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*)

## 1930

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

No. 724

*Frontispiece* PLATE I

New Ore Dressing and Metallurgical Laboratories of the Mines Branch, Department of Mines. (Erected 1931.)

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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, 1930

## I

## GENERAL REVIEW OF INVESTIGATIONS

**W. B. Timm**

*Chief of Division*

The investigative work conducted in the Ore Dressing and Metallurgical Laboratories of this Division of Mines Branch activities is steadily increasing as the facilities provided and the type of work performed is becoming better known to the mining public. The greater demand for assistance from the mining and metallurgical industry for experimental tests and research on ore treatment and on metallurgical problems confronting the industry, is due to the confidence which has been placed in the value and usefulness of the reports issued on the work performed.

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 efficiently and with greater despatch. A new ore dressing and metallurgical laboratory building was nearing completion at the end of the year. The top floor will be entirely devoted to analytical and assay laboratories for the chemical determinations of the various products of test operations. On the second floor is provided hydrometallurgical and electrochemical laboratories, microscopic and spectroscopic laboratories for the examination of ores, ore dressing and metallurgical products, and a chemical research laboratory for chemical problems encountered in conducting the investigative work on ore treatment. On the ground floor is provided office accommodation for the staff of ore dressing engineers, as well as small-scale testing and research laboratories for gravity concentration, flotation, and cyanide tests. In the basement is provided grinding and polishing, metallographic, and mechanical and physical testing laboratories for the examination and testing of metals and alloys.

Complete laboratory facilities for conducting investigations on iron and steel were provided in the pyrometallurgical laboratory building and in the basement of the new laboratory building as noted above. It is hoped with these facilities to give the same measure of assistance to the iron and steel industry as has been accorded the mining industry by the investigations conducted on ore treatment.

Co-operative arrangements were still existent with The Base Metals Extraction Company, Limited; The Cassel Cyanide Company, a subsidiary of Canadian Industries Limited; the American Cyanamid Company, Niagara Falls, Ontario; and the Ross-Playfair interests of Montreal, Quebec. Due to the low market price for the base metals very little investigative work was carried on by the Base Metal Extraction Company. Research fellows were maintained, however, by the Cassel Cyanide Company and the American Cyanamid Company in the laboratories, investigating the suitability of their various products in comparison with other similar manufactured products to the treatment of Canadian complex ores. The tests carried out in conjunction with the Ross-Playfair interests on the Musso semi-direct steel process for the production of steel from iron ore failed to demonstrate that the process in its present state of development had any commercial value. The failure was due to the inadequate means provided by the inventor to carry out his ideas.

The trend of activities in the industry, from the search for new discoveries of base metal minerals to that of the precious metals, principally gold, due to conditions in the world's markets for the base metals is shown in the nature of the work performed in the laboratories. Some fifteen investigations were carried out on gold ores, or where gold was the chief valuable constituent, one on a straight silver ore, nine on base metal ores such as lead-zinc-silver admixtures, copper-zinc, copper-nickel and copper pyrite, and two on cobalt ores.

The investigations were carried out under the direction of W. B. Timm, Chief Engineer, Division of Ore Dressing and Metallurgy. Those dealing with the treatment of metallic ores were under the supervision of C. S. Parsons; with the dressing of non-metallic minerals under the supervision of R. K. Carnochan, and the iron and steel investigations under the supervision of T. W. Hardy. The chemical and analytical work was under the supervision of H. C. Mabee, Chief Chemist.

Section II contains the reports of investigations on the treatment of metallic ores by C. S. Parsons, A. K. Anderson, J. S. Godard, D. S. Halford, and G. B. Walker.

Section III contains the reports of investigations on the dressing of non-metallic minerals by R. K. Carnochan, R. A. Rogers, B. P. Coyne, J. S. Godard and M. F. Goudge.

Section IV contains the reports of investigations of the iron and steel section by T. W. Hardy, H. H. Bleakney, and W. S. Jenkins.

