CANADA

DEPARTMENT OF MINES

HON. W. A. GORDON, MINISTER; CHARLES CAMSELL, DEPUTY MINISTER

MINES BRANCH

JOHN McLEISH, DIRECTOR

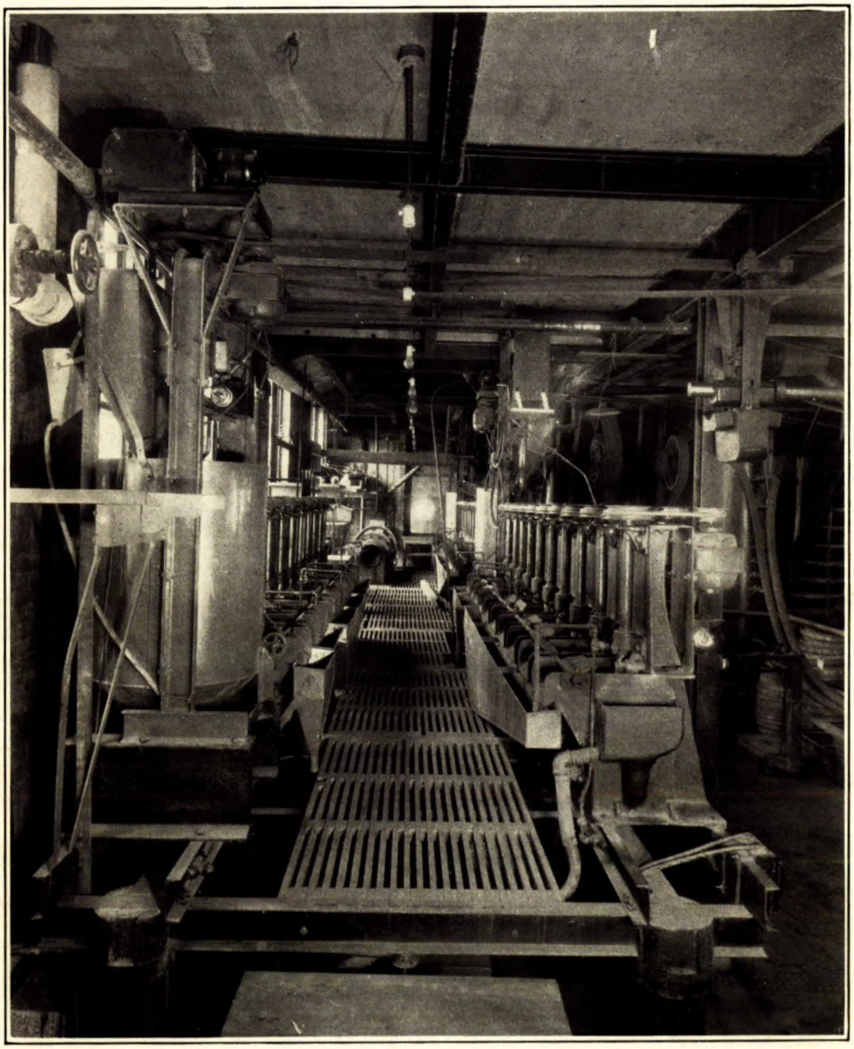
INVESTIGATIONS IN ORE DRESSING AND METALLURGY

(Testing and Research Laboratories)

1931

OTTAWA
F. A. ACLAND
PRINTER TO THE KING'S MOST EXCELLENT MAJESTY
1932

No. 728



New flotation unit, capacity 500 pounds of ore feed an hour, consisting of three sets of ten cells each, with individual motor drives, conditioning tanks, reagents feeders, automatic samplers, etc., for three-mineral float or three-stage flotation. Regrind mill in background. Pumping equipment beneath cell floor.

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Annual reports on Mines Branch investigations are now issued in four parts, as follows:—

Investigations of Mineral Resources and the Mining Industry.

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

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

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

Other reports on Special Investigations are issued as completed.

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MINES BRANCH INVESTIGATIONS IN ORE DRESSING AND METALLURGY, 1931

I

GENERAL REVIEW OF INVESTIGATIONS

W. B. Timm

Chief of Division

The investigative work of the Division of Ore Dressing and Metallurgy is steadily increasing in spite of unfavourable conditions in the base metal section of the mineral industry. This is accounted for by the increased activity in the search for new gold properties; the re-opening and development of old gold mines; by the discovery of rich pitchblende and silver ores at Great Bear lake, Northwest Territories; and by the increased attention being paid to the occurrences of non-metallic minerals in Canada. The confidence which the industry has placed in the work being carried out in the Ore Dressing and Metallurgical Laboratories has resulted in samples of virtually every new ore discovery and of ore from the re-opened old properties, being submitted for examination and investigation as to treatment methods.

In former years, with the exception of 1930, the greater proportion of the investigative work of the metallic ores section, was conducted on base metal ores, such as copper, lead, and zinc. In 1931, there was a decided increase in the investigations on the treatment of gold ores. Of the twenty investigations completed and reported on by this section, sixteen were on gold ores, one on copper-gold, one on a pyrite ore, one on a cobalt ore, and one on a molybdenite ore. From British Columbia, samples of gold ores were submitted from the Lorne mine in the Bridge River district, the Reno gold mine, in the Salmo district, and from a property at Vidette lake in the Kamloops district. The concentration of the molybdenite ore from Alice Arm was also investigated. From Alberta, samples of the black sand concentrate from the placer operations on McLeod river were examined. From northern Ontario mines further investigative work was conducted on Howey gold ore to improve the milling practice and a study was made of the effect on the cyanidation of Howey ore of adding refinery slags to the ore, and of using spent carbide instead of lime. Gold ores from the Moss mine at Kashabowie; the Miller Bay mine, in the Sudbury district; the Telluride gold mine at Englehart, the Hayden gold mine, in the Porcupine district; and from the new discoveries in Chester township, Algoma district, were investigated. Further investigative work was also carried out on mill products from the Kirkland Lake mines. The concentration of a pyrite ore from the Goudreau district, and of a cobalt ore from Werner lake, Patricia district, was investigated and reported on. From northern Quebec, samples of gold ores from a number of new properties in the Pascalis area were submitted for treatment tests, and concentration tests were made on several samples of copper-gold-pyrite ores from the Chibougamau district. Reports on the above ores are to be found in Section II.

An increase is also shown in the investigative work on non-metallic minerals. Ten investigations were conducted and reported on. Silica sands and sandstone from Beauharnois, Quebec; from Melford, Cape Breton; and from Guigues township, Témiscamingue county, Quebec, were washed and prepared for various uses such as glass, moulding and blasting sand, etc. Crushing tests were made on Glauber's salt from Ormiston, Saskatchewan, and on the preparation of slate for slate granules. Tests were conducted on syenite from Methuen township, Peterborough county, Ontario, for the removal of iron impurities. Washing tests were made on shipments of marl from Milton, Ontario. Concentration tests for the purpose of separating rock material from clam shells from Denman island, B.C., were made, and samples of gypsum from Bull River, B.C., were calcined and tested for their plaster-making qualities. Reports on the above non-metallic investigations are to be found in Section III.

The investigation into the treatment of the pitchblende ores from Great Bear lake, Northwest Territories, for the extraction of radium, was commenced and satisfactory progress made, considering the small staff assigned to the work. A study was made of the literature on the treatment of radioactive ores and minerals, on the methods of measuring and determining radium element by electroscopic means and the hazards involved and precautions to be taken. A preliminary laboratory examination was made of the known processes, an electroscopic laboratory fitted up for emanation measurement and determination, and a treatment laboratory equipped on pilot-plant scale. Reports on the progress made are to be found in Section IV.

Five investigations were completed and reported on by the section dealing with ferrous metallurgy. A study was made of the production of low-sulphur sponge iron from ore-coal mixtures. Laboratory concentration tests were made on Texada Island (B.C.) iron ore and on Bathurst (N.B.) iron ore. Series of tests were made on the mechanical properties of certain samples of Monel metal. An investigation was conducted on the semi-direct production of nickel-steel from Sudbury copper-nickel ores. In addition to the above investigations, co-operative work with Canadian steel companies was carried out. These latter investigations concerned either the production in Canada of material hitherto imported, or the improvement in the quality of the product of the manufacturer concerned. Included in these were the production by the basic electric furnace of a steel used in Canada but imported abroad; the annealing of large high-carbon chromium molybdenum steel castings; the production of a modified composition of austenitic manganese steel for special applications; the production of a cast alloy ring die; and the investigation of a failed wedge-bar from a ball mill lining. Thermal analyses, metallographic studies, mechanical tests and normality tests were made for various Canadian concerns. Reports on investigations in ferrous metallurgy are to be found in Section V.

In the chemical section of the Division, the chief chemist reports increased work over previous years. A larger percentage of the samples submitted by those engaged on the investigations involved much fuller and in many cases complete analyses. The report of the chemical laboratory is given in Section VI.

The laboratory facilities are gradually being improved upon, in order to enlarge the scope of the investigative work and to carry out the investigations more thoroughly and with greater despatch. The new ore dressing and metallurgical laboratory building was completed, the equipment installed and the building occupied in June. This addition to the laboratory facilities was an urgent necessity to take care of the increased demands of the mining industry for work on ore treatment and other metallurgical problems. In this new building improved facilities are provided for the microscopic and spectrographic examination of ores and mill products, for the metallographic examination of metals and alloys, for the mechanical testing of iron and steel and other alloys, for research on ore treatment in special ore dressing, hydrometallurgical and electrochemical laboratories, and for the chemical analyses of ores, metals and alloys and metallurgical products from the test operations.

In the main plant laboratory for the treatment of metallic ore a new flotation unit was installed for continuous test runs on large shipments of ore up to carload lots. It consists of three sets of ten-cells each for a three-mineral float or three-stage flotation with accessory equipment, such as conditioning tanks, regrind mill and classifier, reagent feeders, automatic samplers and small centrifugal pumps. It has a capacity of about 500 pounds of ore feed an hour, is a most flexible unit, and ideally suitable for test operations. A photograph of this unit is shown as the Frontispiece of this report.

In the pyrometallurgical laboratory, a briquetting machine and a Dwight and Lloyd laboratory continuous sintering machine were installed for making sponge iron briquettes and sintering iron ore concentrates, etc.

In the hydrometallurgical and electrochemical laboratories, small pilot-plant equipment was installed for the treatment of the pitchblende ores from Great Bear lake, Northwest Territories, for the extraction of radium. Ten-kilogram lots of ore are treated in this plant, which is about one-tenth the size of an ordinary commercial unit. An electroscopic laboratory was also equipped at the Mines Branch building, Sussex street, for the measuring and determining of the radium element in the ores and various products from test operation. Plate II is a photograph of the small pilot-plant.

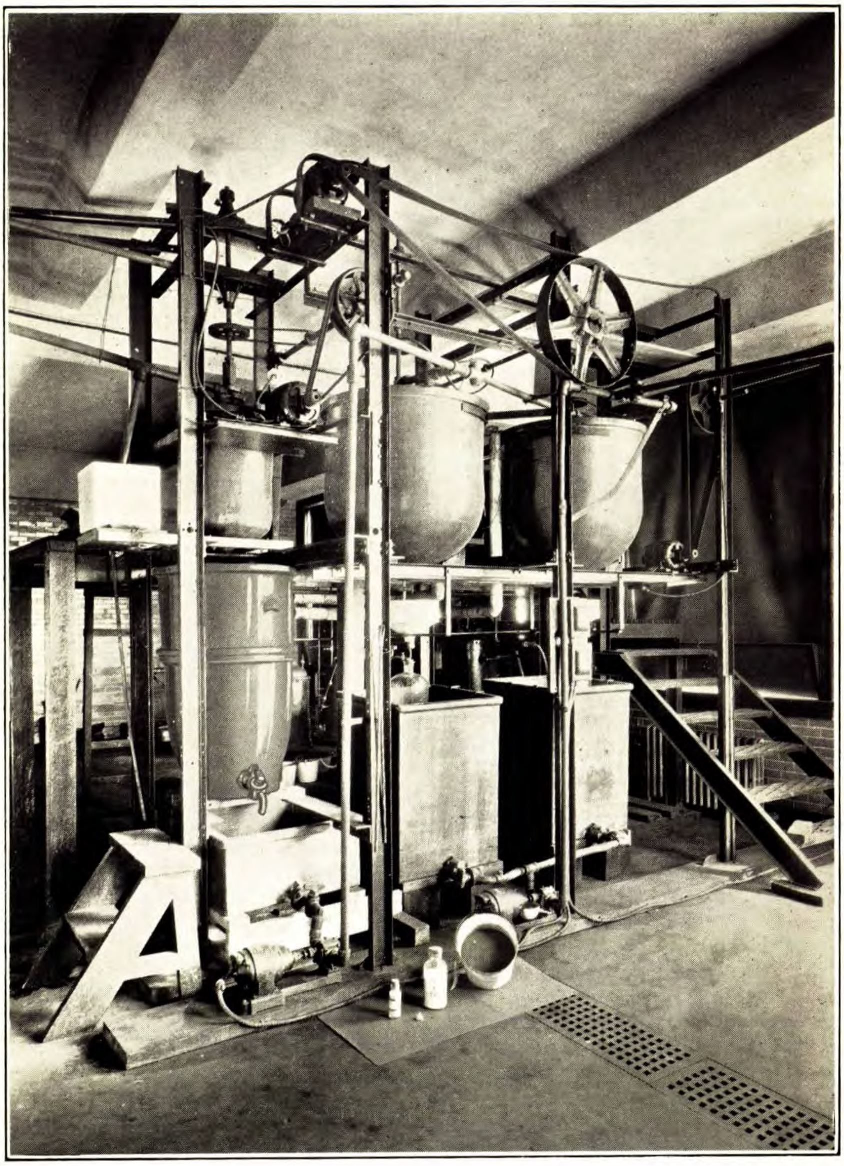
A mineragraphic laboratory was equipped for the microscopic and spectrographic examination of ores and minerals. The equipment consists of a small hydraulic press, electrically heated, for mounting specimens of ore and mill products in bakelite; grinding and polishing machines for the preparation of polished surfaces, thin sections, and micro-chemical analyses; micro-boring apparatus for obtaining pure samples of mineral from polished surfaces; photo-micrographic apparatus; and an Adam Hilger quartz spectrograph with facilities for the production of arc and spark spectra.

The co-operative investigative work carried on in former years with the Base Metal Extraction Company, Limited; the Cassel Cyanide Company, a subsidiary of Canadian Industries, Limited; and with the American Cyanamid Company, Niagara Falls, Ontario, was curtailed, due to economic conditions. The research fellows maintained by the two latter companies were withdrawn and their services on investigative work lost to the Division.

In co-operation with the engineering staff, engineers from the Hardinge Company conducted in the laboratories a practical demonstration of the operation of a new design of classifier. The results obtained were very promising for the introduction of this type of classifier in the milling plants and concentrators of the industry in Canada.

Advantage was taken of the laboratory facilities by the metallurgists from operating companies to carry out investigations on their ore treatment problems. This type of co-operative work is invited and the use of the laboratory facilities is placed at the disposal of the mining industry.

The advice of the staff was solicited by consulting engineers engaged on the design of ore treatment plants. The knowledge gained by carrying out the investigative work was imparted to the industry in this manner.



Pilot plant for investigation of the treatment of Great Bear Lake pitchblende ores for the extraction of radium. Note: in foreground, the charge of ore, 10 kilograms, in pail; the bottle of lead chloride recovered; the small button of silver recovered; the small bottle of crude radium salt.

II

REPORTS OF INVESTIGATIONS: METALLIC ORES SECTION

Report No. 387

CONCENTRATION TESTS ON GOLD ORE FROM THE HOWEY GOLD MINES, LTD., RED LAKE, ONTARIO

C. S. Parsons

Shipment. A shipment of 485 pounds of gold ore was received at the Ore Dressing Laboratories of the Mines Branch, April 17, 1931. The shipment was made by the Howey Gold Mines, Ltd., from their mine at Red Lake, Ontario.

Characteristics of Ore and Analysis. The sample represents the run-of-mine ore from the property. The assay made on the sample showed it to contain 0·32 ounce per ton of gold and 0·12 ounce per ton of silver.

Purpose of Experimental Tests. The following tests were made with two objects in view: first, to determine whether or not, as a preliminary step before cyanidation, a portion of the ore could be discarded at a relatively coarse size—that is between 20 and 40 mesh—by some means of classification and tabling; and second, by flotation at the same sizes.

EXPERIMENTAL TESTS

Test C-1

Ore, 3,900 grammes, was prepared by screening a −14-mesh sample and crushing the +20-mesh material to all pass the 20-mesh screen.

This product was then fed to a Richards hydraulic classifier and the following products made.

Product	Weight		Assay, Au, oz./ton	Distribution of values
	Grms.	Per cent		
Product No. 1	1,720·5	44·1	0·43	69·0
Product No. 2	260·7	6·7	0·17	4·1
Product No. 3	1,918·8	49·2	0·15	26·9
Totals	3,900·0	100·0	0·275	100·0

Test T-1

A table test was made on product No. 1 from the above classification, after the product had been dried, weighed and sampled.

Product	Weight		Assay, Au, oz./ton	Distribution of values
	Grms.	Per cent		
Concentrates	473·4	34·7	1·22	91·5
Tailing	889·6	65·3	0·06	8·5
Totals	1,363·0	100·0	0·46	100·0

A screen analysis was made of the tailing from this table test.

Product	Weight		Assay, Au, oz./ton	Distribution of values
	Grms.	Per cent		
+28 mesh	368·8	42·0	0·07	50·5
−28+ 35 mesh	416·4	47·4	0·05	40·7
−35+ 48 mesh	81·0	9·2	0·04	6·3
−48+ 65 mesh	7·0	0·8	0·04	0·5
−65+100 mesh	1·2	0·1	0·12	0·3
−100 mesh	4·5	0·5	0·19	1·7
Totals	878·9	100·0	0·06	100·0

From Test C-1, 44 per cent of the ore is represented in product No. 1 from the classifier which was tabled in Test T-1. The table concentrate represented 34·7 per cent of the feed to the table or 15·3 per cent of the original ore, and contained 1·22 ounces gold per ton or 91·5 per cent of 69 per cent which is about 63·2 per cent of the gold in the ore. A tailing containing $1.20 in gold was made which, if figured back to the ore, represented 28·8 per cent of the original ore fed to the classifier and represented a loss of 8·5 per cent of 69·0 per cent which is 5·86 per cent. That is to say by classifying the ore at −20 mesh and tabling the coarse product, 28·8 per cent of the ore can be discharged as a table tailing containing $1.20 in gold and representing a loss of 5·86 per cent of the gold in the ore. This does not seem a very attractive proposition.

It was also noted that all the sizes in the table tailing coarser than 65 mesh contained metallic gold still locked in the quartz.

FLOTATION TESTS

Tests F-1 and F-2

These tests are duplicates; 2,000 grammes of ore previously crushed to pass 14 mesh was used in each test. The following reagents were ground with the ore: xanthate, 0·10 pound per ton; coal-tar creosote, 0·18 pound per ton, and pine oil was added to the flotation cell. The grinding of each lot of −14-mesh material was done in a small rod mill. The accompanying screen test made on the tailing from Test F-1 will give approximately the size to which the ore was ground.

Test No.	Product	Weight		Assay, Au, oz./ton	Recovery, per cent
		Grms.	Per cent		
F-1	Concentrates	100·2	5·1	3·64	76·3
	Tailing	1,883·0	94·9	0·06	23·7
	Totals	1,983·2	100·0	0·24	100·0
F-2	Concentrates	104·2	5·2	3·66	83·5
	Tailing	1,885·5	94·8	0·04	16·5
	Totals	1,989·7	100·0	0·23	100·0

A screen analysis was made on the tailing from Test F-1.

Product	Weight		Assay, Au, oz./ton	Distribution of values
	Grms.	Per cent		
+ 35 mesh	32·0	3·6	0·71	42·5
− 35+ 48 mesh	137·0	15·5	0·16	41·3
− 48+ 65 mesh	147·7	16·8	0·03	8·3
− 65+100 mesh	133·3	15·1	0·015	3·8
−100 mesh	431·1	48·9	0·005	4·0
Totals	881·1	99·9	0·06	99·9

These two tests and the screen analysis of the tailing show that recovery at this size of grinding was poor. The screen analysis shows that 83·8 per cent of the values in the tailing is in the screen products coarser than 48 mesh. The gold in these sizes was mostly metallic still unfreed from the quartz.

Test F-3

In this test the grinding was done in a small ball mill. The reagents used were: soda ash, 1·5 pounds per ton; sodium, xanthate, 0·10 pound per ton, and coal-tar creosote, 0·18 pound per ton. A little pine oil was added to the cell.

Product	Weight		Assay, Au, oz./ton	Recovery, per cent
	Grms.	Per cent		
Concentrate	155·7	7·8	2·92	77·3
Tailing	1,842·2	92·2	0·07	22·7
Totals	1,997·9	100·0	0·29	100·0

A screen analysis was made on the tailing.

Product	Weight		Assay, Au, oz./ton	Distribution of values
	Grms.	Per cent		
+ 35 mesh	22·2	1·2	2·62	43·6
− 35+ 48 mesh	86·4	4·7	0·11	7·1
− 48+ 65 mesh	238·4	12·9	0·20	35·7
− 65+100 mesh	319·2	17·3	0·02	4·8
−100 mesh	1,176·0	63·8	0·01	8·8
Totals	1,842·2	99·9	00·7	100·0

When crushing the +35-mesh screen product to 100 mesh in preparation for an assay sample, 1·84 ounces of the 2·62 ounces was found to report as metallic gold, and the 65-mesh product 0·15 ounce of the 0·20 ounce reported as metallic gold—that is about 70 per cent. This metallic gold was partly free at the screen sizes but most of it was still locked up in the quartz and freed on crushing to 100 mesh.

These screen analyses show that the ore must be crushed to practically all to pass 65 mesh in order to free the metallic gold from the quartz.

SUMMARY AND CONCLUSIONS

Table Concentration. It was immediately recognized that in order to prevent the loss of fine sulphides, the feed to the table would have to be classified. This was done in Test C-1 in a Richards hydraulic classifier, and a product obtained which was −20 mesh and only 1·4 per cent −48 mesh, and represented 44·1 per cent of the weight of the ore. This product was tabled and a concentrate made which represented 34·7 per cent of the weight of the feed or a concentration ratio of 1: 2·88. The concentrate assayed 1·22 ounces per ton which gave a recovery of 91·5 per cent. The tailing product contained $1.20 in gold of which about 70 per cent was metallic gold, freed as such when the sample was prepared for assay.

A further point in connexion with these tests is that the tailing from the tables represented 65·3 per cent of the feed and the feed represented 44·1 per cent of the original ore, therefore, the amount discarded was 65·3 per cent of 44·1 per cent which is 28·8 per cent of the original ore fed to the classifier, and contained as stated above $1.20 in gold. For details see Tests C-1 and T-1 and the accompanying screen analysis of the products.

Coarse Flotation Tests. In Tests F-1 and F-2 the ore previously crushed to pass 14 mesh was ground in a small batch rod mill, and in Test F-3 the ore was ground in a small batch ball mill. Roughly, the ore was ground so that 1 to 4 per cent remained on 35 mesh. In the rod mill product only 49 per cent was −100 mesh and in the ball mill product 64 per cent was −100 mesh. These three tests are very conclusive in as much as they show that a product coarser than 48 mesh can not be discarded. Considering the practice at the mill at present, which is to crush only to 40 mesh and cyanide, extracting 95 per cent of the gold, these tests also show that in order to produce as low a tailing by flotation it is necessary to grind finer. The screen tests and analyses made on the tailing from these three tests show this very plainly. In addition, these screen tests showed the presence of metallic gold. In Test F-3 the +35-mesh product in the tailing contained 2·62 ounces per ton in gold of which 1·84 ounces was reported as metallic when preparing the sample for assay. From these screen analyses made on the flotation tailing, the conclusion drawn is that in order to successfully apply flotation the ore must be ground so that practically all will pass 65 mesh.

In connexion with the flotation tests, a factor was brought out which may be of practical value. It will be observed in studying the screen tests made on the tailings from flotation Tests F-1, F-2, and F-3 that the −65-mesh product from all three tests runs less than 40 cents per ton. In order to lower the tonnage treated by cyanidation this fact could be taken advantage of by floating the ore first at 35 mesh, passing the tailing to a bowl classifier and discharging the overflow to waste. The sand rake product could then be returned and mixed with the flotation concentrate, reground and cyanided. By such a flow-sheet nearly 65 per cent of the ore could be discarded before cyanidation.

Report No. 389

THE RECOVERY OF GOLD IN ORE FROM THE LEROY FIEDMONT MINING COMPANY, LIMITED, TOWNSHIP OF LOUVICOURT, ABITIBI COUNTY, QUEBEC.

Alex. K. Anderson

Shipment. A shipment contained in 2 bags, gross weight 240 pounds, was received on February 27, 1931, at the Ore Dressing and Metallurgical Laboratories, Ottawa, forwarded by Oscar Roy, Barraute, Quebec, for the LeRoy Fiedmont Mining Company, Limited, 35 Notre Dame St. W., Montreal, Quebec. This ore was said to have come from the company's holdings in the township of Louvicourt, Quebec.

Characteristics of the Ore. The ore consisted of material taken from the surface of the vein and was composed of white quartz together with highly oxidized sulphides. Some little iron pyrite in a semi-oxidized condition was present. No free gold was detected.

Purpose of Experimental Tests. Tests were undertaken to determine what recovery of gold could be made by standard methods. These showed that 58 per cent of the gold in this shipment can be recovered by amalgamating the ore crushed to pass 48 mesh and that 98 per cent of the gold can be extracted by cyanidation when the ore is ground to pass 150 mesh.

Sampling and Analysis. The entire lot was crushed to pass $\frac{1}{4}$ inch, quartered, and one of these portions further reduced by passing through rolls and again quartered by passing through a Jones riffle sampler. By further grinding and quartering a representative portion passing 100 mesh was obtained for assay. This showed the shipment to contain 0·31 ounce gold per ton, or $6.20.

Experimental Tests

Tests were made to determine the recovery of gold that would be obtained by amalgamating coarsely ground ore. Cyanide tests on different sizes were also made.

AMALGAMATION

Test No. 1

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

Heads	Au	0·31 oz./ton
Amalgamation tailing	Au	0·13 “
Recovery		58·1 per cent

Amalgamation leaves a residue containing $2.60 in gold per ton.

AMALGAMATION AND CYANIDATION

Test No. 2

The tailing resulting from Test No. 1 was cyanided for 48 hours without regrinding with a cyanide solution containing one pound KCN per ton; 1:3 dilution. Lime at the rate of 7 pounds per ton ore was added.

Cyanide heads	Au	0·13 oz./ton
Cyanide tailing	Au	0·06 “
Extraction		53·8 per cent
Consumption: KCN		0·6 lb./ton
CaO		3·7 “

On cyaniding the tailing from the amalgamation operation, 53·8 per cent of the gold remaining in this product is extracted, or a recovery of 80·6 per cent of the gold by amalgamation followed by cyanidation is obtained.

CYANIDATION

Test No. 3

A sample of the ore was crushed to pass 48 mesh and cyanided without prior amalgamation.

Heads	Au	0·31 oz./ton
Cyanide tailing	Au	0·06 “
Extraction		80·6 per cent
Consumption: KCN		0·24 lb./ton
CaO		2·8 “

This shows that the same recovery of gold is obtained by straight cyanidation as by amalgamating and cyaniding the amalgamation tailing.

Test No. 4

A sample of the ore was crushed to pass 100 mesh and cyanided as in Test No. 3.

Heads	Au	0·31 oz./ton
Cyanide tailing	Au	0·01 "
Extraction		97 per cent
Consumption: KCN		0·24 lb./ton
CaO		5·1 "

This shows that material ground to pass 100 mesh yields all but 20 cents of the contained gold by cyanidation at 100 mesh.

CYANIDATION

Test No. 5

A sample of the ore was ground to pass 150 mesh and cyanided as in Test No. 4.

Heads	Au	0·31 oz./ton
Cyanide tailing	Au	0·005 "
Extraction		98·4 per cent
Consumption: KCN		0·96 lb./ton
CaO		6·2 "

Cyaniding the ore ground to pass 150 mesh leaves a tailing assaying 10 cents, representing an extraction of 98·4 per cent.

These tests indicate that to secure a high recovery of gold, the ore should be ground to pass 100 mesh. When finely ground the gold is readily extracted with cyanide, 97 to 98 per cent recovery being obtained. It may be assumed that practically all the gold in this shipment is present as free gold, due to the highly oxidized nature of the deposit at the surface.

As this shipment evidently was taken at or very near the surface, it cannot be considered as representative of the ore-body. Fresh unoxidized ore below the surface may not respond so readily to treatment as this surface material. There may be an increase in sulphide content and a drop in the gold values, or the reverse, at depth.

It is recommended that before any estimate of nature and grade of ore be made, fresh unoxidized vein matter taken below the zone of surface enrichment be secured for test purposes.

Report No. 390

EXPERIMENTAL TESTS ON AN ARSENICAL GOLD ORE FROM THE MILLER BAY GOLD MINES, LTD., SUDBURY DISTRICT, ONTARIO

J. S. Godard

Shipment. A shipment of 110 pounds of ore was received, January 21, 1931, from the Miller Bay Gold Mines, Ltd., per W. E. Smith, Secretary, 287 Bay St., Toronto.

The sample was taken from the property of the Miller Bay Gold Mines, Ltd., situated on the north bank of Howry creek, five miles east of the Algoma Eastern railway, in Sudbury district, Ontario.

Characteristics of the Ore. The sample submitted is an arsenical gold ore. The arsenopyrite is massive and constitutes the main bulk of the ore. The gangue is a hard white quartz.

Metals of the platinum group were reported to be present in the ore, but none was detected in a 10-assay-ton sample.

Analysis of the Ore. Analysis of the head sample is as follows:

Gold	0·85 oz./ton	Nickel	0·06 per cent
Silver	0·08 "	Antimony	nil
Platinum group	nil	Lead	0·07 per cent
Arsenic	29·78 per cent	Zinc	0·03 "
Iron	27·60 "	Sulphur	16·7 "
Copper	Trace	Insoluble	21·10 "
Cobalt	Nil		

Purpose of Experimental Tests. The purpose of the experimental tests was to determine a method of treatment for the recovery of the gold in this ore.

The high arsenical content of this ore precluded any hope of the successful adaptation of amalgamation and concentration methods of treatment.

The ore as represented by this sample might be sold to a smelter either direct or after roasting, or it might be roasted and cyanided, or possibly cyanided without previous roasting.

The experimental tests are based on these latter two methods.

EXPERIMENTAL TESTS

Test No. 1. Flotation

A flotation test was made on this ore for the purpose of checking the head sample and to act as an indicator of the behaviour of the ore.

Results:

Product	Weight %	Assays		Per cent of values	
		Au	As	Au	As
Concentrate	28·0	2·02	36·8	65·3	34·7
Middling	26·1	0·60	35·5	18·1	31·1
Tailing.....+200	5·8	0·48	21·3	3·2	4·1
" −200	40·1	0·29	22·4	13·4	30·1

Head sample from products:
Au........................ 0·87 oz./ton
As........................ 29·79 per cent

The sulphide content of this ore is too high to be amenable to concentration.

ROASTING AND CYANIDATION OF THE ROASTED ORE

Roasting

A sample of the ore, dry crushed to −18 mesh was roasted to drive off the arsenic and sulphur.

Just sufficient heat was applied to start the reaction and to maintain it until nearly all the arsenic and sulphur were driven off. Towards the end of the roast the temperature was raised and the ore well rabbled in the presence of an excess of air. No attempt was made to obtain a dead roast as this was thought to be impractical on a large scale.

Results:

Head sample to roast:	
Au	0·85 oz./ton
As	29·78 per cent
S	16·70 "
Analysis of roasted ore:	
Au	1·28 oz./ton
As	2·60 per cent
S	0·65 "
Loss of weight during roasting	35·4 "

During the roasting 94·4 per cent of the arsenic and 97·4 per cent of the sulphur were eliminated.

Cyanidation

Test No. 1

Roasted ore 300 grammes at −18 mesh; agitated 46 hours in 1:2·5 solution, KCN 0·075 per cent.

Screen Analysis on the Tailing:

Mesh	Weight, per cent	Assay	Per cent of value
		Au, oz./ton	Au
+ 35	28·6	0·24	42·2
+ 48	14·7	0·17	15·4
+ 65	12·6	0·17	13·1
+100	8·8	0·14	7·6
+200	17·2	0·10	10·6
−200	18·1	0·10	11·1

Tailing	Au 0·163 oz./ton
Extraction	87·3 per cent

Reagents, lb./ton:

—	Roasted ore	Original ore
KCN	7·1	4·6
CaO	26·1	17·0

Test No. 2

Roasted ore 300 grammes at −18 mesh, dry crushed to −65 mesh, agitated 46 hours in 1:2·5 solution, KCN 0·075 per cent.

Tailing screened on 100 mesh and 200 mesh:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	8·0	0·16	12·9
+200	42·3	0·11	46·9
−200	49·7	0·08	40·2

Tailing	Au 0·10 oz./ton
Extraction	92·2 per cent.

Reagents, lb/ton:

	Roasted ore	Original ore
KCN	7·6	4·9
CaO	28·7	18·5

Test No. 3

Roasted ore 300 grammes at −18 mesh, dry crushed to −150 mesh, agitated 46 hours in 1:2·5 solution, KCN 0·075 per cent.

Tailing screened on 200 mesh:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	20·2	0·09	20·2
−200	79·8	0·09	79·8

Tailing	Au 0·09 oz./ton
Extraction	93·0 per cent.

Reagents, lb/ton:

	Roasted ore	Original ore
KCN	7·9	5·1
CaO	33·5	21·6

Test No. 4

Cyanidation of Roasted Ore after Water-washing

Roasted ore 300 grammes at −18 mesh, dry crushed to pass 100 mesh, agitated 1¼ hours; 1:2 dilution, water only—filtered and agitated 47 hours in 1:2·5 solution, KCN 0·075 per cent.

Tailing screened on 200 mesh:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	20·8	0·09	22·8
−200	79·2	0·08	77·2

Tailing .... Au 0·082 oz./ton

Extraction .... 93·6 per cent

	Roasted ore	Original ore
KCN	3·5	2·3
CaO	29·6	19·1

Tests Nos. 5, 6, and 7

Cyanidation Tests on the Ore without Previous Roasting

Test No. 5

Ore 1,000 grammes at −18 mesh, ground 15 minutes in ball mill with 700 c.c. cyanide solution, KCN 0·05 per cent, CaO 0·127 per cent, and lime 6·0 pounds per ton.

After grinding, the pulp was diluted to 2·5 : 1 with solution, KCN 0·074 per cent, CaO 0·116 per cent, and agitated 45 hours.

The cyanide tailings were repulped and screened on 100 and 200 mesh.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	2·6	0·27	3·1
+200	25·8	0·29	33·3
−200	71·6	0·20	63·6

Head sample .... Au 0·85 oz./ton
Tailing .... Au 0·225 oz./ton
Extraction .... 73·6 per cent

Reagents, lb/ton:

KCN	1·80
CaO	10·9

Test No. 6

Ore 1,000 grammes at −18 mesh, ground 30 minutes in ball mill with 700 c.c. cyanide solution. KCN 0·05 per cent, CaO 0·127 per cent, and lime 6·0 pounds per ton.

After grinding the pulp was diluted to 2·5:1 with solution, KCN 0·074 per cent, CaO 0·116 per cent, and agitated 45 hours. The cyanide tailings were repulped and screened on 200 mesh.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	5·4	0·27	8·3
−200	94·6	0·17	91·7

Head sample.......Au 0·85 oz./ton
Tailing.......Au 0·175 oz./ton

Extraction.......79·4 per cent

Reagents, lb/ton:

KCN	2·06
CaO	11·9

Test No. 7

Ore 1,000 grammes at −18 mesh; ground 30 minutes in ball mill with 700 c.c. cyanide solution. KCN 0·075 per cent CaO 0·13 per cent, and lime 6·0 pounds per ton.

After grinding the pulp was diluted to 1:2·5 with cyanide solution. KCN 0·074 per cent, CaO 0·119 per cent—then agitated 23 hours, then filtered and reagitated, using the same solution, for an additional 23 hours, after which period the tailings were filtered and repulped, and screened on 200 mesh.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	6·0	0·22	8·6
−200	94·0	0·15	91·4

Head sample.......Au 0·85 oz./ton
Tailing.......Au 0·154 oz./ton

Extraction.......81·9 per cent

Reagents, lb/ton:

KCN	2·64
CaO	14·5

SUMMARY AND CONCLUSIONS

Roasting and cyanidation of the roasted ore showed recoveries of 87 to 93·6 per cent of the gold. Cyanidation of the ore direct gave a maximum recovery of 81·9 per cent. The consumption of cyanide and lime are high.

Better extractions of the gold were obtained by roasting the ore previous to cyanidation than by direct cyanidation.

Water-washing of the roasted ore before cyanidation gave a slightly improved extraction and greatly lessened the cyanide and lime consumption by removing a large part of the soluble salts formed during the roasting. Without this water-washing operation the solution was badly fouled, and a considerable portion of it would have to be discarded which would further increase the reagent cost.

It is believed that the best results on this type and grade of ore would be attained by roasting the ore and water-washing it before cyanidation.

Report No. 391

A STUDY OF THE EFFECTS OF ADDING GOLD REFINERY SLAGS TO RAW GOLD ORE TREATED BY CYANIDATION AND ALSO SPENT CARBIDE USED AS LIME.

Alex. K. Anderson

Shipment. A half-pound sample of refinery slag was received March 26, 1931, by parcel post from the Howey Gold Mines, Ltd., Red Lake, Ontario, accompanied by spent carbide, rejects from miners' carbide lamps.

Characteristics. The slag was that produced on smelting gold precipitates at the cyanide plant of the company. The smelting charge was made up as follows:

100 lb.	precipitates.
40 "	borax.
25 "	soda ash.
10 "	silica sand.

Melting was carried on in a graphite crucible.

The sample was a dark homogeneous substance carrying no visible prills or pellets of metal.

The spent carbide was a fine, dry white powder giving off an odour of acetylene gas when moistened with water.

Purpose of Experimental Tests. The slag was forwarded to determine if it could be added to the mill feed in order to recover the gold contained in it. The spent carbide was sent to see if it could be used in place of lime to furnish protective alkalinity in cyanide solutions without detrimental effects.

These tests indicate that the addition of slag to the ore increases the gold content in the mill tailing. Carbide also slightly lowers the gold recovery.

Sampling and Analysis. The slag was ground to pass 100 mesh and sampled. The metallics remaining on the screen amounted to 4·26 ounces gold per ton. The −100-mesh portion assayed 36·16 ounces per ton.

EXPERIMENTAL TESTS

As there was no Howey ore on hand, a quartz gold ore with a value of $6.20 per ton was used. Cyanide tests made on this ground to pass 100 mesh gave a tailing assaying $0.20 with a cyanide consumption of 0·24 pound KCN per ton and 5·1 pounds lime. Tests were made adding slag ground to pass 100 mesh at the rate of 20 pounds and 100 pounds per ton of ore. These tests also incorporated the use of the carbide.

Settlement tests were made comparing the effect of spent carbide and lime.

Test No. 1

Quartz ore, assay = $6.20 gold per ton.
Slag added at rate of 20 lb./ton.
Cyanide solution—KCN 1 lb./ton; 1 : 3 dilution.
Lime—5 lb./ton ore.
Head assay: Ore at $6.20 + 20 lb. slag at $7.23 = $13.43.
Tailing assay—$1.00.
Tailing from slag: $1.00-$0.20 =$0.80.
Recovery from slag = 88·9 per cent.
Consumption:
  KCN—0·30 lb./ton ore.
  CaO—3·5 lb./ton ore.

Test No. 2

Same as Test No. 1 only with 100 pounds slag per ton ore.

Heads: Ore at $6.20 + 100 lb. slag at $36.15 = $42.35.
Cyanide tailing—$5.60.
Tailing from slag: $5.60-$0.20 = $5.40.
Recovery from slag: 85·1 per cent.
Consumption:
  KCN—0·60 lb./ton.
  CaO—3·1 lb./ton.

Test No. 3

In this test spent carbide was added at the rate of 5 pounds per ton ore instead of lime, together with 100 pounds slag per ton.

Heads: Ore at $6.20 + 100 lb. slag at $36.15 = $42.35.
Cyanide tailing: $5.60.
Recovery from slag: 85·1 per cent.
Consumption:
  KCN—1·5 lb./ton.
  CaO—3·1 lb./ton.

Spent carbide in conjunction with slag has a marked effect, increasing the cyanide consumed to 1·5 pounds per ton ore.

Test No. 4

In this test no slag was added. Spent carbide at the rate of 5 pounds per ton was used in place of lime.

Heads = $6.20.
Tailing = $0.30.
Recovery = 95·1 per cent.
Consumption:
  KCN—0·20 lb./ton.
  CaO—4·7 lb./ton.

The use of carbide increases the gold in the tailing by 10 cents per ton.

Settlement Tests: Carbide vs. lime in equal quantities show that carbide gives a settling rate of 2 feet ½ inch per hour as against 1 foot 8½ inches when lime is used.

At the end of these tests, the alkalinity of the effluent given by 5 pounds lime per ton of ore was 0·044 per cent CaO, while the carbide showed 0·024 per cent CaO. This indicates that the carbide has approximately one-half the available alkalinity of the lime.

These tests show that refinery slag added to the ore gives higher mill tailings. This increase, doubtless, is caused by gold not dissolved in the slag additions. In order to determine whether the practice of disposing of this gold-bearing slag by putting it through the mill would be economical, it would be necessary to know the freight and treatment charges resulting from shipping to a smelter.

There is an increase of 0·06 pound cyanide consumed when slag at the rate of 20 pounds per ton is added. This increases to 1·26 pounds when 100 pounds of slag are added.

The disposal of this material by adding it to the mill feed is not recommended.

The use of spent carbide has no ill effects on the settling rate. There is, however, an indication that a slightly higher tailing loss, as shown in Test No. 4, may result from its use.

This possibility will more than offset the saving made by using this waste material in place of lime.

Report No. 392

THE RECOVERY OF GOLD AND COPPER FROM THE ORE OF THE TELLURIDE GOLD MINES OF CANADA, LTD., ENGLEHART, ONTARIO

Alex. K. Anderson

Shipments. Three boxes containing 280 pounds of ore were received March 11, 1931, from the Telluride Gold Mines of Canada, Ltd., Englehart, Ontario, shipped by H. Hollands-Hurst, General Superintendent.

A 16-ounce bottle of water from lake Skidoo was also received by mail.

Characteristics of the Ore. The shipment consisted of chalcopyrite in a dark, heavy specular iron, a form of iron oxide. The associated country rock is also of a basic nature. No free gold was visible to the naked eye.

Purpose of Experimental Tests. The shipment was made for the purpose of determining the type of concentrator necessary to treat this class of material. A 50-ton per day flotation mill has been operating on the property with varying degrees of success. The flotation tests were made with the object of establishing the best conditions under which to operate this plant.

Sampling and Analysis. The entire shipment was crushed to pass ½ inch, quartered, and by further reduction in size and cuts through a Jones riffle sampler a representative portion −100 mesh was obtained for analysis. This sample showed the shipment to contain:

	Per cent		Oz./ton
Copper	2·67	Gold	0·33
Zinc	0·12	Silver	0·28
Lead	0·02	Arsenic	trace
Iron	33·7		

The sample of water furnished was examined and found to be exceptionally pure, colourless, odourless, and containing no acids. It had a pH value of 7·0. Total dissolved salts present were 53 parts per million, chiefly iron carbonates.

EXPERIMENTAL TESTS

The practice at the mill has been to grind the ore with 5 pounds lime per ton and float with potassium ethyl xanthate and pine oil. The rougher concentrate was cleaned and the cleaner tailing returned to the head of the rougher cells.

The first part of this investigation consisted of a series of flotation tests to determine if this milling practice is the most suitable. Later information disclosed the fact that a maximum recovery of gold as bullion on the property was desired. Amalgamation tests were, therefore, made to see if the gold could be recovered by this method.

It is a well established fact that lime used in flotation depresses gold and gold-bearing pyrite and also has a hardening effect on amalgam, resulting in decreased recoveries. The following flotation tests show the effect of varying amounts of lime and also that of substituting soda ash in place of lime.

STANDARD MILL PRACTICE.

Test No. 1

The ore was ground with 5 pounds lime per ton and floated with 0·08 pound potassium ethyl xanthate and 0·04 pound pine oil per ton. The rougher concentrate was cleaned.

Product	Weight, per cent	Assay				Per cent of values	
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Cu	Au
Heads (cal.)	100·0	2·62	....	0·34	0·27	100·0	100·0
Concentrate	11·6	21·66	0·25	2·28	1·84	96·0	76·9
Cleaner tailing	5·3	1·20	....	0·40	0·43	2·4	6·2
Tailing	83·1	0·05	....	0·07	0·04	1·6	16·9
Rougher concentrate (cal.)	16·9	15·24	....	1·69	....	98·4	83·1

The grinding was much finer than that practised on the property where they grind 65 per cent −200 mesh. A screen analysis of the tailing from this test showed 97 per cent −200 mesh.

The conditions under which this test was made resulted in a tailing containing $1.40 in gold and a rougher concentrate assaying 15·24 per cent copper, 1·69 ounces gold per ton with a recovery of 83·1 per cent of the gold. On cleaning this product, 6·2 per cent of the gold is found in the cleaner tailing which has a slightly higher value than the feed to the cells.

Test No. 2

This test is similar to Test No. 1 with the exception that 5 pounds soda ash per ton was used in place of lime.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·0	2·74	0·33	100·0	100·0
Concentrate	10·3	25·44	2·48	95·5	77·6
Cleaner tailing	5·7	1·45	0·56	3·0	9·7
Tailing	84·0	0·05	0·05	1·5	12·7
Rougher concentrate (cal.)	16·0	16·89	1·80	98·5	87·3

Soda ash has the effect of lowering the tailing from 0·07 ounce gold to 0·05 ounce gold per ton with an increase in extraction from 83·1 per cent to 87·3 per cent in the rougher concentrate. On cleaning, the gold in this product shows more tendency to drop out. The recovery of copper is the same, 98·5 per cent.

Test No. 3

In this test the lime was reduced to 2 pounds per ton. Cyanide at the rate of 0·10 pound per ton was added to depress zinc and iron sulphides.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·0	2·65	0·30	100·0	100·0
Concentrate	10·0	25·48	2·24	96·1	74·1
Cleaner tailing	4·2	1·45	0·84	2·3	11·7
Tailing	85·8	0·05	0·05	1·6	14·2
Rougher concentrate (cal.)	14·2	18·37	1·83	98·4	85·8

This test shows that a reduction in the amount of lime added results in a higher recovery of the gold. It also indicates a tendency for the gold to drop out of the rougher concentrate in the cleaner cell.

Test No. 4

To determine if there was much gold-bearing pyrite in the ore, this test was made using 2·0 pounds soda ash and 0·10 pound cyanide to depress iron pyrite. After removing the first copper concentrate, copper sulphate was added to re-activate any depressed minerals and a second concentrate removed. Potassium ethyl xanthate and pine oil were used as in preceding tests.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads	100·0	2·67	0·33	100·0	100·0
Concentrate No. 1	9·3	26·84	2·32	93·4	65·1
Cleaner tailing	4·5	2·40	1·13	4·0	15·4
Concentrate No. 2	4·7	0·56	0·16	1·0	2·3
Tailing	81·5	0·05	0·07	1·6	17·2

These results show that no great amount of gold is associated with iron pyrite as but 2·3 per cent of the total gold was found in concentrate No. 2. However, the use of cyanide in conjunction with soda ash lowers the recovery of gold. A high-grade concentrate is obtained, but a large amount of gold is found in the cleaner tailing. This product if returned to the head of the rougher flotation cells will result in an increased loss in the tailing.

Test No. 5

In this test the lime and cyanide were reduced to 1 pound and 0·04 pound respectively. Other conditions were unchanged.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·0	2·65	0·30	100·0	100·0
Concentrate	11·0	23·08	1·90	95·7	69·2
Cleaner tailing	3·6	2·00	1·40	2·7	16·7
Tailing	85·4	0·05	0·05	1·6	14·1
Rougher concentrate (cal.)	14·6	17·88	1·78	98·4	85·9

Lime used in quantities of from 1 to 2 pounds per ton results in a higher recovery of gold than that shown in Test No. 1 where 5 pounds lime was used. None of these tests shows as high a recovery as Test No. 2 where 5 pounds soda ash was used. This test also confirms the conclusions in former tests that the gold drops out in the cleaner cell.

