. By this means any inherent irregularity or weakness may be noted and subsequent changes in the worker's health can be determined more accurately.

TESTING OF ROOM AIR FOR RADON

Testing of the atmosphere[15] in the laboratories constitutes another precautionary measure, and enables a check to be maintained on the efficiency of the ventilating system. The method of testing is easily carried out and consists of drawing dust-free air slowly through a calibrated ionization chamber, the air being dried by passing through a jar containing a suitable drying reagent. A Lind type of electroscope was used and the results were recorded in curies per cubic metre.

The average amount of radioactive products present in the atmosphere is about $8 \cdot 3 \times 10^{-11}$ curies per cubic metre.[16] This very small amount is not detected by the usual methods employed for testing room air, a more sensitive apparatus being necessary.

Pitchblende ore has the property of giving off radon.[17] This characteristic is known as the emanating power and varies with different types of ore and the size of the ore particles.

In several offices and rooms of the Department of Mines varying amounts of pitchblende specimens were stored. To determine to what degree these samples would affect the radon content of the air, it was decided to make electroscopic tests on the atmosphere of these several rooms. A record of these tests is found in Table IV.

The most important tests naturally were in the radium laboratories, and results given in Table III show the amounts of radon present at different times. The results shown for February, March, and April indicate radon due to ore, dust, or some other source, as active operations were not being carried on at this time.

TABLE III

Main Laboratory—(40 ft. x 22 ft.)	
11/3/32	1220×10^{-11} curies per cu. metre.
4/5/32	1826×10^{-11} " "
30/9/32	2167×10^{-11} " "
16/1/33	1055×10^{-11} " "
21/2/33	632×10^{-11} " "
16/3/33	473×10^{-11} " "
20/4/33	616×10^{-11} " "
Small Laboratory—(22 ft. x 19 ft.)	
30/9/32	1734×10^{-11} curies per cu. metre.
16/1/33	3259×10^{-11} " "
21/2/33	400×10^{-11} " "
16/3/33	377×10^{-11} " "
19/4/33	968×10^{-11} " "
Offices on Same Floor as above Laboratories—	
Room 207—(20 ft. x 16 ft.)	
18/1/33	518×10^{-11} curies per cu. metre.
17/3/33	140×10^{-11} " "
Room 207A—(16 ft. x 10 ft.)	
18/1/33	80×10^{-11} curies per cu. metre.
20/3/33	None.

The results in Table IV are from tests on certain offices and laboratories located in other buildings of the Department of Mines.

TABLE IV

Office A (15 ft. x 12 ft.)	$1,904 \times 10^{-11}$ curies per cu. metre.
Office B (20 ft. x 12 ft.)	$2,176 \times 10^{-11}$ " "
Specimen Cabinet in B	$1,225,000 \times 10^{-11}$ " "
Office C (15 ft. x 12 ft.)	220×10^{-11} " "
Office D (18 ft. x 16 ft.)	$1,946 \times 10^{-11}$ " "

In office A a number of specimens of pitchblende were stored. In office B a cabinet containing several hundred pounds of pitchblende was situated. A test on one of the lower drawers in this cabinet gave the high result above recorded. About 10 pounds of specimens in a metal case was situated in office C. In office D some 60 pounds of pitchblende was exposed on a table. Several rooms adjoining these offices were tested and no radon was found in the atmosphere.

The results of these tests established the fact that even small amounts of pitchblende kept in a room will make the atmosphere appreciably radioactive. In the case of the large cabinet, steps were at once taken to have it removed to an unoccupied room. The value of these tests lies in the light they may throw on conditions existing in a plant or the underground working of a mine containing radioactive ores. It is at once apparent that very serious study must be given to methods of ventilation. Even such a radioactive atmosphere as shown in some of the above-mentioned offices might become a hazard if a person were exposed to this atmosphere for a prolonged time. With quantities of pitchblende the hazard is greatly increased and may become serious.

CONCLUSIONS

The purpose of the investigation was to examine and determine all possible sources of danger and to carry out measures of precaution to overcome them or reduce them to a minimum.

Expired air tests on workers showed the presence of radioactivity in their exhaled air.

Examination of the blood showed an abnormal condition in two of the workers. After work with radium-bearing materials was discontinued their blood count returned to a normal condition. One worker showed no change in his blood condition.

It was found that the presence of high-grade pitchblende ore in a room produced sufficient radon to render the air distinctly radioactive.

The electroscopic test on expired air and room atmosphere was found to be a very effective and satisfactory method for determining the presence of radon in the lungs or atmosphere.

Hazards are due to radon in the atmosphere and dust from ore or products of treatment in the early stages of radium recovery and, while not as potent as in the refining stages, the effect may be dangerous if spread over a long period of time.

The fact that dangers from this source are more difficult to combat, owing to their physical nature, makes it all the more important that the maximum of precautionary measures be taken.

The results of this investigation have contributed informative data upon the methods employed for the examination for radioactivity in workers, the necessary ventilation of laboratories and buildings, and the handling and storage of radioactive materials.

ACKNOWLEDGMENTS

The writer wishes to acknowledge his indebtedness to the following individuals who furnished valuable information and assistance in the preparation of this report:—

Dr. Herman Schlundt and his associates, of the University of Missouri; Dr. S. C. Lind, of the University of Minnesota; the Deputy Minister of Pensions and National Health and the officers of his department.