Section V contains a summary of the work conducted in the chemical laboratories under the direction and supervision of H. C. Mabee, Chief Chemist. The analytical work was carried out by B. P. Coyne, R. A. Rogers, H. L. Beer, J. D. Johnston, R. W. Cornish, and A. E. Larochelle, while the assay determinations were made by L. Lutes and J. S. McCree. The amount of work performed in the chemical laboratories, which showed an increase of 34 per cent over 1929, is a good indication of the investigative work conducted during the year.

The operation and maintenance of the Ore Dressing and Metallurgical Laboratories was performed by a staff of mill operators, laboratory assistants, mechanics and labourers, under the immediate supervision of B. M. Derry, mill foreman, and his assistant, Alex. Davie.

# II

# REPORTS OF INVESTIGATIONS: METALLIC ORES SECTION

## Report No. 350

## FLOTATION TESTS ON LEAD-SILVER ORE FROM THE MALLARD LAKE GOLD MINES, LTD., KIRKLAND LAKE, ONTARIO

**J. S. Godard**

*Shipment.* A shipment of 3,000 pounds of ore was received November 28, 1929, from G. H. Dickson, Superintendent of the Temiskaming Testing Laboratories, Cobalt. The ore was from the Kirkland Lake property of the Mallard Lake Gold Mines, Ltd., 69 Adelaide St. E., Toronto.

*Characteristics of the Ore.* The ore is a low-grade lead-silver one. Small quantities of chalcopyrite and zinc blende were present and forty cents in gold per ton. The gangue is calcite.

*Sampling and Analysis.* The whole shipment was crushed to ½ inch and a 1/20 cut made using a Vezin sampler. The sample was crushed to −14 mesh and cut to 20 pounds, from which the head sample was taken.

Analyses showed it to contain:

| | | | |
|---|---|---|---|
| Gold | 0·02 oz./ton | Lead | 5·70 per cent |
| Silver | 6·40 " | Copper | 0·47 " |
| | | Zinc | 0·49 " |

*Purpose of Tests.*—Tests were made to indicate the recoveries of lead and silver that might be obtained by flotation.

### EXPERIMENTAL TESTS

### *Test No. 1*

| Product | Weight, per cent | Assays | | Per cent of values | |
|---|---|---|---|---|---|
| | | Pb, per cent | Ag, oz./ton | Pb | Ag |
| Concentrate | 11·8 | 48·6 | 53·50 | 98·2 | 93·5 |
| Tailing | 88·2 | 0·12 | 0·50 | 1·8 | 6·5 |

Copper in lead concentrate, 4·14 per cent.

### *Test No. 2*

| Product | Weight, per cent | Assays | | Per cent of values | |
|---|---|---|---|---|---|
| | | Pb, per cent | Ag, oz./ton | Pb | Ag |
| Concentrate | 12·0 | 49·3 | 55·20 | 97·8 | 94·2 |
| Tailing | 88·0 | 0·15 | 0·46 | 2·2 | 5·8 |

Copper in lead concentrate 4·05 per cent.

*Test No. 3*

| Product | Weight, per cent | Assays | | Per cent of values | |
|---|---|---|---|---|---|
| | | Pb, per cent | Ag oz./ton | Pb | Ag |
| Concentrate | 9·0 | 65·42 | 68·00 | 97·5 | 83·9 |
| Middling | 2·7 | 1·70 | 27·70 | 0·8 | 10·2 |
| Tailing | 88·3 | 0·12 | 0·49 | 1·7 | 5·9 |

| | | | |
|---|---|---|---|
| Copper in lead concentrate | 4·81 per cent | Zinc in middling | 3·00 per cent |
| Copper in middling | 2·14 " | Gold in lead concentrate | 0·15 oz./ton |
| Zinc in lead concentrate | 2·13 " | Gold in middling | 0·10 " |

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

| Product | Weight, per cent | Assays | | Per cent of values | |
|---|---|---|---|---|---|
| | | Pb, per cent | Ag, oz./ton | Pb | Ag |
| +100 | 5·9 | 0·15 | 1·00 | 7·3 | 12·0 |
| −100 | 94·1 | 0·12 | 0·46 | 92·7 | 88·0 |

Average tailing calculated: Pb 0·12 per cent, Ag 0·49 oz./ton.