Test No. 6

The effect of adding a small quantity, 0·06 pound per ton of sodium sulphide together with 1 pound lime and 0·04 pound cyanide was studied in this test.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·0	2·65	0·26	100·0	100·0
Concentrate	10·2	24·54	1·56	94·5	61·7
Cleaner tailing	5·0	2·07	1·30	3·9	25·2
Tailing	84·8	0·05	0·04	1·6	13·1
Rougher concentrate (cal.)	15·2	17·14	1·47	98·4	86·9

Sodium sulphide apparently is of no great benefit. While a higher recovery of gold is noted, the ratio of concentration and grade of concentrate is lower. This would indicate that a lower grade concentrate will probably yield higher gold recoveries.

Test No. 7

In this test the grinding was slightly coarser than in previous tests, 85·3 per cent −200 mesh. The procedure and reagents were the same as in Test No. 2, 5·0 pounds soda ash added to the grinding mill, 0·08 pound potassium ethyl xanthate and 0·08 pound pine oil for flotation.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·0	2·71	0·35	100·0	100·0
Concentrate	15·3	16·38	1·84	92·3	80·7
Cleaner tailing	6·3	2·18	0·82	5·1	14·8
Tailing	78·4	0·09	0·02	2·6	4·5
Rougher concentrate (cal.)	21·6	12·23	1·54	97·4	95·5

This test resulted in a higher recovery of gold in the rougher concentrate, 95·5 per cent as against 87·3 per cent in Test No. 2. This, doubtless, is due to the greater weight of lower grade concentrate produced. However, due to coarser grinding the recovery of copper is slightly lower. The gold again shows a tendency to drop out in the cleaner cell.

Test No. 8

In this test the ore was ground to the same degree of fineness as in Test No. 7 with 5·0 pounds soda ash per ton; 0·04 pound each of xanthate and Aerofloat No. 25 were added to the flotation cell.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·0	2·77	0·37	100·0	100·0
Concentrate	19·7	13·42	1·52	95·4	80·0
Cleaner tailing	9·1	1·00	0·12	3·3	2·9
Tailing	71·2	0·05	0·09	1·0	17·1
Rougher concentrate (cal.)	28·8	9·50	1·08	98·7	82·9

The addition of Aerofloat to the flotation cell does not result in a high recovery of gold, 82·9 per cent of which is found in the rougher concentrate of lower grade than in the preceding test. The gold does not show the same tendency to drop out in the cleaner tailing.

Test No. 9

In this test the ore was ground with 1 pound lime to pass 75 per cent through 200 mesh and floated with 0·04 pound each of potassium ethyl xanthate and Aerofloat No. 25.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads (cal.)	100·0	2·67	0·31	100·0	100·0
Concentrate	17·0	14·64	1·60	93·0	88·2
Cleaner tailing	7·6	1·48	0·28	4·2	6·9
Tailing	75·4	0·10	0·02	2·8	4·9
Rougher concentrate (cal.)	24·6	10·57	1·19	97·2	95·1

The combination of reagents used in this test, 1 pound lime with Aerofloat and xanthate added in the cells, gives higher gold recoveries than those shown in Test No. 8 where soda ash was used. The coarser grind tends to lower the copper recovery.

Test No. 10

In this test the effect of adding Aerofloat No. 25 to the grinding mill was studied. The ore was ground with 1 pound lime per ton and 0·07 pound Aerofloat to pass 96 per cent −200 mesh; 0·04 pound xanthate was added to the flotation cells. After flotation, the tailing was passed over a Wilfley table to note if any additional values could be saved.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Flotation concentrate	18·4	15·08	1·56	97·8	91·7
Cleaner tailing	7·4	0·64	0·12	1·7	2·8
Table concentrate	1·1	....	2·03	....	0·8
Table tailing	73·1	0·02	0·02	0·5	4·7

Due to the small bulk of sample, no copper assays were made on the table concentrate.

Fine grinding results in a higher recovery of copper. The recovery of gold is very little lower than in Test No. 9 where Aerofloat was added to the flotation cells; 25·8 per cent of the weight of ore milled is floated as a rougher concentrate. Aerofloat again has the effect of preventing the gold from dropping out in the cleaner cell. This is probably due to a lower grade cleaned concentrate.

AMALGAMATION

A series of tests was run on different sizes of material to determine if the gold could be saved by amalgamation. The ore was dry crushed to pass the given meshes and agitated in a closed porcelain mill with mercury. After separating the amalgam, the tailings were assayed.

Screen Analysis of Different Tests

Mesh size of material	Test			
	−20	−48	−65	−100
	%	%	%	%
−20 + 48	37·6	....	....	....
−48 + 65	9·1	16·1	....	....
−65 + 100	7·4	14·0	16·2	....
−100 + 150	5·6	14·2	16·0	15·8
−150 + 200	2·7	4·9	9·0	10·1
−200	37·6	50·8	58·8	74·1

The tailing assays and recoveries secured are as follows:

Test No.	Tailing assay, Au, oz./ton	Per cent of gold amalgamated
−20	0·22	33·4
−48	0·19	42·5
−65	0·18	45·5
−100	0·20	39·4

These results indicate that maximum recoveries by amalgamation will be attained at approximately −65-mesh grinding. However, as this shipment contains a large percentage of iron oxide which produces a heavy slime, it is probable that amalgamation of this material in a mill will not yield so high recoveries as shown above. These oxides will be found to coat the amalgamation plates and sicken the mercury.

CONCLUSIONS

It is apparent from the above tests that high recoveries of copper and gold are possible by flotation. Fine grinding, 85 per cent −200 mesh, will be found to give best results. The use of Aerofloat has the effect of keeping most of the gold in the cleaned concentrate. When this reagent is not used, the cleaner tailing, if returned to the head of the flotation cells, will result in increased losses in the mill tailing. The use of much lime is detrimental. This reagent can safely be reduced from 5 pounds per ton to 1 pound. Unless there is a great increase in the zinc content of the ore, it will not be necessary to use cyanide in the circuit.

To recover the amalgamable gold in the ore, it is recommended that plates be installed between the grinding mill and classifier. If the mill discharge is coarse enough to scour the plates, a screen should be installed, allowing the undersize to pass over the plates before entering the classifier. As stated before, any large amount of iron oxide in the ore will probably interfere with amalgamation.

Report No. 393

CONCENTRATION OF A MOLYBDENITE ORE FROM ALICE ARM, BRITISH COLUMBIA

C. S. Parsons

Shipment. A shipment consisting of 2,700 pounds of molybdenite ore was received January 20, 1931. This shipment was made by the Dalhousie Mining Company, Ltd., Room 601, Bank of Toronto Building, Victoria, B.C., from their property at Alice Arm, B.C.

Characteristics and Analysis. The sample contains molybdenite associated with vein-filling rocks consisting principally of quartz and calcite. The molybdenite occurs very finely disseminated as filling in little veinlets throughout the rock and a portion of it is present as a thin coating on the grains of quartz.

An analysis of the shipment shows it to contain:

	Per cent
MoS_2	1·67
Fe	2·41
Insoluble	83·00

Purpose of Experimental Tests. This shipment is a second lot of ore shipped from this property. The experimental work on the first shipment was inconclusive and a further shipment was required to complete the test work.

The object of the experimental tests is to work out a method to concentrate this ore so as to produce a commercial grade of molybdenite concentrate. In order to obtain a ready market, this concentrate should contain 80 per cent MoS_2. Due to the finely disseminated nature of the molybdenite and its intimate association with the quartz, this ore presents a rather difficult problem.

EXPERIMENTAL TESTS

A series of small-scale laboratory batch flotation tests was first made to determine a suitable combination of reagents. The results of these tests follow and show that the reagent combination which produced the best results, taking into consideration the pyrite removal and grade of concentrate as well as the physical character of the product most suitable for thickening preliminary to regrinding, was about one-tenth pound per ton of both cyanide and Aerofloat No. 25.

Test No.	Products	Weight		Analysis, per cent			Recovery, per cent
		Grms.	Per cent	MoS_2	Fe	SiO_2	
1	Concentrate	82·0	5·2	36·75			88·7
	Tails. +100	406·7	40·9	0·30			5·7
	“ +200	191·7	19·2	0·18			1·6
	“ −200	345·2	34·7	0·25			4·0
	Totals	995·6	100·0				100·0
2	Concentrate	45·7	4·5	41·25			92·5
	Tails. +200	377·5	37·4	0·14			2·6
	“ −200	585·2	58·1	0·17			4·9
	Totals	1,008·4	100·0				100·0
3	Concentrate	167·6	8·1	26·17	5·24	51·9	93·2
	Tailing	1,893·3	91·9	0·17			
	Total	2,060·9	100·0	2·3			
4	Concentrate	200·0	9·8	22·93	3·86	55·6	95·8
	Tailing	1,834·7	90·2	0·11			
	Total	2,034·7	100·0	2·35			
5	Concentrate	127·2	6·2	34·30	4·01	44·4	94·8
	Tailing	1,913·5	93·8	0·15			
	Total	2,040·7	100·0	2·3			
6	Concentrate	159·2	7·9	27·54	4·93	51·7	92·6
	Tailing	1,865·4	92·1	0·19			
	Total	2,024·6	100·0	2·3			

Screen Test.—Showing grinding in above Tests Nos. 3 to 6 inclusive.

Mesh	Weight	Per cent
+ 65	1·0	0·5
− 65+100	12·4	6·2
−100+150	29·7	14·8
−150+200	23·4	11·7
−200	133·5	66·8
Total	200·0	100·0

Reagents used in above tests:

Test No. 1—Ore, 1000 grms., kerosene oil, 0·10 lb./ton, pine oil No. 5, 0·02 lb./ton, added to ball mill and ground with the ore.

Reagents added to flotation cell—pine oil 0·02 lb./ton.

Test No. 2—Ore, 1000 grms., kerosene oil 0·10 lb./ton added to ball mill and ground with the ore.

Reagents to cell—0·02 lb./ton of pine oil No. 5.

Note.—Ore was ground finer in this test than in No. 1.

Test No. 3—Ore, 2,000 grms. Reagents added to ball mill—Aerofloat No. 25, 0·08 lb./ton.
Reagents to cell—pine oil, 0·02 lb./ton.

Test No. 4—Ore, 2,000 grms. Reagents added to ball mill—Aerofloat No. 25, 0·08 lb./ton, cyanide 0·10 lb./ton, soda ash 1·5 lb./ton.
Reagents to cell—pine oil, 0·04 lb./ton.

Test No. 5—Ore, 2,000 grms. Reagents added to ball mill—Aerofloat No. 25, 0·08 lb./ton, cyanide 0·10 lb./ton.
Reagents to cell—pine oil, 0·02 lb./ton.

Test No. 6—Ore, 2,000 grms. Reagents added to ball mill—Aerofloat No. 25, 0·08 lb./ton, cyanide 0·10 lb./ton, lime 1·5 lb./ton.
Reagents added to cell—pine oil, 0·02 lb./ton.

Test No. 7

Following the above tests, a series of batch lots of 2,000 grammes each was run—totalling 10,000 grammes, in order to accumulate enough concentrates to regrind and reconcentrate.

Products	Weight		Analysis, per cent			Recovery, per cent
	Grms.	Per cent	MoS_2	Fe	SiO_2	
Final concentrates	239·8	2·4	75·26	2·36	10·7	76·5
Tailing	9,554·0	93·8	0·18	....	....	7·3
Middling No. 1	340·0	3·3	5·85	....	....	3·5
Middling No. 2	53·7	0·5	32·92	....	....	7·7
Totals	10,187·5	100·0	2·31	....	....	100·0

Procedure. The same grinding to produce the above tailing was used as in Tests Nos. 3 to 6. The concentrate from the first separation was thickened and reground to pass a 200-mesh screen. The reground concentrate was again floated, from which procedure a middling No. 1 was produced, and a concentrate which in turn was again floated producing middling No. 2 and the final concentrate. The reagents used were cyanide 0·20 pound per ton and Aerofloat No. 25, 0·06 pound per ton added to the ball mill, also pine oil, 0·02 pound per ton, added to the flotation cell.

Continuous Test No. 8

Following Test No. 7 a continuous test was made on approximately 1,200 pounds of ore in a small laboratory continuous flotation unit with its own ball mill and classifier. The feed rate was 100 pounds of ore an hour. The reagents used were as follows: Reagents added to ball mill and ground with the ore—Aerofloat No. 25, 0·10 pound per ton, cyanide 0·10 pound per ton. During the last three hours of the test a little sodium silicate was added to the cleaner cell. The addition of this reagent seemed to improve the operation of the cells, and produced a lower tailing by cleaning up the froth in the last few rougher cells which up till then had been carrying considerable gangue.

The results of this continuous test are given as follows:

Feed to mill—1·67 per cent MoS_2

		Per cent
	Sample taken first day's run:	
10.30— 4.30	Gave concentrates cleaned once	66·25 MoS_2
	Gave tailing	0·29 "
	Gave recovery by calculation	83·1
	Samples taken second day's run:	
9.30—12.30	Gave concentrates cleaned once	33·51 MoS_2
	Gave tailing cleaned once	0·27 "
	Gave recovery by calculation	84·2
12.30— 3.15	Gave concentrates cleaned once	62·63 MoS_2
	Gave tailing cleaned once	0·17 "
	Gave recovery by calculation	90·2
3.15— 4.45	Gave concentrate cleaned once and by addition of sodium silicate	51·06 MoS_2
	Gave tailing cleaned once	0·15 "
	Gave recovery by calculation	91·4

A screen test made on the feed to the flotation cell during the period 12·30–4·45, in the second day's run, showed the grinding to be as follows:

Screen Test

Mesh	Weights	Per cent
+ 48	3·0	0·6
− 48+ 65	20·8	4·2
− 65+100	80·0	16·0
−100+150	83·6	16·7
−150+200	59·0	11·8
−200	253·6	50·7
Totals	500·0	100·0

Following the above test, series of re-treatment tests were made on the concentrate accumulated from this test.

Re-treatment Test No. 1

About 500 grammes of the concentrate from Test No. 8 was thickened and reground in a small batch pebble mill. Pebbles were used in preference to steel balls. No reagents were added during grinding but about 0·09 pound per ton of cresylic acid and 1·5 pounds per ton of sodium silicate were added to the flotation cell.

Results:

Product	Weight		MoS_2 per cent	Recovery
	Grms.	Per cent		
Final concentrate	270·0	55·7	85·55	89·9
Middling No. 1	165·6	34·3	2·53	1·6
Middling No. 2	49·0	10·0	44·5	8·5
Totals	484·6	100·0	53·0	100·0

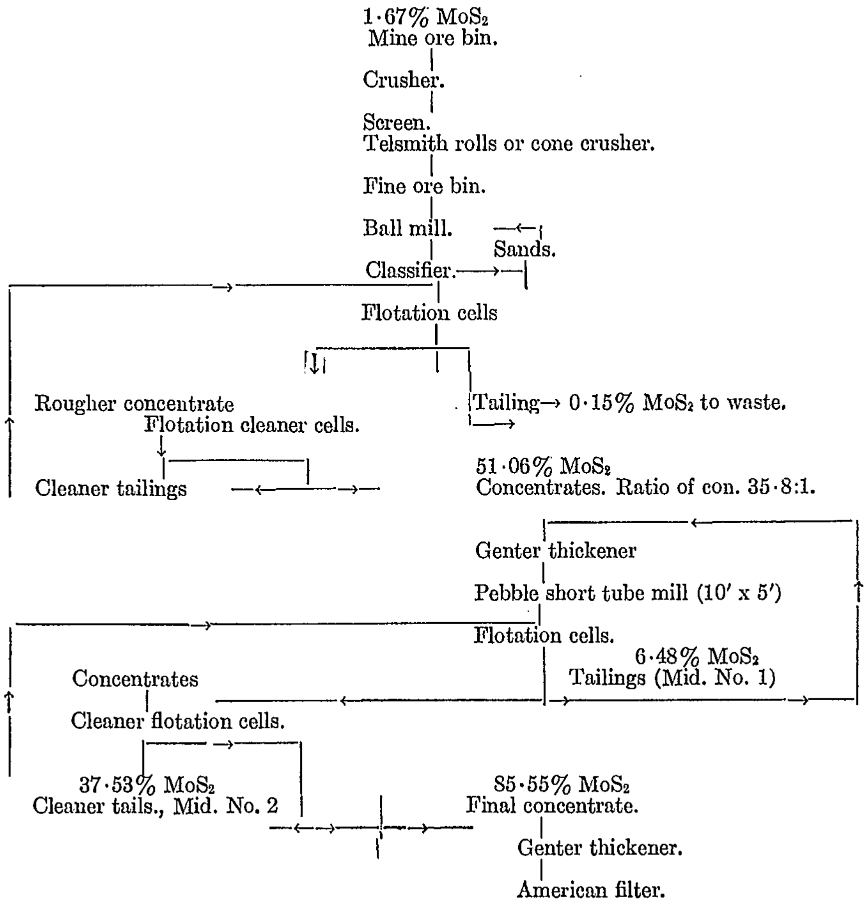
Re-treatment Test No. 2

This test was run under the same conditions as Test No. 1 with the exception that the concentrate was ground a little finer.

Results:

Product	Weight		MoS_2 per cent	Recovery
	Grms.	Per cent		
Final concentrate	285·3	56·6	85·53	89·9
Middling No. 1	175·5	35·0	6·48	4·2
Middling No. 2	42·5	8·4	37·53	5·9
Totals	503·3	100·0	53·90	100·0

Flow-sheet for Dalhousie Molybdenite Ore



SUMMARY AND DISCUSSION OF RESULTS

(1) The tests show that two steps are necessary to produce 80 per cent MoS_2 concentrates. That is, after a preliminary flotation on the ground ore, the concentrate produced from this step will only run about 50 per cent MoS_2 and will contain a large proportion of molybdenite still attached to quartz particles. In order to raise the grade this concentrate must be reground and again concentrated by flotation.

(2) The grinding necessary to produce a low tailing in the first flotation step is relatively coarse, −48 mesh and 50-85 per cent −200 mesh.

(3) The first flotation concentrate must be ground very fine to produce an 80 per cent MoS_2 grade, about 80 per cent −200 mesh.

(4) The recovery should be approximately 90 per cent.

The first concentrate and subsequent middlings obtained from refloating the reground first concentrate were examined under a binocular microscope. This examination showed that the concentrates were being contaminated from two causes, first by middling particles consisting of molybdenite attached to quartz and, second by quartz and calcite particles which had floated with the molybdenite or had become entangled in the froth. The middling product No. 1, obtained from the reflotation of the reground concentrates, was found to consist largely of clean quartz from which the molybdenite had been freed by the regrinding. In the accompanying flow-sheet this product is shown as being returned to the grinding mill after thickening in the Genter. It may be found that this product can be returned to the head of the rougher cells in the first flotation section. The product designated as middling No. 2, obtained from recleaning the reground concentrate, was found to consist largely of true middling particles and free molybdenite which had dropped out in the recleaning. It is possible that this product should be also returned to the grinding mill instead of as shown in the flow-sheet where it is returned to the head of the flotation machines. Only the actual operation of a mill will determine these points.

Recommendations. The accompanying flow-sheet shows an outline of a flow-sheet recommended for a mill to treat this ore.

It will be observed that a pebble mill is recommended for regrinding the flotation concentrate in place of a ball mill. The reason for this is that the molybdenite occurs in the ore as a thin coating on the quartz and must be rubbed off in order to prevent the quartz from floating into the concentrate. For this purpose a long cylindrical pebble mill is recommended.

A Genter thickener is recommended in place of the ordinary Dorr type thickener, because of the quantity of froth which will have to be handled. This froth is the highest grade part of the concentrate and any loss of it would be serious. The Genter can be operated with no overflow and with a low pulp level so that no froth can escape.

Report No. 394

CONCENTRATION OF AN IRON SULPHIDE ORE, ALGOMA DISTRICT, ONTARIO

C. S. Parsons

Shipment. A shipment of 27 bags of ore weighing approximately 2,500 pounds was received at the Ore Dressing and Metallurgical Laboratories on January 24, 1931. The shipment was made by the Algoma Exploration Syndicate, Sault Ste. Marie, Ontario.

Characteristics of Ore. The ore is an iron sulphide containing pyrite and pyrrhotite and also some carbonate of iron.

The analysis of the entire shipment was as follows:

Fe	43·8 per cent	Ni	nil
S	36·0 "	Au	trace
Cu	nil	Ag	"

Purpose of Experimental Work. This sample was said to contain small amounts of copper, nickel, gold, and silver. The purpose of the experimental work was to endeavour to make a copper concentrate containing the gold and silver values and then to make a sulphur concentrate containing 48 to 50 per cent sulphur which could be used in the Freeman pyrite burner. As the sample contained no copper, gold, nor silver, the only tests made were for the purpose of making a sulphur concentrate.

EXPERIMENTAL TESTS

Test No. 1

Products	Weight, per cent	Analysis, S, per cent	Recovery, per cent
Concentrate	37·50	48·82	53·5
Tailing	62·50	25·45	46·5
Totals	100·0	33·6	100·0

Reagents used:

To ball mill—
Lime—5 lb./ton ore.
Aerofloat No. 25—0·06 lb./ton ore.
To cells—
Copper sulphate—0·5 lb./ton ore.
Potassium xanthate—0·25 lb./ton ore.
Pine oil—0·08 lb./ton ore.

Test No. 2

Products	Weight, per cent	Analysis, S, per cent	Recovery, per cent
Concentrate	50·2	46·5	69·0
Tailing	49·8	21·0	31·0
Totals	100·0	33·8	100·0

Reagents used:

To ball mill—
Lime—4·0 lb. /ton ore.
Other reagents as in Test No. 1.

Test No. 3

Products	Weight, per cent	Analysis				Recovery, S, per cent
		S, per cent	Cu, per cent	Ni, per cent	Au, oz./ton	
Concentrate No. 1	4·7		Nil	Nil	0·02	
Concentrate No. 2	53·3	49·15				79·1
Tailings	42·0	16·45				20·9
Totals	100·0	33·1				100·0

Reagents used:

To ball mill—
    Lime—3·0 lb./ton ore.
    Other reagents as in Tests Nos. 1 and 2.

Test No. 4

Products	Weight, per cent	Analysis, S, per cent	Recovery, S, per cent
Concentrate	49·1	49·4	70·2
Tailing	50·9	20·2	29·8
Totals	100·0	34·6	100·0

Reagents used:

To ball mill—
    Soda ash—5·0 lb. / ton ore.
    Coal-tar creosote—0·5 lb./ton ore.
    Pot. xanthate—0·1 lb. / ton ore.

Test No. 5

Products	Weight		Analysis, S, per cent	Recovery, S, per cent
	Grms.	Per cent		
Concentrate	989·0	49·5	50·3	71·0
Tailing	1,011·0	50·5	20·1	29·0
Totals	2,000·0	100·0	35·0	100·0

Test No. 6

Products	Weight		Analysis, S, per cent	Recovery, S, per cent
	Grms.	Per cent		
Concentrate	1,089·0	54·2	51·17	78·7
Tailing	922·0	45·8	16·34	21·3
Totals	2,011·0	100·0	35·2	100·0

Reagents used—Tests Nos. 5 and 6—

To ball mill:
Soda ash—5·0 lb./ton.
Sodium xanthate—0·10 lb./ton.
Coal-tar creosote—0·5 lb./ton.
Pine oil to cell.

Screen test made on feed to cells for Tests Nos. 5 and 6—

Mesh	Weight	Per cent of weight
+48	13·1	0·7
+65	24·4	1·2
+100	183·3	9·1
+150	280·4	13·9
+200	242·5	12·1
−200	1,267·4	63·0
Totals	2,011·1	100·0

Test No. 7

Products	Weight		Analysis, S, per cent	Recovery, S, per cent
	Grms.	Per cent		
Concentrate	1,189·0	59·4	50·39	85·7
Tailing	810·5	40·6	12·32	14·3
Totals	1,999·5	100·0	35·0	100·0

Reagents used:

To rod mill—
Soda ash—5·0 lb./ton.
Sodium xanthate—0·20 lb./ton.
Coal-tar creosote—0·50 lb./ton.
Pine oil to cell—0·01 lb./ton.

Screen tests on feed to cell—

Mesh	Weight	Per cent
+ 65	1·5	0·4
+100	41·7	10·4
+150	86·5	21·6
+200	56·6	14·2
−200	213·7	53·4
Totals	400·0	100·0

SUMMARY

Seven tests in all were made. In Test No. 3 a concentrate was made by flotation which under the conditions maintained would contain any copper, gold, or silver in the ore. This was made to check the head sample, and the result substantiates the head analysis.

A summary of the results of these tests indicate that 55 to 60 tons of sulphide concentrate can be produced from 100 tons of ore and that the concentrate will contain over 50 per cent sulphur.

The tests made also indicate that the coarser the grind within reasonable flotation limits the better the results. The ore was found to be very soft and to grind easily in a ball or rod mill. In Test No. 7 a rod mill was used and produced a more granular product than the ball mill; this will be noted if the screen tests on Tests Nos. 6 and 7 are compared.

In burning pyrite in a hearth-type furnace a granular product is desired.

Analysis of sulphur concentrate from Test No. 7—

	Per cent
Sulphur	50·39
Iron (metallic)	47·00
Copper	nil
Zinc	0·02
Lead	nil
Silica	0·70
Selenium	nil
Tellurium	nil
Arsenic	nil

Report No. 395

EXPERIMENTAL TESTS ON GOLD AND SILVER-BEARING ORES FROM THE MOSS GOLD MINES, LIMITED, KASHABOWIE, THUNDER BAY DISTRICT, ONTARIO.

J. S. Godard

Shipments. A shipment of 265 pounds of ore was received on February 27, 1931, from Moss Gold Mines, Limited. This was designated as shipment No. 1. Two additional shipments designated No. 2 and No. 3 were received on March 24 and April 28 respectively. No. 2 shipment weighed 2,090 pounds and No. 3, 200 pounds. All three shipments were submitted by Mr. Norman Fisher, President of Moss Gold Mines, Limited, 132 St. James St. West, Montreal.

Characteristics of the Ore. Shipment No. 1 was a high-grade gold, low-grade silver ore, the sulphides present being chiefly galena and chalcopyrite. Very little free gold is present as only 15 per cent is amalgamable. The gold is believed to be associated with the chalcopyrite and the silver with the galena. The gangue is a clean hard quartz.

Shipment No. 2 was a low-grade gold ore containing some silver and small quantities of galena, zincblende, chalcopyrite, pyrite and a little tellurium. The gangue is a mixture of quartz and a talcy material.

Shipment No. 3 was a high-grade gold ore containing the same minerals as sample No. 2 in slightly larger quantities. The gangue, however, is a clean hard quartz similar to that in shipment No. 1.

Analyses. Analyses of samples were as follows:

Shipment No.	Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Te, per cent
1	1·255	9·57	0·52	....	0·75	....
2	0·24	1·73	0·13	0·02	0·19	0·02
3	1·07	8·06	0·39	0·75	0·44	0·07

Purpose of Experimental Tests. Tests were desired on the ore of shipment No. 1 in order to secure information that might be of assistance in designing a mill suitable for the recovery of gold and silver. Later a similar series of tests was conducted on ore as represented by shipment No. 2. No tests were made on ore from shipment No. 3, but it was mixed in equal amounts with ore from No. 2 shipment and series of cyanidation tests were made on this composite sample.

Such a composite sample provided a medium grade ore containing a fair percentage of the more refractory sulphides as well as sufficient of the talcy gangue to serve as an indication of what extraction and reagent consumption might be expected in large-scale cyanidation.

EXPERIMENTAL TESTS

A large number of tests on these ore shipments were made, but only the more important tests are included in this report. Considerable difficulty was experienced in obtaining a representative sample of the milling grade of ore. In the case of ore shipment No. 1, a number of tests were conducted and reported on before it was found out that the ore sample was not representative because it ran too high in copper. Shipment No. 2 was then obtained, but unfortunately after a number of tests had been made it was found that it also was not representative. A third shipment was then made for the purpose of making up a composite sample by mixing shipments No. 2 and No. 3. Later a second composite sample was made up.

—	Au, oz./ton	Ag, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Te, per cent
Composite sample No. 1	0·665	4·85	0·23	0·44	0·26	0·04
Composite sample No. 2	0·52	3·75	0·19	....	....	....

Amalgamation and flotation tests were made on shipments No. 1 and No. 2, and also a large number of cyanidation tests. On composite samples No. 1 and No. 2 both small- and large-scale cyanide tests were made. A series of cycle tests was made on composite sample No. 2, to determine the quantity of copper in solution at the end of each cycle and the effect of this dissolved copper on the dissolution of the gold and silver.

AMALGAMATION

Shipment No. 1. A series of tests was made on ore ground to various sizes from 38 per cent −200 mesh to 73 per cent −200 mesh. The recovery varied from 13·5 per cent to 15·1 per cent.

Shipment No. 2. Amalgamation at 54·6 per cent −200 mesh gave 8·3 per cent recovery of gold and 13·3 per cent of the silver, and 92·7 per cent −200 mesh gave 16·6 per cent of gold.

CONCENTRATION

Shipment No. 1. A series of flotation tests was made on this shipment, and in order to save any free gold not floated, the tailings were tabled. These tests showed that flotation would recover between 93 and 94 per cent of the gold and silver with a ratio of concentration of about 9 to 1. The results of a typical test are given below:

Product	Weight, per cent	Assays		Per cent of values	
		Au, oz/ton	Ag, oz/ton	Au	Ag
Flotation concentrate	11·0	11·16	81·00	93·5	93·4
Table concentrate	1·8	0·13	1·18	0·2	0·2
Table tailing + 200	18·6	0·18	1·18	2·5	2·3
Table tailing −200	22·6	0·12	0·71	2·1	1·7
Slimes	46·0	0·05	0·49	1·7	2·4

Shipment No. 2. A series of flotation tests made on this shipment indicated that the talcy nature of the gangue would give considerable trouble. The best results were obtained by grinding to about 80 per cent −200 mesh. This test indicated a ratio of concentration of only about 4 : 1 with a recovery of 93 per cent of the gold and 94 per cent of the silver. The results of Test No. 5 are given below:

Test No. 5

Product	Weight, per cent	Assays		Per cent of values	
		Au, oz./ton	Ag, oz./ton	Au	Ag
Concentrate	24·5	0·98	6·62	93·3	94·3
Tailing +200	5·0	0·06	0·28	1·2	0·8
Tailing −200	70·5	0·02	0·12	5·5	4·9

CYANIDATION

Cyanide Tests on Shipments No. 1 and No. 2.

These tests consisted of both small-batch tests and large-batch tests made on lots of 100 pounds of ore. In the large tests the ore was ground continuously in a small ball mill with cyanide solution. The pulp was collected and cyanided in a batch Pachuca tank holding about 100 pounds of ore.

Small-batch Tests on Shipment No. 1.

The results of a number of tests made indicated that fine grinding was necessary; for example at 77 per cent −200 mesh only 69·8 per cent of the gold was extracted and 53·1 per cent of the silver, while at 94 per cent −200 mesh the extraction was over 90 per cent of the gold. The cyanide consumption was found to be rather high, averaging about 2·4 pounds per ton, but the lime consumption was normal at about 3 pounds. The high copper content of the ore, namely 0·52 per cent, explains the consumption of cyanide.

Small-batch Tests on Shipment No. 2.

This shipment of ore was not so refractory as shipment No. 1. By grinding to 67 per cent −200 mesh about 94 per cent of the gold and 90 per cent of the silver were extracted, and at 88 per cent −200 mesh about 95·5 per cent of the gold and 96 per cent of the silver. The cyanide consumption was only 1 pound per ton as compared to about 2·4 pounds per ton on shipment No. 1. Attention is directed to the fact that shipment No. 2 contained only 0·13 per cent copper and that shipment No. 1 ran 0·52 per cent copper.

Large-scale Cyanide Tests on Shipment No. 1.

These tests were conducted by grinding the ore with cyanide solution in a small rod mill and collecting the ground ore pulp in a Pachuca tank where it was agitated 48 hours.

Results:

Feed to mill	Gold 1·25 oz./ton.	Silver 9·25 oz./ton.
Tailing (average)	Gold 0·065 oz./ton.	Silver 0·83 oz./ton.
Extraction	Gold 94·8 per cent.	Silver 91·0 per cent.

A screen test made on the tailing showed that the ore was ground to 82 per cent −200 mesh.

Attention is directed to the higher extraction obtained in the large-scale test than in the small-batch tests when grinding to the same degree of fineness.

Composite Sample No. 1

Mr. Norman Fisher, President of The Moss Gold Mines, Limited, requested that a few cyanidation tests be made on a composite sample, consisting of equal amounts of ore from shipments Nos 2 and 3. According to this request such a composite sample was made and five cyanidation tests were conducted. Four of these were small-scale tests for the purpose of obtaining the reagents consumption and the fifth a larger scale test using a continuous grinding unit and a Pachuca tank.

The analysis of the composite sample was:

Gold 0·665 oz./ton, silver 4·85 oz./ton.
Lead 0·44 per cent, copper 0·23 per cent.
Zinc 0·26 per cent, and tellurium 0·04 per cent.

SMALL-SCALE CYANIDATION TESTS

Tests Nos. 1 to 4

In this series of tests the ore at −18 mesh was wet ground to varying degrees of fineness in a pebble mill, with lime equivalent to 2·0 pounds per ton, and in cyanide solution KCN 0·075 per cent, CaO 0·09 per cent. The grinding was done at 58 per cent solids. The ground pulp was diluted to 2·5 : 1 with cyanide solution and agitated for 48 hours. The cyanide strength was KCN 0·075 per cent during the agitation period. The tailings were filtered, washed, re-pulped and screened.

Head sample for tests was:

Gold 0·62 oz./ton, silver 4·60 oz./ton.
Lead 0·41 per cent, copper 0·23 per cent, zinc 0·26 per cent.

Results:

Test No.	Mesh	Weight, per cent	Assays		Per cent of values		Average tailing		Extraction, per cent		Consumption, reagents, lb./ton	
			Au, oz./ton	Ag, oz./ton	Au	Ag	Au, oz./ton	Ag, oz./ton	Au	Ag	KCN	CaO
1	+100	6·8	0·09	1·43	14·1	9·8	0·043	0·99	93·2	78·5	1·4	2·2
	+200	30·5	0·07	1·75	49·5	54·0						
	−200	62·7	0·025	0·57	36·4	36·2						
2	+200	22·4	0·05	1·17	42·0	48·4	0·027	0·54	95·7	87·9	1·3	3·1
	−200	77·6	0·02	0·36	58·0	51·6						
3	+200	22·3	0·09	1·45	56·5	48·4	0·036	0·70	94·2	84·9	1·6	2·4
	−200	77·7	0·02	0·48	43·5	51·6						
4	+200	7·8	0·07	1·25	23·0	26·0	0·024	0·37	96·2	92·2	2·0	2·9
	−200	92·2	0·02	0·30	77·0	74·0						

Test No. 5. Cyanidation Test using Continuous Grinding Unit and Pachuca Tank

In this test the ore dry crushed to −18 mesh was fed continuously to a small rod mill in closed circuit with a classifier. Grinding was done at about 60 per cent solids, using cyanide solution KCN 0·07 per cent. Lime equivalent to 4 pounds per ton was mixed with the ore. The classifier overflow was emptied into the Pachuca tank, where it was agitated for 48 hours at 2·3 : 1 dilution. The cyanide strength during agitation was KCN 0·075 per cent.

At the conclusion of the agitation period the tank was sampled in duplicate, and the tailings were filtered, washed, re-pulped and screened on 200 mesh.

Head sample for test:

Au 0·665 oz./ton—Ag 4·85 oz./ton.

Cu....................	0·23 per cent	Pb........................	0·44 per cent
Zn....................	0·26 "	Te........................	0·04 "

Results:

Sample No.	Mesh	Weight, per cent	Assays		Per cent of values		Average tailing		Extraction, per cent	
			Au, oz./ton	Ag, oz./ton	Au	Ag	Au, oz./ton	Ag, oz./ton	Au	Ag
1	+200	17·3	0·05	0·81	41·2	33·9	0·021	0·41	96·8	91·6
	−200	82·7	0·015	0·33	58·8	66·1				
2	+200	15·2	0·05	0·67	30·9	30·0	0·025	0·34	96·3	93·1
	−200	84·8	0·02	0·28	69·1	70·0				

Average tailing—Au 0·023 oz./ton.
Ag 0·38 oz./ton.
Extraction based on average tailing—Au 96·5 per cent.
Ag 92·3 per cent.

Composite Sample No. 2

A second composite sample was made up and the following tests, including a 7-period cycle test, were made.

Analysis of composite sample No. 2:—
Gold 0·52 oz./ton—silver 3·75 oz./ton
copper 0·19 per cent.

CYANIDATION

Test No. 1

Procedure. The ore dry crushed to −18 mesh was ground in cyanide solution, 0·05 per cent KCN, in a rod mill in closed circuit with a classifier. The classifier overflow was emptied into a Pachuca tank where it was agitated for 42 hours. The dilution of the pulp during agitation was 2 : 1.

The ore remaining in the classifier at the completion of grinding was washed out and reground in cyanide solution in a small ball mill then emptied into the tank. The tank was then sampled, Sample No. 1. The assays of this sample were the basis for computing the dissolution of gold and silver in the grinding circuit. After taking No. 1 sample the tank was sampled at 6-hour intervals.

Results of Screening the Various Samples

Sample No.	Mesh	Weight, per cent	Assays, oz./ton		Average assay		Extraction, per cent		Sample taken
			Au	Ag	Au	Ag	Au	Ag	
1	+200 −200	16·1 83·9	0·30 0·09	1·96 2·47	0·124	2·39	76·1	36·3	Completion of grinding.
2	+200 −200	12·6 87·4	0·10 0·04	0·98 0·93	0·048	0·97	90·8	74·2	Six hours after grinding.
3	+200 −200	12·1 87·9	0·05 0·03	0·87 0·62	0·033	0·65	93·6	82·7	12 hours after grinding.
4	+200 −200	11·4 88·6	0·09 0·025	0·72 0·38	0·033	0·42	93·7	88·8	18 hours after grinding.
5	+200 −200	12·2 87·8	0·04 0·02	0·56 0·34	0·023	0·37	95·5	90·2	24 hours after grinding.
6	+200 −200	11·7 88·3	0·04 0·02	0·50 0·27	0·022	0·30	95·8	92·1	30 hours after grinding.
7	+200 −200	11·6 88·4	0·03 0·015	0·51 0·21	0·017	0·25	96·8	93·4	36 hours after grinding.
8	+200 −200	11·7 88·3	0·025 0·010	0·44 0·16	0·012	0·19	97·7	95·0	Tailing repulped 42 hours after grinding.

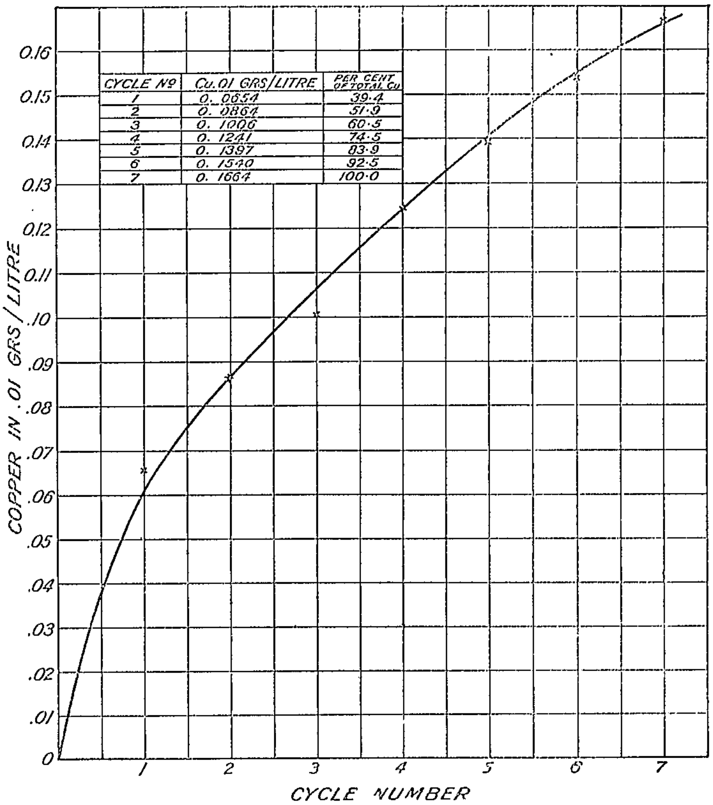
CYCLE TEST

Seven-cycle cyanide tests were made on this sample of ore for the purpose of determining the quantity of copper in solution at the end of each cycle, and the effect of this dissolved copper on dissolution of the gold and silver.

Cycle No. 1 was made in a 100-pound continuous grinding circuit where the ore was ground in cyanide solution and agitated in a Pachuca tank.

The other cycles were made in bottles on ore dry crushed to −150 mesh.

A very small quantity of washings was added to the solution from each cycle. These washings were approximately proportional to the quantity of solution from each cycle.



CYCLE Nº	Cu. 01 GRS/LITRE	PER CENT OF TOTAL Cu
1	0. 0654	39·4
2	0. 0864	51·9
3	0. 1006	60·5
4	0. 1241	74·5
5	0. 1397	83·9
6	0. 1540	92·5
7	0. 1664	100·0

Figure 1. Moss mine, Ontario, No. 2 composite sample, 7-cycle cyanidation test.

The gold in solution at the end of the fourth cycle was precipitated by the addition of zinc dust, and the solution was double filtered before commencing the fifth cycle.

For cycles 2-7, the ore was ground to an average fineness of 21·3 per cent + 200 mesh.

The average copper content of the ore was 0·19 per cent Cu.

Test No. 1

SUMMARY

Cycle No.	Head sample			Tailing, oz./ton		Extraction, per cent		Copper, gr./lit. solution	Reagents consumed, lb./ton	
	Au, oz./ton	Ag, oz./ton	Per cent Cu	Au	Ag	Au	Ag		KCN	CaO
1	0·52	3·75	....	0·012	0·19	97·7	95·0	0·0654	....	....
2	0·545	3·85	....	0·025	1·65	95·4	57·2	0·0864	1·1	2·5
3	0·545	3·82	....	0·036	1·72	93·4	55·0	0·1006	0·8	2·7
4	0·52	3·78	....	0·026	1·92	95·0	49·2	0·1241	0·8	2·6
5	0·55	3·78	....	0·024	2·21	95·6	41·6	0·1397	1·2	2·2
6	0·53	3·79	....	0·033	2·19	93·8	42·2	0·1540	1·2	2·1
7	0·520	3·76	....	0·034	2·13	93·6	43·3	0·1664	1·0	2·3

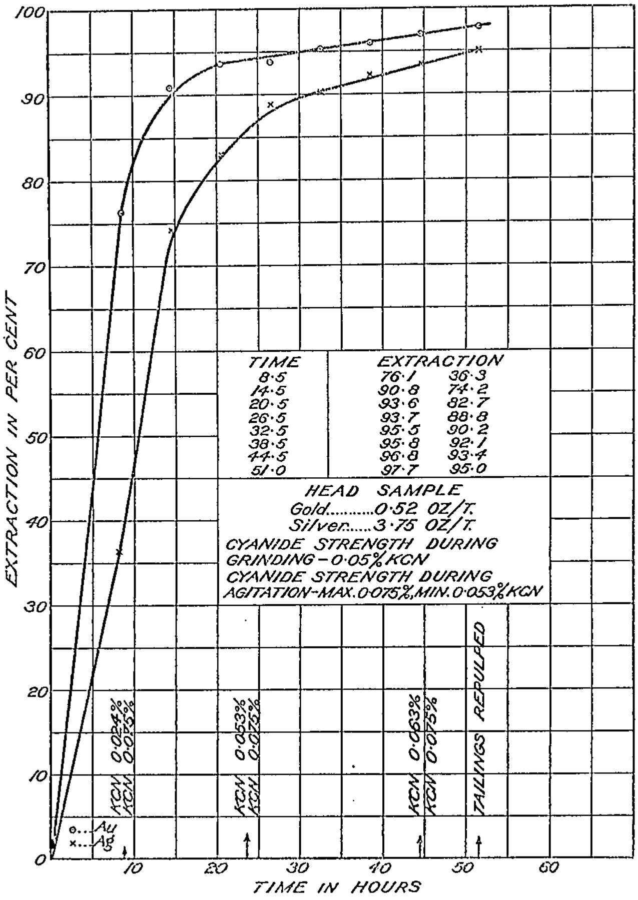


Figure 2. Moss mine, No. 2 composite sample.

From the above summary it will be seen that insofar as the gold extraction is concerned there is but little difference between cycles No. 2 and No. 7—in spite of an increase in copper content from 0·0654 gramme per litre at the end of cycle No. 1 to 0·1664 gramme per litre at the end of cycle No. 7.

There is a decided falling off in the extraction of silver after No. 1 cycle. This is attributed not so much to the fouling of the solution as to the fact that the ore is less finely ground for cycles Nos. 2-7 than for cycle No. 1. Also that as the silver is more refractory than the gold in this ore, the grinding in cyanide solution and the superior agitation and aeration of the Pachuca tank play an important part in the dissolution of this metal.

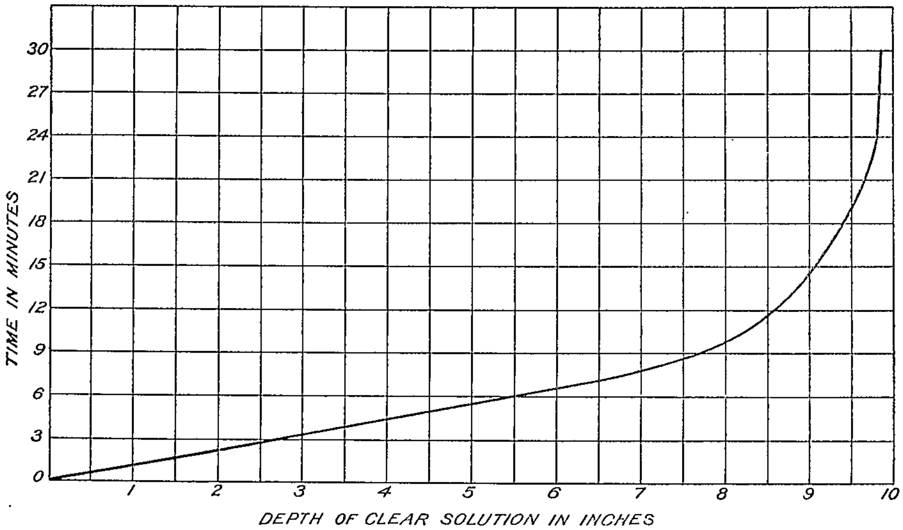


Figure 3. Graph of settling test on No. 2 composite sample from Moss mine.

SETTLING TESTS

A series of settling rate tests was made on the composite sample. The results are plotted in the accompanying diagram for the purpose of estimating the size, type, and number of thickeners required in the mill.

GENERAL CONCLUSIONS

From the test work done, it is believed that cyanidation is the best method of treatment for this ore. The cyanide consumption is moderate and will depend largely on the relative quantities of the various grades of ore that will provide the mill feed.

Report No. 396

PRELIMINARY CONCENTRATION TESTS ON ORES FROM CHIBOUGAMAU DISTRICT, QUEBEC, FOR METALLUM, LTD., MONTREAL

Alex. K. Anderson

Shipments. Seven jute bags containing 12 different samples were received at the Ore Dressing and Metallurgical Laboratories on June 2, 1931, consigned by Dr. Eng. F. Esser, President of Metallum, Ltd., 460 St. John St., Montreal, Que. These samples were taken in the Chibougamau district, Quebec, from the following properties:—

Bag Nos. 1 and 2	Chibougamau Prospectors, Ltd., Camp 1.
Bag No. 4	Chibougamau McKenzie Mines, Ltd., Cedar bay.
Bag No. 6, Sample 23	Chibougamau Prospectors, Ltd., and Chibougamau McKenzie Mines, Ltd., Cedar bay.
Bag No. 6, Samples 10 and 11	Chibougamau Prospectors, Ltd., Cache bay.
Bag No. 7, Sample 29	Chibougamau Prospectors, Ltd., Cache bay.
Bag No. 3, Samples 20 and 21	Chibougamau Prospectors, Ltd., Merrill island, Camp 11.
Bag No. 5, Sample 24	Chibougamau Prospectors, Ltd., near Cedar bay.
Bag No. 7—	
Sample 22	Chibougamau Prospectors, Ltd., near Cedar bay.
Sample 30	Chibougamau Prospectors, Ltd., near Cedar bay.
Sample 32	Chibougamau McKenzie Mines, Ltd., Cedar bay.