Special thanks are due to Dr. N. MacL. Harris and Mr. H. M. Lancaster, of the Department of Pensions and National Health, for their valuable suggestions in reviewing this report.

REFERENCES

[1]Report of the Ontario Royal Commission on the Use of Radium and X-Rays in the Treatment of the Sick, 1932, p. 156.

[2]Rutherford, E.—Radioactive Substances and their Radiations.

[3]"Radium"—Publication by Radium-Belge.

[4]Martland, H. S., Conlon, Philip, and Knef, J. P.: "Some Unrecognized Dangers in the Use and Handling of Radioactive Substances"—Journal of the American Medical Association. December 5, 1925, pp. 1769-1776.

[5]Schlundt, Herman, McGavock, Wm., and Brown, Mildred: "Dangers in Refining Radioactive Substances"—Journal of Industrial Hygiene, vol. XIII, No. 4 (April, 1931.)

[6]Martland, Harrison S., M.D.: "The Occurrence of Malignancy in Radioactive Persons" —The Amer. Jour. of Cancer, vol. XV, No. 4, p. 2439 (October 1931).

[7]Lind, S. C., and Roberts, L. D.: "New Determination of the Absolute Value of the Radium: Uranium Ratio"—Jour. Amer. Chem. Soc., vol. XLII, No. 6 (1920).

[8]Martland, Harrison S., M.D.: "The Occurrence of Malignancy in Radioactive Persons"—The Amer. Jour. of Cancer, vol. XV, No. 4 (October 1931).

[9]Thiele, Rostoski, Saupe and Schmorl: "Uber den Schneeberger Lungenkrebs, Munchen. med. Wchnschr."—71 : 24, 1924.

[10]League of Nations Report—Health Organization Cancer Commission—May 3rd, 1929. Dr. F. Pirchan and Dr. H. Sikl.

Prof. Julius Löwy. Ibid., June 19th, 1929.

[11]Report of the Ontario Royal Commission on the Use of Radium and X-Rays in the Treatment of the Sick, p. 156 (1932).

[12]Details supplied by Dr. Herman Schlundt—University of Missouri, Columbia, Mo., U.S.A.

[13]Personal Communication—1931.

[14]Investigations in Ore Dressing and Metallurgy, 1931—No. 728, pp. 138-148; and 1932—No. 736, pp. 249-267. Investigations in Mineral Resources and the Mining Industry, 1931, pp. 55-92.

[15]Schlundt, McGavock, and Brown: "Dangers in Refining Radioactive Substances"—Journal of Industrial Hygiene, vol. XIII, No. 4 (April 1931).

[16]Kinoshita, S., Nishikawa, S., and Ono, S.: "On the Amount of Radioactive Products Present in the Atmosphere"—Phil. Mag. 1911, 22, 821.

[17]Bulletin 104, U. S. Bureau of Mines—p. 91.

BIBLIOGRAPHY

"X-Ray and Radium Protection"—Engineering, vol. 112, p. 13, (July 1, 1921).

"Industrial Hazards of Radioactive Material"—Scientific American, vol. 136, p. 240 (April 1927).

"Radium Poisoning Regarded as New Occupational Disease"—Chemical and Metallurgical Engineering, vol. 32, p. 654 (July 1925).

"French Experience Regarding Occupational Diseases Caused by X-Rays and Radioactive Substances"—United States Monthly Labour R., vol. 28, pp. 77-79 (January 1929).

"Industrial Poisoning Cases among Watch-dial Painters"—Ibid, vol. 27, pp. 42-43, (July 1928).

"Survey of Industrial Poisoning from Radioactive Substances"—Ibid, vol. 28, p. 1200 (June 1929).

"Effects of Use of Radioactive Substances on the Health of Workers"—Ibid, vol. 22, pp. 958-971 (May 1926).

Hoffman, Frederick: "Radium (Mesothorium) Necrosis"—Journal of American Medical Association, 85, 961-965 (September 26, 1925).

Reitter, G. S., and Martland, H. S.: "Leucopenic Anemia of the Regenerative Type Due to Exposure to Radium and Mesothorium"—Amer. Jour. Roentgenol, 16: 161-167 (1926).

Flinn, F. B.: "Elimination of Radium Salts from the Human Body"—Journal of American Medical Association, 96: 1763-1765 (1931).

Schlundt, Herman, Barker, Howard H., and Flinn, F. B.: "The Detection and Estimation of Radium and Mesothorium in Living Persons"—Amer. Jour. Roentgenol, 21: 345-354 (1929).

United States Department of Labour Bulletin: "Radium Poisoning" (1929).

Rolleston, Humphrey: "Critical Review. The Harmful Effects of Irradiation (X-Rays and Radium)." Quart. Jour. Med. 23: 101 (1930).

Schlundt, Herman: "Refining of Mesothorium"—Journal of Chemical Education, July 1931.

Evans, Robley D., Ph.D.: "Radium Poisoning; a Review of Present Knowledge"—American Journal of Public Health, vol. XXIII, No. 10, pp. 1017-1023 (October 1933).

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