*Reagents*—

*Test No. 1.* To ball mill, soda ash 3·0 lb./ton, cyanide 0·20 lb./ton, thiocarbanilide 0·10 lb./ton. To cell, pine oil 0·05 lb./ton.

*Test No. 2.* To ball mill, soda ash 3·0 lb./ton, cyanide 0·20 lb./ton, thiocarbanilide 0·10 lb./ton. To cell, pine oil 0·05 lb./ton.

*Test No. 3.* To ball mill, soda ash 3·0 lb./ton, cyanide 0·10 lb./ton, thiocarbanilide 0·10 lb./ton, pine oil 0·02 lb./ton.

Rougher concentrate cleaned using NaCN 0·25 lb./ton.

### CONCLUSIONS

Recoveries of 97 per cent of the lead and 94 per cent of the silver were made by flotation. The flotation concentrate assayed, Pb 65 per cent, Ag 68 oz./ton, Cu 4·81 per cent, Zn 2·13 per cent, and Au 0·15 oz./ton. The ratio of concentration was 11:1. Grinding to 94 per cent −100 mesh was found to be sufficiently fine.

## Report No. 351

### THE FLOTATION OF COPPER AND IRON PYRITES IN THE ORE OF THE ALDERMAC MINE, ROUYN AREA, QUEBEC

**A. K. Anderson**

*Shipments.* Four hundred and fourteen pounds of ore was received on December 12, 1929, from the Aldermac Mines, Limited, Rouyn area, Quebec. The shipment was forwarded by Alderson & MacKay, New Birks Bldg., Montreal.

*Characteristics of the Ore.* The shipment consisted of heavy, massive iron sulphides containing a small amount of copper sulphide. Practically no gangue minerals were visible.

*Purpose of Tests.* The purpose of the tests was to determine what recovery of copper could be obtained, and also to produce an iron sulphide concentrate as low in zinc as possible. Twenty-five pounds of the iron sulphide concentrate was required by the Canadian Industries, Ltd., Bell Telephone Bldg., Montreal, for the purpose of determining its suitability for the manufacture of sulphuric acid.

*Sampling and Analysis.* The lot was crushed to pass ¼-inch mesh and after being thoroughly mixed passed through a Jones riffle sampler. One-half was then crushed to pass 14 mesh. After several passes through the sampler a representative part of this was secured for analysis showing the shipment to contain:

| | | | |
|---|---|---|---|
| Copper | 1·98 per cent | Gold | 0·09 oz./ton |
| Zinc | 0·49 " | Silver | 1·18 " |
| Iron | 40·5 " | | |

## EXPERIMENTAL TESTS

All tests were made by flotation. The procedure which was the same in all tests was as follows: First: grinding and conditioning the ore with reagents known to depress zinc and iron sulphides, and floating the copper minerals. Second: reactivating and floating the zinc sulphide in order to remove as much zinc as possible from the remaining pyrrhotite and pyrite. Third: Floating the iron pyrite to secure a concentrate high in sulphur, leaving the pyrrhotite as a tailing too low grade to meet the specified requirements of 48 to 50 per cent sulphur.

All tests were made on 2,000-gramme portions of −14 mesh material ground with an equal weight of water in an iron ball mill containing iron balls. Grinding was such that 85 per cent passed 200 mesh. Each test, while conforming to the main procedure, was made with a separate set of reagents.