Characteristics of the Ore. The greater part of the shipment was made up of copper-iron pyrite ores in siliceous gangues. Three samples contained magnetite while one was a high-grade white zinc sulphide.

Purpose of Experimental Tests. The shippers desired that concentration tests be made to determine if the various samples were amenable to concentration, to recover the valuable constituents in marketable forms.

Sampling and Analysis. Each lot was handled separately, the ore being crushed and ground to succeeding fine sizes with intervening cuts through a Jones riffle sampler until a representative portion −100 mesh was secured for analysis. These head samples are recorded under the individual tests.

EXPERIMENTAL TESTS

Concentration of the copper-iron pyrite samples was made by flotation. Those containing gold also were tested to determine what percentage could be recovered by amalgamation. The iron ore sample was concentrated magnetically.

Bags Nos. 1 and 2

Both bags were combined, sampled, and tested for free gold and concentration of sulphides, making a copper concentrate and an iron pyrite concentrate. A separation of pyrrhotite as a cleaner tailing from the rougher iron pyrite concentrate was attempted.

The lot had the following analysis:

Cu..................	1·48 per cent	Insol.................	63·5 per cent
Fe..................	13·3 "	Au....................	0·06 oz./ton
S....................	7·7 "	Ag....................	0·36 "

AMALGAMATION

Test No. 1

A 1,000-gramme representative portion of the ore was ground to pass 80 per cent −200 mesh and amalgamated with mercury.

—	Au, oz./ton	Ag, oz./ton
Heads..	0·06	0·36
Amalgamation tailing..	0·03	0·31
Recovery..	50 per cent	13·9 per cent

SELECTIVE FLOTATION

Test No. 2

A 1,000-gramme sample was ground to pass 80 per cent −200 mesh, 70 per cent solids with 3 pounds soda ash, 0·10 pound cyanide. A copper concentrate was removed after adding 0·10 pound amyl xanthate and 0·09 pound pine oil per ton.

After the addition of 1 pound copper sulphate, 0·20 pound amyl xanthate, and 0·09 pound pine oil per ton, an iron sulphide concentrate was removed. This was cleaned once to drop out pyrrhotite.

Product	Psr cent of weight	Assay						Per cent of values					
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Fe, per cent	S, per cent	Insol., per cent	Cu	Au	Ag	Fe	S	Insol.
Heads (calculated).......	100·0	1·38	0·06	0·36	14·1	7·7	62·8	100·0	100·0	100·0	100·0	100·0	100·0
Copper concentrate.......	11·1	11·64	0·30	2·80	38·3	34·4	10·2	93·8	80·5	85·4	30·2	49·6	1·8
Iron pyrite concentrate...	5·7	1·08	0·06	0·44	45·4	40·9	6·8	4·5	8·3	6·9	18·4	30·2	0·6
Iron pyrite cleaner tail...	3·4	0·24	0·02	0·12	28·2	14·7	41·7	0·6	1·6	1·1	6·8	6·5	2·2
Tailing..................	79·8	0·02	0·005	0·03	7·9	1·3	76·3	1·1	9·6	6·6	44·6	13·7	95·4

This test indicates that from each 100 tons of ore milled, there are recovered 11·1 tons of concentrate assaying 11·64 per cent copper and $6.00 in gold, and 5·7 tons of iron pyrite concentrate containing 40·9 per cent sulphur.

Test No. 3

In this test a copper concentrate was first removed and by the addition of 0·06 pound hardwood creosote to the reagents used in Test No. 1, an iron sulphide concentrate was obtained. Other conditions were smiliar to Test No. 1.

Product	Per cent of weight	Assay						Per cent of values					
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Fe, per cent	S, per cent	Insol., per cent	Cu	Au	Ag	Fe	S	Insol.
Heads (calculated)........	100·0	1·43	0·04	0·36	14·0	7·6	64·0	100·0	100·0	100·0	100·0	100·0	100·0
Copper concentrate........	13·8	9·43	0·26	2·36	40·2	38·0	6·7	91·3	85·5	91·6	39·7	68·8	1·5
Sulphur concentrate......	7·1	1·07	0·03	0·20	37·3	23·8	23·7	5·3	5·1	4·0	19·0	22·1	2·6
Tailing..................	79·1	0·06	0·005	0·02	7·3	7·3	77·6	3·4	9·4	4·4	41·3	9·1	95·9

This test indicates a recovery of 91·3 per cent copper, 85·5 per cent gold in a concentrate assaying 9·43 per cent copper, $5.20 gold. The sulphur concentrate is low grade and would require cleaning to bring it to a commercial grade.

Bag No. 4

This lot was also a copper-iron pyrite ore, very heavy with iron sulphides. The tests were similar to those made on Bags Nos. 1 and 2.

Head Sample—Analysis:

Cu..................	4·03 per cent	Insoluble............	41·4 per cent
Fe..................	23·4 "	Au..................	1·02 oz./ton
S..................	19·0 "	Ag..................	0·66 "

AMALGAMATION

Test No. 1

—	Au, oz./ton	Ag, oz./ton
Heads..	1·02	0·66
Amalgamation tailing..............................	0·33	0·45
Recovery..	67·6 per cent	20·6 per cent

FLOTATION

Test No. 2

A 1,000-gramme sample was ground to pass 80 per cent through 200 mesh, 70 per cent solids with 10 pounds soda ash and 0·10 pound cyanide per ton. A copper concentrate was removed by the addition of 0·10 pound Minerals Separation 'F' compound and 0·09 pound pine oil per ton.

One pound copper sulphate, 0·20 pound amyl xanthate and 0·06 pound pine oil per ton were then added and an iron pyrite concentrate removed.

A third concentrate called pyrrhotite was then taken after acidifying with 5 pounds sulphuric acid and floating with 0·10 pound amyl xanthate and 0·06 pound pine oil per ton.

Product	Per cent of weight	Assay						Per cent of values					
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Fe, per cent	S, per cent	Insol., per cent	Cu	Au	Ag	Fe	S	Insol.
Heads (calculated).......	100·0	4·01	0·94	0·61	23·5	18·7	41·8	100·0	100·0	100·0	100·0	100·0	100·0
Copper concentrate.......	14·7	26·22	4·00	2·88	30·5	32·2	4·1	96·2	76·3	69·7	19·0	25·3	1·4
Iron pyrite concentrate...	25·9	0·36	0·63	0·45	44·8	50·7	2·8	2·3	17·3	19·2	49·3	70·1	1·7
Pyrrhotite concentrate...	2·9	0·76	0·84	0·76	24·8	18·8	40·7	0·5	2·7	3·6	3·1	2·9	2·9
Tailing..................	56·5	0·07	0·06	0·08	11·9	0·6	69·5	1·0	3·7	7·5	28·6	1·7	94·6

This test shows that 96·2 per cent of the copper and 76·3 per cent of the gold can be recovered in a concentrate assaying 26·22 per cent copper, $98.00 gold per ton. Also from each 100 tons of ore milled, 25·9 per cent of the weight can be recovered as a sulphide concentrate containing 50·7 per cent sulphur.

Bag No. 3—Samples Nos. 20 and 21

These samples were combined and tests run similar to those made on Bag No. 4.

Head Sample:

Cu..........	2·68 per cent	Fe...........	14·2 per cent	Au...........	0·04 oz./ton
Insoluble.....	58·8 "	S.............	7·9 "	Ag...........	1·06 "

AMALGAMATION

Test No. 1

A 1,000-gramme sample ground to pass 80 per cent −200 mesh, and amalgamated.

	Au	Ag
Heads..	0·04	1·06
Amalgamation tailing.......................................	0·04	1·02
Recovery...	nil	nil

FLOTATION

Test No. 2

Reagents and procedure as in Test No. 2, Bag No. 4.

Product	Per cent of weight	Assay						Per cent of values					
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Fe, per cent	Insol., per cent	S, per cent	Cu	Au	Ag	Fe	S	Insol.
Heads (calculated).......	100·0	2·78	0·056	1·00	14·0	7·9	58·9	100·0	100·0	100·0	100·0	100·0	100·0
Copper concentrate.......	16·9	16·04	0·22	5·30	37·6	30·9	8·5	97·3	66·2	89·2	45·2	66·5	2·4
Iron pyrite concentrate...	5·8	0·52	0·14	1·25	38·7	40·4	12·3	1·1	14·5	7·2	16·0	29·8	1·2
Pyrrhotite concentrate...	1·0	0·63	0·32	1·28	19·4	10·9	47·1	0·2	5·7	1·3	1·4	1·4	0·8
Tailing..................	76·3	0·05	0·01	0·03	6·9	0·2	73·8	1·4	13·6	2·3	37·4	2·3	95·6

This test shows that 97·3 per cent of the copper is recovered in a concentrate which assays 16·04 per cent copper and is 16·9 per cent of the total weight of ore milled; 5·8 per cent of the weight of feed is recovered as a rougher iron pyrite concentrate assaying 40·4 per cent sulphur. The grade of this can be raised by cleaning.

Bag No. 5—Sample No. 24

This sample was tested for copper and iron pyrite recovery similarly to the preceding tests.

Head Sample:

Cu..........	4·06 per cent	Fe..........	15·7 per cent	S...........	10·1 per cent
Insoluble....	60·5 "	Au..........	0·02 oz./ton	Ag..........	2·14 oz./ton

An amalgamation test showed that none of the gold and silver is recovered by amalgamation.

Test No. 2

A copper concentrate and one carrying iron pyrite were made as in preceding tests.

Product	Per cent of weight	Assay						Per cent of values					
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Fe, psr cent	S, per cent	Insol., per cent	Cu	Au	Ag	Fe	S	Insol.
Heads (calculated).......	100·0	3·75	0·019	2·19	15·9	10·0	60·2	100·0	100·0	100·0	100·0	100·0	100·0
Copper concentrate.......	20·0	17·86	0·06	9·48	35·9	31·9	5·6	95·4	61·6	86·5	45·1	63·5	1·9
Iron pyrite concentrate...	8·1	1·45	0·04	2·08	49·4	36·7	9·0	3·1	16·6	7·7	25·1	29·6	1·2
Pyrrhotite concentrate...	1·8	1·15	0·04	1·60	38·0	25·7	25·3	0·5	3·7	1·3	4·3	4·6	0·8
Tailing..................	70·1	0·05	0·005	0·14	5·8	0·3	82·6	1·0	18·1	4·5	25·5	2·3	96·1

Bag No. 6—Sample No. 23

This sample contained sulphides of copper and iron, and also iron carbonate—siderite. The tests were made in the same manner as the preceding flotation tests. After flotation, the tailing was passed over a Wilfley table to determine if an iron concentrate could be recovered.

Head Sample:

Cu..........	1·38 per cent	Fe..........	39·3 per cent	S............	3·4 per cent
P...........	0·011 "	Au..........	0·01 oz./ton	Ag..........	0·32 oz./ton
Insoluble....	15·2 per cent				

Product	Per cent of weight	Assay						Per cent of values					
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Fe, per cent	S, per cent	Insol., per cent	Cu	Au	Ag	Fe	S	Insol.
Heads (calculated).......	100·0	1·30	0·01	0·32	38·9	3·3	16·5	100·0	100·0	100·0	100·0	100·0	100·0
Copper concentrate.......	4·1	27·16	0·16	5·28	33·8	30·4	2·0	85·6	65·6	67·7	3·6	37·4	0·5
Iron pyrite concentrate...	4·0	1·29	......	......	46·4	45·0	2·2	4·6	......	......	4·8	53·9	0·5
Table concentrate........	1·6	0·03	......	......	42·9	1·2	11·9	0·1	......	......	1·8	0·6	11·5
Table tailing............	90·3	0·15	......	......	38·7	0·3	16·0	10·3	......	......	89·8	8·1	87·5

This test indicates that a separation of the copper and iron sulphides can be made by flotation. The results of the table concentration with an object of producing an iron concentrate are inconclusive. There is an increase in the iron content over that of the table tailing, and also with an increase of the sulphur content.

Bag No. 7—Sample No. 30

This sample was a very heavy iron sulphide. Tests were made to observe what grade of iron pyrite concentrate could be secured by flotation.

Head Sample:

Cu	nil	Insoluble	11·4 per cent
Fe	34·6 per cent	Au	trace
S	37·9 "	Ag	0·23 oz./ton

Reagents, lb./ton:

Soda ash	10·0	Pine oil	0·18
Cyanide	0·10	Copper sulphate	1·0
"F" compound	0·10	Sulphuric acid	10·0

Product	Per cent weight	Assay						Per cent of values	
		Cu %	Au oz./ton	Ag oz./ton	Fe %	S %	Insol. %	Fe	S
Heads (calculated)	100·0	nil	trace	0·23	34·9	38·0	37·9	100·0	100·0
Iron pyrite concentrate	74·4	nil	0·01	0·16	45·4	50·5	2·3	96·7	98·9
Pyrrhotite concentrate	1·8	nil	......	......	24·5	14·3	21·5	1·3	0·7
Tailing	23·8	nil	......	......	2·9	0·7	......	2·0	0·4

This test shows that 74·4 per cent of the weight of feed is recovered as a pyrite concentrate assaying 50·5 per cent sulphur.

Bag No. 7—Sample No. 32

This sample was a high-grade white zinc sulphide. The ore was ground with 10 pounds soda ash and 0·10 pound cyanide to depress iron pyrite. The zinc was then activated with 1 pound copper sulphate and floated with 0·10 pound potassium ethyl xanthate and 0·09 pound pine oil per ton. The pulp was then acidified with 10 pounds sulphuric acid and an iron pyrite concentrate removed by the addition of 0·20 pound amyl xanthate and 0·09 pound pine oil.

Head Sample:

Cu....0·27 per cent	Zn....28·40 per cent	S....16·7 per cent	
Fe....7·5 per cent	Au....0·02 oz./ton	Ag....0·36 oz./ton	Insol....40·8 per cent

Product	Per cent of weight	Assay							Per cent of values				
		Cu, per cent	Zn, per cent	Au, oz./ton	Ag, oz./ton	Fe, per cent	S, per cent	Insol., per cent	Cu	Zn	Au	Ag	S
Heads (calculated)	100·0	0·28	28·00	0·02	0·36	7·5	16·7	40·8	100·0	100·0	100·0	100·0	100·0
Zinc concentrate	52·3	0·49	50·40	0·03	0·52	6·7	29·7	9·8	91·3	94·1	78·5	75·5	93·0
Pyrite concentrate	5·1	0·23	6·24	......	......	19·8	18·3	41·0	4·1	1·1	......	......	0·6
Tailing	42·6	0·03	3·16	......	......	7·1	......	76·6	4·6	4·8	......	......	......

This test indicates a recovery of 94 per cent of the zinc in a concentrate containing 50·4 per cent zinc.

Bags Nos. 6 and 7—Samples Nos. 10, 11, and 29

This was a sample containing magnetite. It was ground to pass 35 mesh and passed through an Ullrich magnetic separator operated at 110 volts, ½ amp.

Head Sample:

Cu....0·03 per cent Fe......33·7 per cent S....0·19 per cent
P.....0·01 per cent Insol....35·1 per cent

Product	Per cent of weight	Assay				Per cent of values			
		Fe %	S %	P %	Insol %	Fe	S	P	Insol.
Heads (calculated)........	100·0	39·4	0·20	0·009	49·9	100·0	100·0	100·0	100·0
Concentrate...............	90·3	41·2	0·15	0·008	49·0	94·4	68·9	82·3	88·6
Tailing...................	9·7	22·9	0·63	0·016	58·5	5·6	31·1	17·7	11·4

This test indicates the possibility of concentrating this material, reducing the amount of sulphur and phosphorus in the concentrate; with a properly sized feed and the right magnetic intensity better results would doubtless be secured.

This series of tests shows that the ores of the Chibougamau district are amenable to concentration by selective flotation.

Report No. 397

EXPERIMENTAL TESTS ON OLD CYANIDE TAILINGS FROM THE WRIGHT-HARGREAVES MINES, LIMITED, KIRKLAND LAKE, ONTARIO

J. S. Godard

Shipment. A shipment of about 2,500 pounds of old cyanide tailings was received May 28, 1931, from the Wright-Hargreaves Mines, Ltd., Kirkland Lake, Ontario.

Characteristics and Analyses of the Tailings. The tailings from the cyanide mill of the Wright-Hargreaves Mines, Ltd., have been discharged for a number of years into Kirkland lake. During this period a considerable tonnage of these tailings has accumulated and the sample submitted is said to be fairly representative of this material.

Iron pyrites is the chief and predominating sulphide, and, although very small quantities of other sulphides are present, indications are that the iron pyrites carries the bulk of the gold. No visible gold has been noted and it is believed that at least a large part of this gold is very intimately held in this pyrite.

The average assay of a number of samples showed gold 0·093 ounce per ton. Screen analyses indicate that about 6 per cent of this material was +100 mesh and about 62 per cent −200 mesh.

Purpose of Experimental Tests. The present experimental test work was done at the request of Mr. H. V. Wallace, Research Engineer for the Wright-Hargreaves Mines, Ltd., for the purpose of ascertaining if the gold-bearing sulphides could be concentrated by flotation and the flotation concentrates recyanided for the recovery of the gold.

Mr. Wallace was present during part of the experimental work.

Experimental Tests

SMALL-SCALE FLOTATION TESTS

Test No. 1

The rougher concentrates from 6,000 grammes of cyanide tailings were cleaned and recleaned. The middlings were combined.

Results:

Product	Weight, per cent	Assays, Au oz./ton	Per cent of value, Au
Heads	100·0	0·093	100·0
Concentrates	2·2	2·00	47·3
Middlings	6·3	0·08	5·4
Tailings	91·5	0·048	47·3

A sample of the flotation tailing was screened on 200 mesh.

Results:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·048	100·0
+200	36·6	0·08	60·6
−200	63·4	0·03	39·4

Test No. 2

The rough concentrates from 4,800 grammes of cyanide tailings were cleaned once. A sample of the flotation tailing was tabled.

Results—Flotation:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·099	100·0
Concentrates	2·2	2·06	45·8
Middlings	3·3	0·14	4·6
Tailings	94·5	0·052	49·6

A sample of the flotation tailing was screened on 200 mesh.

Results:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·052	100·0
+200	43·6	0·08	67·4
−200	56·4	0·03	32·6

Tabling a sample of the flotation tailing.

Results:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·057	100·0
Table concentrates	2·4	0·26	10·9
Table middlings +100	2·4	1·05	6·3
Table middlings −100	16·1	0·09	25·5
Table tailing +100	2·5	0·13	5·8
Table tailing +200	13·8	0·055	13·4
Table tailing −200	35·0	0·03	18·5
Slimes	27·8	0·04	19·6

Test No. 3

In this test the cyanide tailings were reground before flotation. The rougher concentrate was not cleaned.

Results:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·094	100·0
Concentrates	8·3	0·66	58·4
Tailings +200	28·8	0·07	21·5
Tailings −200	62·9	0·03	20·1

Test No. 4

In this test the cyanide tailings were reground before flotation. The time of grinding was three times as long as in Test No. 3. The rougher concentrate was not cleaned.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·094	100·0
Concentrates	10·5	0·58	64·6
Tailings +200	18·9	0·065	13·0
Tailings −200	70·6	0·03	22·4

Test No. 5

In this test the cyanide tailings were floated, No. 1 concentrate, the flotation tailing deslimed and the sands reground and floated, No. 2 concentrate. The flotation tailing from the second float was screened on 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·095	100·0
Concentrate No. 1	7·5	0·64	50·5
Concentrate No. 2	2·5	0·42	11·1
Classifier overflow	26·7	0·04	11·3
Tailing No. 2 +200	22·5	0·06	14·2
Tailing No. 2 −200	40·8	0·03	12·9

Recovery in concentrates, 61·6 per cent.

CONTINUOUS TEST

Test No. 6

This test was made in a continuous grinding and flotation unit. The feed rate was about 90 pounds per hour, and the flow-sheet was as follows:—

The cyanide tailing was fed into a contact tank, then pumped to a classifier. The classifier overflow passed to No. 2 cell of a six-cell Denver machine where a rougher concentrate was taken from cells Nos 2 to 6. This rougher concentrate was cleaned in No. 1 cell and the middlings returned to No. 2 cell. The flotation tailing was tabled on a quarter-sized Wilfley table.

The classifier return was reground in a rod mill in closed circuit with the classifier. The duration of the test was 8¼ hours.

Samples were taken of the feed, flotation concentrate, flotation tailing, table concentrate, table middling, and table tailing. A number of these samples were screened.

Results:

The head sample No. 1 was screened on 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·092	100·0
+200	29·5	0·12	38·6
−200	70·5	0·08	61·4

The head sample No. 2 was screened on 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·088	100·0
+200	27·9	0·11	34·7
−200	72·1	0·08	65·3

Average of two head samples—Au 0·090 oz./ton.

2. Assays of flotation concentrate samples:

No. 1	Au 2·00 oz./ton.
No. 2	Au 1·56 "
No. 3	Au 2·16 "

3. The flotation tailing samples were screened on 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·032	100·0
+200	11·4	0·05	17·7
−200	88·6	0·03	82·3

4. The table tailing was screened on 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·028	100·0
+200	9·9	0·05	18·2
−200	90·1	0·025	81·8

Flotation of a Sample of the Table Concentrates

A sample of the table concentrates was given a short regrind and then floated for the purpose of ascertaining if the sulphides reporting in the table concentrates were amenable to flotation under more favourable conditions.

Results:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·11	10·0
Flotation concentrate	3·7	1·52	51·1
Flotation tailing, +200	28·0	0·07	17·8
Flotation tailing, −200	68·3	0·05	31·1

It is apparent that about 50 per cent of the gold in the table concentrate may be recovered by flotation. This flotation concentrate consisted of a small amount of fairly bright pyrite that had escaped the first flotation and a considerable quantity of pyrite that had become oxidized and would only float under very favourable conditions and with the addition of sulphidizing reagents. This flotation concentrate, however, represents only a very small proportion of the sulphides in the original cyanide tailings.

Recoveries by Flotation

The calculated recoveries, P, by flotation and the ratios of concentration, R, are as follows:—

Head sample—Au 0·090 oz./ton.

No.	Concentrates, Au, oz./ton	Tailing, Au, oz./ton	P	R
1	2·00	0·032	65·6	34:1
2	1·56	0·028	70·1	25:1
3	2·16	0·032	65·4	37:1

Test No. 7

This test was made in the same unit as Test No. 6. There was, however, a change in the flow-sheet, which for this test was as follows:—

The cyanide tailings were fed to a contact tank, then to No. 2 cell of the Denver machine. A rougher concentrate was taken from cells Nos. 2 to 6, which was cleaned in No. 1 cell, the middlings returning to No. 2 cell. The flotation tailings were classified and the classifier overflow passed to the Wilfley table. The classifier return was reground in the rod mill and the rod mill discharge passed to the contact tank.

As the classifier used in Tests Nos. 6 and 7 was not particularly well adapted for such fine material as the cyanide tailings some alterations were made after the completion of Test No. 6 for the purpose of increasing the quantity of returned material. These alterations consisted of laying a tapered false bottom extending from the upper end for a distance of 2½ feet. The openings in the spiral were closed, and the number of revolutions per minute doubled. The first-named alteration appeared to have a beneficial effect, the latter two did not. After three hours operation a change was made to the original number of revolutions per minute. The increase in the speed of rotation had the effect of an agitator with the result that both fine and coarse material overflowed the discharge end leaving practically nothing to be returned for regrinding.

Samples were taken of the following products, head sample, flotation concentrate, flotation tailing, classifier overflow, table concentrate, table middling, and table tailing.

Results:

1. The head sample was screened on 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·087	100·0
+200	33·5	0·10	38·7
−200	66·5	0·08	61·3

2. The flotation concentrate assayed—Au 2·00 oz./ton.
3. A sample of the flotation tailing was screened on 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·041	100·0
+200	26·7	0·07	46·0
−200	73·3	0·03	54·0

4. A sample of the classifier overflow was screened on 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·031	100·0
+200	3·4	0·06	6·5
−200	96·6	0·03	93·5

5. A sample of the table concentrate was screened on 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·117	100·0
+200	59·6	0·06	30·7
−200	40·4	0·20	69·3

6. The table middling assayed, Au 0·10 oz./ton.
7. The table tailing assayed, Au 0·025 oz./ton.

Summary of Results:

Head sample	Au 0·087 oz./ton.
Flotation concentrate	Au 2·00 "
Flotation tailing, classifier overflow	Au 0·031 "
Recovery	65·5 per cent.
Ratio of concentration	35 : 1

CYANIDATION OF THE FLOTATION CONCENTRATES

Samples of the flotation concentrates from Tests Nos. 7 and 8 were cyanided for the purpose of extracting the gold.

Tests Nos. 8 amd 9

In these two tests the concentrates were ground in cyanide solution, 0·20 per cent KCN, in a ball mill, then diluted 3 : 1 and agitated for 48 hours in cyanide solution, 0·17 per cent KCN. The pulp was then filtered and reagitated for 17 hours in the original solution. The tailings were repulped. Tests were made on approximately 1000-gramme samples and agitation was done in large water bottles.

Results:

Test No.	Heads, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton concentrates		
				KCN	CaO	NH$_4$OH
8	1·64	0·80	51·2	5·2	20·1	....
9	1·64	0·88	46·4	5·1	21·7	5·0

Tests Nos. 10 and 11

The concentrates were ground in a pebble mill in 1 : 1 pulp; KCN 0·10 per cent, then diluted to 3 : 1 and agitated for 48 hours in pails fitted with a stirring device rotating at 100 r.p.m. Cyanide strength during this agitation was 0·10 per cent KCN. The pulps were then filtered and the concentrates reagitated for 24 hours in 3 : 1 pulp with a new solution, 0·125 per cent KCN.

Results:

Test No.	Heads, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton concentrates		
				KCN	CaO	NH$_4$OH
10	1·84	0·98	46·7	4·4	35·0	....
11	1·84	1·01	45·1	4·6	33·8	10·0

Test No. 12

In this test the concentrates were cyanided for 68 hours in 4 : 1 pulp, KCN 0·25 per cent.

Results:

Test No.	Heads, Au, oz./ton	Tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton concentrates	
				KCN	CaO
12	1·82	0·94	48·4	5·55	20·6

Sample No.	Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
2	Heads	100·0	0·028	100·0
	+200	8·0	0·06	17·3
	−200	92·0	0·025	82·7
3	Heads	100·0	0·032	100·0
	+200	8·0	0·05	12·7
	−200	29·0	0·03	87·3

Tabling the Flotation Tailing

Results:

1. Head sample to table screened on 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·033	100·0
+200	12·9	0·05	19·9
−200	87·1	0·03	80·1

2. Assay of table concentrate: Au 0·10 oz./ton.
3. Table middling screened on 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·074	100·0
+200	57·5	0·07	54·2
−200	42·5	0·08	45·8

Test No. 13

A sample of the concentrates was ground in a pebble mill in cyanide solution 0·15 per cent. Density of pulp during grinding was 57 per cent solids. The pulp was then diluted to 2:1 and agitated 48 hours in cyanide solution, 0·10 per cent KCN. After this period of agitation the pulp was filtered and recyanided in a new solution, 0·20 per cent KCN, for 3½ hours in a 3:1 pulp using a Greenawalt flotation machine as an agitator.

Results:

Head sample	Au 1·82 oz./ton
Tailing	Au 0·92 "
Extraction	49·5 per cent

Test No. 14

Cyanidation of the Concentrates after Roasting

As the extraction by cyanidation of the gold in the flotation concentrates was so consistently low, it was decided to roast a sample of these concentrates to drive off most of the sulphur and then cyanide the roasted product.

The roasting was done in a gas-fired muffle at a dull red heat and no attempt was made to make the roast complete.

The analysis of the concentrates before roasting was Au 1·82 oz./ton; Fe 28·84 per cent, S 28·85 per cent, As 0·07 per cent, Pb 0·03 per cent, Cu 0·18 per cent, Insol 28·4 per cent.

The analysis of the roasted concentrates was Au 2·08 oz./ton; S 3·24 per cent.

The loss in weight due to roasting was 17·65 per cent.

From the above figures the gold content by assay in the roasted product is lower than the calculated value, the latter being 2·21 ounces per ton.

Four cyanide tests were made on the roasted concentrates, the details of which are as follows:—

Head sample	Au 2·08 oz./ton
Dilution during agitation	3 : 1
Cyanide strength	KCN 0·20 per cent

Test No.	Tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, lb./ton				Time, hours
			On roasted concentrate		On raw concentrate		
			KCN	CaO	KCN	CaO	
14-A	0·18	91·3	4·6	73·0	3·8	60·1	48
14-B	0·17	91·9	4·9	75·6	4·0	62·2	72
14-C	0·19	91·0	3·6	40·5	3·0	33·4	70
14-D	0·18	91·3	3·95	53·4	3·3	44·0	70

NOTE.—14-C concentrates water-washed before cyanidation.
14-D concentrates ground in water, then filtered before cyanidation.

CONCLUSIONS

Concentration. Concentration by flotation gave fairly satisfactory results on these tailings. The highest recovery obtained was 70 per cent. The concentrate assayed about 1·6 ounces per ton in gold and the ratio of concentration was 25:1. The gold remaining in the flotation tailing was partly associated with pyrite which did not respond to flotation principally because of oxidation, and some probably in pyrite unfreed from gangue.

The reagents used and their approximate quantities per ton of cyanide tailings were as follows: soda ash 3·0, water-gas tar A 0·15, amyl xanthate 0·10, and pine oil 0·10.

For the flotation of these tailings two flow-sheets are suggested. First: The cyanide tailings be pumped to a classifier and the classifier overflow floated. The classifier return, consisting of the coarser sands, be reground in a ball mill in closed circuit with the classifier. Second: The cyanide tailings be first floated and the flotation tailing classified. The classifier overflow would then be the final tailing, and the classifier return would be reground in a ball mill in closed circuit with a second classifier. The overflow from this second classifier would be floated in a second flotation machine of about 25 per cent the capacity of the first machine. The rougher concentrate from this second operation might be united with the feed to the main flotation circuit or it might be cleaned and then combined with the concentrate from the main operation. This second flow-sheet would be the more costly installation but would permit a separate treatment, with other or additional reagents, of the reground sands.

In either case emphasis is made on the importance of allowing sufficient contact of the reagents with the pulp before flotation.

Cyanidation of the Concentrates. Cyanidation of the concentrates was not satisfactory as only about 50 per cent extraction of the gold was obtained. While this result is disappointing it corresponds fairly closely with previous work done, 1927, on a similar concentrate from cyanide tailings from the Wright-Hargreaves Mines, Ltd.

It is reported that extractions of 97 per cent of the gold in similar concentrates have been made by cyanidation. Unfortunately, we have not been able to duplicate these most excellent results.

When these flotation concentrates were first subjected to an oxidizing roast, then cyanided, the extraction was increased to 91 per cent. While no attempt was made to make the roast complete it was sufficient to permit an increase in extraction of over 40 per cent. Were the roasting done on a larger scale, where conditions could be more closely controlled, it is quite probable that a more even and thorough roasted product would be obtained with a consequent better extraction by cyanidation.

It is evident that the success of this project will largely depend on the solution of the treatment of the flotation concentrate.

Report No. 398

INVESTIGATION OF RIFFLE CONCENTRATE FROM McLEOD RIVER MINING CORPORATION, LTD., PEERS, ALBERTA

C. S. Parsons

Shipment. Two samples of riffle concentrates containing gold were received from the McLeod River Mining Corporation, Peers, Alberta, on August 16 and 19, 1931.

Characteristics of Samples. Both these samples are concentrates obtained by riffling one cubic yard of gravel from the dredging operations of the above company.

Sample No. 1 represents only the gravel at point of sampling above water level. The gravel bed at this point is said to be 22 to 24 feet thick of which 4 to 6 feet is above water level.

Sample No. 2 represents the top six feet of soil and gravel near dredge site.

Experimental Tests

Sample No. 1 consisted of black sand and weighed 388 grammes. The sample was examined under the microscope and found to contain flat flakes of gold all smaller than pin head size and also some very fine granular gold. The gold looked bright and clean. No attempt was made to cut a head sample for assay as the character and quantity of free gold would prevent an accurate sample being taken. The whole of this sample was, therefore, amalgamated by placing it in a bottle and shaking with water and mercury. The mercury after amalgamation was carefully collected and dissolved in acid.

Results:

Weight of gold recovered from 388 grammes of sand........449·30 mgs.
Assay of tailing after amalgamation...................... 0·34 oz./ton
Weight of gold in tailings is therefore.................... 4·50 mgs.

$$\frac{0\cdot34 \text{ oz./ton} \times 31103\cdot5 \text{ mgs.} \times 388 \text{ grms.}}{2000 \text{ lb.}}$$

$$= \frac{0\cdot34 \times 31103\cdot5 \times 388}{907185} = 4\cdot50 \text{ mgs.}$$

1 ton, 2000 lb., contains 907185 grms.
Total weight of gold in 388 grms. of sand = 449·30 + 4·50 = 453·80 mgs.
Value of total gold in one yard of gravel is:
1 ounce troy contains 31103·6 mgs.
1 ounce troy of gold is worth $20.67.

$$\therefore 453\cdot8 \text{ mgs. of gold are worth } \frac{20\cdot67 \times 543\cdot8}{31103} = 30\cdot1 \text{ cents.}$$

Value of recovered gold from one yard of gravel:

$$= \frac{20\cdot67 \times 449\cdot30 \text{ mgs.}}{31103} = 29\cdot8 \text{ cents.}$$

$$\text{Percentage of total gold recovered} = \frac{29\cdot8}{30\cdot1} = 99\%$$

Tests and Analyses for the Platinum Group Metals

The tailings, after the amalgamation, were assayed and found to contain[1] gold 0·34 ounce per ton, platinum and palladium 0·40 ounce per ton, and osmium and iridium 0·38 ounce per ton.

A concentration test was made on part of the tailing after amalgamation to determine whether or not the values remaining could be concentrated into a smaller bulk. Therefore 214·3 grammes was passed over a corduroy blanket; amount caught on blanket was 47·1 grammes. The blanket was burnt and the content assayed and found to run as follows:

Concentrate on blanket assayed: gold, 1·51 ounces per ton; platinum and palladium, 1·29 ounces per ton; osmium and iridium, none.

If this is figured back it will be found that it does not check with the sample of the amalgamation tailings.

Original tailing sample: gold, 0·34 ounce per ton; platinum and palladium, 0·40; osmium and iridium, 0·38. Calculated tailing sample: gold, 0·34 ounce per ton; platinum and palladium, 0·28; osmium and iridium, 0·0014. This confirms the above note regarding the assay of this tailing sample, and it is believed that it contains very little osmium or iridium.

Sample No. 2 consisted of a concentrate from one cubic yard of gravel which weighed 1456·2 grammes. The sample contained more light sand than did sample No. 1. The sample examined under the microscope was found to contain all the gold in the form of thin, flat flakes from pin head size to smaller. As this sample was much larger than sample No. 1, an attempt was made to cut out a head sample; 369·2 grammes was cut for a head sample; 719·3 grammes was cut for an amalgamation test; 387·3 grammes was cut and held for examination.

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[1] No reliance should be placed on these assays as some trouble was encountered in making them and there was not sufficient of the sample to check the result.

Results of Amalgamation Test: This test was made in the same manner as on sample No. 1. The amalgam from 719·3 grammes of concentrate was found to contain 492·88 milligrams of gold. Tailing after amalgamation was found to contain gold 0·37 ounce per ton; platinum and palladium, 0·28 ounce per ton; and osmium and iridium, trace.

Recalculating this recovered gold to the total weight of concentrate, namely 1456·2 grammes,

$$\frac{1456{\cdot}2 \times 492{\cdot}88}{719{\cdot}3} = 997{\cdot}82 \text{ mgs.}$$

and weight of gold in tailing assay 0·37 oz./ton is

$$\frac{0{\cdot}37 \times 31103{\cdot}5 \times 1456{\cdot}2}{907185} = 18{\cdot}10 \text{ mgs.}$$

Total weight of gold in 1456·2 grammes of concentrate or 1 cubic yard of gravel = 997·82 + 18·1 = 1015·92 mgs.

Value of total gold in one yard of gravel therefore =

$$\frac{\$20.67 \times 1015{\cdot}92 \text{ mgs.}}{31103} = 67{\cdot}4 \text{ cents.}$$

Value of recovered gold in one yard of gravel =

$$\frac{\$20.67 \times 997{\cdot}82 \text{ mgs.}}{31103} = 66{\cdot}2 \text{ cents.}$$

$$\text{Percentage of total gold recovered} = \frac{66{\cdot}2}{67{\cdot}4} = 98{\cdot}3\%$$

Analysis of head sample: gold, 17·57 ounce per ton; platinum and palladium, 0·66 ounce per ton; osmium and iridium, trace.

Calculated head sample from products of amalgamation test is—gold, 20·35 ounce per ton.

Report No. 399

EXPERIMENTAL TESTS ON GOLD ORE FROM THE BRALORNE MINES, LTD., BRIDGE RIVER, LILLOOET MINING DIVISION, B.C.

J. S. Godard

Shipment. A shipment of 145 pounds of ore was received June 16, 1931, from H. E. Wilmot, manager of the Bralorne Mines, Limited.

The sample was taken from the King vein, main adit level of the Lorne mine, Bridge River, Lillooet mining division, British Columbia.

Characteristics and Analysis of the Ore. The sample submitted was a high-grade gold ore containing considerable free gold of which 20 per cent of the total gold was +18 mesh. Of the remainder about 55 per cent is free and amalgamable at −48 mesh. The balance of the gold is associated with a small quantity of sulphides, of which arsenopyrite and pyrite predominate. The gangue is of hard quartz.

Analyses of the head sample showed gold 1·41 ounces per ton.

Purpose of Experimental Tests. Mr. Wilmot requested that a few tests be made for the purpose of obtaining some information that might be of assistance in the design of a mill to recover the gold in this ore. Preference was expressed for a flow-sheet that would embody the following steps, namely, grinding in a ball mill, amalgamation, classification, regrinding the oversize, flotation, and either blanket concentration of the flotation tailing, or desliming of the flotation tailing and table concentration of the coarser material. The concentrates from these operations to be reground and cyanided for the recovery of the gold.

EXPERIMENTAL TESTS

Tests Nos. 1, 2, and 3

Amalgamation Tests

Ore dry crushed to pass given meshes. Head sample for tests, Au 1·13 ounces per ton.

Results:

Test 1—Ore Crushed—35 mesh

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au	Average tailings, Au, oz./ton	Recovery, per cent
+ 48	9·2	0·61	11·1	0·51	54·8
+ 65	20·6	0·50	20·4		
+100	16·9	0·42	14·0		
+200	21·3	0·41	17·2		
−200	32·0	0·59	37·3		

Test 2—Ore crushed—65 mesh

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au	Average tailings, Au, oz./ton	Recovery, per cent
+100	25·9	0·44	24·6	0·46	59·2
+200	29·5	0·40	25·4		
−200	44·6	0·52	50·0		

Test 3—Ore crushed—150 mesh

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au	Average tailings, Au, oz./ton	Recovery, per cent
+200	26·7	0·35	18·5	0·50	55·8
−200	73·3	0·56	81·5		

Test No. 4

Amalgamation and Concentration

A sample of ore was dry crushed to pass 48 mesh, then amalgamated. The amalgamation tailing was deslimed and the fine product floated, No. 1

concentrate. The coarser product was reground then floated, No. 2 concentrate. The tailing from the second flotation was deslimed and the coarser product tabled.

Results:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
No. 1 concentrate	2·3	5·18	23·8
No. 2 concentrate	3·5	9·68	67·5
Table concentrate	3·2	0·34	2·2
No. 1 tailing	29·1	0·05	2·9
Table tailing +200	24·8	0·03	1·5
Table tailing −200	30·4	0·03	1·8
Slimes	6·7	0·02	0·3

Head sample for test	Au 1·13 oz./ton
Amalgamation tailing	Au 0·50 "
Recovery— I. By amalgamation	55·8 per cent
II. By concentration	41·4 "
Total	97·2 "

Ratio of concentration—11 : 1.

Test No. 5

Amalgamation, Concentration and Treatment of Concentrates

In this test, 18 lots of ore at −48 mesh were treated as in Test No. 4, except that the flotation concentrates from the first flotation were cleaned once and the middlings deslimed. The coarser product from this desliming operation was combined with the feed to the table. By this method of treatment the grade of concentrate was slightly increased.

The concentrates assayed: Au 6·50 ounces per ton, Ag 2·29 ounces per ton, Cu 0·23 per cent, Pb 0·08 per cent, Zn 0·30 per cent, Fe 16·34 per cent, As 9·35 per cent, S 11·15 per cent.

The ratio of concentration was 13·6 : 1.

Test No. 5A

Treatment of Concentrates

A sample of the concentrates was given a short regrind in a pebble mill in 1 : 1 pulp, KCN 0·10 per cent, CaO 10 ounces per ton. The pulp was then diluted 3 : 1 and agitated 48 hours. KCN maximum 0·20 per cent. At the end of this agitation period the pulp was then filtered and reagitated for 20 hours using the original solution.

The cyanide tailing was screened on 200 mesh.

Results:

Mesh	Weight, per cent	Assays, oz./ton		Per cent of values		Average tailing, oz./ton		Extraction, per cent	
		Au	Ag	Au	Ag	Au	Ag	Au	Ag
+200.............	11·0	1·23	0·58	23·3	19·3	0·58	0·33	91·1	85·7
−200.............	89·0	0·50	0·30	76·7	80·7				

Summary:

Gold recovery— I. By amalgamation and cyanidation 97·2 per cent (Ref. Test No. 4).
II. By cyanidation 91·1 per cent.

Total, 88·4 per cent.

Reagents consumed, lb./ton of concentrate, KCN 13·04, CaO 25·6.
Reagents consumed, lb./ton of ore..........KCN 0·96, CaO 1·9.

Test No. 5B

Cyanidation

This test was similar to 5A in detail except that the concentrates were more finely ground before cyanidation.

The cyanide tailing was screened on 200 mesh.

Results:

Mesh	Weight, per cent	Assays, oz./ton		Per cent of values		Average tailing, oz./ton		Extraction, per cent	
		Au	Ag	Au	Ag	Au	Ag	Au	Ag
+200.............	0·7	2·19	1·10	3·2	2·4	0·47	0·33	92·8	85·7
−200.............	99·3	0·46	0·32	96·8	97·6				

Summary:

Gold recovery, I. By amalgamation and concentration, 97·2 per cent (Ref. Test No. 4).
II. By cyanidation, 92·8 per cent.

Total, 90·2 per cent.

Reagents consumed, lb./ton of concentrates, KCN 13·2 , CaO 27·4.
Reagents consumed, lb./ton of ore..........KCN 0·98, CaO 2·02.

Test No. 5C

Amalgamation

A sample of the concentrates, dry crushed −150 mesh was amalgamated to determine the quantity of free and amalgamable gold contained therein.

Amalgamation tailing screened on 200 mesh.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au	Average tailing, Au, oz./ton	Recovery, per cent
+200	11·0	8·28	16·5	5·51	15·2
−200	89·0	5·17	83·5		

Test No. 5D

Amalgamation and Cyanidation

A sample of the concentrates, dry crushed −150 mesh was amalgamated as in Test 5C and the amalgamation tailing was dewatered, and cyanided for 65 hours in 3 : 1 pulp, KCN maximum 0·20 per cent.

The cyanide tailing was screened on 200 mesh.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au	Average tailing, Au, oz./ton	Extraction, per cent Au
+200	10·7	1·14	21·1	0·58	89·5
−200	89·3	0·51	78·9		

Summary:

Head sample to amalgamation	Au 6·50 oz./ton
Amalgamation tailing (Ref. Test No. 5C)	Au 5·51 "
Recovery by amalgamation	Au 15·2 per cent
Heads to cyanidation—Amalgamation tailing	Au 5·51 oz./ton
Cyanidation tailing	Au 0·58 "
Extraction	Au 89·5 per cent
Recovery I. By amalgamation	Au 15·2 per cent
II. By cyanidation	Au 76·0 "
Total	Au 91·2 "

Reagents consumed, lb./ton of concentrates, KCN 11·4, CaO 17·5.
Reagents consumed, lb./ton of ore..........KCN 0·84, CaO 1·30.

Tests Nos. 6, 7, 8

Amalgamation and Cyanidation

In this series of three tests the ore was dry crushed to pass 48 mesh then amalgamated. The amalgamation tailing was then dewatered and cyanided.

In Test No. 6 no regrinding was practical before cyanidation but in Tests Nos. 7 and 8 the amalgamation tailings were reground to different degrees of fineness in cyanide solution before cyanidation.

Cyanidation was done at a dilution of 2·5 : 1, the cyanide strength, KCN (maximum) was 0·075 per cent and the time of agitation was about 45 hours.

All cyanidation tailings were screened and the products assayed.

For calculation purposes a recovery of 55 per cent of the gold is attributed to amalgamation.

Head sample to cyanidation (calculated), Au 0·51 ounce per ton.

Results:

Screening the Cyanidation Tailings

Test No.	Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au	Average tailing, Au, oz./ton	Extraction, per cent
6	+ 65	30·1	0·07	27·6	0·077	85·0
	+100	27·5	0·07	25·2		
	+200	20·7	0·07	18·9		
	−200	21·7	0·10	28·3		
7	+200	51·9	0·06	41·8	0·074	85·5
	−200	48·1	0·09	58·2		
8	+200	29·7	0·05	22·0	0·068	86·7
	−200	70·3	0·075	78·0		
		+				

Summary—Tests Nos. 6, 7, and 8:

Head sample to amalgamation	Au 1·13 oz./ton
Amalgamation tails, head sample to cyanidation	Au 0·51 "

Number	Recoveries, per cent of total gold			Reagents consumed, lb./ton	
	Amalgamation	Cyanidation	Total	KCN	CaO
6	55·0	38·3	93·3	0·36	5·1
7	55·0	38·5	93·5	0·66	6·1
8	55·0	39·0	94·0	0·62	8·9

Test No. 9

Settling Test

A small-scale settling test was made on a sample of cyanide tailings for the purpose of indicating the settling rate of the pulp in a cyanide solution. The details of the test are as follows:

SUMMARY

About 55 per cent of the gold was recovered by amalgamation at −48 mesh. Flotation of the amalgamation tailing recovered an additional 41 per cent of the gold in the form of a concentrate assaying, Au 6·50 ounces per ton, Ag 2·29 ounces per ton, Fe 16·3 per cent, As 9·35 per cent. The ratio of concentration was 13·6 : 1. Cyanidation of this concentrate showed a gold extraction of 92·8 per cent, making an overall recovery of 90 per cent.

Amalgamation and cyanidation of the amalgamation tailing showed a recovery of 94 per cent of which about 55 per cent is attributed to amalgamation and 39 per cent to cyanidation.

CONCLUSIONS

Two methods of treatment were found to be satisfactory for this ore. They are amalgamation and concentration of the amalgamation tailing and cyanidation of the concentrates, and amalgamation and cyanidation of the amalgamation tailing.

By the first method an excellent tailing is secured as well as a good grade of concentrate and a fairly high ratio of concentration. Extraction of the gold in the flotation concentrate was as high as 92·8 per cent making a net recovery of 90·2 per cent.

By the second method net recoveries of 93 to 94 per cent of the gold were obtained.

Of the recoveries obtained about 55 per cent may safely be attributed to amalgamation, which owing to the presence of some relatively coarse free gold would seem to be almost essential in any flow-sheet designed to treat this ore.

Dilution: 4:1.
Solids: 30 per cent +200 mesh.
Cyanide strength: KCN 0·070 per cent; CaO 0·089 per cent.
Cyanide used: Cassel's NaCN 127·5 per cent.
Temperature of pulp: 71·60° F.
Diameter of cylinder: 2·28 in.
Depth of column of pulp: 12·1 in.

Readings:

Time, mins.	Depth of clear solution, in.	Time, mins.	Depth of clear solution, in.
3	1·65	18	6·77
6	3·10	21	7·22
9	4·35	24	7·62
12	5·35	54	8·00
15	6·18		

Report No. 400

THE RECOVERY OF GOLD FROM THE ORE OF THE HAYDEN GOLD MINES CO., LTD., TIMMINS, ONT.

Alex. K. Anderson

Shipments. Two bags of ore, weighing 217 pounds, were received by express from Timmins, Ont., shipped by T. C. Fawcett, Mine Manager, Hayden Gold Mines Co., Ltd., Box 2064, Timmins, Ont. This consignment is called Lot No. 2, as a previous high-grade shipment had already been reported on.