### *Test No. 1*

*Reagents (lb. per ton)—*

| To ball mill | | To flotation machine | | | |
|---|---|---|---|---|---|
| | | — | Copper | Zinc | Iron |
| Soda ash | 5·0 | Potassium xanthate | 0·10 | 0·04 | |
| Cyanide | 0·14 | Amyl xanthate | | | 0·30 |
| | | Copper sulphate | | 1·0 | |
| | | Pine oil | 0·04 | 0·04 | 0·08 |

*Results—*

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Fe, per cent | Cu | Zn | Fe |
| Heads | | 1·98 | 0·49 | 40·5 | | | |
| Cu concentrate | 24·9 | 8·28 | 0·76 | 42·5 | 93·2 | 38·3 | 25·2 |
| Zn concentrate | 15·2 | 0·46 | 1·75 | 43·5 | 3·2 | 53·9 | 15·8 |
| Fe concentrate | 30·7 | 0·08 | 0·05 | 46·5 | 1·1 | 3·1 | 34·0 |
| Tailing | 29·2 | 0·19 | 0·08 | 35·8 | 2·5 | 4·7 | 25·0 |

A bulky, low-grade copper concentrate representing 24·9 per cent of the weight of ore milled, with a recovery of 93·2 per cent of the copper, is produced. This concentrate was not cleaned. In commercial practice however it would be passed through one or two cleaner cells, thus raising the grade of the final product. The tailing from the cleaner cells called middling would be returned to the circuit for further treatment.

The iron concentrate is apparently of the required grade, being very low in zinc. However, due to the large amount of low-grade copper concentrate the weight of iron pyrite recovered is only 30·7 per cent of the total weight.

### *Test No. 2*

In this test the amount of soda ash added to the ball mill was increased over that used in Test No. 1.

*Reagents (lb. per ton)*—

| To ball mill | | To flotation machine | | | |
|---|---|---|---|---|---|
| | | — | Copper | Zinc | Iron |
| Soda ash | 10·0 | Potassium xanthate | 0·10 | 0·04 | ............ |
| Cyanide | 0·14 | Amyl xanthate | ............ | ............ | 0·30 |
| | | Copper sulphate | ............ | 1·0 | ............ |
| | | Pine oil | 0·04 | 0·04 | 0·08 |

*Results*—

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Fe, per cent | Cu | Zn | Fe |
| Heads | ......... | 1·98 | 0·49 | 40·5 | ....... | ....... | ....... |
| Cu concentrate | 24·6 | 7·86 | 0·76 | 42·4 | 92·0 | 38·3 | 28·2 |
| Zn concentrate | 15·5 | 0·50 | 1·75 | 44·3 | 3·7 | 55·6 | 18·6 |
| Fe concentrate | 32·4 | 0·12 | 0·05 | 47·5 | 1·8 | 3·3 | 28·8 |
| Tailing | 27·5 | 0·19 | 0·05 | 32·8 | 2·5 | 2·8 | 24·4 |

Increasing the soda ash does not have any marked beneficial effect. The bulk and grade of the copper concentrate and the recovery of copper are approximately the same. The iron concentrate also is similar to that of Test No. 1.

### *Test No. 3*

In this test the alkaline reagent was changed, lime being used in place of soda ash.

*Reagents (lb. per ton)*—

| To ball mill | | To flotation machine | | | |
|---|---|---|---|---|---|
| | | — | Copper | Zinc | Iron |
| Lime | 6·0 | Potassium xanthate | 0·05 | 0·05 | ............ |
| Cyanide | 0·10 | Amyl xanthate | ............ | ............ | 0·30 |
| | | Copper sulphate | ............ | 1·0 | ............ |
| | | Pine oil | 0·04 | 0·04 | 0·08 |

*Results*—

| Product | Weight, per cent | Assay |  |  | Per cent of values |  |  |
|---|---|---|---|---|---|---|---|
|  |  | Cu, per cent | Zn, per cent | Fe, per cent | Cu | Zn | Fe |
| Heads | ........ | 1·98 | 0·49 | 40·5 | ........ | ........ | ........ |
| Cu concentrate | 7·8 | 18·86 | 0·63 | 36·9 | 75·3 | 13·9 | 6·9 |
| Zn concentrate | 6·7 | 1·32 | 3·72 | 47·7 | 4·5 | 70·5 | 7·7 |
| Fe concentrate | 44·7 | 0·70 | 0·05 | 45·8 | 16·0 | 6·3 | 49·2 |
| Tailing | 40·8 | 0·20 | 0·08 | 36·9 | 4·2 | 9·3 | 36·2 |

The use of lime and cyanide as depressants for zinc and iron sulphides has a marked detrimental effect. It also depresses copper sulphides as is indicated by the lowered recovery of copper in the copper concentrate and the presence of 20·5 per cent copper in the combined zinc and iron concentrates. The grade of copper concentrate is much higher than that produced in Tests Nos. 1 and 2—18·86 per cent as against 8·28 per cent. The bulk is much less—7·8 per cent as against 24·9 per cent.

The amount of iron pyrite recovered is increased. The zinc content of this is low.

### *Test No. 4*

The reagents used for the flotation of copper and zinc were the same as in Test No. 3. After removing the zinc concentrate, the pulp was acidified with sulphuric acid and the iron pyrite floated with amyl xanthate and pine oil.

*Reagents (lb. per ton)*—

| To ball mill |  | To flotation machine |  |  |  |
|---|---|---|---|---|---|
|  |  | — | Copper | Zinc | Iron |
| Lime | 6·0 | Potassium xanthate | 0·05 | 0·05 | ........... |
| Cyanide | 0·10 | Amyl xanthate | ........... | ........... | 0·30 |
|  |  | Copper sulphate | ........... | 1·0 | ........... |
|  |  | Sulphuric acid | ........... | ........... | 17·0 |
|  |  | Pine oil | 0·04 | 0·04 | 0·08 |

*Results*—

| Product | Weight, per cent | Assay |  |  | Per cent of values |  |  |
|---|---|---|---|---|---|---|---|
|  |  | Cu, per cent | Zn, per cent | Fe, per cent | Cu | Zn | Fe |
| Heads | ........ | 1·98 | 0·49 | 40·5 | ........ | ........ | ........ |
| Cu concentrate | 9·9 | 14·72 | 0·76 | 40·1 | 72·1 | 19·5 | 9·5 |
| Zn concentrate | 4·8 | 1·46 | 4·87 | 41·9 | 3·5 | 60·7 | 4·8 |
| Fe concentrate | 67·2 | 0·68 | 0·10 | 47·7 | 22·6 | 17·4 | 76·3 |
| Tailing | 18·1 | 0·20 | 0·06 | 22·0 | 1·8 | 2·4 | 9·4 |

These results bear out the deduction made in the preceding test that lime and cyanide are detrimental to the flotation of copper sulphides.

Acidifying the pulp for the flotation of iron pyrite has a marked effect, increasing the weight of iron concentrate recovered—67 per cent of the total weight of ore treated is obtained as against 44·7 per cent in an alkaline circuit. The percentage of zinc in this product is 0·10 per cent as against 0·05 per cent in preceding tests.

### *Test No. 5*

The four preceding tests were made to establish flotation conditions prior to making 25 pounds of iron pyrite concentrate as low in zinc as possible for shipment to the Canadian Industries Limited, Montreal, Quebec.

As the best grade of material resulted from grinding with soda ash and cyanide and floating in alkaline circuit, these conditions were adopted. Sixty-six pounds of the ore was ground in batches of 2,000 grammes each, and floated. The copper concentrate was cleaned once to note the effect on the grade.

*Reagents (lb. per ton)*—

| To ball mill | | To flotation machine | | | |
|---|---|---|---|---|---|
| | | — | Copper | Zinc | Iron |
| Soda ash | 7·0 | Potassium xanthate | 0·10 | 0·04 | ........ |
| Cyanide | 0·14 | Amyl xanthate | ........ | ........ | 0·30 |
| | | Copper sulphate | ........ | 1·0 | ........ |
| | | Pine oil | 0·04 | 0·04 | 0·06 |