Characteristics of the Ore. The shipment consisted of a siliceous gangue carrying a small amount of fine-grained iron pyrite. No free gold was visible.

Purpose of Experimental Tests. The shipment was made in order to determine the recovery of gold that could be obtained by standard methods of treatment. This test work showed that 61·7 per cent of the gold could be recovered by amalgamating material ground to pass 200 mesh. The same material when cyanided yielded 87·0 per cent of the gold.

Sampling and Analysis. The lot was crushed to pass ½-inch and quartered. By alternate grinding to finer sizes and quartering, a representative portion −200 mesh was secured. This was found to contain 0·23 ounce gold, 0·04 ounce silver per ton.

EXPERIMENTAL TESTS

The investigation included flotation concentration tests, amalgamation and cyanidation of the tailing after amalgamation together with cyanide tests on the raw ore ground to different degrees of fineness.

AMALGAMATION

Test No. 1

One lot of 1000 grammes, wet ground to pass 94 per cent through 200 mesh and amalgamated.

Heads	Au 0·23 oz./ton
Tailing	Au 0·088 "
Recovery	61·7 per cent

A screen analysis of the tailing shows—

Mesh	Weight, per cent	Assay, Au, oz./ton
+100	0·21	trace
−100+150	1·33	trace
−150+200	3·73	0·08
−200	94·73	0·09

Average tailing........0·088 oz./ton.

This screen analysis shows that all gold coarser than 150 mesh is free and easily recovered by amalgamation. The gold in the finer sizes apparently is partly associated with the fine sulphides.

The tailing from amalgamation was cyanided for 48 hours 1 : 3 dilution with a 1 pound per ton cyanide solution. Lime, 8 pounds per ton of ore, was added for protective alkalinity. This gave a tailing containing 0·04 ounce gold per ton, or an extraction of 54·6 per cent of the gold remaining after amalgamation, or a combined recovery by both of 82·6 per cent; 0·6 pound KCN and 6·8 pounds lime were consumed during the cyaniding.

FLOTATION

Test No. 2

This test by flotation consisted of a two-stage operation. The ore was ground 70 per cent solids with 3 pounds soda ash and 0·14 pound Aerofloat No. 25 per ton to pass approximately 35 per cent through 200 mesh or to about 28 mesh. After adding 0·05 pound potassium ethyl xanthate per ton, concentrate No. 1 was removed. The tailing was deslimed giving tailing No. 1. The sands were returned to the mill and reground with 1 pound soda ash, 0·06 pound Aerofloat until all passed 200 mesh. Concentrate No. 2 was taken off after the addition of 0·03 pound xanthate and 0·05 pound cresylic acid to the flotation machine.

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
Heads (calculated)	100·00	0·209	100·0
Concentrate No. 1	9·52	1·50	68·2
Concentrate No. 2	4·13	0·72	14·2
Tailing No. 1	31·25	0·03	4·5
Tailing No. 2	55·10	0·05	13·1
Combined concentrates	13·65	1·26	82·4
Combined tailings	86·35	0·04	17·6

This flotation test shows that the same recovery can be made by flotation as is secured by amalgamation and cyanidation. However, the gold still remains to be extracted from the concentrate. Fine grinding is necessary to secure a high recovery as 14·2 per cent of the gold is recovered after regrinding the sands from the first flotation.

Test No. 3

A series of tests was made on the raw ore ground to different degrees of fineness and cyanided for 48 hours with a 1 pound KCN per ton solution, 1 : 3 dilution.

Mesh grind	Heads, Au, oz./ton	Tailings, Au, oz./ton	Extraction, per cent	Consumption, lb./ton	
				KCN	CaO
− 48	0·23	0·06	73·9	0·6	7·1
−100	0·23	0·05	78·3	0·75	7·1
−150	0·23	0·04	82·7	0·90	7·5
−200	0·23	0·03	87·0	1·05	7·7

Three lots of the ore were ground to pass 200 mesh and cyanided for 18, 24, and 36 hours with the following results:—

Time of agitation, hours	Heads, Au, oz./ton	Tails, Au, oz./ton	Extraction, per cent
18	0·23	0·03	87·0
24	0·23	0·03	87·0
36	0·23	0·025	89·1

These tests show that fine grinding is necessary for highest recovery. The gold is rapidly dissolved, as much going into solution in 18 hours as in 48 hours.

Cyanidation of this ore leaves the same amount of gold in the tailing, 0·03 ounce, as did tests made on shipment No. 1, much higher in free gold. It would appear, therefore, that this amount is in a form not readily recovered.

The above tests indicate that cyanidation will yield the highest metallurgical recovery. A flow-sheet similar to Hollinger practice incorporating concentration of the sulphides with re-treatment will doubtless yield good recoveries.

Report No. 401

THE RECOVERY OF GOLD FROM THE ORE OF THE LEROY MINES, LTD., TOWNSHIP OF LOUVICOURT, QUE.

Alex. K. Anderson

Shipment. A sample of ore, weighing 150 pounds, was received October 27, 1931, at the Laboratories of the Ore Dressing and Metallurgical Division, Department of Mines, Ottawa. This was expressed from the property of the LeRoy Mines, Ltd., situated in the township of Louvicourt, northwestern Quebec, with head office at 660 Ste. Catherine West, Montreal, Que., Mr. H. Lefebvre, President.

Characteristics of the Ore. The shipment consisted of white quartz with stringers of basic rock cutting through. Considerable fine-grained iron pyrite was present lying along these stringers. No free gold was visible to the naked eye.

Purpose of Experimental Tests. This investigation was made to determine the best method to recover the contained values. Tests showed that 74·5 per cent of the gold was recovered by amalgamation at 48-mesh grinding and 89·7 per cent at 150-mesh grinding. Amalgamation followed by cyaniding gave a recovery of 99·6 per cent.

Sampling and Analysis. The entire lot was crushed to pass 20 mesh and quartered through a Jones riffle sampler. By passes through the sampler with intervening grinding to finer sizes a representative portion −100 mesh was obtained for analysis. This showed the shipment to contain 4·47 ounces gold and 0·19 ounce silver per ton.

EXPERIMENTAL TESTS

The investigation included tests by amalgamation at different degrees of fineness followed by cyanidation of the tailing; amalgamation followed by flotation concentration with cyanidation of the flotation tailing; amalgamation and table concentration, and straight cyanidation.

Test No. 1

A representative portion of the ore was ground to pass 48 mesh and amalgamated with mercury. After removing amalgam, the residue was found to contain 1·14 ounces gold per ton, representing a recovery of 74·5 per cent.

A portion of the amalgamation tailing was agitated for 24 hours, 1 : 3 dilution with a 1·0 pound KCN solution. Lime, 9 pounds, was added for protective alkalinity.

Part of this amalgamation tailing was reground to pass 200 mesh and cyanided.

Amalgamation:

Heads	Au 4·47 oz./ton.
Amalgamation tailing	Au 1·14 "
Recovered by amalgamation	74·5 per cent.

Cyanidation of unground amalgamation tailing:

Heads	Au 1·14 oz./ton.
Cyanide tailing	Au 0·40 "
Extraction	64·9 per cent.
Recovered by amalgamation and cyanidation	91·1 per cent.

Cyanidation of reground amalgamation tailing:

Heads	Au 1·14 oz./ton.
Cyanide tailing	Au 0·03 "
Extraction	97·4 per cent.
Recovered by amalgamation and cyanidation	99·3 per cent.

Test No. 2

In this test, a sample of the ore was ground to pass 100 mesh and treated as in Test No. 1.

Amalgamation:

Heads	Au 4·47 oz./ton.
Amalgamation tailing	Au 0·63 "
Recovered by amalgamation	85·9 per cent.

Cyanidation of unground amalgamation tailing:

Heads	Au 0·63 oz./ton.
Cyanide tailing	Au 0·17 "
Extraction	73·0 per cent.
Recovered by amalgamation and cyanidation	96·2 per cent.

Cyanidation of reground amalgamation tailing:

Heads	Au 0·63 oz./ton.
Cyanide tailing	Au 0·03 "
Extraction	95·3 per cent.
Recovered by amalgamation and cyanidation	99·3 per cent.
Reagent consumption	KCN 0·3 lb./ton. CaO 7·8 "

The gold in the ore is comparatively fine as but 0·12 ounce per ton remains on 100 mesh.

Test No. 3

A sample of the ore was ground to pass 150 mesh and treated as in preceding tests.

Amalgamation:

Heads	Au 4·47 oz./ton.
Amalgamation tailing	Au 0·46 "
Recovered by amalgamation	89·7 per cent.

Cyanidation of unground amalgamation tailing:

Heads	Au 0·46 oz./ton.
Cyanide tailing	Au 0·09 "
Extraction	80·4 per cent.
Recovered by amalgamation and cyanidation	98·0 per cent.

Cyanidation of reground amalgamation tailing:

Heads	Au 0·46 oz./ton.
Cyanide tailing	Au 0·02 "
Extraction	95·7 per cent.
Recovered by amalgamation and cyanidation	99·6 per cent.

Test No. 4

In this test, a sample was ground to pass 200 mesh and treated as in preceding tests.

Amalgamation:

Heads	Au 4·47 oz./ton.
Amalgamation tailing	Au 0·47 "
Recovered by amalgamation	89·5 per cent.

Cyanidation of amalgamation tailing:

Heads	Au 0·47 oz./ton.
Cyanide tailing	Au 0·07 "
Extraction	85·1 per cent.
Recovered by amalgamation and cyanidation	98·4 per cent.

It is apparent from the results of cyaniding this amalgamation tailing without regrinding that there is some gold in a form that requires to be reground to secure maximum recoveries. The final tailing from this test is 0·07 ounce gold per ton while that of the cyanidation of reground tailing in Test No. 3 is 0·02 ounce.

Test No. 5: Amalgamation, Flotation, and Cyanidation

A representative portion of the ore was wet ground to pass 200 mesh and amalgamated. After removing amalgam, the pulp was diluted to 1 : 4 and agitated in a flotation machine for 5 minutes together with 2 pounds soda ash per ton. A concentrate was then removed by the addition of 0·10 pound potassium ethyl xanthate, 0·10 pound Aerofloat No. 25, 0·06 pound hardwood creosote, and 0·06 pound cresylic acid per ton.

The tailing after flotation was dewatered and cyanided 1 : 3 dilution for 24 hours with a 1·0 pound KCN solution and 9 pounds lime per ton.

Product	Weight, per cent	Assay, Au, oz./ton	Per cent of values
Heads	100·0	4·47	100·0
Amalgamation tailing	....	0·26	*94·2
Flotation concentrate	3·2	6·26	4·5
Flotation tailing	96·8	0·06	1·3
Cyanide tailing	....	0·01	....

*Recovered by amalgamation.

This shows that wet grinding to −200 mesh followed by amalgamation recovers 94·2 per cent of the gold. Flotation recovers an additional 4·5 per cent in a pyrite concentrate assaying 6·26 ounces per ton. The flotation tailing contains 0·06 ounce per ton which is reduced to 0·01 ounce by cyaniding.

No cyanide tests were made on the concentrate but judging by the behaviour of the ore in previous tests no trouble should be experienced in making a high recovery of the gold contained in it by cyanidation.

However, as the flotation tailing contains 0·06 ounce gold per ton, and is treated by cyanidation, no advantage is to be gained by concentration.

Test No. 6: Flotation

A test was made on the ore to note the results obtained by flotation. This gave a concentrate containing 46·60 ounces gold per ton with a recovery of 85·4 per cent and left 0·82 ounce in the tailing. Amalgamating this product left a residue containing 0·12 ounce gold per ton.

Flotation, therefore, is not applicable to this ore.

Test No. 7

Amalgamation followed by table concentration gave a concentrate assaying 12·06 ounces gold per ton representing 3·1 per cent of the values, and a tailing assaying 0·14 ounce.

This method does not yield highest recoveries.

Straight Cyanidation

Representative portions of the ore were ground to pass 48, 100, 150, and 200 mesh and cyanided for 48 hours, 1 : 3 dilution with a 1·0 pound KCN solution and 9 pounds lime per ton.

Results:

Mesh grind	Heads, Au, oz./ton	Tailings, Au, oz./ton	Extraction, per cent	Reagent consumption, lb./ton	
				KCN	CaO
48	4·47	0·38	91·5	0·9	7·9
100	4·47	0·16	96·4	0·3	7·9
150	4·47	0·08	98·2	0·3	8·2
200	4·47	0·055	98·8	0·3	8·4

Samples taken after 24 hours agitation were practically the same as those after 48 hours showing that the gold is readily soluble in cyanide solution.

SUMMARY AND CONCLUSIONS

The treatment of this ore presents no great metallurgical difficulties. The major amount of the gold present in the shipment is in the form of finely divided metal, 74·5 per cent of which can be recovered by amalgamating at −48-mesh grinding and 89·7 per cent at −150-mesh. The gold not caught on the amalgamating plates is readily soluble in cyanide solution after regrinding, yielding a total recovery of 99·3 per cent.

As the gold is readily recovered by amalgamation and cyanidation, flotation and table concentration are not applicable as the tailings from these operations are too high to discard.

The process recommended for treatment of this grade of ore is to grind not finer than 65 mesh to avoid losses in float gold in subsequent thickening operations, amalgamate, regrind the thickened tailing to −200 mesh and cyanide.

Care should be taken to determine if the nature and grade of the ore-body is likely to change with depth. A decrease in the gold content and higher tailing losses would possibly mean a change in milling practice.

Report No. 402

FLOTATION TESTS ON A SAMPLE OF HIGH-GRADE COBALT ORE FROM KENORA PROSPECTORS AND MINERS, LIMITED, TORONTO, ONTARIO

J. S. Godard

Shipments. A small shipment consisting of 4,420 grammes was received October 22, 1931, from Mr. N. F. Parkinson, Secretary-Treasurer for Kenora Prospectors and Miners, Ltd., 100 Adelaide Street West, Toronto, Ontario. The sample was taken from claims K.R.L. Nos. 10590 to 95 inclusive, Werner Lake district, Red Lake mining division, Ontario.

Characteristics of the Ore. The sample submitted was that of a high-grade cobalt ore. It is presumed that the cobalt is present as cobaltite and while no head sample analysis was made, because of the small quantity of the sample submitted, the products from three tests showed an average of 15·81 per cent cobalt in the ore. This sample is unique because of the presence of a quantity of iron sulphide and a small amount of zincblende in the ore.

Purpose of Tests. Mr. Parkinson requested that concentration by flotation of the cobalt be tried with the main object in view of making a high-grade cobalt concentrate.

Experimental Tests. Three small-scale flotation tests were made on this sample. The ore was wet ground to about 70 per cent −200 mesh before flotation.

Results:

Test No.	Product	Weight, per cent	Assays, Co per cent	Per cent of value, Co
1	Heads	100·0	15·84	100·0
	Concentrates	37·5	27·06	64·1
	Middlings	7·0	13·85	6·1
	Tailings	55·5	8·51	29·8
2	Heads	100·0	16·11	100·0
	Concentrates	39·9	29·56	73·3
	Middlings	10·6	21·44	14·1
	Tailings	49·5	4·12	12·6
3	Heads	100·0	15·46	100·0
	Concentrates	60·6	24·77	97·1
	Tailings	39·4	1·14	2·9

CONCLUSIONS

From the above three tests it is evident that the grade can be increased from 15·81 per cent to 24·77 per cent in the flotation concentrate with a recovery of 97 per cent of the cobalt. Attempts to increase the grade of concentrate were successful to a point where a concentrate assaying 29·5 per cent cobalt was obtained. This higher grade concentrate was made at the expense of the recovery which, for this grade of concentrate, would approximate about 85 per cent.

Owing to the small differences in the flotability of cobalt, iron pyrite, and zincblende it is not thought possible to drop these minerals by a selective action without incurring a serious loss of cobalt.

Report No. 403

CONCENTRATION TESTS ON SAMPLES OF TWO MILL PRODUCTS FROM THE KIRKLAND LAKE GOLD MINING COMPANY, LIMITED, KIRKLAND LAKE, ONT.

J. S. Godard

Shipments. A shipment consisting of 900 pounds marked B.U. and 1,000 pounds marked M.T. was received October 28, 1931, from the Kirkland Lake Gold Mining Company, Limited, Kirkland Lake, Ontario.

Characteristics of the Products. The product designated B.U. is equivalent to the return from the bowl classifier in the grinding circuit of a cyanide mill practising fine grinding. Such a product consists of the coarser sulphides and sands and owing to the concentrating action in the classifiers it is relatively higher in gold-bearing sulphides than the original feed though lower in gold value owing to the dissolving action of the cyanide to this stage in the circuit.

The sample of this product contained 16 per cent moisture as cyanide solution, and as dissolution is continually taking place the relationship between the undissolved and dissolved gold value is a variable one depending upon the time elapsed between the taking of the sample and the commencement of the test work and also between tests.

The following average screen analysis indicates the fineness of grinding to this stage:

Mesh	Weight, per cent
+100	4·8
+200	58·7
−200	36·5

The sample designated M.T. (mill tailings) is the final tailing as discharged from the filter. As this product also contained some cyanide solution, a somewhat similar relationship between undissolved and dissolved gold value exists as in the previously described product.

A screen analysis on this product showed that 95 per cent was −200 mesh.

Purpose of Experimental Tests on Product B.U. The product B.U. was subjected to three small-scale flotation tests and a medium-sized table test for the purpose of ascertaining if the sulphides could be removed at this stage and subjected to a special grinding and agitation before being repassed to the main cyanide circuit to be ultimately discharged with the tailing.

Some data on the feed rate to the table were desired in order to estimate the number of tables that would be required to yield results at least equal to those obtained in the table tests and to treat about 150 tons per twenty-four hours.

Results of Screen Analyses on Products:

1. Head sample—Au 0·156 oz./ton.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	6·4	0·20	8·2
+200	59·3	0·12	45·7
−200	34·3	0·21	46·1

2. Table concentrate—Au 0·524 oz./ton.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	18·3	0·36	12·6
+200	44·1	0·34	28·6
−200	37·6	0·82	58·8

3. Table tailing—Au 0·126 oz./ton.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	6·3	0·20	10·0
+200	63·0	0·11	55·8
−200	30·7	0·14	34·2

Summary of Table Test:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Head	100·0	0·156	100·0
Concentrate	6·2	0·524	21·6
Tailing	93·8	0·126	78·4

1. Product B.U.

Experimental Tests

Mr. John Dixon, mill superintendent for the Kirkland Lake Gold Mining Company, Limited, was present during, and co-operated in, this test work.

Test No. 1—Flotation

Results:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·15	100·0
Concentrate	0·8	3·30	2·0
Middling	1·5	0·20	2·0
Tailing	97·7	0·124	80·5

A sample of the flotation tailing was screened on 200 mesh.

+200	72·0	0·11	63·8
−200	28·0	0·16	36·2

Average flotation tailing—Au 0·124 oz./ton.

Test No. 2—Flotation

Results:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·178	100·0
Concentrates	2·6	1·82	26·6
Middling	1·4	0·90	7·1
Tailing	96·0	0·123	66·3

A sample of the flotation tailing was screened on 200 mesh.

+200	68·8	0·11	61·7
−200	31·2	0·15	38·3

Average flotation tailing—Au 0·123 oz./ton.

Test No. 3—Flotation

In this test the sample was water-washed before flotation.

Results:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Heads	100·0	0·11	100·0
Concentrate	6·5	0·84	48·8
Middling	2·1	0·16	3·0
Tailing	91·4	0·059	48·2

A sample of the flotation tailing was screened on 200 mesh.

+200	70·8	0·05	60·2
−200	29·2	0·08	39·8

Average flotation tailing—Au. 0·059 oz./ton.

Test No. 4—Table Test

For this test a quarter-deck Wilfley table, fitted with slime riffles was used.

Two products only were taken from the table, namely, a concentrate and a tailing. What would ordinarily be a middling product was allowed to report with the tailing.

The average feed rate was 3,098 grammes per minute.

Samples of the feed to the table, table concentrate and tailing were taken at 10-minute intervals. All samples were taken wet and filtered before drying to avoid inaccuracies due to dissolved gold.

SUMMARY AND CONCLUSIONS

By flotation on a sample assaying Au 0·18 ounce per ton a recovery of 26·6 per cent of the gold was obtained in a concentrate assaying Au 1·82 ounces per ton with a ratio of concentration of 38·5 : 1. Also on a sample assaying Au 0·11 ounce per ton practising water-washing to remove the cyanide and soluble lime before flotation, a recovery of 48·8 per cent of the gold was obtained in a concentrate assaying Au 0·84 ounce per ton, with a ratio of concentration of 15·4 : 1.

By tabling a sample assaying Au 0·156 ounce per ton a recovery of 21·6 per cent of the gold was obtained in a concentrate assaying Au 0·524 ounce per ton and with a ratio of concentration of 16·1 : 1.

From the preceding tests on this product it is evident that flotation after water-washing would produce the best metallurgical results. From an economic standpoint we are inclined to favour table concentration.

As the tabling may be done in cyanide solution, it eliminates the necessity of filtration and avoids an excess of cyanide solution. While it is true that the recovery by tabling was only 21·6 per cent this recovery includes the coarser refractory sulphides.

It is proposed that the table concentrates be subjected to a special regrinding and agitation, then repassed to the main circuit. It is suggested that after regrinding the table concentrates be returned to the table, thus retaining in the grinding circuit any sulphide particle until sufficiently fine to report with the table tailing.

The ratio of concentration of 16 : 1 might be reduced to say 12 : 1 thereby including some sulphides unfreed from gangue that would ordinarily be a middling product. The introduction of this step would depend largely on the capacity of the regrinding circuit.

In the table test a feed rate of 3,095 grammes per minute was maintained. This rate is equivalent to 4·9 tons per day for a quarter-deck or 19·6 tons per day for a full-size table. Four double-deck tables would then be sufficient to take the daily tonnage of 150 tons.

It should be borne in mind that a high recovery was not to be obtained at the expense of capacity because the table tailing is not a final product but was to be reground and agitated in the main circuit.

2. Mill Tailing

The product mill tailing was subjected to three flotation tests for the purpose of ascertaining the percentage of gold remaining in the cyanide tailing that could be recovered by flotation.

As the mill tailing contained some cyanide solution and soluble lime and as both of these reagents have a depressing effect on the pyrite, with which the greater part of the gold is associated, tests were made before and after water-washing to remove these reagents.

Medium-scale 100 pounds per hour Flotation Tests

On a sample assaying Au 0·061 ounce per ton, a recovery of 57·6 per cent of the gold was obtained in the flotation concentrate which assayed Au 0·54 ounce per ton, with a ratio of concentration of 15·5 : 1.

After water-washing before flotation 67·9 per cent of the gold was recovered in the concentrate, which assayed Au 0·64 ounce per ton with a ratio of concentration of 15·5 : 1.

EXPERIMENTAL TESTS

Mr. John Dixon was present during, and co-operated in, the following test work.

In order to show the difference in assay value between the washed and unwashed tailing two samples were taken. Each sample is composed of a number of smaller samples taken at 15-minute intervals during one of the tests, No. 1 sample is considered to be representative of 500 pounds; No. 2 of 350 pounds. Both samples were screened on 200 mesh.

Results:

No. 1 Sample after Water-washing

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	5·0	0·08	6·5
−200	95·0	0·06	93·5

Average assay—Au 0·061 oz./ton.

No. 1 Sample without Water-washing

Mesh	Weight, per cent	Assays, Au oz./ton,	Per cent of value, Au
+200	4·8	0·10	6·7
−200	95·2	0·07	03·3

Average assay—Au 0·071 oz./ton.

The difference between the two samples is Au 0·01 ounce per ton which is attributed to dissolved gold included in the mill tailing.

FLOTATION TESTS

Test No. 1—Sample Unwashed before Flotation

Results of Flotation:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Head	100·0	0·046	100·0
Concentrate	4·4	0·46	44·3
Middling	3·9	0·015	1·3
Tailing	91·7	0·027	54·4

A sample of the flotation tailing was screened on 200 mesh.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	4·1	0·07	10·8
−200	95·9	0·025	89·2

Average assay of flotation tailing—Au 0·027 oz./ton.

Test No. 2—Sample Washed before Flotation

Results of Flotation:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Head	100·0	0·049	100·0
Concentrate	4·8	0·68	66·6
Middling	6·7	0·02	2·7
Tailing	88·5	0·017	30·7

A sample of the flotation tailing was screened on 200 mesh.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	3·8	0·08	17·2
−200	96·2	0·015	82·8

Average assay of flotation tailing—Au 0·017 oz./ton.

Test No. 3

FLOTATION TEST IN A CONTINUOUS UNIT

A flow-sheet of the unit was as follows: The mill tailing was passed through a 4-mesh screen, to break the lumps, then fed to a small contact tank set about 15 minutes contact. From the contact tank the pulp was pumped to an Akins classifier. The classifier overflow passed to No. 2 cell of a six-cell Denver flotation machine. A rougher concentrate was taken from cells Nos. 2 to 6 inclusive, which was cleaned in No. 1 cell. The final concentrate was taken from No. 1 cell. The middling repassed through the other cells.

It was intended that the classifier return be reground in a small rod mill in closed circuit with the classifier, but as the feed was too fine for the classifier no return was made and the rod mill was cut from the circuit.

The test was a comparative one for the purpose of comparing the results obtainable by flotation on tailings after water-washing to remove the soluble lime and cyanide, and on the tailings as received.

Two series of samples were taken during the flotation of the water-washed tailing. The purpose of these samples was to determine whether the use of sodium sulphide as an auxiliary flotation reagent was instrumental in increasing the recovery.

The results were as follows:—

Head Sample: Water-washed, Screened on 200 Mesh

Mesh	Weight, per cent	Assay, Au, oz./ton	Per cent of value, Au
+200	5·0	0·08	6·5
−200	95·0	0·06	93·5

Average assay—Au 0·061 oz./ton.

Head Sample: Unwashed, Screened on 200 Mesh

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	4·8	0·10	6·7
−200	95·2	0·07	93·3

Average assay—Au 0·071 oz./ton.

Series No. 1:

Feed water-washed before flotation.

Flotation concentrate	Au 0·70 oz./ton
Flotation tailing	Au 0·032 "

A sample of the flotation tailing was screened on 200 mesh.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	3·2	0·08	8·2
−200	96·8	0·03	91·8

Average assay—Au 0·032 oz./ton.

Series No. 2:

Feed water-washed before flotation.

Flotation concentrate	Au 0·64 oz./ton
Flotation tailing	Au 0·021 "

A sample of the flotation tailing was screened on 200 mesh.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	1·8	0·08	6·7
−200	99·2	0·02	33·3

Average assay—Au 0·021 oz./ton.

Series No. 3:

Feed not water-washed before flotation.

Flotation concentrate	Au 0·54 oz./ton
Flotation tailing	Au 0·028 "

A sample of the flotation tailing was screened on 200 mesh.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	3·7	0·11	14·5
−200	96·3	0·025	85·5

Reagents:

The following reagents were used in their approximate quantities:

Series No. 1.—Feed washed before flotation:
Na_2CO_3, 3·0 lb./ton..........Added to contact tank.
$CuSO_4$, 0·75 lb./ton..........Added to classifier overflow.
Sodium amyl xanthate, 0·10 lb./ton..........Added to classifier overflow.
Pine oil, 0·20 lb./ton..........Added to classifier overflow.
Time of series..........2½ hours.

Series No. 2—Feed water-washed before flotation.
Na_2CO_3, $CuSO_4$, sodium amyl xanthate and pine oil as in Series No. 1.
Na_2S 0·75 lb./ton added to pump after contact tank. Time of series, 2 hours.

Series No. 3—Feed not washed before flotation:
Reagents as in series No. 2. Time of series, 2 hours.

Summary of Results:

Head sample —Au 0·061 ounce per ton.

Series No.	Concentrates Au, oz./ton	Tailings, Au, oz./ton	Recovery, per cent	Ratio of concentration
1	0·70	0·032	50·0	23 : 1
2	0·64	0·021	67·9	15·5 : 1
3	0·54	0·028	57·6	15·5 : 1

SUMMARY

Small-scale Flotation Tests

On a head sample assaying Au 0·046 ounce per ton, a recovery of 44·3 per cent of the gold was obtained in a concentrate assaying Au 0·46 ounce per ton with a ratio of concentration of 23 : 1.

On a sample assaying Au 0·049 ounce per ton practising water-washing before flotation a recovery of 66·6 per cent of the gold was made in a concentrate assaying Au 0·68 ounce per ton with a ratio of concentration of 21 : 1.

CONCLUSIONS

Using given reagents better and more consistent results will be obtained by water-washing the cyanide tailing before flotation.

From the results of the above tests it is evident that sodium sulphide as a flotation reagent is effective as a means of increasing the recovery. It is probable that it acts as a sulphidizer on the fine particles of pyrite that have been pitted by the action of the cyanide during the cyanidation treatment. It has a tendency to lower the grade of concentrate so that the quantity used should be held within fairly narrow limits.

Samples of the table concentrate and the flotation concentrate were forwarded to Mr. Dixon for further experimental work at Kirkland Lake.

Report No. 404

THE TREATMENT OF A GOLD-COPPER ORE FROM GOGAMA, ONTARIO

Alex. K. Anderson

Shipments. Four bags of ore, gross weight 400 pounds, were received by freight at the Ore Dressing and Metallurgical Laboratories on November 10, 1931. These were shipped by J. A. Shannon, Sudbury, Ontario, for C. T. Young, No. 1 Toronto St., Toronto. This sample was said to come from a property in Chester township, west of Gogama, Algoma district, Ontario.

Characteristics of the Ore. The ore consisted of a quartz gangue carrying copper and iron sulphides, No free gold was visible.

Purpose of Experimental Tests. The shipment was made to determine the best method to apply for the recovery of the contained values. The tests indicated that 79 per cent of the gold could be recovered by amalgamation and an additional 17 per cent in a flotation concentrate containing 98 per cent of the copper.

Sampling and Analysis. The entire lot was crushed to pass $\frac{1}{4}$ inch and quartered. One quarter was then ground to pass 8 mesh and again quartered. By further grinding to pass succeeding finer screens with intervening cuts through a Jones riffle sampler, a representative portion −100 mesh was secured for analysis. This showed the shipment to contain 6·21 ounces gold, 4·12 ounces silver per ton, 3·82 per cent copper and 0·05 per cent arsenic. Considerable metallic gold was present, 1·28 ounces per ton remaining on 100 mesh.

Experimental Tests

The investigation included tests by amalgamation, amalgamation followed by flotation concentration and cyanidation of the flotation tailing.

AMALGAMATION

Two representative portions of the ore were dry ground to pass 48 and 100 mesh, and amalgamated with mercury.

Results:

Grinding	Heads, Au, oz./ton	Amalgamation tailing Au, oz./ton	Recovery, per cent
− 48 mesh	6·21	2·94	54·4
−100 mesh	6·21	1·48	76·2

These results indicate that comparatively fine grinding is necessary to obtain a high recovery of gold by amalgamation.

Cyanidation of the tailing from these tests left a residue containing approximately $4.00 in gold. A very heavy cyanide consumption, 26 pounds per ton, resulted making this process impractical.

Cyanidation of the ore without amalgamation also was found to give a high tailing and a heavy cyanide consumption.

AMALGAMATION AND FLOTATION

Test No. 1

A sample of the ore was ground 70 per cent solids in a porcelain mill containing iron balls until 97 per cent passed 200 mesh. After removing the balls, the pulp was transferred to a flotation machine and conditioned for 5 minutes with soda ash equivalent to 4 pounds per ton of ore, 0·10 pound potassium ethyl xanthate, 0·10 pound Aerofloat No. 25 and 0·06 pound cresylic acid were then added and a concentrate removed.

The tailing from flotation was de-watered and cyanided 1 : 3 dilution for 24 hours with a 4 pound per ton KCN solution. Lime, 9 pounds, was added for protective alkalinity.

Product	Weight, per cent	Assay			Per cent of value		
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
Heads	100·00	3·62	6·21	4·12	100·0	100·0	100·0
Amalgamation tailing	....	....	1·29	1·59	....	79·3†	61·3†
Flotation concentrate	24·17	14·72	4·48	6·25	98·3	17·4	36·7
Flotation tailing	75·83	0·08	0·27	0·11	1·7	3·3	2·0
Cyanide tailing	....	....	0·01	0·03	....	....	....

†Recovered by amalgamation.

Recovered by Cyanidation. Cyanidation of the flotation tailing which has a content of 0·27 ounce gold per ton, reduces this to 0·01 ounce or 20 cents. This gives an additional recovery of 3·2 per cent of the gold which added to that recovered by amalgamation and flotation gives a total recovery of 99·9 per cent. However, there is sufficient copper in the flotation tailing to cause a loss of 10·2 pounds KCN per ton. This would tend to make the cyanide treatment very troublesome due to fouling of solutions and the necessity of cyanide regeneration.

Test No. 2

A second test was made to see if a flotation tailing lower in gold than in Test No. 1 could be made. Apart from the use of different reagents, the test is the same as the preceding one.

Flotation Reagents, lb./ton:

Soda ash	4·0
Sodium amyl xanthate	0·10
Sodium sulphide	1·0
Hardwood creosote	0·10
Pine oil	0·06

Product	Weight, per cent	Assay			Per cent of values		
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
Heads	100·00	3·72	6·21	4·12	100·0	100·0	100·0
Amalgamation tailing	....	....	2·35	1·89	....	62·1†	54·0†
Flotation concentrate	24·48	14·66	9·44	7·28	96·4	37·2	43·3
Flotation tailing	75·52	0·18	0·06	0·15	3·6	0·7	2·7

†Recovered by amalgamation.

In this test, the recovery of gold by amalgamation is lower than in the preceding test. However, the total recovery is higher. The copper recovery dropped leaving more copper in the tailing while the gold in this residue was reduced to $1.20 as against $5.40 in Test No. 1. The gold recovery by amalgamation and flotation is 99·3 per cent.

As the copper in the flotation tailing is higher than in Test No. 1, cyanidation of this product can be depended on to consume more than 10 pounds cyanide per ton.

SUMMARY AND CONCLUSIONS

The tests show that 66 to 79 per cent of the gold can be recovered by amalgamation and an additional saving made by flotation to bring the overall recovery of gold up to 96 to 99 per cent.

Cyanidation of the ore should be avoided as the copper acts as a strong cyanicide.

From 96 to 98 per cent of the copper can be recovered in a rougher concentrate containing 14 per cent copper, 9 ounces gold and 7 ounces silver per ton with a ratio of concentration of approximately 4 : 1. A still higher grade of concentrate could be obtained by cleaning this product, resulting in a cleaner tailing which would require further treatment.

The process recommended for this class of ore is amalgamation followed by flotation. The gold recovered from the amalgam would be shipped as bullion and the copper concentrate shipped to a smelter.

As a safeguard and indicator of recoveries, the tailing from flotation should be passed over concentrating tables to recover any coarse or rusty gold not caught in preceding circuits.

Report No. 405

AMALGAMATION TESTS ON A SAMPLE OF GOLD ORE FROM THE BEAUFOR GOLD MINES, LTD., PASCALIS TOWNSHIP, NORTHERN QUEBEC

J. S. Godard

Shipment. A shipment of 668 pounds of ore was received October 19, 1931. It was consigned by Mr. W. W. Davis, Box 1, Amos, Quebec, and is said to be a representative sample of ore from the property of Beaufor Gold Mines, Ltd., Pascalis township, Quebec.

Characteristics and Assay of the Ore. The ore is a high-grade gold one. Some native gold about 65-mesh size was seen, but the major portion is probably loosely held in the pyrite crystals, which are of various sizes, and are embedded in a gangue, which is chiefly quartz.

The free gold seen during the examination of a number of hand specimens was in quartz, but in close proximity to a number of tourmaline crystals. These latter crystals may possibly have exerted some influence on the free gold deposition.

The head sample assayed gold 1·82 ounces per ton.

Purpose of Experimental Tests. Mr. Davis requested that some amalgamation tests be made for the purpose of ascertaining if a sufficiently good recovery of the gold could be obtained by amalgamation.

EXPERIMENTAL TESTS

Tests Nos. 1, 2, 3, and 4: Amalgamation Tests

For these tests the ore was dry crushed to pass given screens. The amalgamation tailings were screened and the products assayed.

Test No. 1: Ore crushed −18 mesh.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+ 28	28·9	1·64	40·5
+ 35	12·2	1·81	18·9
+ 48	11·9	1·24	12·6
+ 65	10·9	1·10	10·2
+100	10·0	0·83	7·1
+200	11·3	0·59	5·7
−200	14·8	0·40	5·0

Average tailing—Au, 1·17 oz./ton.

Test No. 2: Ore crushed −35 mesh.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+ 48	9·8	1·30	16·8
+ 65	21·9	1·22	35·4
+100	18·0	0·81	19·3
+200	22·3	0·60	17·8
−200	28·0	0·29	10·7

Average tailing, Au 0·76 oz/ton.

Test No. 3: Ore crushed −65 mesh.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	9·9	0·72	21·2
+200	34·5	0·49	50·5
−200	55·6	0·17	28·3

Average tailing—Au, 0·34 oz./ton.

Test No. 4: Ore crushed −150 mesh.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	22·5	0·31	37·3
−200	77·5	0·15	62·7

Average tailing—Au, 0·19 oz./ton.

Summary Tests 1-4:

Head Sample Au, 1·84 oz./ton.

Test	Mesh	Au,oz./ton, average tailing	Recovery, per cent
1	− 18	1·17	36·4
2	− 35	0·76	58·7
3	− 65	0·34	81·5
4	−150	0·19	89·6

Test No. 5

Amalgamation Test in Continuous Unit

In this test the ore dry crushed to −18 mesh was fed to a small rod mill in closed circuit with a drag classifier; 1,500 grammes of mercury was poured into the well of the classifier. The classifier overflow passed over an amalgamation plate 72 inches long with a $\frac{7}{8}$-inch cascade at the centre.

Samples of the feed were taken at 30-minute intervals. Samples of the plate tailing were taken at 15-minute intervals. The plate tailing samples were divided into two parts, No. 1 and No. 2. Some of the plate tailing was floated in batch lots and the flotation concentrate was reground and cyanided. A sample of the plate tailing was tabled on a laboratory

Wilfley table. Two cyanidation tests were made on samples of the plate tailing. In one of these tests the tailing was cyanided direct, in the other test regrinding was practised before cyanidation.

The running time of the unit was 6 hours.

Results:

Head sample for test assayed, Au 1·86 oz/ton.

No. 1 Sample of Plate Tailing

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+ 65	1·6	0·21	1·3
+100	6·3	0·46	10·9
+200	26·6	0·38	38·2
−200	65·5	0·20	49·6

Average assay, No. 1 sample—Au 0·265 oz./ton.

No. 2 Sample of Plate Tailing

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+ 65	2·4	0·24	2·0
+100	10·5	0·43	15·8
+200	28·4	0·39	38·9
−200	58·7	0·21	43·3

Average assay, No. 2 sample—Au 0·285 oz./ton.
Average assay of plate tailing, both samples Au 0·275 oz./ton.
Recovery, 85·1 per cent.

Tabling a Sample of Amalgamation Tailing

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Concentrate	10·7	1·86	74·6
Tailing +100	5·1	0·04	0·8
+200	18·4	0·025	1·7
−200	35·6	0·07	9·3
Slimes	30·2	0·12	13·6

Amalgamation tailing from products—Au 0·27 oz./ton.

Recovery—By amalgamation	85·5 per cent
By tabling	10·8 "
Total	96·3 "

Flotation Test on a Sample of Amalgamation Tailing

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Concentrate	9·4	2·98	94·8
Tailing +100	10·5	0·02	0·7
+200	25·3	0·02	1·7
−200	54·8	0·015	2·8

Amalgamation tailing from products—Au 0·296 oz./ton.

Recovery—By amalgamation	84·20 per cent
By flotation	14·98 "
Total	99·18 "

Cyanidation of a Sample of Flotation Concentrate

The concentrates from a number of flotation tests on the amalgamation tailing were combined and reground in cyanide solution, then diluted to 3 : 1 and agitated 52 hours. The cyanide strength during grinding was KCN 0·075 per cent and the maximum strength during agitation was KCN 0·150 per cent.

Cyanide Tailing Screened on 200 Mesh

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	9·1	0·70	31·7
−200	90·9	0·14	68·3

Average tailing—Au 0·20 oz./ton.

Head sample	Au 1·86 oz./ton
Amalgamation tailing	Au 0·30 oz./ton
Recovery by amalgamation	83·9 per cent
Recovery by flotation of amalgamation tailing	94·8 per cent
Assay of flotation concentrate	Au 2·24 oz./ton
Assay of cyanide tailing	Au 0·20 oz./ton
Extraction by cyanidation	91·1 per cent
Total net recovery 83·9 + (94·8 x 16·1 x 91·1) =	97·8 per cent

Reagents consumed lb./ton of concentrates: KCN 3·23, CaO 13·7

Reagents consumed lb./ton of ore: KCN 0·40; CaO 1·7.

Ratio of concentration 8·1 : 1.

Cyanidation Tests on Amalgamation Tailing

Two cyanidation tests were made on samples of the amalgamation tailing. In No. 1, no regrinding was practised on the amalgamation tailing before cyanidation, but in No. 2, the amalgamation tailing was reground in cyanide solution KCN 0·046 per cent before agitation. In both tests the agitation was done in 2 : 1 pulp, maximum KCN strength 0·05 per cent, and the time was 27 hours.

The head sample to cyanidation, amalgamation tailing, was screened and the products assayed.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	10·5	0·44	4·62
+200	28·0	0·45	12·60
−200	61·5	0·21	12·92

Average assay of amalgamation tailing—Au 0·30 oz./ton.

No. 1 cyanidation tailing was screened.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	8·8	0·10	20·4
+200	27·0	0·08	50·0
−200	64·2	0·02	29·6

Average assay of cyanidation tailing — Au 0·043 oz./ton.

A comparison of the assays of the various screen products before and after cyanidation shows the following extraction of gold to have been made.

Mesh	Extraction, per cent
+100	77·2
+200	82·2
−200	90·5

No. 2 cyanidation tailing was screened on 200 mesh.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	19·1	0·06	41·5
−200	80·9	0·02	58·5

Average assay of cyanidation tailing—Au 0·028 oz./ton.

Summary: Cyanidation of the Amalgamation Tailing

No.	KCN tailing, Au, oz./ton	Extraction, per cent	Extraction, per cent of total Au	Total recovery, per cent	Reagents consumed, lb./ton	
					KCN	CaO
1	0·043	85·7	13·8	97·7	0·76	4·3
2	0·028	90·6	14·6	98·5	0·98	6·4

SUMMARY

Recoveries varying from a minimum of 36·4 per cent at −18 mesh to a maximum of 89·6 at −150 mesh were obtained by amalgamation.

On a larger scale test a recovery of 85·1 per cent was made at about 2 per cent +65 mesh and 63 per cent −200 mesh. A flotation test on the amalgamation tailing recovered 94·8 per cent of the gold in the form of a concentrate assaying, gold 2·98 ounces per ton, and with a ratio of concentration of 11 : 1. The total recovery by amalgamation and flotation was 99·18 per cent. A flotation concentrate assaying gold 2·24 ounces per ton was cyanided with a recovery of 91·1 per cent of the gold. The net overall recovery was 97·8 per cent. Two cyanidation tests were made on the amalgamation tailing. Without regrinding the extraction by cyanidation amounted to 85·7 per cent or 13·8 per cent of the total gold making a total recovery of 97·7 per cent. Regrinding the amalgamation tailing before cyanidation increased the total recovery to 98·5 per cent of which 14·6 per cent was due to cyanidation.

CONCLUSIONS

On this ore ground to 100 mesh, 65 per cent −200 mesh, amalgamation should recover about 85 per cent of the gold. This figure, it is believed, will be that approximately obtainable on a practical scale using plate amalgamation.

The tests show that an additional 12 to 13 per cent of the gold may be recovered by practising either cyanidation of the amalgamation tailing, or flotation of the amalgamation tailing and cyanidation of the flotation concentrates. Metallurgically there is but little difference which of the latter two methods may be chosen.

Report No. 406

EXPERIMENTAL TESTS ON TWO SAMPLES OF GOLD ORES FROM THE RENO GOLD MINES, LTD., SALMO, NELSON MINING DIVISION, B.C.

J. S. Godard

Shipment. A shipment weighing 85 pounds and consisting of equal weights of sulphide and oxidized ores was received December 2, 1931, from Mr. T. J. Mateer, Superintendent for Reno Gold Mines, Ltd.

Characteristics and Analyses of the Ores. Both samples were high-grade gold ores.

In the sulphide ore about 58 per cent of the gold is free and amalgamable at −65 mesh. The remainder is associated with the sulphides which constitute about 37 per cent of the ore. The sulphides present in their quantitative order are pyrite, zincblende, galena, pyrrhotite, chalcopyrite, and arsenopyrite. The gangue material is mainly quartz.

In the oxidized ore there is a considerable quantity of free gold in a coarser state than in the sulphide ore. Much of this free gold is of a rusty nature, which does not lend itself readily to plate amalgamation. The remainder of the gold appears to be associated with deeply oxidized sulphides, which resemble limonite in appearance. The gangue is a hard rusty-coloured quartz.

The analyses of the two ores are as follows:

Ore	Au, oz./ton	Cu, per cent	Pb, per cent	Zn, per cent	Fe, per cent	S, per cent	Insol., per cent
Sulphide	3·25	0·06	3·30	6·05	11·64	13·47	58·16
Oxidized	5·58	0·15	0·48	0·84	6·83	0·41	86·02

Purpose of Experimental Tests. During the present year a number of experimental tests were made on a sample of sulphide ore from the Reno mine. Mr. Mateer, the present superintendent, requested that an additional test be made on this ore, as well as a series of tests on a composite sample consisting of 50:50 mixture of sulphide and oxidized ore. The present mill feed consists of ore in the proportion of 3 oxidized to 1 sulphide. As the proportion of oxidized to sulphide ore is gradually decreasing it was thought that, with the information gained from two years' milling of oxidized ore, the previous and present tests on sulphide ore, and the present work on the composite sample, the field would be fairly well covered and that mill changes of a permanent nature might be undertaken.

EXPERIMENTAL TESTS

I. Sulphide Ore

Amalgamation at −65 mesh, flotation of the amalgamation tailing and cyanidation of the flotation tailing gave a recovery of 97·6 per cent of the gold. The total recovery was made up as follows: by amalgamation 55·1 per cent; by flotation 40·3 per cent; by cyanidation 2·2 per cent.

AMALGAMATION, FLOTATION, AND CYANIDATION OF THE FLOTATION TAILING

Two tests of the above type were made on the sulphide ore. The ore in each test was dry crushed to −65 mesh before amalgamation. The amalgamation tailing was floated and the flotation tailing sampled and cyanided. In Test No. 1 a smaller quantity of flotation concentrate was purposely removed than in Test No. 2, where as much of the sulphides was floated as was possible.

Test No. 1

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Concentrate	9·1	8·64	63·2
Tailing	90·9	0·503	36·8

Amalgamation tailing—Au 1·24 oz./ton.

The flotation concentrate assayed — Ag 26·68 oz./ton, Pb 37·34 per cent, Zn 18·17 per cent, Cu 0·36 per cent.

A sample of the flotation tailing was screened on 100 and 200 mesh:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	28·0	0·58	52·0
+200	37·1	0·51	37·6
−200	34·9	0·15	10·4

Average flotation tailing—Au 0·503 oz./ton.

Cyanidation of the Flotation Tailing

The flotation tailing was cyanided for 30 hours in 2 : 1 pulp, KCN maximum 0·05 per cent.

The cyanide tailing was screened on 100 and 200 mesh:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	29·2	0·77	61·8
+200	34·4	0·30	28·3
−200	36·4	0·10	9·9

Average cyanide tailing—Au 0·364 oz./ton.
Heads to cyanidation, flotation tailing—Au 0·503 oz./ton.
Extraction, 27·6 per cent.

Test No. 2

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Concentrate	35·9	3·64	89·8
Tailing	64·1	0·233	10·2

Amalgamation tailing—Au 1·46 oz./ton.
The flotation concentrate assayed Pb 8·74 per cent, Zn 30·57 per cent, Fe 19·23 per cent.

A sample of the flotation tailing was screened on 100 and 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	21·9	0·52	48·9
+200	41·7	0·18	32·2
−200	36·4	0·12	18·9

Average flotation tailing—Au 0·233 oz./ton.

Cyanidation of the Flotation Tailing

The details of this part of the test were similar to those in the corresponding part in Test No. 1. The cyanide tailing was screened on 100 and 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	30·6	0·26	65·0
+200	37·9	0·08	24·4
−200	31·5	0·04	10·6

Average cyanide tailing—Au 0·123 oz./ton.
Heads to cyanidation, flotation tailing—Au 0·233 oz./ton.
Extraction, 47·2 per cent.