*Results*—

| Product | Weight, per cent | Assay | | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent | Cu | Zn | Fe |
| Heads | ........ | 1·98 | 0·49 | 40·5 | ........ | ........ | ........ | ........ |
| Cu concentrate | 6·7 | 13·03 | 0·24 | 38·9 | ........ | 43·6 | 4·5 | 6·4 |
| Cu middling | 10·5 | 8·98 | 0·36 | 41·7 | ........ | 47·0 | 10·6 | 10·7 |
| Zn concentrate | 6·1 | 1·54 | 4·15 | 42·1 | ........ | 4·7 | 70·8 | 6·3 |
| Fe concentrate | 40·6 | 0·10 | 0·08 | 46·5 | 50·9 | 2·0 | 9·1 | 46·2 |
| Tailing | 36·1 | 0·15 | 0·05 | 34·5 | ........ | 2·7 | 5·0 | 30·4 |

Under these conditions every 100 tons of ore milled would yield 40·6 tons of iron concentrate with an analysis of 50·9 per cent sulphur, 46·5 per cent iron, 0·10 per cent copper, and 0·08 per cent zinc.

Cleaning the rougher copper concentrate raises the grade to 13 per cent copper. A second cleaning would result in a still higher grade product. In practice the middling would be returned to the circuit for further treatment.

### *Test No. 6*

The preceding tests indicated that flotation in a circuit alkaline with soda ash produces a bulky, low-grade copper concentrate. In this test the amount of collecting agent added for the flotation of copper was reduced and the cyanide increased. The iron pyrite was floated from an acid pulp.

*Reagents (lb. per ton)*—

| To ball mill | | To flotation machine | | | |
|---|---|---|---|---|---|
| | | — | Copper | Zinc | Iron |
| Soda ash | 7·0 | Potassium xanthate | 0·05 | 0·05 | |
| Cyanide | 0·20 | Amyl xanthate | | | 0·30 |
| | | Copper sulphate | | 1·0 | |
| | | Sulphuric acid | | | 17·0 |
| | | Pine oil | 0·04 | 0·04 | 0·06 |

*Results*—

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Fe, per cent | Cu | Zn | Fe |
| Heads | | 1·98 | 0·49 | 40·6 | | | |
| Cu concentrate | 12·0 | 14·73 | 0·41 | 38·1 | 89·7 | 14·2 | 11·2 |
| Zn concentrate | 4·3 | 1·82 | 5·85 | 40·1 | 4·0 | 72·7 | 4·2 |
| Fe concentrate | 48·1 | 0·06 | 0·05 | 48·3 | 1·5 | 6·9 | 56·8 |
| Tailing | 35·6 | 0·27 | 0·06 | 31·9 | 4·8 | 6·2 | 27·8 |

Under these conditions, 89·7 per cent of the copper is recovered in a rougher concentrate assaying 14·2 per cent copper. This grade would be raised by cleaning. The precious metals in this product were found to be 0·06 oz. gold and 5·96 oz. silver per ton, representing 36·0 and 60·6 per cent respectively of the values in the ore.

The grade and recovery of the iron pyrite is good.

### *Test No. 7*

Lime and cyanide in reduced amounts were used in this test, and the iron pyrite floated in acid circuit.

*Reagents (lb. per ton)*—

| To ball mill | | To flotation machine | | | |
|---|---|---|---|---|---|
| | | — | Copper | Zinc | Iron |
| Lime | 1·5 | Potassium xanthate | 0·05 | 0·05 | |
| Cyanide | 0·05 | Amyl xanthate | | | 0·30 |
| | | Copper sulphate | | 1·0 | |
| | | Sulphuric acid | | | 7·1 |
| | | Pine oil | 0·04 | 0·04 | 0·06 |

*Results—*

| Product | Weight, per cent | Assay |  |  | Per cent of values |  |  |
|---|---|---|---|---|---|---|---|
|  |  | Cu, per cent | Zn, per cent | Fe, per cent | Cu | Zn | Fe |
| Heads | .......... | 1·98 | 0·49 | 40·5 | ........ | ........ | ........ |
| Cu concentrate | 7·4 | 18·91 | 0·35 | 35·2 | 73·0 | 9·2 | 6·3 |
| Zn concentrate | 4·1 | 5·85 | 2·57 | 40·8 | 12·5 | 37·4 | 4·1 |
| Fe concentrate | 47·9 | 0·15 | 0·22 | 47·4 | 3·7 | 37·5 | 55·0 |
| Tailing | 40·6 | 0·51 | 0·11 | 35·1 | 10·8 | 15·9 | 34·6 |

The results further bear out the conclusions arrived at in previous tests that the use of lime and cyanide results in poor copper recoveries and also increases the impurities in the pyrite concentrate.

The copper concentrate assays 0·08 oz. gold and 8·28 oz. silver per ton, representing recoveries of 29·6 and 39·0 per cent respectively.

### *Test No. 8*

As the lime-cyanide combination has a strong depressing action on the copper sulphides in this ore, a test was made to note the effect of lime alone, no cyanide being added. The iron pyrite was floated in acid circuit.

*Reagents (lb. per ton)—*

| To ball mill |  | To flotation machine |  |  |  |
|---|---|---|---|---|---|
|  |  | — | Copper | Zinc | Iron |
| Lime | 6·0 | Potassium xanthate | 0·06 | 0·05 | ............ |
|  |  | Amyl xanthate | ............ | ............ | 0·30 |
|  |  | Copper sulphate | ............ | 1·0 | ............ |
|  |  | Sulphuric acid | ............ | ............ | 17·0 |
|  |  | Pine oil | 0·04 | 0·04 | 0·06 |

*Results—*

| Product | Weight, per cent | Assay |  |  |  | Per cent of values |  |  |
|---|---|---|---|---|---|---|---|---|
|  |  | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent | Cu | Zn | Fe |
| Heads | .......... | 1·98 | 0·49 | 40·5 | ........ | ........ | ........ | ........ |
| Cu concentrate | 7·5 | 22·00 | 0·47 | 33·7 | ........ | 85·3 | 12·9 | 6·0 |
| Zn concentrate | 2·9 | 5·65 | 6·78 | 33·1 | ........ | 8·5 | 71·9 | 2·3 |
| Fe concentrate | 60·9 | 0·08 | 0·04 | 48·6 | 48·2 | 2·5 | 8·9 | 70·8 |
| Tailing | 28·7 | 0·25 | 0·06 | 30·4 | ........ | 3·7 | 6·3 | 20·9 |

Omitting the cyanide to the ball mill raises the recovery of copper from 75·3 per cent, as indicated in Test No. 3, to 85·3 per cent. A high-grade rougher concentrate assaying 22·00 per cent copper is produced. By taking off a lower grade concentrate and cleaning, a higher recovery would likely result.

Very good results are obtained in the flotation of the iron pyrite. This product analyses 48·2 per cent sulphur, 0·04 per cent zinc, and represents 60·9 per cent of the total weight of ore milled.

The copper concentrate assays 0·07 oz. gold and 6·49 oz. silver per ton, representing 26·3 per cent and 41·2 per cent of the gold and silver in the ore.

### *Test No. 9*

This test is the same as No. 8 with the exception that after the flotation of zinc, the pulp was dewatered and repulped with fresh water before acidifying for the flotation of the iron pyrite.

*Reagents (lb. per ton)*—

| To ball mill | To flotation machine | | | |
|---|---|---|---|---|
| | —— | Copper | Zinc | Iron |
| Lime.............. 6·0 | Potassium xanthate............ | 0·06 | 0·05 | |
| | Amyl xanthate................. | | | 0·30 |
| | Copper sulphate................ | | 1·0 | |
| | Sulphuric acid................. | | | 5·0 |
| | Pine oil....................... | 0·04 | 0·04 | 0·06 |