Summary—Tests Nos. 1 and 2

No.	Tails—Au, oz./ton			Recovery, per cent of total Au				Ratio of conc.	Reagents consumed, lb./ton	
	Amal.	Flot.	Cyan.	Amal.	Flot.	Cyan.	Total		KCN	CaO
1	1·24	0·503	0·364	61·9	24·1	3·9	89·9	11·0 : 1	4·06	8·04
2	1·46	0·233	0·123	55·1	40·3	2·2	97·6	2·8 : 1	2·67	6·90

CONCLUSIONS

The test work on this ore may be considered as complementary to that done previously during the present year and these tests are chiefly concerned with the cyanidation of the flotation tailing.

The results of the test work, Test No. 2, show that 3·9 per cent of the total gold in the head sample was recovered by cyanidation of the flotation tailing. On a head sample assaying gold 3·25 ounces per ton such a step would prove profitable as the operation could be carried out well within the economic limit of 3·9 per cent of 3·25 ounces of gold per ton of ore, roughly $2.50, but the writer is not in a position to state just where the line of profitable operation would be.

In the circumstances it is almost essential that the flotation concentrates be treated at the property and for this purpose cyanidation appears to be the most practical method of treatment. As it is proposed to cyanide both the flotation concentrates and the tailings it is believed that the introduction of flotation into the flow-sheet would be superfluous. Flotation should be attempted only if it alone, or if followed by table concentration, be sufficient to reduce the gold content of the ore to such a point as to permit the discarding of a final tailing, at the same time securing a ratio of concentration of at least 2·5 : 1.

II. Composite Sample

For the following tests the sulphide and oxidized ores were mixed in equal proportions to make the composite sample, which assayed gold 4·52 ounces per ton.

Test No. 1

Amalgamation and Cyanidation of the Amalgamation Tailing

In this test the ore was dry crushed −65 mesh before amalgamation. The amalgamation tailing was sampled and then cyanided in two parts, A and B. In part A, the amalgamation tailing was cyanided direct. In part B, the amalgamation tailing was reground before agitation.

Results:

The amalgamation tailing was screened on 100 and 200 mesh.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	25·5	2·76	43·8
+200	29·6	1·61	29·7
−200	44·9	0·95	26·5

Amalgamation tailing—Au 1·61 oz./ton.

In addition to the gold in the amalgamation tailing there was rusty gold equivalent to 0·42 ounce per ton that would not amalgamate. This brings the amalgamation tailing to 2·03 ounces per ton.

Recovery by amalgamation 55·1 per cent.

Cyanidation of the Amalgamation Tailing

Part A. The amalgamation tailing was cyanided for 46 hours in 2·5 : 1 pulp, KCN maximum 0·075 per cent. The cyanide tailing was screened on 100 and 200 mesh.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	27·2	0·23	34·2
+200	29·7	0·16	25·9
−200	43·1	0·17	39·9

Average cyanide tailing—Au 0·183 oz./ton.
Head sample to cyanidation, amalgamation tailing—Au 1·61 oz./ton.
Extraction, 88·7 per cent.

Part B. The amalgamation tailing was reground in cyanide solution, KCN 0·05 per cent, then diluted to 2·5 : 1 before agitation, KCN strength maximum during agitation was 0·075 per cent and the time was 45 hours.

The cyanide tailing was screened on 200 mesh.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	18·6	0·14	24·2
−200	81·4	0·10	75·8

Average cyanide tailing—Au 0·11 oz./ton.
Head sample to cyanidation, amalgamation tailing—Au 1·61 oz./ton.
Extraction, 93·2 per cent.

Summary:

Head sample—Au 4·52 oz./ton.

Test No.	Recovery per cent total Au			Reagents consumed, lb./ton	
	Amal.	Cyan.	Total	KCN	CaO
1—A	55·1	39·8	94·9	3·39	8·3
1—B	55·1	41·8	96·9	3·87	10·7

Test No. 2

Amalgamation, Flotation, Tabling the Flotation Tailing

A sample of the ore, dry crushed to pass 65 mesh was amalgamated. The amalgamation tailing was floated and the flotation tailing tabled. The rusty gold that did not amalgamate was united with the table concentrate. The table tailing was screened on 100 and 200 mesh.

Results:

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
Flotation concentrate	24·8	4·08	52·7
Table concentrate	15·5	4·96	40·1
Table tailing +100	15·1	0·30	2·4
" +200	16·7	0·19	1·6
" −200	13·9	0·17	1·2
Table slimes	14·0	0·27	2·0

The flotation concentrate assayed Ag 7·36 oz/ton, and the table concentrate assayed Ag 2·48 oz/ton.

The amalgamation tailing assayed Au 1·92 oz/ton.

Summary:

Head sample, Au 4·52 oz./ton.
Recovery by amalgamation, 57·6 per cent.
Recovery by concentration = 92·8 x 42·4 = 39·4 per cent.
Total recovery, Au = 97·0 per cent.

Test No. 3

Amalgamation, Flotation and Cyanidation of the Flotation Tailing

A sample of the ore, dry crushed to pass 65 mesh, was amalgamated, the amalgamation tailing was floated and the flotation tailing was sampled. The remainder of the flotation tailing divided into two parts and each part was cyanided. In part A the flotation tailing was cyanided without regrinding. In part B the flotation tailing was reground before agitation.

Results:

Product	Weight, per cent	Assays, Au, oz./ton,	Per cent of value, Au
Concentrate	20·6	7·28	71·8
Tailing	79·4	0·74	28·2

Amalgamation tailing from products—Au 2·09 oz/ton. Recovery by amalgamation 53·7 per cent.

Rusty gold amounting to 0·44 ounce per ton that did not amalgamate was removed at this stage. When this is added to the amalgamation tailing, the assay value is Au 2·53 ounces per ton, and the recovery is 44·1 per cent.

A sample of the flotation tailing was screened on 100 and 200 mesh.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	22·0	1·84	54·4
+200	32·8	0·51	22·5
−200	45·2	0·38	23·1

Average assay of flotation tailing—Au 0·74 oz./ton.

The recovery of the gold to this stage is:

By amalgamation	44·1 per cent
By flotation, 71·8 x 55·9 =	40·1 "
Total	84·2 "

Cyanidation of the Flotation Tailing

In parts A and B the dilution during agitation was 2 : 1 and the cyanide strength was KCN, maximum, 0·05 per cent and the time was 29 hours in part A, 42 hours in part B.

Results—Part A:

The cyanide tailing was screened on 100 and 200 mesh.

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	31·8	0·54	64·4
+200	31·0	0·15	17·4
−200	37·2	0·13	18·2

Average assay of cyanide tailing—Au 0·27 oz./ton.

Part B:

The flotation tailing was reground in cyanide solution KCN 0·05 per cent before agitation.

The cyanide tailing was screened on 200 mesh.

Product	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	25·1	0·40	64·1
−200	74·9	0·075	35·9

Average assay of cyanide tailing—Au 0·157 oz./ton.

Summary Test No. 3:

		A	B
Head sample	Au, oz./ton	4·52	4·52
Recovery amalgamation and flotation	per cent	84·1	84·1
Head to cyanidation	Au, oz./ton	0·74	0·74
Cyanidation	"	0·27	0·157
Extraction, per cent of gold in flotation tailing		63·5	78·7
Extraction, per cent of total gold		10·1	12·5
Total recovery		94·2	96·6
KCN consumed	lb./ton	1·37	1·89
CaO consumed	"	4·7	8·3

Test No. 4

Amalgamation, Flotation of the Amalgamation Tailing and Cyanidation of the Flotation Concentrates

Because of the high freight and smelter charges, amounting to $22-$26 per ton of concentrates, it is almost essential, that any concentrates that may be obtained either by flotation or tabling, be treated at the mine.

This test was made at the request of Mr. Mateer, for the purpose of obtaining some information on the behaviour of the flotation concentrates from the amalgamation tailing, when subjected to cyanidation.

A quantity of the composite sample was dry crushed to −65 mesh, then amalgamated. The amalgamation tailing was floated in batch lots and the flotation concentrates were combined. Two samples of the concentrate were cyanided. In part A the concentrate was cyanided direct, in part B, it was reground in cyanide solution before agitation.

As in all previous tests on the composite sample, a quantity of rusty gold that would not amalgamate was removed before flotation.

The flotation concentrate, head sample to cyanidation, assayed:—

Au	4·92	oz./ton
Ag	8·00	"
Pb	6·55	per cent
Zn	22·83	"
Fe	19·38	"
Cu	0·21	"

The ratio of concentration was 46 : 1.

Cyanidation of Flotation Concentrate

Part A. The concentrates were agitated for 48 hours in 3 : 1 pulp, KCN maximum 0·20 per cent. The cyanide tailing was repulped and screened on 100 and 200 mesh.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+100	14·2	6·65*	69·9
+200	25·0	0·92	17·0
−200	60·8	0·29	13·1

* Evidently free gold.
Average cyanide tailing, Au 1·35 oz./ton.

Part B. The concentrates were reground in 1 : 1 pulp, KCN 0·10 per cent, then diluted to 3 : 1 before agitation. The KCN strength (max.) was 0·10 per cent, and the time was 67 hours. The cyanide tailing was screened on 200 mesh.

Results:

Mesh	Weight, per cent	Assays, Au, oz./ton	Per cent of value, Au
+200	3·9	0·09	4·4
−200	96·1	0·08	95·6

Average cyanide tailing—Au 0·09 oz./ton.

Summary Test No. 4:

Head sample to cyanidation—Au 4·92 oz./ton.

Test No.	Tailing, Au, oz./ton	Extraction, per cent	Reagents consumed, in lb./ton			
			Of concentrate		Of ore	
			KCN	CaO	KCN	CaO
4—A	1·35	72·4	9·5	19·3	2·06	4·2
4—B	0·08	98·3	13·0	28·2	2·82	6·1

SUMMARY

By amalgamation and cyanidation of the amalgamation tailing a recovery of 96·9 per cent of the gold was obtained.

By amalgamation, flotation, and tabling the flotation tailing 97 per cent of the gold was recovered. The ratio of the concentration was 2·5 : 1.

By amalgamation, flotation, and cyanidation of the flotation tailing a total recovery of 96·6 per cent of the gold was made.

Cyanidation of the reground flotation concentrate extracted 98·3 per cent of the gold.

CONCLUSIONS

Grinding. For good metallurgical results on this sample the ore should be ground to −100 mesh with 70 per cent −200 mesh, and two-stage grinding with classification should be practised.

Amalgamation. Due to uneven distribution of the rusty gold liberated from the oxidized ore portion of the sample some difficulty was experienced in getting good check results. The average of those obtained should closely approximate those obtainable at the size indicated in actual practice using plate amalgamation.

Plate amalgamation using a number of short plates is suggested rather than blanket amalgamation because of the quantity of sulphides present which would tend to fill the pores with sulphides. A fairly steep slope to the plates would keep the sulphides in motion.

Flotation. Flotation tests on the amalgamation tailings showed the ore to divide itself into two natural selections. For the first part of the flotation the following reagents were used: Na_2CO_3 2·0 ounces per ton, water-gas tar A 0·20 pound per ton, Na_2S 0·40 pound per ton, amyl xanthate 0·03 pound per ton and Risor pine oil 0·08 pound per ton. On the completion of this float, the addition of $CuSO_4$ 0·30 pound per ton and amyl xanthate 0·06 pound per ton raised additional sulphides which were mainly zincblende, pyrite, and pyrrhotite. The addition of $CuSO_4$ in the presence of xanthate made the cell more difficult to froth, requiring an additional 0·12 pound per ton of Risor pine oil. Should flotation be introduced on this ore and air cells be chosen it is suggested that the flotation be done in two stages with a contact tank between stages for the mixing of the second group of reagents. While the quantity of sulphides removed in the second stage is less than in the first, the sulphides are slower to respond and hence the stages should be of equal capacity. With mechanical cells a 25-50 per cent over capacity for the tonnages and quantity of sulphides is suggested. In this type of machine the second group of reagents could be added about half way down the series.

Cyanidation. Cyanidation is effective once the free coarser gold is removed. The concentrates cyanide quite readily after regrinding. The reagents consumption is fairly high but might possibly be cut down by shortening the time. Agitation using a plentiful supply of air would probably save cyanide at the expense of lime, by oxidizing some of the soluble sulphides to the sulphate state.

Generally there appears to be two methods of treating such an ore as this one, namely: two-stage grinding in cyanide solution with blanket concentration and cyanidation of the blanket tailing. If single-stage grinding should be practised it is suggested that an agitation period of 12 hours should precede the thickening operation. The second method is plate amalgamation followed by flotation and tabling the flotation tailing. This latter operation will catch the rusty gold and the oxidized gold-bearing sulphides. The table concentrate could be treated in a pan amalgamator with mercury and cyanide. The cyanide solution and the sulphide-oxide material could then be passed to the cyanide circuit for treatment with the flotation concentrates.

The extremely high gold content of the composite sample and the great differences in the nature of the two ores that constitute the sample make the metallurgy, at best, somewhat complicated.

Report No. 407

THE RECOVERY OF GOLD FROM A COPPER-GOLD ORE FROM VIDETTE LAKE, BRITISH COLUMBIA

Alex. K. Anderson

Shipments. Two shipments of ore were received by freight at the Ore Dressing and Metallurgical Laboratories, shipped by Douglas B. Sterrett, Plaza Hotel, Kamloops, British Columbia, and a third was delivered personally by Mr. Sterrett. These were said to have been taken from a copper-gold deposit at Vidette lake, 45 miles north of Savona, B.C.

Shipment No. 1, weighing 95 pounds, was received on July 23, 1931. This sample consisted of thoroughly oxidized surface material, unfit for testing purposes. A second shipment was requested. This was received October 7, 1931, weighing approximately 90 pounds. Shipment No. 3, weighing about 20 pounds, was received on December 13, 1931.

Characteristics of the Ore. Shipments Nos. 2 and 3 on which the test work was done consisted of white quartz impregnated with particles of chalcopyrite. Many of these showed surface oxidation being surrounded by the characteristic green stain of copper carbonate. Much of the quartz was also stained by malachite and red iron oxide.

Purpose of Experimental Tests. The shipper requested that tests be made to determine the recoveries that could be made by amalgamation and concentration. A test made on Shipment No. 1, highly oxidized material, showed that 30 per cent of the gold could be recovered by amalgamation with an additional 24 per cent in a flotation concentrate containing 70·7 per cent of the copper.

Tests on Shipment No. 2, slightly less oxidized material, showed a recovery of 51 per cent of the gold by amalgamation with an additional 36·6 per cent in a copper concentrate assaying 14·5 per cent copper, 4·50 ounces gold per ton.

Tests on Shipment No. 3 showed recoveries of 91 per cent of the gold by amalgamation and flotation with an additional 0·5 per cent by tabling the flotation tailing.

Sampling and Analysis. Shipment No. 2 was crushed to pass ¼ inch and quartered. One quarter was progressively reduced to pass succeeding finer meshes with cuts through a Jones riffle sampler until a representative portion −100 mesh was secured for assay. This showed the shipment to contain 0·62 per cent copper, 0·585 ounce gold and 0·91 ounce silver per ton.

Shipment No. 3 was ground to pass 20 mesh, quartered and a representative portion −100 mesh secured as above. This sample contained 0·96 per cent copper, 0·66 ounce gold, and 1·10 ounces silver per ton.

EXPERIMENTAL TESTS

The investigation included tests by amalgamation at different sizes and amalgamation followed by flotation concentration.

Shipment No. 2

AMALGAMATION

Test No. 1

A representative portion of the ore was dry ground to pass 48 mesh and amalgamated with mercury.

Results:

Heads	Au	0·585 oz./ton
Amalgam tailing	Au	0·38 “
Recovery		35·0 per cent

Test No. 2

A similar portion of the ore was dry ground to pass 100 mesh and amalgamated.

Results:

Heads	Au	0·585 oz./ton
Amalgam tailing	Au	0·28 “
Recovery		50·4 per cent

Test No. 3

Amalgamation and Flotation

A representative 1,000-gramme portion of the ore was ground 70 per cent solids in a porcelain mill containing iron balls until 76 per cent passed 200 mesh. After removing the balls, the pulp was amalgamated with mercury. After removing mercury and amalgam, the pulp was transferred to a flotation machine and conditioned for 5 minutes with 4 pound per ton soda ash and 1 pound sodium sulphide. A concentrate was then taken off following the addition of 0·10 pound Aerofloat No. 25; 0·10 pound potassium ethyl xanthate, and 0·06 pound cresylic acid per ton. The concentrate was not cleaned.

Results:

Product	Weight, per cent	Assay			Per cent of values		
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
Heads	100·00	0·62	0·585	0·91	....	....	....
Amalgamation tails—flotation heads	....	....	0·286	0·96	....	*51·1	....
Flotation concentrate	4·76	14·52	4·50	15·34	90·1	36·6	76·2
Flotation tailing	95·24	0·08	0·075	0·24	9·9	12·3	23·8

* Recovered by amalgamation.

This test shows that amalgamation of the ore ground to pass 76 per cent through 200 mesh gives a recovery of 51·1 per cent of the gold, very little greater than that obtained on −100-mesh material.

Flotation recovers an additional 36·6 per cent of the gold, 76·2 per cent of the silver and 90 per cent of the copper in a concentrate representing 4·76 per cent of the weight of ore milled. This has an assay value of 14·52 per cent copper, 4·50 ounces gold and 15·34 ounces silver per ton.

The combined processes give a total recovery of 87·7 per cent of the gold and a tailing containing $1.50 in gold.

A screen analysis of the flotation tailing shows:

Screen size	Weight, per cent	Assay, Au, oz./ton
+100	0·6	0·05
−100+150	4·8	0·06
−150+200	18·6	0·06
−200	76·0	0·08

These results indicate that there is no object in grinding finer than 100 mesh to liberate the gold.

Test No. 4

In this test, sodium sulphide was omitted; other conditions were the same as in Test No. 3.

Product	Weight, per cent	Assay		Per cent of values	
		Cu, per cent	Au, oz./ton	Cu	Au
Heads	100·00	0·62	0·585	100·0	100·0
Amalgamation tailing	....	....	0·30	....	*49·0
Flotation concentrate	3·98	13·25	5·56	84·6	37·8
Flotation tailing	96·02	0·10	0·08	15·4	13·2

* Recovered by amalgamation.

The results of this test show lower recoveries of gold and copper than those of Test No. 3, indicating that sodium sulphide is beneficial.

Shipment No. 3

This sample which had a value of 0·96 per cent copper, 0·66 ounce gold per ton, was tested in the same manner. In Test No. 4, the flotation tailing was passed over a Wilfley table to note if any additional saving could be made.

Test No. 1

Amalgamation

A representative portion of the shipment was dry ground to pass 100 mesh and amalgamated.

Results:

Heads	Au 0·66 oz./ton.
Amalgam tailing	Au 0·28 oz./ton
Recovery	57·6 per cent

A second test was made similar to the above adding soda ash, 4 pounds per ton, to the pulp during the amalgamation. The results were identical, no benefit being derived nor detrimental effect on the mercury noticed.

Test No. 3

Amalgamation and Flotation

A sample of the ore was ground 70 per cent solids to pass 86 per cent through 200 mesh and amalgamated. After removing amalgam, the pulp was conditioned for 8 minutes with 4 pounds soda ash and 1 pound sodium sulphide per ton. A flotation concentrate was then removed following the addition of 0·10 pound sodium amyl xanthate, 0·06 pound hardwood creosote and 0·06 pound pine oil. The concentrate was not cleaned.

Results:

Product	Weight, per cent	Assay			Per cent of values		
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
Heads	100·0	0·96	0·66	1·10	100·0	100·0	100·0
Amalgamation tailing	....	....	0·323	....	....	*51·0	....
Flotation concentrate	6·2	12·74	4·00	13·12	85·7	37·7	73·9
Flotation tailing	93·8	0·14	0·08	0·20	14·3	11·3	....

*Recovered by amalgamation.

Test No. 4

Amalgamation, Flotation and Table Concentration

This test is similar to the preceding one with the exception that the flotation tailing was passed over a Wilfley table.

Product	Weight, per cent	Assay			Per cent of values		
		Cu, per cent	Au, oz./ton	Ag, oz./ton	Cu	Au	Ag
Heads	100·0	0·98	0·66	1·10	100·0	100·0	100·0
Amalgam tailing	....	....	0·286	....	....	*56·6	....
Flotation concentrate	5·9	14·24	3·84	13·28	87·5	34·4	71·2
Flotation tailing	....	....	0·063	....	....	....	....
Table concentrate	0·6	Nil	0·60	....	....	0·5	....
Table tailing	93·5	0·15	0·06	0·40	12·5	8·5	....

* Recovered by amalgamation.

This test shows higher recoveries by amalgamation than the preceding one. The flotation tailing also is lower indicating higher overall recoveries of gold. Passing the flotation tailing over concentrating tables results in a slight saving. The concentrate, 12 pounds of which is recovered from one ton of mill feed, has a value of $12.00 per ton.

SUMMARY AND CONCLUSIONS

The results obtained in the investigation indicate that for highest recoveries, the ore should be ground to pass 100 mesh. Approximately 50 per cent of the gold can be recovered by amalgamation of this size of material. Flotation concentration of the amalgamation tailing results in recovering an additional 36 per cent of the gold together with 76 per cent of the silver in a concentrate assaying 14·5 per cent copper, 4·50 ounces gold and 15·34 ounces silver per ton. This product represents 4·76 per cent of the weight of ore milled. If a still higher grade of concentrate were desired, this rougher concentrate could be cleaned and the resulting cleaner tailing given additional treatment.

As the investigation calls for tests by amalgamation and concentration no cyanide tests were made to recover the $1.50 remaining in the tailing. The nature of the ore, containing as it does oxidized copper, would indicate the probability of high cyanide consumption entailing costly additions to the plant.

The installation of concentrating tables following flotation will act as a safeguard. Any coarse or rusty gold escaping preceding circuits would be recovered. This concentrate if of sufficient value could be treated in an amalgamation barrel for the recovery of the contained gold.

III

REPORTS OF INVESTIGATIONS: NON-METALLIC MINERALS SECTION

Report No. 408

SCREEN ANALYSIS AND EXAMINATION OF SANDSTONE SAMPLES FROM BEAUHARNOIS CONSTRUCTION COMPANY, BEAUHARNOIS, QUEBEC

L. H. Cole and R. K. Carnochan

Thirteen samples of sandstone, designated A, B, C, D, and Nos. 1 to 9 inclusive were submitted by the Beauharnois Construction Company, P.O. Box 50, Beauharnois, Quebec, for screen analysis and examination.

The samples, each averaging about five pounds, consisted of sandstone about 2 inches in size or larger. As requested each sample was divided into two parts, one part being kept intact while the other was used for the tests.

Preparation. Half the original sample was crushed to pass a 10-mesh screen and then quartered down until a suitable portion was obtained for screen test. Screen tests were made on each sample. Results are given in the accompanying table. (Page 109).

Two hundred grammes of the material was washed by agitating with water and decanting until the water came off clear. The percentage of material lost in washing is also given in the accompanying table.

After washing the material was dried and quartered down. Results of screen tests made on each sample are shown in the table.

Microscopic Examination. A microscopic examination made on each of the washed samples showed them to be so similar in character that one description covers them all. Sample A. was therefore taken for detailed examination. The washed sample was screened and the product on each screen examined separately. The material from 10 to 28 mesh was composed principally of agglomerates of small, well rounded grains of quartz cemented together with a siliceous bond and the voids filled with calcareous material. The natural grain of the sand runs from 35 to 65 mesh and it is only when the 65 mesh is reached that the grain clusters are entirely absent. The material finer than the 65 mesh is all angular being composed of broken grains and the crushed bonding material.

It is probable that the original sand from which the sandstone was formed consisted of well rounded uniformly sized grains between 35 and 65 mesh and the sandstone is such that on crushing the fracturing does not always release the rounded grains, but in many cases the fractures go through the grain as well as the bonding material. In consequence it is not possible to produce a sand composed entirely of rounded grains and the product obtained is made up of a mixture of rounded and angular particles.

Screen Analysis

Samples from Beauharnois Construction Company

—	A		B		C		D		1		2		3		4		5		6		7		8		9	
	C	W	C	W	C	W	C	W	C	W	C	W	C	W	C	W	C	W	C	W	C	W	C	W	C	W
Retained on:																										
10 mesh..........	0·5	0·9	0·2	0·8	0·6	1·1	0·4	0·6	0·1	0·8	0·8	0·5	0·4	0·6	0·2	0·3	1·8	0·4	0·5	0·3	0·4	0·5	0·3	0·3	0·5	1·0
14 "	7·7	7·9	9·7	11·5	10·8	13·9	6·7	9·6	6·6	8·7	9·3	11·0	7·7	9·0	7·4	6·8	6·6	7·0	6·4	8·1	6·6	8·2	6·5	7·7	8·4	11·6
20 "	8·2	8·4	9·6	11·5	10·0	12·3	7·3	10·9	7·2	8·9	9·0	10·2	7·8	9·7	7·7	7·0	8·3	8·9	7·3	8·7	7·8	8·2	7·0	8·1	10·8	12·7
28 "	8·2	8·7	8·8	10·9	9·2	11·9	7·5	9·8	6·7	8·7	8·5	11·3	8·1	10·0	6·8	7·0	9·3	10·3	7·1	9·1	6·5	7·6	6·5	8·5	11·0	13·7
35 "	17·5	20·8	15·2	17·5	15·2	18·7	15·8	16·8	13·3	15·9	16·2	19·4	13·7	16·4	13·0	16·6	15·7	17·3	13·9	17·3	13·9	17·1	12·9	17·5	22·3	20·0
48 "	18·6	23·7	16·6	20·6	15·6	17·8	17·8	19·5	20·1	22·9	16·8	19·2	15·7	18·6	18·1	22·3	16·6	19·2	15·6	20·3	23·2	22·7	19·3	22·9	13·1	16·9
65 "	17·1	20·1	17·1	17·8	14·7	14·3	18·2	18·6	22·5	22·3	15·6	16·6	17·3	19·6	23·6	24·9	17·8	20·5	19·1	19·8	16·3	22·1	24·7	21·5	16·6	14·9
100 "	9·9	7·2	9·9	7·3	9·7	7·2	11·3	10·6	9·6	8·9	9·4	8·6	12·6	11·0	8·8	11·9	10·7	11·6	9·9	11·2	9·0	10·1	10·0	10·4	8·1	6·6
150 "	4·2	1·9	4·3	1·7	5·2	2·3	5·5	3·1	4·3	2·3	4·0	2·5	4·9	3·9	4·5	2·7	4·5	3·6	5·3	4·0	5·5	2·9	6·8	2·7	4·0	2·1
200 "	2·3	0·3	2·3	0·3	2·4	0·4	2·6	0·4	2·4	0·4	2·5	0·5	2·7	0·8	2·7	0·4	2·5	0·8	3·4	0·8	2·2	0·5	1·6	0·3	1·0	0·4
Through 200 mesh...	5·8	0·1	6·3	0·1	6·6	0·1	6·9	0·1	7·2	0·2	7·9	0·2	9·1	0·4	7·2	0·1	6·2	0·4	11·5	0·4	8·6	0·1	4·4	0·1	4·2	0·1

C.=Crude material crushed to pass 10-mesh screen.
W.=Crude material washed and dried.

PERCENTAGE OF CRUDE LOST IN WASHING TEST

—	A	B	C	D	1	2	3	4	5	6	7	8	9
Per cent........................	12·8	14·9	13·9	15·0	15·2	15·8	16·0	15·5	13·0	17·0	16·5	15·4	11·1

The material retained on each screen reacts strongly with hydrochloric acid and the fine material below 150 mesh carries a high percentage of calcareous material. All of the samples reacted with hydrochloric with the exception of sample D, and in this case virtually no effervescence was noticeable.

The material on all the coarser screens contained numerous particles of dark minerals, probably hornblende and argillaceous substance, and many of the groups of grains are heavily iron-stained. These impurities persist throughout the products on all the finer screens but not to such a noticeable degree.

CONCLUSIONS

1. The amount lost on washing is high but not excessive. However, to produce a sand from these sandstones it would be necessary to crush to a finer mesh than 10 mesh and this would increase the fines, and since washing in a commercial plant would be more violent than the laboratory washing the loss would be still greater.

2. There is an appreciable amount of calcareous material in the samples and while a certain portion of this enters the fines and is removed by washing, there is still calcareous material present in the washed sand as evidenced by the acid test.

3. The amount of impurities such as iron, etc., is probably too high to allow the sand produced from this sandstone to be used even for ordinary grades of glass, and the only possible use in glass-making would be for the poorer grades of dark green glass bottles. Since such sand is a low-priced commodity any method to remove the impurities would make the cost of the sand to the consumer prohibitive.

4. It is possible that the sand might be used for steel foundry work, but only in the case of not being able to obtain a better grade sand free from lime, and since such sands are now available on the Montreal market the marketing of this sand for such a purpose is not hopeful. The size of grain is also a little finer than is customary for such use.

5. From the examination of these samples, therefore, there does not seem to be much hope of obtaining a market of sufficient size for the product to make a profitable industry.

Report No. 409

THE CRUSHING OF GLAUBER'S SALT FROM THE HORSESHOE LAKE MINING CO., LTD., ORMISTON, SASK.

R. K. Carnochan

Shipment. A box containing 108 pounds of Glauber's salt was received on May 15, 1931, from the Horseshoe Lake Mining Co., Ormiston, Sask.

Purpose of Experimental Tests. Tests were desired to determine if it would be possible to crush the Glauber's salt to 2 mm. size (9 mesh approximately) commercially.

Characteristics of Glauber's Salt. The salt ranged in size from pieces 3 inches in diameter down to about $\frac{1}{4}$ inch. There were only a small number of the large pieces, the bulk of the material being about $\frac{3}{4}$-inch in size. A few of the large pieces contained blue clay as well as salt; one piece observed, appeared to be about 60 per cent clay. The salt contained sufficient moisture to wet the surfaces of all the pieces.

EXPERIMENTAL TESTS

Jaw Crusher. The salt was first cut in halves by means of a Jones riffle. One half, 52 pounds, was crushed in a small 2- by 6-inch jaw crusher in $2\frac{1}{2}$ minutes to about $\frac{1}{2}$ inch. The crusher worked without any difficulty and judging by the feel of the crushing plates there did not seem to be any temperature rise.

Small Rolls. The product from the jaw crusher was fed to a small set of 8- by 5-inch rolls. On the size of material fed, $\frac{1}{2}$ inch, these rolls did not work at all well. They crushed a small amount and then blocked. The blocking being caused by the larger pieces of salt slipping on the rolls against their motion and so not entering the rolls to be crushed. Any salt that did pass the rolls crushed fine but caked badly.

Hammer Mill. The material that could not be crushed in the small rolls was crushed in a small Sturtevant No. 00 hammer mill fitted with $\frac{1}{8}$-inch grates in 10 minutes.

The product from the hammer mill was very fine, about 20 mesh, and damp enough that when pressed in the hand it held its shape.

The mill was difficult to feed, a stick having to be used to force the feed in. When the mill was opened all the grates were tightly blocked. Apparently as the salt is pushed into the feed opening, it is ground and pressed through the grates.

A quantity of salt, 27 pounds, was put through the same hammer mill fitted with $\frac{1}{2}$-inch grates in 2 minutes. The product was the same as from the $\frac{1}{8}$-inch grates, and the feeding was just as difficult.

Large Rolls. A set of 12-inch by 12-inch rolls was tried out on a few pieces of salt and it was found that the larger pieces of salt rode the rolls unless the rolls were opened up. The remaining salt, 28 pounds, which had dried somewhat by standing in a tub for 3 days, was then put through the rolls set $\frac{1}{2}$ inch apart; the product was again put through the rolls, set almost touching, in 3 minutes.

After having been put through the rolls twice the salt was almost all -6 mesh. No difficulty was experienced in operating the rolls but the feed should be regular, otherwise too much salt enters the roll at one time and gives a caked product.

The -6-mesh salt was crushed in the 8-inch by 5-inch rolls. At first the rolls were set tight but this gave too much caked material. The rolls were gradually opened up till only a small amount of caked material was obtained. At this setting the remainder of the -6 mesh, 22 pounds, was run through the rolls in 4 minutes.

The product obtained was almost all -8 mesh, except a small amount of caked material.

Report No. 410

THE PREPARATION OF SILICA SAND FROM THE RIVER DENYS SAND AND CLAY CO., LTD., MELFORD, CAPE BRETON, N.S.

R. K. Carnochan

Shipment. Three bags of sand were received on September 25, 1930, from the River Denys Sand and Clay Co., Melford, Inverness county, Cape Breton, N.S. The shipment consisted of one bag of coarse sand, 92 pounds net weight, and two bags of fine sand, 168·5 pounds net weight.

Purpose of Experimental Tests. Tests were desired on both the coarse and fine sand to determine if sand suitable for making clear glass or for sand-blasting could be prepared from them.

Characteristics of the Sand. Both lots of sand were sharp and contained quite an amount of light-coloured clay, and a little white mica. The coarse sand ranged in size from about 6 mesh down and the fine from about 10 mesh.

Sampling and Analysis. A head sample was cut out of each lot by means of a Jones riffle having slots 1 inch wide. In cutting the fine sand a piece of rusty agglomerated sand and a piece of rock would not pass through the 1-inch slots. These weighed 126·3 grammes or 0·28 pound. They were kept separate and called +1 inch from fine sand.

Analysis gave:

	Fe_2O_3, per cent	Per cent of heads
Head sample coarse sand	0·31	100·0
Sample fine sand −1 inch	0·19	99·83
+ 1 inch from fine sand	15·49	0·17

EXPERIMENTAL TESTS

The test work was divided into two parts, the first being preparation of glass sand and the second, preparation of blast sand.

Preparation of Glass Sand

Four different methods were tried for the preparation of glass sand. These were washing; washing and magnetic separation; washing and tabling; and washing, tabling and magnetic separation.

Washing. Small samples of the coarse and fine sand were agitated for 1 hour in a revolving porcelain jar. The jar contents were washed by decantation from a pail three times. The washed sand and decanted sand and clay were dried. Screen tests were made on cut samples of the washed sands from each lot.

Product	Coarse lot			Fine lot		
	Grms.	Per cent of heads	Fe_2O_3 per cent	Grms.	Per cent of heads	Fe_2O_3 per cent
Feed	1,140·2	100·0	0·31	1,137·9	99·83	0·19
Washed sand	1,062·0	93·14	0·13	994·5	87·25	0·11
Decanted sand and clay	74·0	6·49	1·27	136·7	11·99	0·90

Screen Tests on Washed Sands

Size	Coarse lot			Fine lot		
	Weight, per cent	Per cent of heads	Fe_2O_3, per cent	Weight, per cent	Per cent of heads	Fe_2O_3 per cent
+ 20	10·61	9·88	1·43	2·04	1·78	2·42
− 20+ 28	10·90	10·15	0·076	3·99	3·48	0·12
− 28+ 35	44·88	41·80	0·070	25·69	22·42	0·040
− 35+ 48	21·20	19·75	0·086	36·00	31·41	0·034
− 48+ 65	7·96	7·41	0·16	20·99	18·31	0·074
− 65+100	2·98	2·78	0·21	7·90	6·89	0·17
−100	1·47	1·37	1·43	3·39	2·96	1·14
	100·00	93·14	0·250	100·00	87·25	0·144

Sand for making clear glass should not contain over 0·06 per cent Fe_2O_3 plus TiO_2. Only two of the sizes in the screen tests were 0·06 per cent or less Fe_2O_3 alone without considering TiO_2. These sizes were −28 + 35 and −35 + 48 in the washed sand from the fine lot. These sizes were not run for TiO_2 because the washing, tabling, and magnetic separation tests on the fine lot gave more than 0·06 per cent Fe_2O_3 plus TiO_2 in both these sizes. Hence it was concluded that in a test using washing only these two sizes would be over the required limit.

Washing alone will not give any products suitable for making clear glass.

Washing and Magnetic Separation. Cut samples of the washed sand of each lot were run over an Ullrich magnetic separator using a current of 10 amperes. The Ullrich removes tourmaline, magnetite, grains of quartz with tourmaline attached, flakes of white mica with tourmaline embedded in them, and also a very small amount of other minerals amongst which is garnet. In the non-magnetic are very fine grains of tourmaline both free and attached, discoloured quartz grains, and white mica. Screen tests were made on the non-magnetic from each lot.

Product	Coarse lot		Fine lot	
	Grms.	Per cent of heads	Grms.	Per cent of heads
Ullrich non-magnetic	474·3	92·90	462·4	86·84
Ullrich magnetic	1·2	0·24	2·2	0·41

Screen Tests on Ullrich Non-Magnetic

Size	Coarse lot				Fine lot	
	Weight, per cent	Per cent of heads	Fe_2O_3, per cent	TiO_2, per cent	Weight, per cent	Per cent of heads
+ 8	1·26	1·17	....	....	....	....
− 8+ 10	1·46	1·36	....	....	....	....
− 10+ 14	2·29	2·13	....	....	0·81	0·70
− 14+ 20	5·16	4·79	....	....	0·97	0·84
− 20+ 28	10·58	9·83	0·038	0·014	3·74	3·25
− 28+ 35	44·35	41·20	0·029	0·015	23·32	20·25
− 35+ 48	22·52	20·92	....	....	35·74	31·04
− 48+ 65	7·91	7·35	....	....	21·29	18·49
− 65+100	3·03	2·81	....	....	10·44	9·07
−100	1·44	1·34	....	....	3·69	3·20
	100·00	92·90	....	....	100·00	86·84

Only the two sizes −20 + 28 and −28 + 35 from the coarse lot were analysed for Fe_2O_3 and TiO_2 as the washing, tabling and magnetic separation test showed only these two sizes below the required limit, and it was concluded that any size over the limit in the washing, tabling and magnetic separation tests would be over the limit in a washing and magnetic separation test.

The two sizes −20 + 28 and −28 + 35 from the coarse lot could be used to make clear glass. Together they represent 51·03 per cent of the coarse lot.

Washing and Tabling. A small cut sample of each lot of sand was agitated for one hour in a revolving porcelain jar and was then washed three times by stirring in a pail, letting the pail contents stand for one minute and decanting.

The washed sand was then dried and run over a small Wilfley table making two products. The product reporting higher up on the table was called table concentrates and consisted mostly of sand. In these table concentrates were some large grains of quartz, rock and rusty agglomerated sand as well as some tourmaline and a small amount of magnetite. The product reporting lower down was called tabled sand and contained no large grains, a small amount of fine tourmaline and magnetite and some white mica. Screen tests were made on the tabled sands from each lot.

Product	Coarse lot				Fine lot		
	Grms.	Per cent of heads	Fe_2O_3, per cent	TiO_2, per cent	Grms.	Per cent of heads	Fe_2O_3, per cent
Feed	1,290·7	100·00	0·31	....	1,175·4	99·83	0·19
Washed sand	1,217·3	94·31	....	....	1,056·7	89·75	....
Table concentrate	86·5	6·70	1·00	....	93·6	7·95	0·85
Tabled sand	1,125·7	87·22	0·052	0·032	959·2	81·47	0·076

Screen Tests on Tabled Sands

Size	Coarse lot				Fine lot			
	Weight, per cent	Per cent of heads	Fe_2O_3, per cent	TiO_2, per cent	Weight, per cent	Per cent of heads	Fe_2O_3, per cent	TiO_2, per cent
+ 10	1·13	0·98	0·19	....	....	....	....	....
− 10+ 14	2·49	2·17	0·078	....	0·29	0·24	1·35	....
− 14+ 20	5·63	4·91	0·064	0·019	1·25	1·02	0·25	....
− 20+ 28	13·50	11·77	0·049	0·024	4·48	3·65	0·08	....
− 28+ 35	47·53	41·46	0·033	0·024	28·26	23·02	0·048	0·021
− 35+ 48	19·36	16·89	0·042	0·029	37·51	30·56	0·048	0·035
− 48+ 65	6·21	5·42	0·08	....	18·37	14·97	0·062	0·045
− 65+100	2·12	1·85	0·14	....	6·14	5·00	0·09	....
−100	2·03	1·77	0·45	....	3·70	3·01	0·37	....
	100·00	87·22	0·055	....	100·00	81·47	0·073	....

Only the one size, −28 + 35 from the coarse lot, is low enough in iron and titanium to allow it to be used for making clear glass. This size represents 41·46 per cent of the coarse lot.

Washing, Tabling and Magnetic Separation. Cut samples of the tabled sand from each lot were run over an Ullrich magnetic separator using a very strong current, 10 amperes. The separation secured was similar to that described under washing and magnetic separation. Screen tests were made on the non-magnetic from each lot.

Product	Coarse lot			Fine lot		
	Grms.	Per cent of heads	Fe_2O_3, per cent	Grms.	Per cent of heads	Fe_2O_3, per cent
Feed	555·7	87·22	0·052	483·2	81·47	0·076
Ullrich non-magnetic	554·7	87·06	0·065	482·2	81·30	0·08
Ullrich magnetic	0·5	0·08	18·80	0·5	0·08	14·25

Screen Tests on Ullrich Non-Magnetic

Size	Coarse lot				Fine lot			
	Weight, per cent	Per cent of heads	Fe_2O_3 per cent	TiO_2, per cent	Weight, per cent	Per cent of heads	Fe_2O_3, per cent	TiO_2, per cent
+ 10	1·43	1·24	0·13	....	....	....	....	....
− 10+ 14	2·51	2·18	0·065	....	0·45	0·37	1·14	....
− 14+ 20	5·75	5·01	0·049	0·013	1·22	0·99	0·18	....
− 20+ 28	14·09	12·27	0·040	0·013	5·42	4·41	0·08	....
− 28+ 35	49·98	43·51	0·037	0·015	32·92	26·76	0·057	0·020
− 35+ 48	16·78	14·61	0·049	0·022	33·47	27·21	0·057	0·025
− 48+ 65	5·42	4·72	0·069	....	16·57	13·47	0·07	....
− 65+100	2·00	1·74	0·12	....	6·24	5·07	0·10	....
−100	2·04	1·78	0·32	....	3·71	3·02	0·29	....
	100·00	87·06	0·051	....	100·00	81·30	0·078	....

Only two sizes of the coarse lot are low enough in iron and titanium to allow them to be used for making clear glass. These sizes are the −20 + 28 and the −28 + 35 and together they represent 55·78 per cent of the heads.

SUMMARY OF TESTS FOR THE PREPARATION OF GLASS SAND

It will be noticed that the analyses of some of the products that were screened have been calculated from the analyses of the screened sizes, and that the analyses obtained in this way do not always check closely the analyses obtained from the products before screening. Also, if the analysis of a separated product is calculated from the analyses of the products into which it was separated the value obtained does not always approximate the value obtained by direct analysis. Moreover, if the analyses of the prepared sizes are compared, small errors are revealed by the fact that some sizes give lower iron values by washing and magnetic treatment or by washing and tabling than by washing, tabling and magnetic treatment, and also that some sizes in the fine lot give lower iron values by washing than by washing and tabling and washing, tabling and magnetic separation.

However, the results have been put down just as obtained and it is believed that they are sufficiently correct for the purpose intended, besides this method will give a good idea of the accuracy of the work.

No sand suitable for making clear glass was obtained from the fine lot. The following table gives the percentage of sand suitable for making clear glass obtained by the different methods of treatment from the coarse lot,—

Method of Treatment	Glass sand recovered, per cent
Washing	0·00
Washing and magnetic treatment	51·03
Washing and tabling	41·46
Washing, tabling and magnetic treatment	55·78

Considering that magnetic treatment by means of a high power machine, similar to the one used in the tests, is expensive and that glass sand is a cheap commodity, it would seem advisable to remove deleterious material by simply washing and tabling the coarse sand. The recovery by this method would be 41·46 per cent.

Preparation of Blast Sand

The remaining sand of the coarse lot and the remaining −1-inch sand of the fine lot were each washed four times by means of an Akins classifier. The four washes in the Akins would be equivalent to the treatment usually given when washing a sand. The washed sands were dried and screened on a 20-mesh Hummer screen. The following table gives the results of this work,—

Product	Coarse lot		Fine lot	
	Pounds	Per cent of heads	Pounds	Per cent of heads
Sand before washing	88	100·00	160	99·83
Sand after washing	81	92·05	140	87·35
+20 mesh	14	15·91	7	4·37
−20 mesh	67	76·14	133	82·98

A screen test was made on each of the −20-mesh products with the following results,—

Screen size, mesh	Coarse lot, per cent	Fine lot, per cent
+ 20	0·10	0·02
− 20+ 28	10·32	2·25
− 28+ 35	51·21	29·72
− 35+ 48	24·34	37·84
− 48+ 65	9·00	19·63
− 65+100	3·28	8·12
−100	1·75	2·42
	100·00	100·00

Sixty-five pounds of the −20 from the coarse lot and 100 pounds of the −20 from the fine lot were shipped to the C.P.R. at Montreal, where they were used in tests with a regular sandblast outfit in the Angus shops.

Besides testing the River Denys sand the C.P.R. tested several other Canadian sands and an imported sand. The results of these tests have been published by the Mines Branch in Report No. 727-1, entitled "The Suitability of Certain Canadian Sands for Use in Sandblasting."

The sandblasting tests showed that both the lots of River Denys sand that were tried were good blast sand.

CONCLUSIONS

1. No glass sand suitable for making clear glass can be obtained from the fine lot.
2. By washing, tabling and screening 41·46 per cent of the coarse lot can be recovered as sand suitable for making clear glass.
3. By washing and screening 76·14 per cent of the coarse lot and 82·98 per cent of the fine lot can be recovered as good blast sand.

Report No. 411

THE TESTING OF SILICA SAND FROM GUIGUES TOWNSHIP, TÊMISCAMINGUE COUNTY, QUEBEC, FOR SANDBLASTING

R. K. Carnochan

Shipment. A carload of silica sand, shipping weight 47,100 pounds, was received on October 11, 1930. The carload had been sent in by the Development Department of Canadian Pacific Railway, Montreal. The sand came from a property in Guigues township, Têmiscamingue county, Quebec, on the east side of lake Timiskaming.

Purpose of Experimental Tests. The C.P.R. desired that the sand be prepared and samples sent to them to try out in a regular sandblasting outfit at their Angus shops in Montreal in comparative tests with an imported sand and other Canadian sands.

Characteristics of the Sand. The sand ranges in grain size from over 4 mesh to under 100 mesh. The colour of the sand is light buff. Most of the sand is composed of silica grains but small amounts of clay and iron oxide are present.

EXPERIMENTAL TESTS

The test work was divided into several parts. These were washing, drying, screening, and preparation of samples for sandblasting.

Washing. In order to remove clay, iron oxide, and fine sand, the whole carload of sand was washed by feeding the crude sand with plenty of water through a $\frac{5}{8}$-inch screen to a 2-inch Wilfley centrifugal pump, which discharged into a 24-inch Akins classifier. As the sand was only given one treatment in the Akins instead of the customary four treatments in screw flight washers, as much wash water as possible was used in order to secure a clean sand.

The maximum rate of feed obtained during the washing was 1,500 pounds per hour.

A sample of the sand fed to the washing circuit was obtained by taking a spoonful out of each tubful of sand before feeding. This sample when wet weighed 50 pounds and when dried weighed 47·5 pounds. This indicates 5 per cent moisture in the feed. The dried sample contained 3·7 per cent −100 mesh.

During the washing a small amount of material was picked off the $\frac{5}{8}$-inch screen by hand and kept separate. This $+\frac{5}{8}$-inch when dried weighed 92 pounds.

Samples of the Akins oversize and overflow were taken during all the washing. These samples were secured by catching a small dishful of the product every 10 minutes. The samples represented runs of about 4 or 8 hours, depending on whether the run lasted a half or a full day.

The oversize samples ran from 1·2 to 2·8 per cent −100 mesh and the overflow samples from 74·6 to 91·8.

The washed sand was clean.

Drying. The washed sand was put into piles on a cement floor and allowed to drain. After standing a few days all the sand in a pile except the bottom 6 inches contained about 3·5 per cent of moisture and was ready to dry. After the removal of the tops of the piles the bottoms were heaped into one pile and allowed to drain out. The bottom of this pile was heaped into a small pile and after the small pile had drained it was dried without separating out the wet bottom.

Some of the sand was dried by putting it in bags and placing the bags on steam coils. This sand was called coil dried.

Some of the sand was dried in pans on a gas-fired dryer. This sand was called pan dried.

During the drying 218 pounds of sand was discarded because it got slightly dirty.

The reason for drying in two different ways was to determine if pan drying injured the sand and made it break up more easily when used for sandblasting.

Screening. The coil dried and pan dried sands were screened separately on a Hummer screen into various sizes. The 20, 26, 35, 42, and 60 mesh screens used on the Hummer have very nearly the same openings as the Tyler standard screens 20, 28, 35, 48, and 65. The following table gives the results of the screening:—

Size	Coil dried		Pan dried	
	Weight		Weight	
	lb.	%	lb.	%
+ 4	6·5	0·08	32·0	0·10
− 4 + 5½	31·0	0·38	378·0	1·18
− 5½+14	1,689·0	20·87	4,947·0	15·50
−14 +20	1,758·0	21·72	8,102·0	25·39
−20 +26	1,660·0	20·51	6,128·0	19·21
−26 +35	1,476·5	18·24	5,781·0	18·12
−35 +42	488·0	6·03	3,183·5	9·98
−42 +60	631·0	7·80	2,564·5	8·04
−60	334·0	4·37	790·0	2·48
Total	8,094·0	100·00	31,906·0	100·00

Preparation of Samples for Sandblasting. Two small samples of an imported silica sand, used extensively for sandblasting, were obtained from the C.P.R. These two samples were of different grades, one having less fines in it than the other. Screen tests were made on both samples with Tyler standard screens. The following table gives the results:—

Size	No. 1 sample	No. 2 sample
	Weight, per cent	Weight, per cent
+ 20	0·02	0·48
− 20+ 28	3·17	37·37
− 28+ 35	56·97	58·34
− 35+ 48	26·89	3·55
− 48+ 65	9·39	0·17
− 65+100	2·90	0·06
−100	0·66	0·03
Total	100·00	100·00

The screen tests show that sample No. 1, which is the regular grade, contains more fine sand than sample No. 2, which is a special grade. Both samples are practically all −20 mesh and sample No. 2 is approximately all +48 mesh.

Lots of 100 pounds each and similar in grain size to the two grades of imported sand were prepared from both the pan and coil dried sands. These lots were made up according to the following schedule:—

Size	No. 1 mix		No. 2 mix	
	lb.	oz.	lb.	oz.
−14+20	0	0	0	8
−20+26	3	3	37	6
−26+35	57	0	38	5
−35+42	26	14	3	9
−42+60	9	6	0	3
−60	3	9	0	1
Total	100	0	100	0

No. 1 mix corresponds to No. 1 sample and No. 2 mix to No. 2 sample.

Lots of 100 pounds each were prepared from the coil dried sand to represent what would have been obtained by screening the washed and coil dried sand on 20 mesh and taking the -20, and also by screening on 20 and 42 and taking the $-20\ +42$. The following schedule gives the weights of each size used to make up the -20 and the $-20\ +42$.

Size	−20		−20+42	
	lb.	oz.	lb.	oz.
−20+26	36	0	45	13
−26+35	32	1	40	12
−35+42	10	9	13	7
−42+60	13	11		
−60	7	11		
Total	100	0	100	0

Lots of 100 pounds each were weighed out from the $-5\frac{1}{2}\ +14$ and the $-14\ +20$ sizes of the coil dried sand.

The following lots of sand, each of 100 pounds, were shipped to the C.P.R. at Montreal for sandblasting tests:—

2 lots of No. 1 mix, pan dried.
1 lot of No. 2 mix, pan dried.
1 lot of No. 1 mix, coil dried.
1 lot of No. 2 mix, coil dried.
1 lot of -20, coil dried.
1 lot of $-20+42$, coil dried.
1 lot of $-5\frac{1}{2}+14$, coil dried.
1 lot of $-14+20$, coil dried.

Sandblasting Tests. The nine lots of Témiscamingue sand were tested in a regular sandblasting outfit by the C.P.R. in their Angus shops at Montreal. Besides the Témiscamingue sand the C.P.R. tested the two grades of the imported sand and several Canadian sands.

The results of the sandblast have been published by the Mines Branch.[1]

CONCLUSIONS

The following conclusions are based on the results of the sandblasting tests.

1. The Témiscamingue sand makes a good blast sand.
2. Drying the sand in pans does not injure it.
3. Some other Canadian sands were found to be good for sandblasting, hence before developing the Témiscamingue sand it would be necessary to consider freight rates as the nearest good sand deposit to any market will have an advantage in that market.

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[1] Mines Branch, Dept. of Mines, Canada: "The Suitability of Certain Canadian Sands for Sandblasting," Rept. No. 727-1.

Report No. 412

THE TESTING OF SILICA SAND AND SANDSTONE FROM BEAUHARNOIS POWER CORPORATION, LTD., BEAUHARNOIS, QUEBEC, FOR SANDBLASTING

R. K. Carnochan

Shipments. Three lots or shipments were received. The first consisted of six bags, shipping weight 600 pounds, and was received January 21, 1931. The second, one bag, shipping weight 170 pounds, arrived January 27, 1931. The third, received on March 4, 1931, contained six bags, shipping weight 670 pounds. All the shipments were from Beauharnois, Quebec, and were sent in by Mr. Herbert Cantwell of the Beauharnois Power Corporation, Ltd., University Tower Building, Montreal, Quebec.

Purpose of Experimental Tests. It was desired that sands suitable for blasting be prepared from the three lots and sent to the C.P.R. at Montreal, Quebec. The C.P.R. agreed to test these sands out in a regular sandblast outfit at their Angus shops in Montreal in comparative tests with two grades of an imported sand and several other Canadian sands.

Characteristics of the Lots. Lot No. 1 consisted of a damp, partly fine crushed sandstone, Lot No. 2 was a buff-coloured sand, and Lot No. 3 was sandstone the same as in Lot No. 1 but in larger pieces.

EXPERIMENTAL TESTS

Each lot was treated separately.

Lot No. 1

A moisture sample was taken on Lot No. 1 and indicated 2·5 per cent moisture present. The whole lot less the moisture sample was washed in an Akins classifier. The washed material was too coarse so it was dried and crushed to all pass 20 mesh by means of a set of rolls and a Hummer screen. The −20 mesh was then washed four times in the Akins classifier and the washed sand was dried and screened by means of the Hummer screen into:—

Size	lb.
−20+26	63
−26+35	140
−35+42	39
−42+60	99
−60	70
Total	411

The screens used on the Hummer 20, 26, 35, 42, and 60 correspond very closely in screen opening to the Tyler standard screens 20, 28, 35, 48 and 65.

Two samples of an imported sand were obtained from the C.P.R. These samples, called No. 1 and No. 2, were respectively the regular grade and a special grade containing less fines than the regular grade. Screen tests with Tyler standard screens were made on these two samples with the following results:—

Size	No. 1 sample, weight, per cent	No. 2 sample, weight, per cent
+ 20	0·02	0·48
−20+ 28	3·17	37·37
−28+ 35	56·97	58·34
−35+ 48	26·89	3·55
−48+ 65	9·39	0·17
−65+100	2·90	0·06
−100	0·66	0·03
Total	100·00	100·00

The screen tests show that sample No. 1, which is the regular grade contains more fine sand than sample No. 2, which is a special grade. Both samples are practically all −20 mesh and sample No. 2 is approximately all +48 mesh.

Two lots of 100 pounds each were prepared from the screened sand. One of these lots called No. 1 mix was similar in grain size to the No. 1 sample of the imported sand and the other lot called No. 2 mix was similar to the No. 2 sample. These lots were made by mixing the screened sizes according to the following schedule:—

	No. 1 mix		No. 2 mix	
	lb.	oz.	lb.	oz.
−20+26	3	3	37	14
−26+35	57	0	58	5
−35+42	26	14	3	9
−42+60	9	6	0	3
−60	3	9	0	1
Total	100	0	100	0

The two lots of prepared sand were shipped to the C.P.R. at Montreal for sandblasting tests.

Lot No. 2

Lot No. 2 contained 170 pounds net of sand, all of which was washed four times in an Akins classifier. The washed sand was dried and screened by means of a Hummer screen into:—

Size	lb.	oz.
+20	13	2
−20+26	5	0
−26+35	67	14
−35+42	14	4
−42+60	38	10
−60	18	0
Total	156	14

As large a lot as possible was made up similar in size to sample No. 1 of the imported sand. This was done by mixing the screen sizes according to the following schedule:—

Size	lb.	oz.
−20+26	1	11
−26+35	30	3
−35+42	14	4
−42+60	5	0
−60	1	14
Total	53	0

A lot of 53 pounds was obtained, all the −35+42 being used. This lot called Lot No. 2, No. 1 mix, was sent to the C.P.R. for sandblasting.

Lot No. 3

Lot No. 3 contained 652·5 pounds net of sandstone, all of which was crushed to 20 mesh by means of gyratory crusher, disk crusher, rolls and Hummer screen. The −20 mesh was washed four times in an Akins classifier. The washed sand was dried and screened on a 42-mesh Hummer screen giving:—

Size	lb.
−20+42	319
−42	219
Total	538

The −20+42 mesh is similar to the No. 2 sample of the imported sand in grain size, and it was all shipped to the C.P.R. for sandblasting.

Sandblasting Tests. Lot No. 1, No. 1 mix, and No. 2 mix, and Lot No. 2, No. 1 mix were tested in a regular sandblasting outfit by the C.P.R. in their Angus shops at Montreal. Besides the Beauharnois sands the C.P.R. tested the two grades of the imported sand and several Canadian sands.

The −20+42 from Lot No. 3 was reserved by the C.P.R. for a large test under commercial conditions. This test has not been made. As Lot No. 1 and Lot No. 3 consisted of the same sandstone, the blasting tests made with the sand from Lot No. 1 are sufficient to determine if the Beauharnois sandstone makes a good sandblast sand or not.

The results of the sandblast tests have been published by the Mines Branch in Report No. 727-1, entitled "The Suitability of Certain Canadian Sands for Use in Sandblasting."

CONCLUSIONS

The following conclusion is based on the results of the sandblasting tests.

Good blast sand can be prepared from both the Beauharnois sandstone and sand.

Report No. 413

THE TESTING OF SYENITE FROM METHUEN TOWNSHIP, PETERBOROUGH COUNTY, ONTARIO, FOR THE REMOVAL OF IRON

R. K. Carnochan

Shipment. A small sample of syenite, net weight 14 pounds, was brought in on May 8, 1931, by Mr. Norman B. Davis of Ottawa. The sample came originally from Methuen township, Peterborough county, Ontario.

Purpose of Experimental Tests. Syenite used in glass-making should run 0·10 per cent or less Fe_2O_3. Mr. Davis desired tests to be made to determine whether a product suitable for glass-making could be obtained from the sample submitted.

Characteristics of the Syenite. The syenite is composed of white feldspars, albite and microcline, and nepheline. A small amount of magnetite is present, and a very little white mica.

EXPERIMENTAL TESTS

The whole sample was crushed to all −20 mesh by means of a small jaw crusher, a small set of rolls and a 20-mesh hand screen. The screen was used after each crushing so as to keep the −20-mesh product as coarse as possible. Two tests were made on the −20-mesh material.

Test No. 1

One quarter of the −20-mesh material was run dry through the Ullrich magnetic separator using a very strong current, 10 amperes. The following products were obtained:—

Non-magnetic	999·0 grammes	0·14 % Fe_2O_3
Magnetic	37·7 "	12·50 % "

This test did not give a product sufficiently low in iron so a second test was made.

Test No. 2

One quarter of the −20-mesh material was screened on 40 mesh and 100 mesh. The three sizes obtained were run dry over the Ullrich magnetic separator using a very strong current.

The following products were obtained:—

− 20+ 40	Non-magnetic	588·5 grammes	0·09% Fe_2O_3
− 20+ 40	Magnetic	39·0 "	9·18% "
− 40+100	Non-magnetic	604·2 "	0·12% "
− 40+100	Magnetic	23·9 "	18·37% "
−100	Non-magnetic	344·7 "	0·17% "
−100	Magnetic	9·7 "	13·48% "

The test gave one product suitable for use in glass-making, this product being the −20 +40 non-magnetic.

CONCLUSIONS

1. In order to obtain a product suitable for glass-making it will be necessary to crush the syenite in a jaw crusher and sets of rolls to 20 mesh, screening after each crushing, the −20 mesh will have to be sized on about 40 mesh and the −20 +40 mesh run over a strong magnetic machine.

2. A product suitable for glass-making equal to 36·5 per cent of the feed was obtained in Test No. 2. This might be increased a little by using a slightly finer screen in place of the 40 mesh. This would give a product running slightly higher than 0·09 per cent Fe_2O_3, but so long as it did not run over 0·10 per cent it would be suitable for glass-making.

Report No. 414

THE SEPARATION OF ROCK FROM CLAM SHELLS FROM DENMAN ISLAND, B.C.

R. K. Carnochan

Shipment. Two bags of clam shell, shipping weight 200 pounds, were received on October 30, 1931, from Mr. Gordon Runkle of Alexander Murray and Company, 7 Alexander St., Vancouver, B.C. The shell came originally from Denman Island, B.C.

Purpose of Experimental Tests. Mr. Runkle wished tests to be made to determine if the rock could be removed from the shell and, if so, the best method of doing it. He stated that he proposed to use the cleaned shell for poultry and pointed out that for such use the shell would be worth only about $10 a ton.

Characteristics of the Shell. The shipment consisted of clam shell, rock, and a little wood. The shell was made up mostly of broken pieces and a few unbroken shells. The unbroken shells ranged in size from 5 inches to 3 inches long, the broken shell from 2 inches to very small pieces. The rock ranged in size from a little over 1 inch to very small pieces and nearly all the pieces were well rounded. The wood was in pieces from 2 inches long down to very small particles, and was all dark and partly decayed.

EXPERIMENTAL TESTS

Different methods of separation were tried out, by far the best results being obtained by the use of a large two-compartment James jig. Two tests were made with this jig.

Test No. 1

A lot of shell and rock, 148 pounds in weight and slightly damp, was screened on 1 inch, $\frac{3}{4}$ inch, and 6 mesh.

The +1 inch was spread out on a table and the rock picked out of it in $5\frac{1}{2}$ seconds. The −1 +$\frac{3}{4}$-inch mesh was picked in a similar manner in 27 seconds.

The −$\frac{3}{4}$ inch +6 mesh was jigged in a two-compartment James jig, the concentrate or shell product which came to the top in the first jig overflowed into the second jig. The concentrate from the second jig overflowed into a tank. The tank overflow ran through a fine screen which caught any wood which floated. A tailing was taken from each jig and after the run as much of the jig bed as possible was run out of the tailing discharge. What remained in each jig was called jig bed and consisted almost entirely of rock.

The tails from the second jig were rerun in the first jig making a concentrate and a tailing, the bed being put with the tailing.

The concentrates from the first run were rerun in the first jig to remove wood that did not float. This wood comes to the surface in the jig and flows off with the water.

The −6 mesh was tabled on a quarter-size Wilfley table to remove wood and slime. All the products were dried. The table concentrate was screened on 20 mesh.

Samples were cut from all shell and rock products +20 mesh and these samples were carefully hand-picked into shell and rock and the per cent of shell in each figured.

The following table gives the products obtained from the test. The concentrates or shell products are given first and the tailing or discard products last. The −20 table concentrates are put with the tailing products because shell for poultry should be +20 mesh.

Product	Pounds	Shell, per cent	Shell, pounds
Jig concentrate, Run No. 3	72·50	90·0	65·25
Jig concentrate, Run No. 2	10·00	87·4	8·74
Table concentrate +20	14·00	91·7	12·84
Jig and +20 table concentrate	96·50	90·0	86·83
Hand-picked shell +1 inch	2·06	100·0	2·06
Hand-picked shell −1 +¾ inch	1·54	100·0	1·54
Total shell products	100·10	90·3	90·43
Jig tailings, Run No. 1	18·50	30·3	5·61
Jig tailings, Run No. 2	6·50	38·2	2·43
No. 1 jig bed, Run No. 1	4·00	7·0	0·28
No. 2 jig bed, Run No. 1	1·50	6·2	0·09
Wood from jig, Run Nos. 1 and 3	0·28	....	....
Table concentrate, −20	3·50	38·4	1·34
Wood from table	0·72	....	....
Hand-picked rock, +1 inch	0·79	....	....
Hand-picked rock, −1 +¾ inch	1·09	....	....
Feed	136·98	....	....

The test shows that a shell product 90·3 per cent pure can be obtained by hand-picking, jigging and tabling.

Jigging and tabling give a shell product 90·0 per cent pure.

A test on the table concentrate −20 mesh with acid showed it to be approximately 38·4 per cent shell. The discard products would be 26·6 per cent shell, and the discard products not including the wood would be 27·3 per cent shell.

Test No. 2

All the products from Test No. 1 were mixed, screened on ¾ inch and the oversize crushed in a small jaw crusher till it passed the ¾-inch screen. The −¾ inch was screened on 6 mesh.

The −¾ inch + 6 mesh was jigged in the two-compartment James jig in the same way as described under the first jig run of Test No. 1 except that the jig beds were not run out into the tailings.

The jig concentrates were rerun in one jig to remove wood, the jig bed of this run being put with the jig concentrates.

The two jig beds and the jig tail from the second jig, all of the first run, were mixed and rerun in the following way. First they were run through one jig set so as to take off no tailing. This gave a bed called No. 1 bed Run 3 and an overflow product. The bed was removed and kept separate. The overflow product was fed back to the jig and the overflow was called No. 1 concentrate Run 3. The discharge openings of the jig were lowered and the overflow was called No. 2 concentrate Run 3. The bed was called No. 2 bed Run 3.

The −6 was tabled in the same way as described under Test No. 1.

All products after drying were treated as in Test No. 1.

The following table gives the results:—

Product	Pounds	Shell, per cent	Shell, pounds
Jig concentrate, Run No. 2	66·50	89·2	59·32
No. 1 jig concentrate, Run No. 3	3·31	90·1	2·98
No. 2 jig concentrate, Run No. 3	9·31	87·1	8·11
Table concentrate, +20	13·87	89·6	12·43
Jig and +20 table concentrate	92·99	89·1	82·84
No. 2 jig bed, Run No. 3	10·62	62·2	6·61
No. 1 jig bed, Run No. 3	10·25	12·4	1·27
No. 1 jig tailing, Run No. 1	16·50	29·0	4·78
Wood from jig, Run No. 1	0·24	....	....
Wood from jig, Run No. 2	0·08	....	....
Table concentrate, −20	4·12	38·4	1·58
Wood from table	0·71	....	....
Feed	135·51	....	....

Test No. 2 shows that a shell product 89·1 per cent pure can be obtained by crushing to pass ¾ inch before jigging and tabling.

A small sample of the table concentrate −20 mesh was treated with hydrochloric acid and 38·4 per cent of it was dissolved. This product can be taken, therefore, as being approximately 38·4 per cent shell. If this is done then the discard products average 33·5 per cent shell, and the discard products not including the wood average 34·3 per cent shell.

SUMMARY OF EXPERIMENTAL TESTS

Test No. 1 gives slightly better results than Test No. 2. Even if only the shell recovered by jigging and tabling is considered, Test No. 1 is the better.

CONCLUSIONS

The most suitable method to treat the shells and rock would be to screen on two screens, taking out anything too large or too small for jigging. The large screen could be about 1 inch and the small one about 6 mesh.

The +1 inch could be hand-picked, the −1 inch + 6 mesh could be jigged, and the −6 mesh could be tabled.

In hand-picking it would be best to use a belt and pick out the rock.

Three jigs had best be used for jigging. The first to take out a clean tail, the second to take out a middling which could be returned to the first jig, and the third to produce a shell product and a wood product.

The −6 mesh could be tabled, the concentrates dried and screened on 20 mesh and the −20 mesh discarded.

Before the shell product could be used for poultry it would have to be crushed to $-\frac{5}{8}$ inch and screened on 10- and 20-mesh screens. The $-\frac{5}{8}$ inch +10 mesh would be hen shell and the −10 +20, chicken shell. The −20 would be discarded. The crushing can be done in a hammer mill with a $\frac{5}{8}$-inch grate or what is said to be better a Sturtevant rotary crusher. Any good vibrating screen could be used for screening.

Due to the small quantities used in the tests nothing definite can be said about the capacities of jig or table. As the table was used merely to remove wood and slime there is no reason to doubt that it would have a good capacity. The feed to the jigs was very slow to allow time for adjustment, however, the rock and shell seemed to separate fairly fast and it is believed that fair capacities could be obtained. Before being certain it would be necessary to try out a much larger quantity.

Report No. 415

THE CRUSHING OF SLATE, FROM PULVERIZED PRODUCTS, LIMITED, MONTREAL, QUE., FOR THE PRODUCTION OF SLATE GRANULES

R. K. Carnochan

Shipment. One bag of grey slate, shipping weight 58 pounds, was received June 19, 1931, from Pulverized Products, Limited, 4820 Fourth Avenue, Rosemount, Montreal, Quebec.

Purpose of Experimental Tests. It was desired that experiments be made to determine if granules suitable for making ready roofing could be prepared from the slate, and if so what per cent of the slate would be recovered in granules.

Arrangements for Experimental Tests. Mr. Frank T. St. George of Pulverized Products, Limited, arranged to be present for the first part of the test work.

Characteristics of the Slate. The slate is blue-grey in colour and has a tendency to split into long flat needles, not flat flakes.

Experimental Tests

Mr. St. George submitted two samples of slate granules used for coating ready roofing and requested that the slate be prepared similar to the finer sample. A screen test made on this finer sample showed it to be all −8 mesh and only 9·69 per cent −20 mesh. It would, therefore, seem to have been produced by screening on 8 and 20 mesh.

Nearly all the slate, 54¾ pounds, was crushed once in a small jaw crusher and seven times in a small set of rolls. After each crushing it was screened on 8 mesh and only the +8 mesh was returned to be crushed. After the seven passes through the rolls all the slate was −8 mesh.

The −8 mesh was screened on 14 mesh. As there were a lot of long needles in the −8+14 mesh, it was screened three times on 8 mesh to remove them. The needles were crushed in the rolls eight times or until everything was −8 mesh. After each crushing the crushed product was screened four times on an 8-mesh screen so as to return to the crusher any unbroken needles.

The product from the last crushing was screened on 14 mesh, and the −8+14 and −14 were added respectively to the first −8+14 and −14. The combined −14 was screened on 16, 18, and 20 mesh. A screen test showed the −8 +14 to contain less fines than the fine sample of slate granules so the −14+16 was added. A screen test on the −8+16 showed less fines than the sample of granules so the −16+18 was added. The −8+18 contained just a little less fines than the sample so the −18+20 was added to the −20, giving a −18-mesh product.

SUMMARY

The slate granules produced from the slate seem by their appearance to be suitable for coating ready roofing.

The final −8+18 product weighed 25½ pounds, so the recovery of slate granules from the slate would be 25½ ÷ 54¾ × 100 = 46·58 per cent.

CONCLUSIONS

The best way to treat the slate to prepare slate granules would be to crush in a jaw crusher and screen the discharge on a two-surface vibrating screen using 8 and 18 mesh. The +8 would go to a set of rolls and the roll discharge back to the screen. The −18 would be a discard unless some use was found for it. The −8+18 would go to a three-surface vibrating screen, each surface being 8 mesh. This screen would remove long needles which would go to the rolls, the minus from the screen would be the slate granules −8+18 mesh.

It might be advisable to use pieces of tin on the three-surface screen to cover part of the area of each screen, so that the needles would not have sufficient time to pass the screen. The first screen would perhaps be covered the last two thirds of its length, the second the first and last thirds, and the last the first two thirds. The tin on the second and last screen under the open part of the screen above would place the needles onto the screen flat and thus give them a better chance to go into the oversize. A piece of tin could be added to the feed end of the first screen for this purpose also.

Report No. 416

THE WASHING OF MARL FROM THE GENERAL CALCIUM CORPORATION, LTD., MILTON, ONTARIO

R. K. Carnochan

Purpose of Experimental Tests. The General Calcium Corporation have a deposit of marl near Milton, Ontario. They wished to have tests made on the marl to determine a method of treatment to prepare a product suitable for making linoleum. It was also desired that large samples of the finished product be prepared and sent to linoleum manufacturers for trial.

Arrangements for Experimental Tests. Mr. Brickenden, the engineer of the General Calcium Corporation, arranged to be present when the tests were being made.

Shipments. Two shipments of wet marl were received, the first consisting of 5 large metal containers, shipping weight 1,200 pounds, on July 22, 1930, and the second, of 115 containers, shipping weight 30,000 pounds, on November 5, 1930.

Characteristics of the Marl. The marl as received contained a lot of water, about 53 per cent. The marl is light buff in colour, and in it are pieces of wood and leaves, and shells. The coarser pieces of the marl itself consist of small tubes and broken parts of tubes.

Analysis. The only product analysed was the finished fine product. This ran as follows:—

	Per cent
CaO	50·75
Insoluble	0·45
Al_2O_3	0·15
Fe_2O_3	0·45
MgO	1·50
Organic matter	6·00
CO_2	38·20

EXPERIMENTAL TESTS

The first shipment called Lot No. 1 was used up in small washing tests, a small grinding test, small bleaching and washing tests, and a medium-sized washing test. The second shipment called Lot No. 2 was used for a large-scale washing test and some small classifying and thickening tests. All tests made will not be described but the large-scale test will be given and such of the small tests that give additional information.

Large-Scale Test

Washing. In the large-scale test the marl was fed along with water to a 16-inch by $4\frac{1}{2}$-foot Hardinge mill lined with Silex and running without pebbles. This mill served to mix the marl and water into a thin pulp. The shells present in the marl remained in the mill and assisted in the mixing action somewhat like pebbles but not grinding. Near the end of the test the shells had collected in the mill sufficiently to nearly fill it and be discharged. At the end of the test the cleanout of the mill consisted of clean shells.

The Hardinge mill discharge was pumped by a 2-inch Wilfley pump into a 7-foot Dorr bowl classifier. This classifier separated the fine marl from the coarse. Plenty of wash water was added to the back of the classifier. The speed of the rakes was 12 strokes per minute and the bowl mechanism ran at 2 r.p.m. during the first half of the test and 3 strokes per minute and ½ r.p.m. during the last half. This change of speed had no effect as both speeds used were slow enough to give good results.

As the overflow from the bowl classifier contained a small amount of material that floated in water, such as pieces of wood and leaves and a few shells, it was pumped by a 2-inch centrifugal pump to a No. 7 Rotex screen fitted with a 50-mesh wire screen cloth, size 40 inch by 84 inch. This screen removed the floating material from the fine marl.

The fines from the screen went to a 12-foot, two-tray Dorr thickener, where they were allowed to accumulate. The overflow from the thickener was cloudy. During a part of the run some aluminium sulphate was added to the circuit and it cleared the overflow up somewhat.

The test was divided into a number of parts called A, B, C1, C2, etc. The following table gives the details of the different parts:—

Part	Wet marl fed, pound	Duration of parts, hours	Feed wet marl, lb./hour	Feed water plus wash water, lb./minute
A	5,798	6·00	966	146
B	5,541	3·83	1,447	148
C1	3,291	1·75	1,881	157
C2	2,351	1·25	1,881	157
D1	1,455	1·0	1,455	140
D2	1,212	1·0	1,212	144
E1	1,906	1·0	1,906	200
E2	2,384	1·0	2,384	252
F1	2,362	1·67	1,414	200
F2	472	0·4	1,180	200
F3	472	0·5	944	200
	27,244			

The rise in the bowl classifier was calculated in feet per hour for the different parts of the test. The dilution and fineness of the bowl overflow had been obtained by sampling. From the rise and the dilution the amount of dry marl in the overflow was calculated. The following table gives the results:—

Part	Feet, per hour	Pounds, per hour	Dilution	Classifier fines, per cent −325	Rotex fines, per cent −325
A	3·92	300	31·0	99·34	99·80
B	4·08	446	21·6	98·42	96·18
C1	4·42	798	12·9	....	92·18
C2	4·42	727	14·2	97·70	98·18
D1	3·88	502	18·2	98·06	....
D2	3·92	476	19·4	98·08	....
E1	5·50	821	15·7	98·50	....
E2	6·92	876	18·6	97·02	....
F1	5·37	614	20·6	99·28	....
F2	5·30	562	22·3	99·12	....
F3	5·24	324	38·5	99·46	....

The fines from part C1 are the coarsest obtained, 92·18 per cent −325 mesh; this is due to putting 5 pounds of aluminium sulphate in the feed in the proportion of 1 pound to 235 pounds of wet marl. The aluminium sulphate coagulates the marl in the bowl classifier and the coagulated fine marl entraps and carries over coarse marl into the bowl overflow.

If part C1 is omitted and the other parts plotted using for ordinates the feet per hour and abscissæ the pounds per hour, and putting down at each point plotted the fineness of the classifier overflow, then lines representing 97, 98, 99, and 99·5 per cent −325 mesh can be drawn in as graphs. These graphs show that 5·5 feet per hour gives the best results. The following table shows the best dilutions to use for different products and the amounts of fine marl produced per hour by the bowl classifier for the dilutions given:—

Per cent −325	Dilution	Pounds per hour
97·0	12·3	1,040
98·0	14·4	900
99·0	20·3	640
99·5	38·5	340

The thickener overflow in parts A and B was very cloudy. In part C1 which lasted 1·75 hours 5 pounds of aluminium sulphate was put in the feed in the first 37·5 minutes, this being at the rate of 1 pound to 235 pounds of wet marl. This aluminium sulphate coagulated the bowl overflow and gave a coarse product and for this reason was discontinued and the aluminium sulphate was allowed to wash out of the bowl before starting part C2. In part C2, which lasted 1·25 hours, 5 pounds of aluminium sulphate dissolved in water was fed to the bowl overflow during the 1·25 hours, this was at the rate of 1 pound of aluminium sulphate to 470 pounds of wet marl fed to the circuit. The addition of the aluminium sulphate during parts C1 and C2 reduced the cloudiness of the thickener overflow both during parts C1 and C2 and during the rest of the test. Samples of the overflow were taken from time to time, the following table gives the data obtained:—

Part	Thickener overflow dilution
B	357
C	404
D	1,943
E	573
F	487

A study of the above table shows that there was a slight improvement as soon as the aluminium sulphate was added in part C and more improvement when the aluminium sulphate had mixed into the thickener contents in part D. In parts E and F the effect was beginning to wear off.

Samples of the material stored up in the thickener were drawn off at different intervals from the bottom of the thickener. These samples were taken to show how long it took for the material to settle down into a thick product.

The following table gives the results:—

Days from start of test	Dilution of thickener underflow
1	1·90
2	1·86
3	1·60
4	1·38
7	1·36

The above table shows that the fine marl settles down very rapidly.

Filtering. Five hundred pounds net wet weight of thickener underflow, dilution 1·356 to 1, were put into two metal containers for Mr. Brickenden who intended to have a steam drum dryer test made on them. This lot contained 212 pounds of dry marl.

The remainder of the thickener contents was pumped by means of a No. 2 Dorrco pump into two 3-foot by 3-foot high agitation tanks, and from the tanks by a Patterson No. 1 heavy duty slip pump into a Patterson 28-inch diameter, 36 chambers round rail filter press. When the press filled the pumping was stopped and the press emptied, closed, and the pumping started again. In all, the press was filled 14 times. The filter cakes obtained averaged about 40 per cent moisture. Samples taken of each filter press gave an average fineness of 98·36 per cent −325 mesh.

Drying. Most of the filter cake was dried in shallow pans over steam coils at a temperature of about 60° C. When dried in this way the filter cake was not discoloured.

A small amount of the filter cake was dried in a small Freas No. 645 electric oven at temperatures ranging from 70° to 180° C. The marl dried in this way had a very slight yellow discolorization.

Some of the filter cake was dried in pans on a gas-heated dryer. This method of drying discoloured the marl very badly, turning it quite brown.

The dried filter cake was broken up by means of a small Sturtevant No. 00 hammer mill. When pulverized in this way the coil and oven dried marl were practically the same colour and they were combined. The marl that had been dried in pans on the gas dryer was much darker and was kept separate.

The other products from the test were dried in pans on steam coils, and all dried products were weighed. A sample for chemical analysis was obtained from the combined coil and oven dried marl.

Summary. The following table gives the weights of the different products obtained from the large-scale test:—

Product	Dry weight, pounds	Per cent	Per cent −325
Mill cleanout	82	0·64	....
Bowl cleanout	548	4·29	44·0
Bowl oversize	1,286	10·06	14·0
Rotex oversize	24	0·19	52·0
Filter press cake	9,276	72·58	98·36
Loss	1,564	12·24	....
Feed	12,780	100·00	84·18

The mill cleanout consisted of white shells. The bowl oversize and cleanout were composed of medium-sized marl. The Rotex oversize was mostly fine wood, and the filter press cake was fine marl.

The loss 12·24 per cent is high and is due mostly to marl going over in the thickener overflow. Some of the loss is due to small leaks and spills during the test work and to losses in dewatering the different products. A part of the loss may not be a real loss but due to the head sample, which was taken over a period of a week and kept in a covered container, drying out before the moisture in the sample was determined. This sample ran 53·09 per cent moisture, which figure may be somewhat low.

Small-scale Tests

Classifying. A large number of small-scale classifying tests were made with a 6-inch diameter classifier. These tests showed that there was a certain rate of rise in the classifier that gave the best results and that different finenesses could be best obtained by using the best rate and altering the dilution.

City water was found to give the same results as distilled.

The use of aluminium sulphate in the feed gave a coarser overflow.

The results of the small tests were plotted and graphs drawn in for different finenesses, 97, 98, 99, and 99·5 per cent −325 mesh. These graphs aided in drawing similar graphs for the large-scale test. For instance in the large-scale test no data were obtained for the area between 5 and 6 feet per hour and over 900 pounds per hour. In this case the 97 per cent line was picked up outside the area and sketched in similar to the small-scale graph.

Thickening. A number of small-scale thickening tests were made in a glass tube 2½ inches in diameter and 5 feet long. The following table gives the data obtained in these tests:—

Water used	Reagent used	Dilution	Method of settling	Fine, grms.	Slime, grms.	Slime, per cent of fine	Fall in 1 hour, feet
City		43·4	Up	88·9	8·1	9·1	
Distilled		45·9	Up	84·2	7·9	9·4	
“		19·9	Up then down	203·9	4·3	2·1	
“		10·4	Up then down	385·3	14·7	3·8	
“		7·82	Down	519·4	4·0	0·8	1·89
“	5 grms. aluminium sulphate	10·5	Down	397·0			2·60

Under "Method of settling," "Up" means from the bottom upwards and "Down" means from the top down. "Up then down" means the settled material grew up from the bottom and after awhile this deposit began to settle down. Under "Fine" the material that settled in 24 hours is given, and under "Slime" the material that did not.

City water gives the same results as distilled.

Unless a coagulant is used a good deal of the marl does not settle.

Without a coagulant the initial fall can be taken at 1·9 feet per hour, with enough coagulant the fall would be 2·6 feet per hour. In the last test even the large amount of aluminium sulphate used, 5 grammes to a feed of 2,000 grammes of wet marl, was not enough to completely clear the water.

What thickening results will be obtained when the marl is treated with the water present at the marl deposit cannot be determined without samples of the water. However, the results are likely to resemble those obtained by using distilled water. If so, it seems that the best method would be to use no coagulant and to use the thickener overflow over again.

Oil Absorption

Large samples of the washed marl were shipped to two companies making linoleum or a similar product. One of these companies reported back that the washed marl had an oil absorption of 65·2, and that this was too high to allow the marl to be used for making linoleum, as their materials for this purpose range in oil absorption from 19·6 to 26·0. These oil absorptions are expressed in c.c. of oil needed to wet 100 grammes of material. Similar tests on oil absorption made by the writer confirmed the opinion of this company.

In order to secure a product with a lower oil absorption it was decided to grind the marl instead of washing it. At first wet grinding was tried but no improvement was obtained. Dry grinding was then tried out and it was found to give much better results.

Dry grinding tests were made in a small pebble jar, a large pebble jar, and a small 2-foot by 3-foot pebble mill, and good results were obtained with them all. To secure a suitable product it was found necessary to grind for 4 hours.

Tests made with the 2-foot by 3-foot pebble mill and 30-inch Gayco air separator showed that a suitable product could be prepared by grinding four hours in the mill and then separating, or by grinding one hour in the mill, separating and then grinding the Gayco fines for four hours.

In large-scale continuous grinding with pebble mill and air separator the material being ground stays in the grinding circuit longer than in a small intermittent test, therefore it might be possible to secure a product with a low oil absorption by grinding dry by pebble mill and air separator.

Bleaching

A number of small bleaching tests, using a bleaching liquor prepared with chlorine and hydrated lime, were made on washed marl, but when it was found that the marl would have to be ground dry to secure a product with a low oil absorption, more bleaching tests were made on dry ground marl. The tests on the dry ground marl gave better results due to the fact that the dry ground marl contains a great deal more of very fine material than the washed marl.

The bleaching tests show that the more bleach liquor used the whiter the product up to a certain point beyond which no improvement can be obtained. A pure white product cannot be produced, the best colour obtained might be called white with a slight cast.

The amount of hydrated lime used influences the colour, below or above a certain amount gives a poorer colour.

Fair results were obtained by bleaching 500 grammes of marl, with 450 c.c. of bleach solution containing 33·84 grammes chlorine and 5 grammes of hydrated lime per litre.

Assuming hydrated lime to cost $12 a ton and chlorine 2 cents a pound, the cost of the chemicals to bleach a ton of marl would be about $2. This is a very high cost and quite likely would make bleaching of the marl unprofitable.

CONCLUSIONS

1. The marl can be washed and a fine product obtained but this product has too high an oil absorption to allow it to be used for making linoleum.
2. Small dry grinding tests indicate that a product with a low oil absorption can be obtained. Before being certain that the dry ground product can be used for making linoleum it will be necessary to prepare large samples and submit them to linoleum manufacturers for trial and report.
3. The dry ground marl can be bleached to almost white, but the cost is very high.

Report No. 417

THE TESTING OF GYPSUM FROM BULL RIVER, BRITISH COLUMBIA

R. A. Rogers

Shipment. Four bags of gypsum rock, net weight 210 pounds, were received on October 3, 1931, from Bull River, British Columbia, submitted by Mr. E. G. Hazell, Summit Lime Works, Lethbridge, Alberta.

Purpose of Tests. The tests were to determine the plaster-making qualities of the gypsum.

Sampling and Analysis. The crude rock had a massive, compact texture, some of it being slightly crystalline. Part of it was banded dark grey and white while the remainder was mottled light grey and white.

The sample was crushed in a small jaw crusher and swing-sledge mill, and then ground in a burr mill to 99 per cent through 100 mesh. A screen analysis was made of this product.

Mesh—	Per cent
+100	1·0
+150	1·8
+200	6·9
−200	90·3
	100·0

The burr mill product was cut in a Jones riffle sampler, and the sample chemically analysed by the writer. The results of the analyses are given in the following table:—

Sample dried at 45° C. for 2 hrs.	Per cent
Insoluble	9·50
Ferric oxide (Fe_2O_3)	0·30
Alumina (Al_2O_3)	0·50
Lime (CaO)	28·95
Magnesia (MgO)	3·40
Sulphur trioxide (SO_3)	35·47
Combined water (H_2O)	15·58
Carbon dioxide (CO_2)	6·13
	99·83

Recalculation of these results gives:—	Per cent
Gypsum ($CaSO_4 \cdot 2H_2O$)	74·44
Anhydrite ($CaSO_4$)	1·45
Magnesium carbonate ($MgCO_3$)	7·11
Calcium carbonate ($CaCO_3$)	5·50
Other ingredients	11·33
	99·83

EXPERIMENTAL TESTS

Two calcining tests were made. Fifty pounds of the sample was taken in each case and calcined in a small electric batch kettle to the first settle. This calcined gypsum was tested. The water remaining after calcination, the testing consistency, and time of setting were determined. Briquettes and cylinders were made, dried in the air to constant weight, and broken in Olsen-Boyd and Olsen testing machines to find the tensile and compressive strengths.

Screen Analysis of Calcined Gypsum

Mesh	Test No. 1	Test No. 2
	%	%
+100	0·8	0·9
+150	1·3	1·1
+200	5·0	5·2
−200	92·9	92·8

Summary of Tests

—	Test No. 1	Test No. 2
$\frac{\text{Water after calcination}}{\text{Water before calcination}}$	0·272	0·282
Final temperature of calcination, °F	290	282
Water after calcination, per cent	4·23	4·39
Testing consistency, c.c. per 100 grms	62	61
Time of setting in minutes	19	19
Tensile strength, lb./sq. in	328	309
Compressive strength, lb./sq. in	1,692	1,702
Colour of ground gypsum	light grey	light grey
Colour of calcined product after set	light grey	light grey

CONCLUSIONS

The tests show that this gypsum would be suitable for the manufacture of structural materials having a gypsum base, such as hardwall plaster, gypsum board, gypsum blocks, etc. It would not make a white finishing plaster.

IV

REPORTS OF INVESTIGATIONS:—SECTION OF HYDROMETALLURGY AND ELECTROCHEMISTRY

Report No. 418

Part A

PRELIMINARY REPORT ON METHOD OF TREATING GREAT BEAR LAKE PITCHBLENDE FOR THE EXTRACTION OF RADIUM

R. J. Traill

In May, 1930, Gilbert LaBine, prospecting in the vicinity of Great Bear lake in the Northwest Territories, made a discovery of pitchblende and silver at Echo bay on the eastern end of the lake.

Late in the same year a small shipment was received at the Mines Branch Laboratories for identification and report.

Tests showed the material to consist largely of pitchblende, a heavy black lustrous mineral, consisting mainly of uranium oxides, and consequently important because of its radium-bearing content.

The test work on this shipment was reported in March, 1931, and published in Memorandum Series No. 48.

In August, 1931, H. S. Spence, under direction of the Mines Branch, visited the property and in his report, published October, 1931, Memorandum Series No. 51, verified the discovery and reported it to constitute beyond question a very valuable source of radium.

In July, 1931, a shipment of 90 pounds hand-cobbed ore was submitted to this laboratory by Gilbert LaBine, Managing Director of the Eldorado Gold Mines, Limited, for test work in developing a method of extracting the radium, and a further shipment of approximately 300 pounds was received in September.

H. S. Spence in the course of his examination of the discovery selected many samples representative of the ore deposits and the country rock for microscopic and chemical examination. Analyses of these samples are shown in Table I. (Page 139).

Summarizing the results as given respecting the analysis of the ore it may be observed that the ore varies quite distinctly along the veins. The uranium content, reported as U_3O_8, varies from 30 per cent to 62 per cent, the silica content ranges from 53 per cent to 1 per cent, sulphides vary somewhat but tend to increase with the uranium content with a maximum of 1·09 per cent sulphide sulphur. Alkaline earth minerals appear up to 2·75 per cent with manganese up to 1·97 per cent and some samples carry barium sulphate. Arsenic, antimony, cobalt, and phosphates are present in small amounts. Lead naturally appears in all samples in combination with the uranium, but some samples appear to carry galena in varying amounts usually accompanied by silver. Soluble sulphates vary from 0·13 per cent up to 1·45 per cent (SO_3), and carbonates are present mainly as rhodocrosite, siderite, and calcite.

TABLE I

Analyses of Pitchblende Ore from Great Bear Lake

—	* Vein No. 1 Pit No. 1	* Vein No. 1 Pit No. 2	* Vein No. 2 Pit No. 7	* Vein No. 2 Pit No. 8	* Vein No. 2 Pit No. 9	* Vein No. 2 Pit No. 10	† Vein No. 2 Pit No. 1	† Vein No. 2 Pit No. 2	† Vein No. 2 Pit No. 3	† Vein No. 2 Pit No. 4	† Vein No. 2 Pit No. 5	† Vein No. 2 Pit No. 6
PbO	8·19	6·27	10·68	6·96	14·43	12·65	9·93	12·44	9·20	10·42	3·20	5·98
as Pb	7·60	5·82	9·91	6·46	13·40	11·75	9·22	11·55	8·54	9·67	2·99	5·55
U_3O_8	43·82	30·34	50·80	31·45	58·00	62·10	54·20	56·84	55·17	50·68	30·82	45·17
as U	37·14	25·73	43·09	26·68	49·19	52·68	45·97	48·21	46·79	42·99	26·14	38·31
Fe_2O_3	3·00	3·78	2·46	4·07	3·22	7·00	1·74	1·21	2·18	1·82	3·21	1·74
as Fe	2·15	2·64	1·72	2·85	2·25	4·90	1·22	0·85	1·52	1·27	2·24	1·22
Cu	0·24	0·25	0·84	1·48	1·44	1·85	0·60	0·37	0·59	0·36	0·42	1·01
MnO	0·14	0·20	0·25	0·20	1·48	2·54	0·05	0·20	0·013	0·32	0·55	0·20
as Mn	0·11	0·15	0·19	0·15	1·15	1·97	0·04	0·16	0·010	0·25	0·43	0·16
Ni	nil	trace	nil	trace	nil	0·01	none	none	none	none	trace	none
Co	0·12	0·05	trace	0·03	0·15	0·25	0·03	0·02	0·03	0·04	0·03	0·03
Sb	0·25	0·05	0·11	0·08	0·18	0·22	none	none	none	none	none	none
MoO_3	trace	0·04	0·09	0·37	trace	trace	0·12	0·12	0·12	0·12	0·10	0·12
Zn	0·40	....	....	....	0·25	0·15	none	none	none	none	none	none
(ThO_2—CeO_2)	nil	nil	nil	nil	nil	nil	none	none	none	none	none	none
SiO_2	36·80	53·46	28·28	47·24	1·00	1·10	24·71	23·68	23·16	29·45	52·02	37·33
$BaSO_4$	nil	nil	nil	nil	4·73	nil	none	none	none	none	none	none
As_2O_3	0·40	0·06	0·09	0·13	0·46	0·46	none	none	none	none	none	none
as As	0·30	0·05	0·07	0·10	0·35	0·35	none	none	none	none	none	none
Sulphide S	0·53	0·29	0·42	1·06	0·62	0·84	1·09	0·35	0·33	0·12	0·16	0·30
P_2O_5	0·23	0·21	0·32	0·15	0·45	0·60	0·17	0·15	0·68	0·26	0·20	0·61
Al_2O_3	1·20	0·52	0·33	0·40	0·15	0·65	0·92	0·60	0·85	0·72	1·10	0·69
CaO	0·66	1·11	0·96	0·77	2·70	1·65	0·69	1·39	0·65	0·39	1·00	0·56
MgO	0·12	0·55	0·14	0·44	0·05	0·18	0·49	0·43	0·22	0·37	1·02	0·56
CO_2	0·22	1·25	0·76	2·09	1·25	1·64	0·41	1·07	0·19	0·28	0·16	0·06
($K_2O + Na_2O$)	1·78	0·92	1·00	1·40	1·91	2·55	1·63	0·90	0·53	0·49	0·41	0·36
H_2O	1·05	0·98	2·20	2·08	3·45	3·75	2·11	0·93	2·50	2·15	2·50	2·19
Au, oz./ton	0·08	trace	0·10	0·07	nil	nil	trace	0·05	none	0·07	none	0·02
Ag, oz./ton	1·82	0·13	2·86	14·28	936·7	49·4	1·32	0·28	1·32	2·19	24·90	2·12
Sp. Gr.	4·10	3·42	4·70	3·74	6·10	5·90	4·74	5·52	4·88	4·78	3·62	4·16

*Determinations by B. P. Coyne and H. L. Beer.
†Determinations by A. Sadler and C. L. O'Brian.

In selecting a process of treatment for recovery of radium this varying composition should be kept in mind and, if possible, a single process that would take care of these variations should be used providing that such a process can show a satisfactory extraction of radium content at a reasonable cost.

Several processes have been used and others suggested for the treatment of radium-bearing ores, but each one has been particularly adapted to the type or character of the ore to be treated. No one general process is economically suited to all ores. The radium content of an ore, in comparison with the other constituents, is exceedingly small ranging perhaps from 5 milligrams to 150 milligrams per ton of ore.

In pitchblende the radium content is proportional to the uranium content, the ratio being one part radium per three million parts uranium.

For instance, an ore carrying 50 per cent U_3O_8 has a radium content of approximately 130 milligrams per ton.

This exceedingly small amount of radium, therefore, necessitates the use of a highly specialized process which must be very carefully controlled and which is, as a rule, long and tedious. In its general chemical behaviour radium resembles barium—the chloride, bromide, and nitrate being soluble and the sulphate and carbonate insoluble. The process to be used, therefore, resembles a process that would be employed were barium to be extracted and recovered. The main factors that govern the type of process to be used are the presence of sulphides, sulphates, silica, silicates, and acid-consuming minerals such as the alkaline earth and other carbonates. All processes may be said to be three stage:—

1. Decomposition of ore.
2. Concentration of radium content.
3. Refining of radium concentrate.

The main difference in the processes employed pertains to the first-stage treatment, namely decomposition of the ore, and varies according to the chemical composition of the ore.

The second stage or the concentration of radium content entails simply the precipitation of the radium as sulphate in the presence of a large excess of barium sulphate, and the final stage comprises the conversion of the radium-barium sulphate concentrate to soluble salts with separation of the radium by fractional crystallization of the combined salts, usually in the form of chlorides, the radium chloride being the less soluble.

For the decomposition of the ore the processes may be divided into three main types:

(1) Fusion of ore with sodium sulphate, bisulphate sodium carbonate or hydrate. In sulphate fusions, acid-leaching removes uranium, iron, and other soluble salts, leaving silica, and the insoluble sulphates such as barium, lead and calcium and radium as sulphate. The insoluble material is then converted to a soluble form by leach or fusion with alkalis and after water-leaching and careful washing to get rid of sulphates, the carbonate precipitate is dissolved in hydrochloric acid and the radium precipitated with excess barium by the addition of barium chloride and sulphuric acid or a soluble sulphate. This precipitate constitutes the radium concentrate.

In alkali fusions, the fused mass is leached with water to dissolve soluble salts such as silicates, sulphates and some uranate, and the insoluble material dissolved in hydrochloric acid with subsequent precipitation of the radium with excess barium sulphate.

(2) Alkali leaching has been practised on some ores more particularly on carnotites. Roasting with or without alkali has been resorted to in some cases prior to leaching. The leached and washed residue is treated with hydrochloric acid and the radium precipitated with excess barium sulphate.

(3) Acid leaching has given satisfactory results in several cases.

Some ores are readily decomposed with sulphuric acid, the silica, sulphates of lead, barium, calcium, and the radium being found in the insoluble.

Lead can be removed by brine leaching and calcium by hydrochloric acid, leaving impure barium and radium sulphates, these are converted to carbonates by alkali treatment, silica being mostly removed as silicate. The carbonates are then dissolved in hydrochloric acid and the radium re-precipitated if necessary in purer form with excess barium sulphate. Hydrochloric acid leaching has been used and, provided the ore is low in sulphates, brings the radium directly into solution from which it may be precipitated by the addition of barium chloride and a soluble sulphate. Nitric acid was proposed and utilized by the U.S. Bureau of Mines investigators working on the Colorado carnotites.

It also brings the radium directly into solution, provided the ore is low in sulphates and sulphides. The radium is concentrated from the nitric acid solution by precipitation with barium salts after neutralizing most of the acid. Many other processes have been suggested but few, if any, have reached the stage of commercial application.

PRELIMINARY EXPERIMENTAL WORK ON GREAT BEAR LAKE PITCHBLENDE

The preliminary experimental work was carried out on a 90-pound sample of hand-cobbed ore. The sample was crushed to −10 mesh and a sample cut for analysis. Portions of the sample were ground to −35 mesh and −100 mesh for the experimental tests. An analysis of this sample shows the principal constituents as follows:—

	Per cent
Uranium oxide (U_3O_8)	54·5
Silica (SiO_2)	21·7
Lead (Pb)	10·58
Copper (Cu)	0·60
Sulphur (total) (S)	0·82
Iron (Fe)	1·52
Lime (CaO)	0·54
Magnesia (MgO)	trace
Barium sulphate ($BaSO_4$)	trace

From an examination of the chemical composition of the ore as shown by the above analysis and taking into consideration the varying character of ore as summarized in Table I, it would seem that a process that would directly put the radium into solution at the start would be the best.

Processes such as fusion with sodium sulphate or bisulphate and leaching with sulphuric acid would put the radium into an insoluble form mixed with silica and sulphates of lead and calcium. A concentration of probably only 3 to 1 would be obtained and the presence of such a large proportion of silica would be a distinct disadvantage in the further purification and concentration of the radium content. Whether alkaline leaching or alkaline fusion of this insoluble were resorted to, followed by water leaching to remove sodium silicate, sulphate, etc., a certain amount of sodium silicate would remain with the carbonate residue which on acid treatment would form a gelatinous silicic acid that would clog any filtering medium that might be used and carry by absorption a fair proportion of the radium.

On low silica ore this process probably would be quite satisfactory, but in the present instance it does not appear very encouraging.

The same objections as noted would apply to direct alkaline leach or fusion and also to the direct sulphuric acid leach.

To make certain that the above opinion was reasonably correct a few tests using 5 to 10 grammes of the ground ore were treated in the manner suggested.

Fusion with sodium sulphate was quite unsatisfactory, the ore being attacked but slightly and even with the additional action of sulphuric acid leach a concentration of approximately 2 to 1 was obtained. The residue consisted largely of silica, lead sulphate, and undecomposed pitchblende.

Fusion of this residue or of the ore with sodium carbonate also proved unsatisfactory from the standpoint of the formation of the gelatinous silica and the consequent difficulty in filtering. The gelatinous silica so formed also carried appreciable quantities of radium.

Fusion with sodium bisulphate gave a much better decomposition with a residue of about 30 to 35 per cent of the original weight used after leaching, which contained practically all the silica with some lead sulphate. The same objections, however, are met in the subsequent alkaline treatment. Failing the development of a more direct and satisfactory method, the treatment of this residue from bisulphate fusion with hydrofluoric acid might be considered a possibility if costs could be kept within reason by recovery and re-use of the hydrofluoric acid.

Leaching with sulphuric acid gave somewhat disappointing results inasmuch as this ore seems to be less soluble in this medium than reports would indicate with respect to similar ores. The results obtained showed a residue of 55 per cent of the original weight of ore used. The addition of nitric acid helps considerably in obtaining a more complete decomposition, the residue thus obtained being comparable to that of the bisulphate fusion.

In the series of tests conducted, by far the best results were obtained by leaching with nitric acid or with hydrochloric acid. With these acids in the concentrations of approximately 1 to 1, residues, comparatively low in alpha radioactivity, averaging 23 per cent of the original weight used, were obtained. With large excess acid the results obtained with the respective solvents are in close agreement, but as the excess is reduced, and judging solely by the alpha radioactivity of the residue, the hydrochloric acid appears to give the best extraction of radium. It should be observed here, however, that the apparent radioactivity of the residue may not be a true measure of radium element content, as decomposition products of radium of high alpha activity may be present in the residue.

At the time these tests were being made, suitable equipment for accurate radium determinations was not available and the only basis of comparison available was the alpha electroscope. A separate laboratory is being provided for the accurate measurement of radium by the emanation method and will be referred to later.

From the preliminary examination as briefly outlined above, it appeared that an interesting process might be developed in the hydrochloric acid treatment method. Its advantages would be, direct solution of radium, with lead chloride the only major interfering impurity, and ready precipitation of radium sulphate with barium sulphate from the chloride solution. The highly objectionable silica separation would be thus conveniently overcome by primary elimination. Tests using 100- to 400-gramme samples were therefore made to determine the possibilities of this method, the results of which are briefly set forth as follows:

Initial tests indicated an extraction of radium that varied somewhat from 60 to 90 per cent, with similar extraction of the uranium value, using approximately two and a half parts 18° Bé. hydrochloric acid to one part ore. Re-treatment of the residue with hydrochloric acid showed the remaining uranium mineral to be difficultly soluble in that medium. Further investigation, however, showed that by the addition of an oxidizing agent to the acid these residues gave up 95 per cent of the uranium and radium. This step was naturally followed up and eventually embraced in the leaching process with quite encouraging results.

A convenient oxidizing agent was obtained in sodium nitrate and the suggested process may be described briefly as follows:—

The ore is leached with 2½ parts of 18° Bé. hydrochloric acid and 1½ parts water, at a temperature above 90° C. for 3 hours. A solution of sodium nitrate is then added in small amounts at short intervals so that oxidation will not be too rapid and to prevent the frothing from becoming too excessive. The amount of sodium nitrate required depends on the uranium content of the ore and is approximately 14 per cent of the weight of uranium oxide contained.

This oxidation operation requires about 1½ hours. The leach is now complete and the residue is separated by filtration. Filtration must be carried out while the leach liquor is hot, as lead chloride separates out rapidly with cooling. Washing with brine solution to remove the lead chloride from the residue and finally with hot water completes the first stage of operations.

The next step consists of separating the lead chloride from the leach liquor by filtration, and precipitating the radium in the clear filtrate by the addition of barium chloride and sodium sulphate, obtaining thereby a concentrate of radium sulphate in barium sulphate.

The residue obtained by this method of leaching was dried and weighed and after a 24- to 48-hour interval of time was tested under the alpha electroscope for radioactivity. The results so obtained indicated that a satisfactory extraction was being obtained and that the method had some merit.

The following table summarizes briefly some of the results obtained:—

Sample weight, grammes	Residue weight, per cent	Radio-activity	Extraction
100	24·5	2·2	
100	25·2	5·1	
100	23·7	1·5	
300	23·4	5·2	97·3
400	23·7	3·2	97·6

The radioactivity figure reported is an arbitrary figure measured by alpha electroscope on the dried residue and is supposed to approximately indicate the percentage of radium remaining in the residue. The extraction figures were obtained by actual determination of radium content electroscopically measured by the emanation method from fusions.

A partial analysis made on the last residue shows the following composition:—

	Per cent
Uranium oxide (U_3O_8)	0·15
Lead (Pb)	0·03
Copper (Cu)	0·76
Iron (Fe)	0·90
Sulphuric anhydride (SO_3)	0·04
Total sulphur (S)	3·15
Elemental sulphur (CS_2 soluble)	1·80

An examination of this residue would indicate that about 87 per cent of the sulphur content of the ore remains in the residue unoxidized and therefore ineffective as a precipitant of radium. Sulphur is also eliminated during leaching as hydrogen sulphide and the amount of oxidized sulphur as found in the solution is usually in the order of about half a gramme per litre. The lead chloride which crystallizes from the filtrate carries a very small amount of radium but there is an indication that soluble silica present in solution and precipitating on standing carries radium in appreciable amounts. Practically complete precipitation of the radium has readily been effected by the addition of barium chloride, followed by sodium sulphate, to the hot solution with a ratio of one part radium to one hundred thousand parts barium being obtained.

SUMMARY

Preliminary test work on a high-grade pitchblende sample would indicate that hydrochloric acid with the addition of an oxidizing agent offers a satisfactory method of decomposing this type of ore and obtaining a high solution extraction of the contained radium. Extractions of radium content of upwards of 90 per cent have been obtained.

A radium concentrate of the ratio of one part in one hundred thousand seems to be readily attained.

This preliminary test work has taken into consideration only one type of ore, namely a medium high silica ore, and as the ore varies somewhat in its chemical composition as shown by the analyses given, further test work on representative samples must be carried out to determine the suitability of the above suggested process.

Larger scale tests employing 20 to 25 pounds to a test are being planned for the purpose of checking results and obtaining some idea of cost and operating data.

The chemical analyses in connexion with the laboratory work were made by B. P. Coyne and H. L. Beer of the chemical laboratory of this Division.

Radium and radioactive determinations were made by W. R. McClelland of this laboratory. A description of the equipment and methods employed follows:—

Part B

THE METHODS EMPLOYED IN RADIUM MEASUREMENTS

W. R. McClelland

The amount of radium which is found in pitchblende or its concentration products is so minute that it is not possible to determine it by the usual methods of chemical analyses. Advantage, however, is taken of the radioactivity possessed by radium and methods of measurement based on this physical property are utilized. In brief, radioactivity is that property of a class of substances such as uranium, radium, thorium, etc., and their compounds which spontaneously emit radiations capable of passing through substances opaque to ordinary light. Besides their penetrating power these radiations have three definite characteristics:—

(1) Action on a photographic plate,

(2) Power of ionizing gases,

(3) Causing marked phosphorescence on certain substances.

The radiations consist of three types, the alpha particle which is a charged atom of helium; the beta particle, an electron carrying one negative electrical charge; and the gamma ray which carries no charge and is a wave-motion propagated with the velocity of light. The alpha particle is the least penetrating, while the gamma ray will penetrate great thicknesses of matter.

In practical measurements it is the ionizing power of the radiations that is utilized and the instrument most commonly employed is the electroscope. The simplest type of instrument is known as the alpha electroscope and consists of a chamber in which a plate, attached to an insulated rod is suspended over the material to be tested. The rod, which extends to an upper container, is fitted with a narrow leaf of gold or aluminium foil. A telescope with a graduated scale for reading the fall of the leaf is affixed to the front of the upper container. When an electrical charge is placed upon the rod the leaf will rise. Discharge of the instrument with consequent fall of the leaf occurs when a radioactive substance is placed in the lower chamber and the rate of fall of the leaf is proportional to the radioactivity of the material being tested. This method, while limited in its scope for accurate determinations, serves, however, as a ready qualitative test and for control work in plant operation.

For accurate measurements the emanation method is the one most usually employed, provided the amount of radium to be measured is sufficiently small (not over 5×10^{-5} milligrams). This method is based upon the fact that radium emanation or radon, the disintegration product of radium, is a gas which can be completely separated from a solution, the radium content of which is proportional to the equilibrium quantity of emanation.

As the instruments used in the measurement of radium are highly sensitive to any source of radioactivity, it was deemed advisable to carry out this work at a distance from where the treatment of the pitchblende was being carried on. Accordingly, a laboratory was fitted up in the Mines Branch Building containing all the necessary apparatus for the emanation method of radium determination.

The method of carrying out a determination requires very careful manipulation and is dependent on numerous radioactive conditions. A solution containing radium is boiled in a flask to expel all the emanation and then sealed, the time being noted. Emanation now accumulates at a definite rate and in 3·825 days one half the equilibrium amount will be present in the solution. The amount present at any specific time can be calculated or determined from tables of the progressive accumulation of radium emanation. The flask is connected to a gas burette which is filled with hot sodium hydroxide solution and heat is applied to the flask. When the gas pressure in the flask is sufficient the stop-cock is opened and the emanation is boiled off and collected in the burette, the alkaline solution being forced back into a collecting funnel. After eight or ten minutes' boiling the time is recorded and the collected gas is drawn through a drying tube into an evacuated gas-tight ionization chamber. Air from outside is then drawn through the system until the pressure in the chamber has reached that of the atmosphere. The chamber is then set aside for three hours to allow the emanation to come into equilibrium with the three succeeding disintegration products, radium A, radium B, and radium C. The head containing the leaf system and telescope is fitted on to the chamber and the instrument is charged at a suitable potential for 15 minutes. The rate of fall of the leaf is then read and the amount of radium calculated. The type of instrument used in this laboratory for the determination of radium was the Lind interchangeable electroscope. Calibration of the instrument is effected by transferring the emanation from a standard solution of known radium content into the ionization chamber either by boiling, as described above, or by bubbling air through the solution. The fall of the leaf in divisions per minute will represent a known amount of radium and is used as the factor for that chamber in determining a solution of unknown radium content.

The method used for determining the radium in leached residues and the first sulphate concentrate consists in making a fusion of an accurately weighed sample of the material with potassium bisulphate in a pyrex test tube. The fusion is boiled long enough to expel the last traces of emanation and sealed while hot with a rubber stopper containing two outlet tubes, the ends of which are drawn to capillaries and sealed. After standing for a sufficient time to allow the emanation to accumulate, the tube containing the sample is connected in series with three tubes, one containing strong sodium hydroxide, the next calcium chloride, and the last sulphuric acid. An evacuated chamber is attached to the sulphuric acid tube and the system placed under a slight vacuum. An aspirating bulb containing sulphuric acid is placed ahead of the sample tube with a cock between. By adjusting the stop-cock on the ionization chamber, the flow of air through the system may be regulated. The capillary tips are broken and heat is applied to the tube and the fusion boiled. After all the emanation has been transferred to the chamber, the tap is closed and the chamber set aside for three hours before readings are taken.

The following data are made use of in the necessary calculations:—

x = Number of milligrams of radium contained in solution.
t = Time of accumulation of emanation in hours.
N = Divisions per minute leaf fall given by emanation from solution.
n = Divisions per minute leaf fall due to natural drift of electroscope.
c = Number of milligrams of radium in equilibrium giving with the same electroscope 1 division fall per minute.
λ = Transformation constant of emanation = $\cdot 1800$ $(\text{day})^{-1}$
$=e^{-\lambda t}$ = Fraction of emanation remaining after time t.
$1-e^{-\lambda t}$ = Fraction of equilibrium amount of emanation formed in t days.

$$\text{Then } x = \frac{(N - n)\,c}{1-e^{-\lambda t}}$$

The factor $1 - e^{-\lambda t}$ is obtained from tables for any time t.

Part C

PRECAUTIONS FOR WORKERS IN THE TREATING OF RADIUM ORE

W. R. McClelland

The hazards involved in the handling of high-grade radioactive materials make necessary the adoption of certain precautions. Recent investigations in the field of radium poisoning have led to the conclusion that precautions are necessary even in the handling of substances of low radioactivity. The ingestion of small amounts of radioactive dust or emanation over a long period of time will cause a building up of radioactive material in the body, which eventually may have serious consequences. Lung cancer, bone necrosis, and rapid anaemia are possible diseases due to the deposition of radioactive substances in the cell tissue or bone structure of the body.

A careful study of the literature on this subject has been made and precautionary measures adopted for the protection of the staff employed on radium work.

The hazards incident to the work of this Division are primarily the inhalation of emanation and radioactive dust. To reduce this hazard ventilating fans were installed for carrying off the fumes from active solutions. Rubber gloves are worn when handling the higher grade concentration products and personal cleanliness on the part of the workers is emphasized.

During the carrying out of large-scale experiments electroscopic tests on the expired air of the workers are carried out regularly in order to ascertain to what extent they may have absorbed radioactive material. This test is an extremely sensitive one and indicates whether the radioactivity is increasing or being eliminated from the body. A test on one of the workers showed the equivalent of $0 \cdot 163 \times 10^{-2}$ micrograms of radium. Four days later a test gave negative results. The radioactivity was probably due to the inhalation of emanation and during the period between the tests it had been eliminated from the body. Another test on the same worker after he had been working on a large-scale test gave negative results. Improved methods of ventilation were probably responsible for his freedom from radioactivity. Should the tests continue to show that the amount of radioactivity is increasing in the worker, it is necessary to remove him from the work until such time as the tests give a negative or decreasing result.

By a careful check on the workers and the adoption of all necessary precautions, it is possible to reduce the hazards to a minimum. The fact, however, that radium or radioactive substances once deposited in the bone structure of the body are impossible to eliminate makes the taking of every precaution a most necessary factor in the treatment of pitchblende for the recovery of radium.

V

REPORT OF INVESTIGATIONS: SECTION OF FERROUS METALLURGY

Thos. W. Hardy

The application of low-temperature reduction (sponge iron) methods to Canadian ores has continued to engage the attention of the staff during the past year. In summarizing the work for the year 1930, it was pointed out that for the production of sponge iron suitable for steelmaking, only those ores that were amenable to concentration to a high degree could be considered suitable, and that, for this reason, laboratory studies on the concentration of our iron ores might well precede the study of their reducibility. Such studies are not only valuable in connexion with the sponge iron problem, but, applied to our low-grade ores, are of even greater usefulness in connexion with the beneficiation of iron ores for blast furnace use. In line with this idea, laboratory concentration tests were carried out on such iron ores as were available. Reports on such tests on Bathurst (New Brunswick) ore and Texada Island (British Columbia) ore will be found in the following pages. These reports show that although Bathurst ore can hardly be considered suitable for the manufacture of sponge iron, it is capable of producing a high-grade concentrate for blast furnace use. Texada Island ore, on the other hand, is indicated as a very satisfactory ore for the production of a high-grade sponge iron, being low in sulphur and phosphorus.

In sponge iron processes, such as the Musso, in which bituminous coal is mixed with the ore and acts as the reducing agent, it has been found by experience that the sulphur content of the coal is found largely associated with the concentrated sponge. Since such high-sulphur sponge iron is not acceptable to the steelmaker, it is very desirable that some means be devised to prevent this sulphur contamination. As a result of a considerable amount of investigation the writer has found that if lime be charged with the ore-coal mixture the sulphur content of the mixture associates itself with the lime, forming calcium sulphide, and that if the magnetic separation practice is efficient, this calcium sulphide is largely rejected in the concentration of the sponge. A report of the work along these lines will be found elsewhere.

The commercial utilization of the considerable iron content of the nickel-copper ores of the Sudbury district has been a matter of considerable interest to those concerned with the conservation of our natural resources, since, under conditions obtaining at present, this iron is wasted. During the past year a process has been worked out whereby this iron is made available as a nickeliferous sponge iron. That this process produces a nickel iron sponge, low in copper and quite suitable for conversion into nickel steels and irons, is shown by a report to be found in the following pages.

During the year, several Canadian industrial concerns have sought the aid and advice of the staff in solving some of the difficulties encountered in their efforts to supply Canadian-made metal products to the trade. While this work is not spectacular and is usually necessarily of a confidential nature, it is of considerable value to the industries concerned.

Report No. 419

THE PRODUCTION OF LOW-SULPHUR SPONGE IRON FROM ORE-COAL MIXTURES

T. W. Hardy, H. H. Bleakney, and W. S. Jenkins

Object of Investigation. In the production of sponge iron from a mixture of iron ore and a solid reducing agent it has been found necessary to use ores and reducing agents substantially free from sulphur if a concentrated sponge low in sulphur is required. This is a serious limitation on an otherwise desirable method of reduction. The object of this investigation was to devise a method or methods by which a sponge iron comparatively low in sulphur may be produced from ores and coals of commercial grades.

General Considerations. For the production of sponge iron for subsequent conversion into steel, several processes have been proposed in which reduction is effected by heating a mixture of ore and solid carbonaceous material under suitable conditions. Among these is the Musso process in which a mixture of crushed ore and bituminous coal is fed into and carried through an externally heated, rotating retort. With this process, as with others, it has been found that a large percentage of the sulphur content of the raw materials (ore and reducing agent) is found in the magnetically concentrated sponge iron.

It is at present not difficult to obtain ores so low in sulphur, that, as far as they are concerned, there should be no difficulty in producing a sponge iron comparatively low in this impurity. A solid reducing agent low in sulphur is, however, not so simple. In some cases it has been found possible to use materials such as wood waste, but as far as the most obvious and otherwise desirable and commercial reducing agent, bituminous coal, is concerned, its use in the manufacture of low sulphur sponge iron has been found impracticable, since such coals, low enough in sulphur, are not commercially available.

Since the sulphur is found in the magnetic concentrate it is clear that it must be present either as mechanically free FeS, which is magnetic, or as a FeS film or particle adhering to the reduced iron. Obviously then, if this sulphur is to be kept out of the magnetic concentrate, it must be converted into some mechanically free, non-magnetic form, thus making it feasible to reject it in the magnetic separation. Under the limitations of the sponge iron process, this non-magnetic form must be either solid or gaseous.

In view of these considerations, it was decided to make a series of comparative tests involving the use of different possible methods of removing or lowering the sulphur content of the magnetically concentrated sponge iron. These tests, which, to provide a basis for comparison, included one in which no attempt was made to influence the sulphur content of the sponge, were four in number.

Group 1. Reduction without any attempt to desulphurize.
Group 2. Reduction in the presence of hydrogen, internally generated.
Group 3. Reduction in the presence of hydrogen, externally generated.
Group 4. Reduction in the presence of lime.

Raw Materials. For these four groups of experiments, Bell (Messabi Range) ore and Fairmont (Pa.) coal, both ground to pass a 20-mesh Tyler screen, were used.

The analyses of these materials are as follows:—

Bell iron ore (as charged)	Per cent
Iron	57·78
Sulphur	0·26

Fairmont coal (as charged)	Per cent
Moisture	1·8
Volatile matter	35·4
Fixed carbon	54·6
Ash	8·2
Sulphur	1·3

As Bell ore, as mined, is said to carry only 0·04 per cent sulphur, it is evident that the ore used for these tests was contaminated with sulphides in the grinding operation.

Experimental Runs

Group 1. Reduction Without any Attempt to Desulphurize.

This group of reductions was carried out simply to provide a standard against which the results of subsequent experiments might be compared.

In each of the tests of this group the following procedure was adopted. An intimate mixture of 50 pounds of Bell ore and 18¼ pounds of Fairmont coal, both ground to pass a 20-mesh Tyler screen, was charged into an electrically heated, rotating retort, which had an initial temperature of 500° F. when charged. The retort and its contents were then brought up to a temperature of 1700 °F. in about 4 hours and held at that temperature for a period of exactly 4 hours. The crude sponge was then discharged into a container provided with a tightly sealing cover and allowed to cool. During the reduction, the distillation and reduction gases burned freely from a pipe extending from the plug forming the door of the retort.

When cold, the crude sponge iron was magnetically concentrated on a Stearns drum-type magnetic separator. Since the problem of obtaining a sponge iron concentrate low in sulphur is necessarily influenced by the efficiency of the magnetic separation, it was necessary to adopt a concentrating practice that would eliminate the iron-magnetic material as completely as possible, and, at the same time, see that all the runs of this and other groups of experiments were concentrated under precisely similar conditions. The procedure adopted in concentrating the crude sponge iron in all the experiments recorded in this report is as follows:—

The crude retort product was first given two passes on the Stearns machine, which operations removed the bulk of the excess coke and ash. The rough concentrate was then ground to pass a 40-mesh Tyler screen after which it was given three more passes on the Stearns magnetic separator. The final −40 concentrate and the combined tailings from the five passes were then carefully sampled and chemically analysed.

In order to measure the efficiency of the dry concentration, a small-scale wet magnetic separation test was also made on the −40-mesh material. This wet concentration test was made in the Davis magnetic tube, special precautions being taken to prevent the oxidation of the products during the test. Although the results of these Davis tube tests probably cannot be duplicated in commercial practice, they are of interest here in indicating the efficiency of the dry separations.

Separations were also made on the material ground to pass a 100-mesh Tyler screen. For these tests the final −40-mesh concentrate was ground to −100 mesh and given two passes on the Stearns machine. The concentrate and tailing were then sampled and analysed. In the tabulated results the analyses and weight given for the 100-mesh tailing are not those of the product of these two final passes but are the values for the combined tailings resulting from the seven passes to which the retort product was subjected. In other words the weights and analyses given for the products of the 100-mesh concentration are those obtained from the concentration of the crude retort product, ground to 100 mesh.

As in the case of the −40-mesh material, wet magnetic concentration tests in the Davis tube were also made on the −100-mesh material.

The results obtained in this group of tests are given in Table I. These results are discussed later in this report.

Group 2. Reduction in the Presence of Hydrogen, Internally Generated.

In view of the opinion held by some that sulphur may be removed from sponge iron by hydrogen as hydrogen sulphide, it was decided to make a series of reductions in the presence of hydrogen.

In each run of this group the charge was subjected to the action of hydrogen gas during a considerable part of the reduction period, the hydrogen being generated within the retort by the action of steam on the carbon charged with the ore. To provide this carbon, an additional amount of coal, and hence of sulphur, was added.

In the carrying out of these tests conditions were kept as far as possible the same as obtained in the comparison tests of Group 1, the object being to eliminate all variables but the one being investigated. The procedure adopted was as follows:—

A mixture of 50 pounds of Bell ore and 24¼ pounds Fairmont coal, both ground to pass a 20-mesh Tyler screen, was charged into an electrically heated rotary retort, which had an initial temperature of 500° F. The retort and its contents were then brought up to a temperature of 1700° F. in about 4 hours and held at that temperature for exactly 4 hours. At the end of the second of these final four hours, the introduction, drop by drop of 1,800 c.c. (3·97 pounds) of water into the retort through a pipe in its rear axis, was begun and continued for an hour, so that the final hour of reduction was free from any unusual circumstance. The crude sponge iron was then discharged into the special container, and, when cold, concentrated in precisely the same manner as described for the tests in Group 1.

The results obtained in this group of tests are given in Table II, and are discussed later in this report.

Group 3. Reduction in the Presence of Hydrogen, Externally Generated.

The method of generating hydrogen within the retort by the action of steam on carbon is open to criticism for two reasons: first, the problem of desulphurization is greatly increased because of the additional sulphur introduced by the carbon provided for the water-gas reaction; and second, the steam may react with the reduced iron instead of the carbon and thus make reduction more difficult.

In view of these objections it was deemed wise to carry out a few tests in which reduction took place in the presence of hydrogen, externally generated. Since Ottawa city gas contains 40 to 45 per cent hydrogen, and it was convenient to use, it was utilized as the source of hydrogen.

In the carrying out of each run of this group, the following method was followed:—

A mixture of 50 pounds of Bell ore and 18¼ pounds of Fairmont coal, both ground to pass a 20-mesh Tyler screen, was charged into an electrically heated rotary retort which had an initial temperature of about 500° F. The retort and its contents were then brought up to a temperature of 1700° F. in about 4 hours and held at this temperature for exactly 4 hours. During the last two of this final 4-hour period a stream of city gas containing 45 per cent of hydrogen was passed through the retort. At the end of the 4-hour reduction period the crude sponge iron was discharged into a special container and allowed to cool.

In each case, the crude sponge iron was magnetically concentrated in precisely the same manner as described for the tests of Group 1.

The results obtained in this group of tests are given in Table III and are discussed later in this report.

Group 4. Reduction in the Presence of Lime.

In view of the fact that in smelting operations the use of a high-lime slag makes desulphurization commercially feasible, through the formation of calcium sulphide, the introduction of lime with the ore-coal mixture into the sponge iron reduction chamber has been suggested by several workers, although they apparently were not enthusiastic about its probable efficiency in removing the sulphur from the iron. "In some cases 10 per cent of lime mixed with the ore-coal charge increased the elimination of sulphur by 10 or 15 per cent but not enough experiments were made to determine the probable value of such treatment in large-scale operations. It is doubtful, however, if the use of lime will go far in solving the problem of eliminating sulphur."[1]

In order to obtain some first-hand data on the effect of lime in promoting the elimination of sulphur, a series of reduction in which lime was charged with the ore-coal mixture was carried out. At first a considerable excess of lime was used with the idea of cutting this excess down should the method prove successful.

In each of these tests the following procedure was adopted. A mixture of 50 pounds of Bell ore, 18¼ pounds Fairmont coal, and 12 pounds of burned lime was charged into an electrically heated rotary retort, which had an initial temperature of about 500° F. The retort and its contents

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[1] U.S. Bureau of Mines, Bull. 270, p. 140.

were then brought up to a temperature of 1700° F. in about 4 hours and held at that temperature for exactly 4 hours. The crude sponge iron was then discharged into a special container to cool. When cold, the crude sponge iron was magnetically concentrated in precisely the same manner as described for the tests in Group 1.

The results obtained in this group of tests are given in Table IV and are discussed later in this report.

Results Obtained

From a study of the data presented in Table I, it is apparent that the concentrated sponge iron produced from straight ore-coal mixtures such as were used in these experiments, is so high in sulphur as to be entirely unsuitable for direct conversion into steel. The tabulated results show that over 50 per cent of the sulphur contained in the raw materials fed into the metallizing furnace is found in the 40-mesh dry concentrate and even if we grind to −100 mesh and concentrate wet, the concentrates still retain about 40 per cent of the sulphur content of the charge. The results of this group of tests show that no matter how efficient the concentration practice may be, the sulphur content of the sponge iron concentrate will average well over 0·5 per cent, which is ten times as much as the sulphur content of pig iron and scrap steel ordinarily used in steelmaking.

That the presence of hydrogen in the reducing atmosphere during the low-temperature reduction process does not result in the lowering of the sulphur content of the sponge iron concentrate is made clear from a study of Tables II and III. In each of these groups of experiments, the sulphur content of the concentrate (resulting from the magnetic concentration of the crude sponge iron) is still well over 0·5 per cent and on this account is totally unsuitable for direct conversion into steel.

That a low sulphur sponge iron can be obtained from such ore-coal mixtures by charging lime into the metallizing retort with the ore and coal is clear from the data presented in Table IV. A study of this data makes it clear that although the crude retort product still contains about as much sulphur as it would have, had no lime been charged, yet the sulphur has been transferred from the iron particles to the lime, with which it is apparently associated as sulphide of lime. This sulphide of lime, being nonmagnetic, is largely removed from the iron in the magnetic concentration.

It will be noted that the average sulphur content of the −40-mesh concentrate from the ore-coal-lime mixture is about 0·14 per cent and from the −100-mesh material is about 0·12 per cent. That these residual contents of sulphur are largely if not entirely mechanically-held calcium sulphide is evidenced by the fact that by wet magnetic concentration these values are reduced to 0·06 per cent and 0·04 per cent respectively. In view of this and of what is commonly known of the difficulty of separating in a dry way finely divided magnetic materials from finely divided nonmagnetics, it is reasonable to assume that with ore ground to a coarser mesh than was used in these tests, dry magnetic concentration of the sponge produced from it in the presence of lime would result in a product with a sulphur content more closely in line with that obtained from the wet separation of the finely ground materials.

TABLE I

Group 1. Mixture Ore and Coal

Charge heated to 1700° F. and held for 4 hours.

Charge—

Bell ore....................	50·0 lb.
Fairmont coal..............	18·2 lb.

CONCENTRATION PRACTICE

A. Separation of —40-Mesh Material

1. Dry Magnetic Separation—
   - *(a)* Retort product—2 passes, Stearns machine
   - *(b)* Rough concentrate, *(a)* ground to —40 mesh
   - *(c)* —40-mesh rough concentrate—3 passes on Stearns machine
2. Wet Magnetic Separation—
   - *(a)* —40-mesh dry concentrate treated in Davis tube

B. Separation of —100-Mesh Material

1. Dry Magnetic Separation—
   - *(a)* Retort product—2 Passes, Stearns machine
   - *(b)* Rough concentrate *(a)* ground to —40 mesh
   - *(c)* —40 mesh rough concentrate—3 passes Stearns machine
   - *(d)* —40-mesh concentrate *(c)* ground to —100 mesh
   - *(e)* —100-mesh product *(d)*—2 passes
2. Wet Magnetic Separation—
   - *(a)* —100-mesh dry concentrate—Davis tube.

Crude Sponge Iron					Dry Magnetic Concentration								Wet Magnetic Concentration								Iron Recovery		Sulphur Distribution							
					Concentrate				Tailing				Concentrate				Tailing													
Test	Wt.	Analysis			Wt.	Analyses			Wt.	Analyses			Wt.	Analyses			Wt.	Analyses					Dry Separation				Wet Separation			
No.	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Dry separation	Wet separation	Retort charge	Gases	Conc.	Tail.	Retort charge	Gases	Conc.	Tail.
Minus 40 mesh separation																														
100	100	62·86	59·87	0·62	72·10	82·00	81·20	0·68	27·90	13·43	4·78	0·47	69·69	83·08	82·59	0·71	30·31	16·40	7·62	0·41	94·04	92·11	100	29·39	55·85	14·76	100	29·39	59·24	11·37
101	100	63·35	59·99	0·61	72·55	82·40	80·80	0·64	27·45	13·00	4·99	0·54	70·08	82·60	79·20	0·63	29·92	18·25	15·01	0·57	94·37	91·38	100	29·71	53·43	16·86	100	29·71	50·88	19·41
102	100	63·04	59·89	0·62	71·90	81·85	80·80	0·65	28·10	14·90	6·40	0·54	68·50	82·60	82·60	0·64	31·50	20·51	10·51	0·59	93·35	89·75	100	28·55	53·76	17·69	100	28·55	50·51	20·94
Minus 100 mesh separation																														
100	100	62·86	59·87	0·62	71·00	83·60	82·50	0·63	29·00	12·14	4·48	0·61	64·84	87·06	86·47	0·55	35·16	18·23	10·82	0·74	94·40	89·80	100	29·39	50·93	19·68	100	29·39	40·54	30·07
101	100	63·35	59·99	0·61	71·13	83·60	82·50	0·62	28·87	13·46	4·53	0·59	65·91	85·76	84·16	0·55	34·09	20·13	13·26	0·73	93·88	89·20	100	29·71	50·82	19·47	100	29·71	41·77	28·50
102	100	63·04	59·89	0·62	69·70	82·90	82·90	0·58	30·30	17·39	6·96	0·71	62·94	87·00	86·70	0·54	37·06	22·35	14·36	0·75	91·66	86·86	100	28·55	46·55	24·90	100	28·55	39·17	32·28

TABLE II

Group 2. Mixture Ore and Coal

Reduction in presence of hydrogen generated within the retort.

Charge heated to 1700° F. and held constant for a 4-hour period. At the end of the 2nd hour of the final period, 1800 c.c. of water are added to the retort over a period of 45 minutes.

Charge—	
Bell ore	50·0 lb.
Fairmont coal	24·2 lb.
Water	1800·0 c.c.

CONCENTRATION PRACTICE

A. SEPARATION OF −40-MESH MATERIAL

1. Dry Magnetic Separation—
   (*a*) Retort product—2 passes, Stearns machine
   (*b*) Rough concentrate (*a*) ground to −40 mesh.
   (*c*) −40-mesh rough concentrate—3 passes
2. Wet Magnetic Separation—
   (*a*) 40-mesh dry concentrate treated in Davis tube

B. SEPARATION OF −100-MESH MATERIAL

1. Dry Magnetic Separation—
   (*a*) Retort product—2 passes, Stearns machine
   (*b*) Rough concentrate (*a*) ground to −40 mesh
   (*c*) −40-mesh rough concentrate—3 passes
   (*d*) −40-mesh concentrate ground to −100 mesh
   (*e*) −100 mesh product (*d*)—2 passes
2. Wet Magnetic Separation—
   (*a*) −100-mesh dry concentrate treated in Davis tube

Crude Sponge Iron					Dry Magnetic Concentration								Wet Magnetic Concentration								Iron Recovery		Sulphur Distribution							
					Concentrate				Tailing				Concentrate				Tailing													
Test	Wt.	Analysis			Wt.	Analyses			Wt.	Analyses			Wt.	Analyses			Wt.	Analyses					Dry Separation				Wet Separation			
No.	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Dry separation	Wet separation	Retort charge	Gases	Conc.	Tail.	Retort charge	Gases	Conc.	Tail.
Minus 40 mesh separation																														
103	100	60·95	58·16	0·68	72·70	79·60	78·21	0·64	27·30	11·29	4·75	0·77	70·04	80·98	78·01	0·69	29·96	14·12	11·75	0·66	94·95	93·06	100	34·34	44·83	20·83	100	34·34	46·66	19·00
104	100	58·64	55·32	0·65	69·70	79·00	77·52	0·67	30·30	11·84	4·28	0·61	67·60	78·63	74·72	0·65	32·40	16·97	14·84	0·65	93·89	90·64	100	33·58	47·71	18·71	100	33·58	44·90	21·52
105	100	60·04	55·80	0·65	68·58	80·13	78·63	0·68	31·42	16·20	5·97	0·57	66·90	80·13	76·61	0·71	33·10	19·43	13·74	0·53	91·53	89·28	100	33·94	47·40	18·66	100	33·94	48·26	17·80
Minus 100 mesh separation																														
103	100	60·95	58·16	0·68	71·23	80·70	79·99	0·63	28·77	12·06	4·10	0·80	64·82	85·14	83·06	0·55	35·18	16·37	12·29	0·92	94·31	90·54	100	34·34	43·33	22·33	100	34·34	34·42	31·24
104	100	58·64	55·32	0·65	68·90	80·40	78·70	0·61	31·10	10·42	3·50	0·74	63·62	82·78	81·79	0·55	36·38	16·43	9·04	0·82	94·47	89·81	100	33·58	42·95	23·47	100	33·58	35·75	30·67
105	100	60·04	55·80	0·65	67·68	81·00	80·55	0·62	32·32	16·17	3·97	0·71	60·64	85·37	84·18	0·54	39·36	21·04	12·08	0·82	91·30	86·22	100	33·94	42·65	23·41	100	33·94	33·28	32·78

TABLE III

Group 3. Mixture Ore and Coal

Charge heated to 1700°F. and held constant for a period of 4 hours. City gas was passed into the retort during the final 2 hours.

Reduction in presence of hydrogen generated externally.

Charge—

Bell ore................	50·0 lb.
Fairmont coal.........	18·2 lb.

CONCENTRATION PRACTICE

A. SEPARATION OF −40-MESH MATERIAL

1. Dry Magnetic Separation—
    (a) Retort product—2 passes, Stearns machine
    (b) Rough concentrate *(a)* ground to −40 mesh
    (c) −40-mesh rough concentrate—3 passes
2. Wet Magnetic Separation—
    (a) −40-mesh dry concentrate treated in the Davis tube

B. SEPARATION OF −100-MESH MATERIAL

1. Dry Magnetic Separation—
    (a) Retort product—2 passes, Stearns machine
    (b) Rough concentrate *(a)* ground to −40 mesh
    (c) Rough concentrate *(b)*—3 passes
    (d) −40-mesh concentrate ground to −100 mesh
    (e) −100-mesh concentrate *(d)*—2 passes
2. Wet Magnetic Separation—
    (a) −100-mesh dry concentrate treated in Davis tube

Crude Sponge Iron					Dry Magnetic Concentration								Wet Magnetic Concentration								Iron Recovery		Sulphur Distribution							
					Concentrate				Tailing				Concentrate				Tailing													
Test	Wt.	Analysis			Wt.	Analyses			Wt.	Analyses			Wt.	Analyses			Wt.	Analyses					Dry Separation				Wet Separation			
No.	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Dry separation	Wet separation	Retort charge	Gases	Conc.	Tail.	Retort charge	Gases	Conc.	Tail.
Minus 40 mesh separations																														
106	100	64·14	62·72	0·55	79·02	78·69	77·69	0·54	20·98	9·32	6·34	0·58	76·91	79·38	77·99	0·54	23·09	13·38	11·87	0·59	96·95	95·19	100	37·85	48·22	13·93	100	37·85	46·93	15·22
107	100	63·50	61·75	0·57	77·35	79·28	78·29	0·55	22·65	9·61	5·25	0·65	74·51	80·37	78·78	0·52	25·49	14·20	11·97	0·72	96·57	94·35	100	33·27	49·84	16·89	100	33·27	45·38	21·35
108	100	63·43	61·88	0·59	77·92	78·79	77·97	0·57	22·08	9·22	5·09	0·68	75·58	79·48	78·56	0·53	24·42	13·76	10·23	0·78	96·79	94·70	100	31·32	51·70	16·98	100	31·32	46·63	22·05
Minus 100 mesh separation																														
106	100	64·14	62·72	0·55	78·30	80·07	78·69	0·51	21·70	6·68	5·12	0·69	72·82	82·46	81·08	0·49	27·18	15·06	13·54	0·71	97·77	93·62	100	37·85	45·12	17·03	100	37·85	40·32	21·83
107	100	63·50	61·75	0·57	76·48	79·50	78·49	0·54	23·52	11·22	7·32	0·67	69·60	83·26	82·06	0·48	30·40	18·25	15·25	0·78	95·85	91·27	100	33·27	48·36	18·37	100	33·27	39·10	27·63
108	100	63·43	61·88	0·59	77·22	79·68	78·76	0·59	22·78	8·35	4·66	0·58	72·07	81·64	80·96	0·48	27·93	16·44	12·63	0·86	97·00	92·76	100	31·32	53·03	15·65	100	31·32	40·27	28·41

TABLE IV

Group 4. Mixture Ore, Coal, and Lime

Reduction in presence of lime

Charge heated to 1700°F. and held constant for a period of 4 hours.

Charge—

Bell ore................	50·0 lb.
Fairmont coal..........	18·2 lb.
Lime...................	12·0 lb.

CONCENTRATION PRACTICE

A. SEPARATION OF −40-MESH MATERIAL

1. Dry Magnetic Separation—
   - *(a)* Retort product—2 passes, Stearns machine
   - *(b)* Rough concentrate *(a)* ground to −40 mesh
   - *(c)* −40-mesh rough concentrate—3 passes
2. Wet Magnetic Separation—
   - *(a)* −40-mesh dry concentrate *(c)* treated in Davis tube

B. SEPARATION OF −100-MESH MATERIAL

1. Dry Magnetic Separation—
   - *(a)* Retort product—2 passes, Stearns machine
   - *(b)* Rough concentrate *(a)* ground to −40 mesh
   - *(c)* Rough concentrate *(b)*—3 passes
   - *(d)* −40-mesh concentrate ground to −100 mesh
   - *(e)* −100-mesh material *(d)*—2 passes
2. Wet Magnetic Separation—
   - *(a)* −100-mesh dry concentrate treated in Davis tube.

Crude Sponge Iron					Dry Magnetic Concentration								Wet Magnetic Concentration								Iron Recovery		Sulphur Distribution							
					Concentrate				Tailing				Concentrate				Tailing													
Test	Wt.	Analysis			Wt.	Analyses			Wt.	Analyses			Wt.	Analyses			Wt.	Analyses					Dry Separation				Wet Separation			
No.	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Per cent	Total iron	Metallic iron	Sulphur	Dry separation	Wet separation	Retort charge	Gases	Conc.	Tail.	Retort charge	Gases	Conc.	Tail.
Minus 40 mesh separation																														
109	100	51·13	47·89	0·55	62·48	75·99	72·22	0·12	37·52	9·72	7·38	1·26	53·53	84·24	80·56	0·03	46·47	12·99	10·27	1·15	92·86	88·19	100	22·12	10·62	67·26	100	22·12	2·27	75·61
110	100	52·21	44·83	0·60	63·90	74·90	68·59	0·18	36·10	12·06	2·78	1·33	54·53	83·45	78·47	0·09	45·47	14·74	4·49	1·21	91·67	87·16	100	14·23	16·44	69·33	100	14·23	7·02	78·75
111	100	52·04	48·32	0·59	62·14	77·18	74·40	0·11	37·86	10·77	5·52	1·37	52·61	86·45	81·36	0·05	47·39	13·84	11·64	1·19	92·16	87·40	100	16·06	9·72	74·22	100	16·06	3·74	80·20
Minus 100 mesh separation																														
109	100	51·13	47·89	0·55	61·77	76·98	75·60	0·09	38·23	9·37	3·11	1·30	49·62	87·30	84·60	0·04	50·38	15·51	11·73	1·05	93·00	84·72	100	22·12	7·87	70·01	100	22·12	2·81	75·07
110	100	52·21	44·83	0·60	63·15	75·90	69·19	0·16	36·85	11·61	3·09	1·35	50·10	85·90	83·34	0·03	49·90	18·39	6·16	1·17	91·80	82·43	100	14·23	14·44	71·33	100	14·23	2·15	83·62
111	100	52·04	48·32	0·59	61·31	79·08	76·57	0·10	38·69	9·19	3·55	1·37	50·27	87·89	85·90	0·04	49·73	15·80	10·33	1·16	93·17	84·90	100	16·06	8·72	75·22	100	16·06	2·86	81·08

TABLE V

Group 5. Coarse Ore, Fine Coal

Reduction in presence of Lime

Charge heated to 1700°F. and held constant for a period of 4 hours.

Charge—

Bell ore sinter (−6+10 mesh)	50 lb.
Fairmont coal	18·2 lb.
Lime	12·0 lb.

Concentration Practice—

A. As Metallized—
  (1) Retort product—3 passes on Stearns drum-type machine.

B. Minus 40-mesh separation—
  (1) Retort product—3 passes on Stearns drum-type machine.
  (2) Rough concentrate (1) ground to −40 mesh.
  (3) −40-mesh rough concentrate—2 passes on Stearns machine.

Dry Magnetic Concentration

Test No.	Crude Sponge Iron				Concentrate				Tailing				Iron recovery	Sulphur Distribution			
	Weight, per cent	Analyses			Weight, per cent	Analyses			Weight, per cent	Analyses				Dry separation			
		Total iron	Metallic iron	Sulphur		Total iron	Metallic iron	Sulphur		Total iron	Metallic iron	Sulphur	Dry separation	Retort charge	Gases	Conc.	Tailing
Separation as metallized (−6+10) mesh.																	
116	100	52·45	48·30	0·36	71·20	72·37	67·04	0·07	28·80	3·22	1·98	1·07	98·23	100	19·87	11·15	68·98
117	100	52·85	46·38	0·37	69·33	73·36	65·14	0·07	30·67	6·48	3·96	1·04	96·23	100	16·95	11·05	72·00
Separation −40-mesh.																	
116	100	52·45	48·30	0·36	70·54	72·66	68·31	0·07	29·46	4·07	0·37	1·06	97·72	100	19·87	10·88	69·25
117	100	52·85	46·38	0·37	68·62	73·06	65·14	0·07	31·38	8·67	5·35	1·03	94·86	100	16·95	11·35	71·70

In order to confirm this assumption two runs were made similar to those summarized in Table IV except that the ore was first sintered and crushed to pass through a 6-mesh Tyler screen and remain on a 10 mesh. In each of these runs, the crude retort product was first given three passes on the drum-type separator in order to get rid of as much of the mechanically-free non-magnetic material as possible. This rough concentrate was then ground to −40 mesh and given two passes more on the same machine. The results obtained are shown in Table V. These results show that the sulphur content of the concentrate, after only three passes, has been reduced to 0·07 per cent, and while it must be admitted that the sulphur content of the charge was lower than that of the previous charges due to the sinter containing only 0·06 per cent sulphur, it is reasonable to assume that the low sulphur content of the final concentrate is due to the ready separation of the calcium sulphide from the comparatively coarse sponge in the dry magnetic concentration.

SUMMARY

Several suggested methods of producing a low sulphur sponge iron from ore-coal mixtures have been investigated. Of these but one proved satisfactory.

The method consists of charging lime with the mixture of ore and coal into the reduction chamber. By so doing the sulphur is caused to associate itself with the lime and it is, therefore, feasible to remove it from the reduced iron by magnetic separation. The extent to which this removal takes place depends largely upon the efficiency of the magnetic concentration practice.

No attempt has been made to study the effect of different proportions of lime. The proportion used in these experiments was adequate to fix the sulphur as calcium sulphide and is probably unnecessarily large.

Report No. 420

THE LABORATORY CONCENTRATION OF TEXADA ISLAND IRON ORE

T. W. Hardy and H. H. Bleakney

Object of Investigation. To determine the degree to which Texada Island ore may be beneficiated, with particular reference to its adaptability to the manufacture of sponge iron.

Shipment. A shipment of 2,496 pounds of iron ore was received on November 4, 1930, from Mr. John D. Galloway, Provincial Mineralogist, Province of British Columbia.

Nature of Ore and Chemical Analysis of Shipment. The ore is a high-grade magnetite. The shipment was crushed and carefully sampled to provide a head sample. This sample, on chemical analysis, gave the following results:

(Dried at 105° C.)

Fe_3O_4	85·59	Fe	62·06
$CuFeS_2$	0·13	S	0·15
FeS_2	0·20	P	0·056
MnO_2	0·16	Mn	0·10
SiO_2	8·41	Cu	0·045
P_2O_5	0·13		
CaO	2·80		
MgO	1·17		
Al_2O_3	1·31		
	99·90		

The analysis indicates a very good ore. Much of the gangue present exists in comparatively coarse particles visible to the naked eye, so that it should be readily removable by crushing and magnetic concentration. As the sulphur content of the ore exists as sulphides in the gangue, it should also be easily eliminated by concentration.

General Considerations. With a high-grade ore such as this, the question of beneficiation does not ordinarily come up as its grade is already considerably higher than the average blast furnace feed. In considering the suitability of an ore for the manufacture of sponge iron, however, the question of beneficiation is always important since, for conversion into steel in a steel-melting furnace, the sponge iron should not contain more than 5 to 10 per cent of iron metallic matter. A little consideration will make it apparent that in order to bring this about, the gangue present in the ore must be largely eliminated by concentration methods either before or after the low-temperature reduction operation.

Experimental Method

For these small-scale tests, it was most convenient to use the method described in detail in the report on "The Laboratory Concentration of Wabana Iron Ore."[1] The reducing roast was omitted, however, as the iron is already in the form of the magnetic oxide. The method, in brief, involves the crushing of the ore to various degrees of fineness and then wet-magnetic concentration of the different sized samples in the Davis magnetic tube. Separations were made on samples ground to pass 10, 20, 40, 60, and 100-mesh Tyler screens and, in each case, the magnetic concentrate and non-magnetic tailing were weighed and assayed.

Results Obtained:

—	Size of ore fed to magnetic tube (mesh)				
	−10	−20	−40	−60	−100
	per cent	per cent	per cent	per cent	per cent
Tube Feed—					
Iron	61·20	61·20	61·20	61·20	61·20
Insoluble	11·75	11·75	11·75	11·75	11·75
Sulphur	0·14	0·14	0·14	0·14	0·14
Phosphorus	0·056	0·056	0·056	0·056	0·056
Tube Concentrates—					
Weight	91·7	91·00	89·3	87·7	85·7
Iron	65·60	65·95	67·00	68·35	69·70
Insoluble	7·90	7·20	5·90	4·40	3·00
Sulphur	0·04	0·04	0·03	0·01	0·01
Phosphorus	....	....	....	0·038	....
Tube Tailings—					
Iron	13·55	12·75	12·30	10·65	10·40
Insoluble	53·65	57·15	60·20	63·60	63·85
Sulphur	1·25	1·15	1·06	1·07	0·92
Ratio of Concentration	1·09 : 1	1·10 : 1	1·12 : 1	1·14 : 1	1·17 : 1
Iron Recovered	98·3	98·1	97·8	97·9	97·6
Insoluble Rejected	38·3	44·2	55·2	67·1	78·1
Sulphur Rejected	73·8	74·0	80·8	93·7	93·9
Phosphorus Rejected	....	....	....	40·5	....
Theoretical iron content of sponge iron producible from concentrate	88·0	88·7	90·5	93·0	95·6

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[1] Mines Branch, Dept. of Mines, Canada, Investigations, Ore Dressing and Metallurgy, 1930, pp. 195-198.

SUMMARY AND CONCLUSIONS

The results of these tests show that Texada Island ore responds very well to magnetic concentration. The concentrates from the −40 and finer material will all make a sponge iron containing 90 per cent iron or better, if complete metallization be assumed. These high-iron contents are accompanied by a high recovery (over 97 per cent) and a virtually complete elimination of sulphur.

The tests show that the ore is eminently suitable for the manufacture of sponge iron. Not only is the gangue material readily removed by magnetic concentration but also the sulphur, yielding a concentrate high in iron and low in sulphur and phosphorus.

Should parts of the deposit from which this sample was taken run higher in sulphur than does this shipment, beneficiation would probably be necessary in order to make it suitable for blast furnace use. In this case, the ease with which the sulphur is removed by magnetic concentration suggests that this method rather than sintering be utilized for desulphurization, particularly as the copper associated with the sulphur might then be recovered from the tailing. The ground ore would, of course, have to be sintered in any case before it could be used in the blast furnace, but it is doubtful if sintering can remove the sulphur as completely as has been done by magnetic concentration in these tests.

The suitability of this ore for the manufacture of sponge iron, and the fact that the manufacture of sponge iron lends itself readily to comparatively small-scale operation, suggest that the establishment of a sponge iron industry in British Columbia might be more economically feasible at the present time than the establishment of a conventional blast furnace-open-hearth steel plant which is necessarily large and expensive.

Report No. 421

THE LABORATORY CONCENTRATION OF IRON ORE FROM BATHURST, N.B.

T. W. Hardy and G. S. Farnham

Object of Investigation. To determine the extent to which Bathurst iron ore may be beneficiated, with particular reference to its adaptability to the manufacture of sponge iron.

Shipment. A shipment of 9,865 pounds of iron ore, which was mined near Bathurst, New Brunswick, was received on June 25, 1931, from J. E. McAloney, Canada Iron Foundries, Montreal, Quebec.

Nature of Ore and Chemical Analysis of Shipment. The ore is a magnetite-hematite mixture. To obtain a sample representative of the shipment, it was first crushed to ½ inch. From this ½-inch material a sample of 985 pounds was cut and crushed to pass a 10-mesh Tyler screen. From this −10-mesh material a sample of 100 pounds was cut and ground to pass a 40-mesh Tyler screen. A laboratory sample, cut from this −40-mesh material, gave the following results:—

(Dried at 105° C.)

	Per cent		Per cent
Fe_2O_3	42·76	Fe	45·93
Fe_3O_4	22·15	Mn	1·98
SiO_2	20·46	S	0·26
CaO	2·25	P	0·65
MgO	1·08		
MnO	2·56		
Al_2O_3	1·95		
P_2O_5	1·48		
SO_3	0·66		
Loss on ignition	4·65		

General Considerations. The analysis of the ore indicates that it will be necessary to effect a considerable elimination of silica, phosphorus, sulphur, and other impurities before it can be used for the manufacture of sponge iron.

While the ore, as mined, is suitable for blast furnace use, it is not very desirable principally on account of its high silica content. The appearance of the ore suggests that some of this siliceous material may be freed by grinding to about 60 mesh; it might, therefore, be possible to make a concentrate reasonably low in silica with an iron content that would compare favourably with the better grades of Lake Superior ores.

Experimental Method

In view of the large proportion of hematite in this ore and of the practical difficulties in the way of concentrating such ores in the finely divided condition, it was deemed advisable to convert all the iron to the magnetic oxide by means of a reducing roast, thus making it suitable for wet magnetic concentration.

For these small-scale tests it was convenient to use the method described in detail in the report on "The Laboratory Concentration of Wabana Iron Ore."[1] This method, in brief, involves the roasting of the ore to the magnetic oxide, the grinding of this roasted product to various degrees of fineness, and finally the wet magnetic concentration of these different sized samples in the Davis magnetic tube. Separations were made on samples ground to pass 60, 80, 100, 150 and 200-mesh Tyler screens and in each case the magnetic concentrate and the non-magnetic tailing were weighed and assayed.

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[1] Mines Branch, Dept. of Mines, Canada, Investigations, Ore Dressing and Metallurgy, 1930, pp. 195-198.

—	Size of ore fed to magnetic tube (Tyler mesh)				
	−60	−80	−100	−150	−200
	per cent	per cent	per cent	per cent	per cent
Tube Feed—					
Iron	49·75	49·75	49·75	49·75	49·75
Insoluble	22·95	22·95	22·95	22·95	22·95
Sulphur	0·32	0·32	0·32	0·32	0·32
Phosphorus	0·69	0·69	0·69	0·69	0·69
Magnetic Concentrate—					
Weight	80·66	78·33	75·66	74·33	72·00
Iron	57·51	59·10	60·89	61·49	63·38
Insoluble	14·31	12·70	11·74	10·38	8·85
Sulphur	0·23	0·23	0·21	0·21	0·21
Phosphorus	0·53	0·44	0·43	0·38	0·35
Non-Magnetics—					
Iron	17·34	15·95	15·12	15·75	14·70
Insoluble	59·00	60·00	57·80	59·34	59·21
Sulphur	0·69	0·65	0·66	0·64	0·60
Phosphorus	1·36	1·59	1·50	1·59	1·57
Ratio of Concentration	1·24 : 1	1·28 : 1	1·32 : 1	1·35 : 1	1·39 : 1
Iron Recovered	93·25	93·05	92·60	91·87	91·73
Insoluble Rejected	49·72	56·66	61·31	66·36	72·25
Sulphur Rejected	42·03	43·76	50·35	51·25	52·50
Phosphorus Rejected	38·04	50·05	52·85	59·07	63·48
Theoretical iron content of sponge iron producible from concentrate	73·65	76·29	79·29	80·32	83·57

SUMMARY AND CONCLUSIONS

The results tabulated indicate fairly clearly that even after grinding to −200 mesh, the concentrate obtained is unsuitable for the manufacture of sponge iron, chiefly on account of its phosphorus content, although its comparatively high sulphur and residual gangue contents are also objectionable.

From the blast furnace point of view the results are encouraging, since they indicate that by grinding to −80 mesh it is possible to reject nearly 57 per cent of the insoluble, nearly 44 per cent of the sulphur and about 50 per cent of the phosphorus contained in the ore, which results in a concentrate carrying 59·10 per cent iron, 12·70 per cent insoluble, and 0·44 per cent phosphorus. A concentrate of this analysis is very satisfactory for the production of basic pig iron.

Report No. 422

THE MECHANICAL PROPERTIES OF CERTAIN SAMPLES OF MONEL METAL

T. W. Hardy and H. H. Bleakney

Object of Tests. These tests were made with the object of placing on record for the information of this and other Government departments the mechanical properties that may be expected from Monel metal bars commercially available.

Shipments. The following material was received on March 16, 1931, from Peckover's Limited, Toronto:

5 bars hot-rolled, Monel metal, 1-inch diameter.
1 bar cold-rolled, Monel metal, ¾-inch diameter.
1 bar cold-rolled, high tensile Monel metal, ½-inch diameter.

A second shipment was received on April 8, 1931, from the Huntington, West Virginia, plant of the International Nickel Company.

5 bars cold-rolled, high tensile Monel metal, 1-inch diameter.
1 bar cold-rolled, high tensile Monel metal, ¾-inch diameter.
1 bar cold-rolled, high tensile Monel metal, ½-inch diameter.

All of these bars are said to have been selected at random and to be representative of the general run of hot-rolled and cold-rolled Monel metal.

Method of Testing. The tensile tests were all made on an Amsler tensile testing machine of 200,000 pounds capacity. Bars under ¾ of an inch in diameter were tested in the full or as rolled size. Bars ¾ inch in diameter were tested in both the full, as rolled section and in the machined condition. The 1-inch bars were all tested in the machined condition. Machined bars were turned to 0·505-inch diameter on a gauge length of 2 inches and were provided with threaded ends.

As this metal does not exhibit a well defined yield point, this stress is taken as that which produces a deformation of 0·01 inch over a gauge length of 2 inches. A Berry strain-gauge was used to measure this deformation.

The Izod impact tests were made on an Amsler single-blow impact testing machine.

Results Obtained: (See pages 162 and 163.)

CONCLUSIONS

The results obtained are in agreement with the trade data published by the International Nickel Company.

SERIES 1. Bars Supplied by Peckover's, Limited

A. Hot-rolled bars

Bar No.	Diameter as rolled	Dimensions of test bar		Tensile properties, lb./sq. in.				Izod	Brinell
		Diameter	Gauge length	Yield point	Tensile strength	Elongation	Reduction	Impact	Hardness
	in.	in.	in.			%	%		
1	1	0·505	2	39,900	85,100	43·0	62·8	....	146
2	1	0·505	2	40,100	85,300	43·0	61·8	....	146
3	1	0,505	2	40,100	85,100	43·0	63·3	....	143
4	1	0·505	2	35,300	78,500	45·0	69·0	96 ft./lb.	134
5	1	0·505	2	40,300	85·400	43·5	64·1	....	146

B. Cold-rolled bars

Bar No.	Diameter as rolled	Dimensions of test bar		Tensile properties, lb./sq. in.				Izod	Brinell
		Diameter	Gauge length	Yield point	Tensile strength	Elongation	Reduction	Impact	Hardness
	in.	in.	in.			%	%		
6	Half inch (·501″)	0·501	2	....	102,000	24·0	64·1	....	207
7	¾ inch (·750″)	0·750	2	84,400	96,200	39·0	67·5	....	202
7	“	0·505	2	83,630	95,350	27·0	61·3	102 ft./lb.	202

SERIES 2. Bars Shipped from Huntington, West Virginia.

Cold-rolled high tensile bars.

Bar No.	Diameter as rolled	Dimensions of test bar		Tensile properties, lb./sq. in.				Izod	Brinell
		Diameter	Gauge length	Yield point	Tensile strength	Elongation	Reduction	Impact	Hardness
	in.	in.	in.			%	%		
8	1	0·505	2	91,160	102,210	22·0	66·0	95 ft./lb.	217
9	1	0·505	2	93,230	103,210	23·0	65·5	....	217
10	1	0·505	2	90,850	101,250	22·5	63·5	....	217
11	1	0·505	2	91,610	101,600	23·0	66·5	....	207
12	1	0·505	2	91,600	101,750	23·0	65·5	....	217
13	¾	0·505	2	84,530	93,110	26·0	63·5	107	217
13	¾	0·748	2	86,000	95,600	34·0	70·0	....	207
14	½	0·505	2	99,100	113,300	22·5	64·5	....	241

Report No. 423

THE SEMI-DIRECT PRODUCTION OF NICKEL STEEL FROM SUDBURY ORE

T. W. Hardy and H. H. Bleakney

Introductory

The metallurgical practice at present followed in the treatment of nickel ores such as exist in the Sudbury district is characterized by two features:

(1) The isolation of the nickel in pure form, practically free from iron and other metals.
(2) The wasting of the larger content of iron in the ore as slag.

The facts that these ores contain large amounts of iron and that a large proportion of the nickel so laboriously separated from the iron is subsequently alloyed with iron again in the manufacture of nickel irons and steels, have been the incentives for several attempts to produce nickel-iron alloys from these ores. Probably the most interesting of such attempts is that of G. M. Colvocoresses, who developed and patented a process whereby he produced a nickel-copper pig iron containing nickel, copper, and iron in the same proportions as they existed in the ore.

The Colvocoresses process may be outlined as follows:

(1) The roasting of the nickel-copper-iron sulphides to oxides.
(2) The smelting of these mixed oxides, after suitable agglomeration to a nickel-copper pig iron in an electric shaft furnace or in an ordinary blast furnace.
(3) The conversion of this nickel-copper pig iron into a nickel-copper steel in an open-hearth furnace or Bessemer converter.

Several hundred tons of steel were made experimentally by the Colvocoresses process, but while independent tests more or less confirmed his contention that the copper present in his steel, which was as much as one-third of the nickel content, was equivalent to and acted in the same manner as the same amount of nickel, the process has not been put into commercial use. Among the factors entering into the failure of the Colvocoresses process to find commercial application, it seems probable that the prejudice of the steel trade against copper in steel is not the least important. A contributing factor is undoubtedly the fact that the process would involve considerable capital expenditure and large-scale operation, since both blast furnace and open-hearth plants would be required.

Experimental Methods

In the process described in this report, an attempt bas been made to minimize or eliminate these undesirable features and advantage has been taken of the following facts:

1. Copper sulphide may to a considerable extent be separated from the sulphides of nickel and iron by selective flotation, thus making it possible to produce from the Sudbury and similar ores a high-grade nickel-iron sulphide concentrate, low in copper.

2. This nickel-iron sulphide concentrate may be roasted to a nickel-iron oxide of low-sulphur content in a Herreschoff or other roasting furnace.

3. Should, as is sometimes the case, the nickel-iron oxide contain more gangue than is desirable, it may be converted into a magnetic oxide by a reducing roast, which may follow and be part of the main roasting operation, and be concentrated by wet grinding and wet-magnetic separation.

4. After agglomeration, if necessary, this nickel-iron oxide may be reduced to metallic iron and metallic nickel by any of the low-temperature (sponge iron) reduction processes, such as the Hornsey, Thornhill-Anderson, Wiberg, Norsk-Staal, etc.

Any copper present will also be reduced and found with the iron and nickel.

5. The reduced metals may be separated from any mechanically-free non-magnetic matter, such as coke, ash and gangue, by magnetic concentration. Any free copper present, being non-magnetic, would also be rejected.

6. The nickel-iron sponge may be briquetted and thus made available for melting in an electric furnace, open-hearth furnace, or cupola and converted into any nickel-iron alloy desired.

Raw Material. The ore used in these tests was a shipment of disseminated ore from the Falconbridge mine. Under the supervision of C. S. Parsons this material was concentrated by selective flotation, a concentrate of the following composition being obtained:

	Per cent
Iron	48·02
Nickel	3·52
Copper	0·50
Sulphur	31·75
Insoluble	9·20
	92·99

The 9·20 per cent insoluble and the approximately 7 per cent unaccounted for make up a gangue content of about 16·20 per cent. Upon desulphurization and metallization, this will be concentrated to 23·7 per cent in the metallic briquette, unless previously removed. Since a gangue content this high would be undesirable in a briquette intended for use in a steel-melting furnace, it is most desirable to remove it partly, at least, before the briquetting operation.

Roasting. Under the supervision of R. J. Traill, the above concentrate was roasted in a Herreschoff furnace. The average composition of the calcined material was reported as follows:

	Per cent
Iron	54·96
Nickel	4·07
Copper	0·49
Sulphur	0·21
Insoluble	10·18

Roasting to Magnetic Oxide and Wet-Magnetic Concentration. In order to make possible the separation of at least part of the gangue present in the mixed oxides by wet magnetic concentration about 400 pounds of the calcined material was converted to magnetic oxide by means of a reducing roast. In carrying out this operation, a batch of about 100 pounds of calcine was charged into an electrically heated rotary retort and heated to 1,100° F. for two hours, during which time a stream of city gas was passed through the retort. The hot magnetic oxide was then discharged into a special container provided with a tightly fitting cover and allowed to cool.

It is reasonable to expect that, in commercial operation, by discharging the hot calcine from the oxidizing roast continuously into a second furnace or retort in which a reducing atmosphere is maintained, the conversion of the oxide to the magnetic form would be cheaply and quickly obtained.

The analysis of this magnetic oxide was reported as follows:

	Per cent
Iron	58·21
Nickel	4·23
Copper	0·47
Sulphur	0·27
Insoluble	10·35

This magnetic oxide was wet ground to — 200 mesh and concentrated on a Gröndal wet magnetic separator, the following being the analysis of the concentrate obtained.

	Per cent
Iron	61·70
Nickel	4·45
Copper	0·42
Sulphur	0·16
Insoluble	5·99

Due to the comparatively small amount of oxide available for concentration on the Gröndal machine, which was designed to handle much larger amounts, it was not possible to work out the best practice. That the grade of concentrate obtained can probably be improved upon and that the recoveries of iron and nickel will be high, are indicated by Table I, which shows the results of a laboratory-scale concentration of the same magnetic oxide in the Davis magnetic tube.

Sintering. Sintering of the finely divided concentrate from the wet magnetic separation is desirable for three reasons; first, reduction takes place much more readily when the charge is somewhat porous than when it is a compact mass of finely divided material; second, as has been shown elsewhere, the removal of excess carbon, coal ash, or calcium sulphide in the dry magnetic concentration that follows the metallizing operation takes place to a greater extent when the materials are moderately coarse; and third, coarse material briquettes more satisfactorily than fine. Since sintering locks up any gangue present in the ore, this operation should be carried out only on material that has already been concentrated to the desired extent.

Due to the small amount of oxide available no sintering tests were actually carried out, since a satisfactory sinter cannot be made from such finely divided material until a sufficient amount of fine sinter or "returns" is available to mix with the oxide in order to give it porosity and also to keep it from being drawn through the grates.

TABLE I

Laboratory Wet Magnetic Concentration of Falconbridge Oxide

	Per cent
Separator Feed—	
Iron	58·20
Nickel	4·20
Copper	0·44
Sulphur	0·27
Insoluble	10·35
Concentrate—	
Weight	90·3
Iron	62·37
Nickel	4·52
Copper	0·43
Sulphur	0·11
Insoluble	5·20
Tailing—	
Iron	19·38
Nickel	1·24
Copper	0·53
Sulphur	1·75
Insoluble	58·25
Ratio of Concentration	1·11 to 1·00
Recoveries—	
Iron	96·77
Nickel	97·14
Copper	88·19
Insoluble Rejected	54·59

Metallization

Reductions were carried out only on the relatively high insoluble content material that was the product of the oxidizing roast, since sintered material from the wet magnetic concentrate was not available.

These reductions were carried out in the electrically heated rotary retort used in the low-temperature reduction of iron ores. This alloy retort, which forms the heating chamber of a 60 k.w. furnace, has an inside diameter of 14¾ inches and an effective loading length of 48 inches. In this work, bituminous coal was used as the reducing agent, the following being a typical charge:

	Pounds
Falconbridge oxide	50·0
Fairmont coal	18·25
Lime	12·0

In carrying out these reductions, the retort and its contents were brought up to a temperature of 1750° F. and held at that temperature for three hours, after which the crude sponge was discharged into a special container provided with a tightly sealing cover. A total of about 400 pounds of calcine was reduced in this manner.

Reductions may, of course, be made in other ways, and there appears to be no reason why this operation should not be carried out according to any of the low-temperature reduction processes such as the Hornsey, the Thornhill-Anderson, the Wiberg, the Norsk-Staal, etc.

The average analysis of the crude retort product, before the dry magnetic concentration, was:

	Per cent
Iron	46·60
Nickel	3·65
Copper	0·36
Sulphur	0·66

Dry Magnetic Concentration of Sponge

With the object of separating the reduced metals from the excess coke, lime, etc., the crude retort product was subjected to dry magnetic concentration by giving it ten passes on the Stearns drum-type magnetic separator. The results of this concentration are given in Table II.

It will be noted that the combined contents of iron, nickel, and copper in the concentrate from this separation amount only to 79·38 per cent. This low value is largely due to the fact that the original sulphide concentrate contained over 16 per cent gangue, which, not being mechanically free, has become concentrated during the processing. As has been pointed out, by introducing a reducing roast and wet magnetic concentration operations after the oxidizing roast, this gangue content can be reduced to about half, as shown in Table I. That the low-gangue oxide shown in this table will produce a sponge containing about 90 per cent metals is shown by Table III which shows the results of a small-scale laboratory reduction of the high-grade oxide, by means of hydrogen, without subsequent concentration.

TABLE II

Dry Magnetic Concentration of Sponge from Low-grade Oxide

	Per cent
Magnetic Separator Feed—	
Iron	46·60
Nickel	3·65
Copper	0·36
Sulphur	0·66
Concentrate—	
Weight	59·70
Iron	72·80
Nickel	6·00
Copper	0·58
Sulphur	0·08
Tailing—	
Weight	40·30
Iron	7·80
Nickel	0·17
Copper	0·03
Sulphur	1·53
Ratio of Concentration	1·67 to 1
Recoveries—	
Iron	93·48
Nickel	98·08
Copper	96·11
Sulphur Rejection	92·79

TABLE III

Laboratory Reduction of Falconbridge—Low-gangue Concentrate

	Per cent
Oxide Reduced—	
Iron	62·37
Nickel	4·52
Copper	0·43
Sponge Produced—	
Weight	75·2
Iron	82·96
Nickel	6·10
Copper	0·61
Total metallic content of sponge	89·67
Gangue content (by difference)	10·33
	100·0

Briquetting

The concentrated sponge was briquetted in a 70-ton Southwark briquetting machine. This type of machine has been supplied to several plants engaged in the experimental and semi-commercial production of sponge iron. It has been found that very finely divided material is very difficult to briquet and that to obtain the best results a certain amount at least of comparatively coarse material should be present.

Melting

The briquetted nickel-iron sponge briquettes are in effect ferro-nickel, and like ferro-nickel form a convenient and practical means of introducing nickel into steel or cast iron.

In order to demonstrate the usefulness of these briquettes as a nickel-iron base in steelmaking, a series of nickel steel ingots was made. The melts were made in a 50-pound Ajax-Northrup, high-frequency, induction furnace, the charges consisting of suitable weights of boiler plate punchings and nickel-iron briquettes, together with the amounts of ferro-silicon, ferro-manganese and other ferro-alloys necessary to produce the specification desired. To provide a standard of comparison against which the physical properties of the steels so made might be compared, a similar set of ingots was made from boiler plate punchings and electrolytic nickel.

TABLE IV

Production of Steels from Falconbridge Sponge

(a) S. A. E. 3135 Steel

Charge	Contents (lb.)				
	Iron	Nickel	Copper	Chromium	Sulphur
37·4 lb. boiler punchings	37·0				0·011
9·0 lb. briquettes	6·5	0·54	0·052		0·007
0·4 lb. ferro-chrome	0·1			0·28	
	43·6	0·54	0·052	0·28	0·018

—	Weight of ingot obtained =41·75 lb.								
	C	Mn	Ni	Cu	Cr	P	S	Si	Fe (by diff.)
Ingot analysis....%	0·41	0·58	1·25	0·12	0·59	0·016	0·050	0·18	96·8
Metallic content (lb.)			0·52	0·05					40·4
Metallic recovery%			96·3	96·1					92·6

TABLE IV—Continued

(b) S. A. E. 4615 Steel

Charge	Contents (lb.)				
	Iron	Nickel	Copper	Molyb.	Sulphur
36·75 lb. boiler punchings	36·40				0·011
13·00 lb. briquettes	9·40	0·78	0·075		0·010
0·19 lb. ferro-molyb	0·07			0·114	
	45·87	0·78	0·075	0·114	0·021

—	Weight of ingot obtained =44·0 lb.								
	C	Mn	Ni	Cu	Mo	P	S	Si	Fe (by diff.)
Ingot analysis...%	0·12	0·35	1·74	0·18	0·28	0·024	0·045	0·29	96·97
Metallic content (lb.)			0·765	0·079					42·7
Metallic recovery%			98·1	100·0					93·06

(c) S. A. E. 2325 Steel

Charge	Contents (lb.)			
	Iron	Nickel	Copper	Sulphur
36·0 lb boiler punchings	35·6			0·011
22·3 lb briquettes	16·2	1·34	0·13	0·017
	51·8	1·34	0·13	0·028

—	Weight of ingot obtained =50·0 lb.							
	C	Mn	Ni	Cu	P	S	Si	Fe (by diff.)
Ingot analysis.......%	0·27	0·70	2·68	0·30	0·017	0·049	0·36	95·63
Metallic content (lb.)			1·34	0·15				47·82
Metallic recovery.......%			100·0	100·0				92·3

(d) S. A. E. 3330 Steel

Charge	Contents (lb.)				
	Iron	Nickel	Copper	Chromium	Sulphur
30·0 lb. punchings	29·70				0·009
26·5 lb. briquettes	18·70	1·59	0·16		0·021
1·13 lb. ferro-chrome	0·26			0·79	
	48·66	1·59	0·16	0·79	0·030

TABLE IV—Concluded

—	Weight of ingot obtained =49·0 lb.								
	C	Mn	Ni	Cu	Cr	P	S	Si	Fe (by diff.)
Ingot analysis....%	0·31	0·52	3·24	0·35	1·64	0·017	0·52	0·29	93·58
Metallic content (lb.)..........	....	....	1·59	0·17	....	....	....	....	45·85
Metallic recovery%	....	....	100·0	100·0	....	....	....	....	94·3

(e) S.A.E. 2512 Steel

—	Contents (lb.)			
	Iron	Nickel	Copper	Sulphur
19·5 lb. boiler punchings......................	19·3	....	....	0·006
35·0 lb. briquettes...........................	25·4	2·10	0·20	0·028
	44·7	2·10	0·20	0·034

—	Weight of ingot obtained =41·0 lb.							
	C	Mn	Ni	Cu	P	S	Si	Fe (by diff.)
Ingot analysis............%	0·07	0·31	4·88	0·49	0·017	0·055	0·13	94·05
Metallic content (lb.)......	....	....	2·0	0·20	....	....	....	38·56
Metallic recovery.........%	....	....	95·2	100·0	....	....	....	86·3

From these summaries of melting tests it will be seen that the recoveries of nickel and copper are practically perfect, as might have been expected. That the recovery of iron is not so high may be ascribed to three causes; first, the iron in the briquettes was not completely metallized, the metallization being about 95 per cent; second, due to the high-gangue content and finely divided nature of the sponge, the density of the briquettes was not so high as it might otherwise have been; and third, the addition of the large volume of briquettes to the crucible took a considerable time thus greatly prolonging the time during which the bath was subject to oxidation.

Hot Working (Rolling) Quality of Steels Made

Through the kindness of the management of Canadian Atlas Steels Ltd., of Welland, Ontario, the five ingots made from Falconbridge sponge and the five comparison ingots made from scrap and electrolytic nickel were rolled into one-inch rounds. All of these ingots are reported to have rolled well without any evidence of red-shortness.

Mechanical Properties of the Steels Produced

While steel made in 40- to 50-pound heats and cast into small ingots under laboratory conditions is not likely to be equal in quality to steel made in larger heats in practical operations, it was thought that a comparison of the mechanical properties of these steels made from Falconbridge sponge with those of similar steels made from scrap and electrolytic nickel would be of interest.

Tables V to IX inclusive summarize the results of mechanical tests on these steels after certain heat treatments. For convenience, the steels made from nickel-iron sponge are designated "Falconbridge" steels and those made from scrap and electrolytic nickel "Regular" steels.

In each case, test piece lengths of 1-inch round section of each Falconbridge steel and of its companion Regular steel were heat treated similarly and then machined to standard A.S.T.N. tensile test pieces, 0·505-inch diameter by 2·0-inch gauge length, with threaded ends. For some of the steels similar pairs of one heat treated 1-inch rounds were also machined into standard Izod (square) impact test pieces.

The tensile tests were made on an Amsler tensile and compression testing machine of 200,000-pound capacity. The Izod tests were made on an Amsler single-blow impact testing machine.

TABLE V

Mechanical Properties of Steels made to Specification S.A.E. 3135

—	C	Mn	Ni	Cu	Cr	P	S	Si
Falconbridge steel........%	41	58	1·25	0·12	0·59	0·016	0·050	0·18
Regular steel.............%	31	57	1·25	nil	0·65	0·019	0·036	0·25

Section heat treated—1-inch round.

Dimensions of test pieces—
Tensile tests—0·505-inch diam. by 2·00-inch gauge length
Impact tests—standard square Izod.

Steel	Quenching		Drawing temp. °F	Tensile strength lb.=sq.in.	Yield point lb.=sq.in.	Elon. per cent	Red. in area	B. H. N.
	Temp. °F	Medium						
Falconbridge...........	1500	Water	700	222,500	205,000	9·0	26·8	401
Regular................	1500	Water	700	181,500	172,500	12·5	43·2	352
Falconbridge...........	1500	Water	800	196,005	185,000	11·0	34·4	363
Regular................	1500	Water	800	159,000	151,500	16·0	50·4	321
Falconbridge...........	1500	Water	900	171,000	161,000	15·0	45·5	331
Regular................	1500	Water	900	138,000	124,500	20·0	55·8	277
Falconbridge...........	1500	Water	1000	151,500	141,000	16·0	46·9	293
Regular................	1500	Water	1000	129,000	112,500	21·0	58·1	262
Falconbridge...........	1500	Water	1100	136,000	123,500	20·0	54·1	255
Regular................	1500	Water	1100	122,000	105,000	23·0	60·8	248
Falconbridge...........	1500	Water	1200	122,500	108,000	23·0	58·6	235
Regular................	1500	Water	1200	113,500	94,000	25·0	62·8	223
Falconbridge...........	Annealed	at 1500°	F	94,000	63,500	30·0	54·7	170
Regular................	Annealed	at 1500°	F	86,500	57,500	31·0	57·3	159

TABLE VI

Mechanical Properties of Steels made to Specification S.A.E. 4615

—	C	Mn	Ni	Cu	Mo	P	S	Si
Falconbridge steel....%	0·12	0·35	1·74	0·18	0·28	0·024	0·045	0·29
Regular steel.........%	0·09	0·47	1·94	nil	0·28	0·019	0·033	0·27

Section heat treated—1-inch round

Dimensions of test pieces—
Tensile tests—0·505-inch diameter by 2·00-inch gauge length
Impact tests—standard square Izod.

Heat treatment—
1. Heat to 1525° F. and quench in oil
2. Heat to 1420° F. and quench in oil
3. Draw as shown in table

Steel	Drawing temp. °F	Tensile strength lb./sq. in.	Yield point lb./sq. in.	Elon. per cent	Red. in area, per cent	Izod ft. lb.	B. H. N.
Falconbridge..................	Not drawn	86,000	46,000	32·5	61·8	77	170
Regular.......................	Not drawn	88,500	47,500	33·5	58·6	60	170
Falconbridge..................	100	84,500	47,500	34·0	64·7	78	170
Regular.......................	100	86,500	47,500	31·5	56·2	59	174
Falconbridge..................	200	85,000	47,500	34·0	63·5	81	167
Regular.......................	200	87,500	48,000	33·0	59·4	65	174
Falconbridge..................	300	84,500	46,500	34·0	64·2	84	170
Regular.......................	300	85,500	46,500	31·5	61·6	69	167
Falconbridge..................	400	83,500	48,000	35·0	66·1	88	167
Regular.......................	400	84,500	48,000	32·5	61·6	74	167
Falconbridge..................	500	83,000	47,000	31·5	64·4	93	170
Regular.......................	500	85,000	47,000	31·0	60·8	77	170

TABLE VII

Mechanical Properties of Steels made to Specification S.A.E. 3330

—	C	Mn	Ni	Cu	Cr	P	S	Si
Falconbridge steel........%	0·31	0·52	3·24	0·35	1·64	0·017	0·052	0·29
Regular steel.............%	0·32	0·57	3·54	nil	1·56	0·029	0·037	0·30

Section heat treated—1-inch round

Dimensions of test pieces—
Tensile tests—0·505-inch diameter by 2·00-inch gauge length
Impact tests—standard square Izod.

TABLE VII—Concluded

Steel	Quenching		Drawing temp. °F	Tensile strength lb./sq. in.	Yield point lb./sq. in.	Elon. per cent	Red. in area, per cent	B. H. N.
	Temp. °F	Medium						
Falconbridge.....	1450	Oil	800	195,500	188,500	11·0	40·1	388
Regular..........	1450	Oil	800	195,500	185,200	13·0	40·9	388
Falconbridge.....	....	....	900	177,000	168,000	14·5	48·1	363
Regular..........	....	....	900	163,700	152,300	16·0	51·9	352
Falconbridge.....	....	....	1000	158,000	147,500	17·0	48·6	321
Regular..........	....	....	1000	157,500	147,500	18·0	54·1	321
Falconbridge.....	....	....	1100	140,500	124,000	19·0	54·7	285
Regular..........	....	....	1100	138,900	123,600	21·0	58·7	285
Falconbridge.....	....	....	1200	117,000	99,500	25·0	60·8	241
Regular..........	....	....	1200	119,800	101,200	24·0	59·5	241
Falconbridge.....	Annealed at 1450° F.			100,500	72,500	29·0	60·3	207
Regular..........	Annealed at 1450° F.			101,800	61,150	27·0	54·1	207

TABLE VIII

Mechanical Properties of Steels made to Specification S.A.E. 2325

—	C	Mn	Ni	Cu	P	S	Si
Falconbridge steel...............%	27	70	2·68	30	0·017	0·049	0·36
Regular steel.....................%	26	70	3·08	Nil	0·035	0·040	0·35

Section heat treated—1-inch round

Dimensions of test pieces—
Tensile tests—0·505-inch diameter by 2·00-inch gauge length
Impact tests—standard square Izod.

Steel	Quenching		Drawing temp. °F	Tensile strength lb./sq. in.	Yield point lb./sq. in.	Elon. per cent	Red. in area, per cent	B. H. N.
	Temp. °F	Medium						
Falconbridge....	1475	Water	700	141,500	123,500	18·5	51·4	293
Regular.........	1475	Water	700	151,000	136,000	17·5	58·2	321
Falconbridge....	1475	Water	800	134,500	116,000	20·0	54·7	285
Regular.........	1475	Water	800	138,500	125,500	21·0	59·3	285
Falconbridge....	1475	Water	900	130,000	113,500	20·5	56·8	269
Regular.........	1475	Water	900	128,000	116,800	21·0	59·4	262
Falconbridge....	1475	Water	1000	115,000	98,500	24·0	58·6	235
Regular.........	1475	Water	1000	115,500	102,500	24·0	63·7	241
Falconbridge....	1475	Water	1100	106,000	90,000	26·5	62·3	212
Regular.........	1475	Water	1100	106,500	95,000	27·5	66·5	223
Falconbridge....	1475	Water	1200	100,000	85,750	28·0	63·8	212
Regular.........	1475	Water	1200	99,500	86,500	29·0	66·5	223
Falconbridge....	Annealed at 1475° F.			86,500	67,500	31·5	55·2	159
Regular.........	Annealed at 1475° F.			84,000	64,000	32·0	56·7	156

TABLE IX

Mechanical Properties of Steels made to Specification S.A.E. 2512

—	C	Mn	Ni	Cu	P	S	Si
Falconbridge steel................%	0·07	31	4·88	0·49	0·017	0·055	0·13
Regular steel.....................%	0·08	41	5·40	nil	0·034	0·038	0·25

Section heat treated—1-inch round

Dimensions of test pieces—
Tensile tests—0·505 inch diam. by 2·00-inch gauge length
Impact tests—standard square Izod

Heat treatment—
1. Heat to 1475° F. and quench in oil
2. Heat to 1350° F. and quench in oil
3. Draw as shown in table

Steel	Drawing temp. °F	Tensile strength lb./sq. in.	Yield point lb./sq. in.	Elon. per cent	Red. in area, per cent	Izod ft. lb.	B. H. N.
Falconbridge.................	Not drawn	90,500	62,000	38·0	65·9	78	179
Regular.......................	Not drawn	96,500	67,500	34·0	61·1	94	187
Falconbridge.................	100	90,700	61,500	34·0	60·0	78	183
Regular.......................	100	94,500	63,500	35·0	64·0	92	192
Falconbridge.................	200	89,000	59,000	35·0	65·9	77	179
Regular.......................	200	95,250	64,500	36·5	64·9	97	192
Falconbridge.................	300	88,500	60,000	35·0	62·3	81	179
Regular.......................	300	96,500	66,000	35·0	63·5	100	192
Falconbridge.................	400	87,500	57,000	36·5	62·1	81	174
Regular.......................	400	95,500	65,000	35·0	64·7	92	192
Falconbridge.................	500	88,000	60,000	33·5	62·8	82	179
Regular.......................	500	95,000	64,500	31·0	65·4	99	192

If due allowance is made for differences in composition, these tables indicate that the steels made from Falconbridge sponge are quite comparable to the steels made from the usual raw materials. This is the only conclusion that the writers believe should be drawn from the tabulated results since no attempt was made to eliminate the many variables in melting practice and heat treatment that must be taken into account before it is possible to state with any degree of accuracy that steels made from nickel-iron sponge are, as a class, better or worse than steels made by the usual methods.

SUMMARY

A process has been described by which a nickel-iron sponge may be produced from sulphide ores such as occur in the Sudbury district of Ontario. The results of steel making experiments in which standard

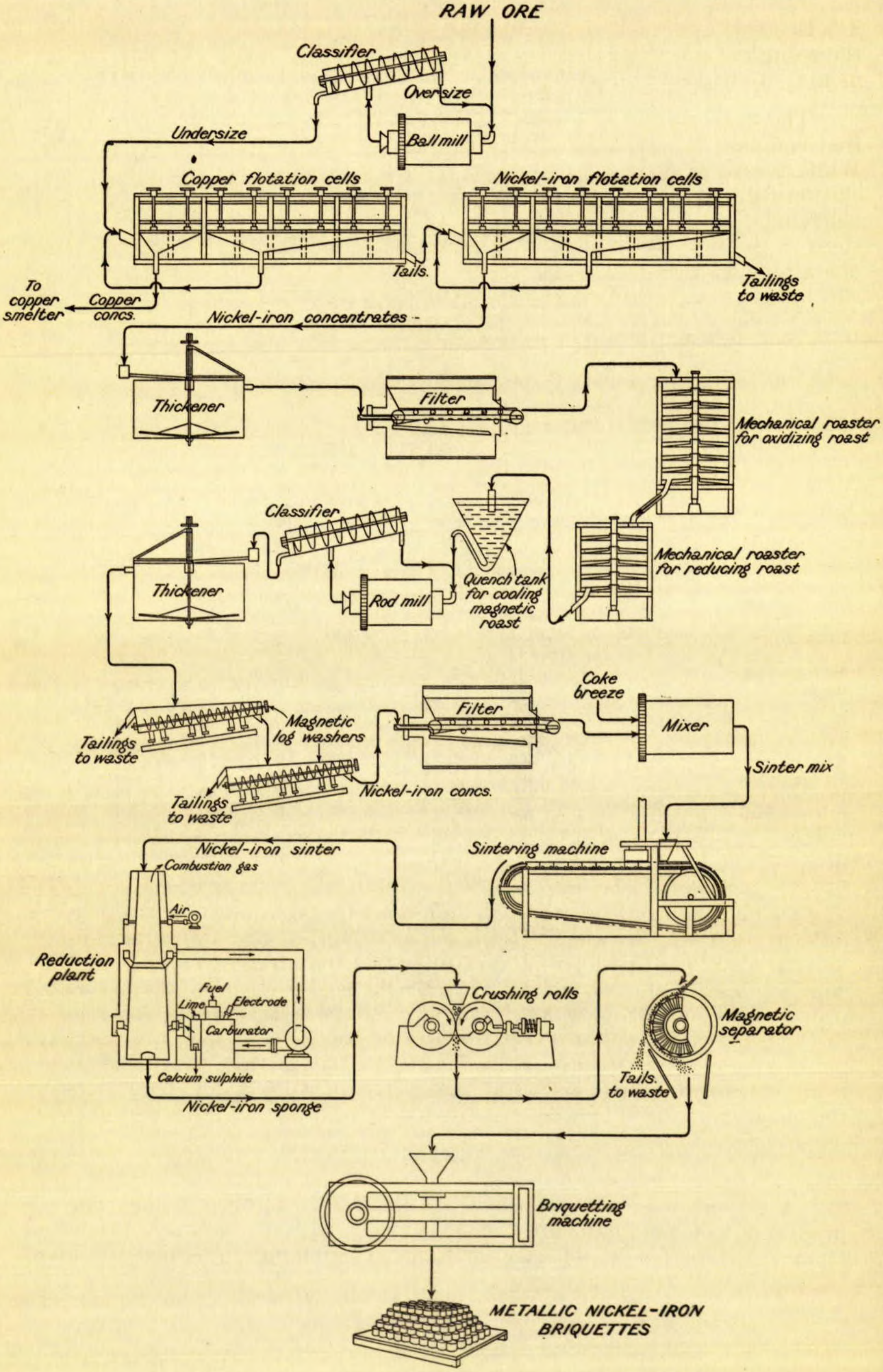

Figure 4. Diagrammatic flow-sheet showing proposed method of producing nickeliferous sponge iron from Sudbury ores.

S.A.E. steels were made from this sponge have been presented, as have also the results of mechanical tests on these steels after commercial heat treatments.

The data presented show that steels have been made by this process that compare favourably with steels made by the commonly used method. While the work done does not permit of the formation of a definite conclusion regarding the economics of the process, there are good reasons for believing that it has commercial possibilities, since none of the various steps in the process are difficult and most of them are already in practical operation in other processes.

VI

REPORT OF THE CHEMICAL LABORATORY SECTION

H. C. Mabee

Chief Chemist

During the year, a total of 2,918 samples of ores, ore dressing and metallurgical products were received, analysed, and reported on. This is only an increase of ten over the previous year but the work performed on a larger portion of the samples submitted was much more exhaustive, involving in many cases complete analyses. Considerably more than 10,000 determinations were made.

Of the samples submitted for assay and analyses, 1,160 were classified as gold ores or the metallurgical products from the experimental tests on gold ores. This number, however, represents but a portion of the total on which assay work was performed as many of the other samples were assayed for their precious metal content in addition to the determinations of the other mineral constituents.

A rather extensive investigation was carried out on the physical and chemical nature of a series of pitchblende and silver samples taken by H. S. Spence of the Mineral Resources Division, from the deposits at LaBine point, Echo bay, Great Bear lake, Northwest Territories. This work involved over 300 determinations and occupied the time of two chemists for several weeks.

In the following table is given a list of the samples upon which reports of analyses were issued and shows the wide variety of analytical and assay work performed in the chemical laboratories.

Samples of ores, minerals, metals, and their test products	Number of samples	Samples of ores, minerals, metals, and their test products	Number of samples
Anhydrite	41	Graphite	24
Antimony	1	Gypsum	42
Apatite	32	Iron ores and iron products	530
Bleaching solutions	55	Vanadium-bearing iron ores	41
China clay	5	Lead-zinc-silver	85
Chrome	51	Manganese	35
Cobalt	39	Marl	3
Cobalt-copper	37	Mercury	28
Copper	13	Molybdenite	54
Copper-iron concentrates	45	Nickel-copper	252
Cyanite	16	Radioactive ores and minerals	49
Diatomite	2	Silica sands	39
Drill core	1	Steels (alloy)	53
Gold	1,160	Syenite rock	8
Gold-copper	13	Sulphide concentrates	14
Gold-platinum	7	Talc	42
Gold-silver	101		

The analytical, assay, and investigative work, carried out in the chemical laboratories was under the direction and supervision of H. C. Mabee, Chief Chemist. B. P. Coyne, H. L. Beer, and R. W. Cornish performed the analyses on non-ferrous metallic ores and their products. J. D. Johnston and A. E. LaRochelle conducted the analyses on iron ores, metals, and alloys. L. Lutes and J. S. McCree performed the fire assay work for the precious metals. R. A. Rogers, R. W. Cornish, and H. L. Beer conducted the analyses on non-metallic minerals and their test products.

On the examination of the pitchblende and silver samples from Great Bear lake, Northwest Territories, B. P. Coyne and H. L. Beer devoted several weeks; and on the continuation of the investigation of gypsum and anhydrite from Canadian deposits, R. A. Rogers was employed for a large portion of his time.

In the completion of the new ore dressing and metallurgical laboratory building, the chief chemist actively co-operated with the Architects Branch of the Department of Public Works. The top floor of this building is entirely devoted to chemical laboratories and offices of the chemical staff.

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