*Results*—

| Product | Weight, per cent | Assay | | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent | Cu | Zn | Fe |
| Heads.................. | | 1·98 | 0·49 | 40·5 | | | | |
| Cu concentrate........... | 7·2 | 23·28 | 0·46 | 33·1 | | 86·2 | 13·1 | 5·9 |
| Zn concentrate........... | 2·6 | 6·60 | 5·85 | 33·6 | | 8·9 | 60·3 | 2·2 |
| Fe concentrate........... | 53·7 | 0·07 | 0·05 | 47·3 | 48·2 | 1·9 | 10·6 | 62·6 |
| Tailing.................. | 36·5 | 0·16 | 0·11 | 32·5 | | 3·0 | 16·0 | 29·3 |

The results of this test duplicate those obtained in No. 8 with the exception that less pyrite concentrate was produced. There was 28·8 per cent of the gold and 41·5 per cent of the silver in the copper concentrate which assayed 0·08 oz. gold and 6·80 oz. silver per ton.

Twelve pounds per ton less of sulphuric acid was used in this test. It is apparent that it is advantageous to dewater the pulp prior to acidifying, thus getting rid of lime in solution which would require acid to neutralize it.

## SUMMARY AND CONCLUSIONS

A comparison of results secured in the copper flotation when using soda ash-cyanide, lime-cyanide, and lime alone for conditioning agents is of interest.

| Conditioning agents | Test No. | Assay Cu, per cent | Per cent of values, Cu | Per cent of weight of feed |
|---|---|---|---|---|
| Soda ash-cyanide............................ | 6 | 14·75 | 89·7 | 12·0 |
| Lime-cyanide................................ | 7 | 18·91 | 73·0 | 7·4 |
| Lime........................................ | 9 | 23·28 | 86·2 | 7·2 |

It is evident that a lime-cyanide combination results in low copper recoveries.

When using lime alone the bulk of concentrate produced is much lower than when soda ash-cyanide is used, but a higher grade product is obtained. Due to this feature, that a rougher concentrate assaying 23·28 per cent is produced when using only lime, an increase in the recovery can be expected if a lower grade rougher concentrate be taken off. This concentrate can then be re-floated in cleaner cells to give a high-grade finished copper concentrate and a middling product which will be returned to the circuit.

Acidifying the pulp after the zinc flotation yields the highest recoveries of iron pyrite—50 to 60 per cent of the ore milled can be recovered in a product containing 48 to 50 per cent sulphur, and containing from 0·10 to 0·05 per cent zinc.

Dewatering the tailing from the zinc flotation cells prior to acidifying will result in a saving of acid without affecting the recoveries of iron pyrite.

It is recommended that grinding be done in a circuit alkaline with lime. The use of cyanide is not advised as it results in lowered copper recoveries. A saving in cost of reagents and royalties is also effected by its omission. The amount of lime necessary will be determined by actual mill operation, only enough to depress the iron pyrite and maintain sufficient alkalinity for best copper flotation conditions being needed.

The removal of the zinc presents no difficulties, as the ore does not contain any great quantity. However, as it will contaminate the iron pyrite concentrate if not removed, it is floated with a minimum addition of reagents and run to waste.

A saving in acid will be effected by dewatering the pulp before entering the iron pyrite flotation circuit. This can be achieved by using any of the standard thickeners or dewaterers followed by dilution of the thickened product while passing to the flotation cells.

Filtering of the iron pyrite concentrate should give no trouble as the moisture content is reduced very rapidly on a suction filter.

The flotation of this ore presents no extraordinary metallurgical problems. However, due to the extremely massive nature of the ore, oxygen depletion of the pulp may result in the grinding mill thus affecting flotation. It would then be necessary to condition the pulp prior to flotation.

## Report No. 352

### EXPERIMENTAL TESTS ON GOLD ORE FROM THE BEARDMORE GOLD MINES, LTD., PORT ARTHUR AREA, ONTARIO

**J. S. Godard**

*Shipments.*—A shipment of 240 pounds was received from the Beardmore Gold Mines, Ltd., Jellicoe, Port Arthur area, Ontario, Nov. 22, 1929. The shipment consisted of two samples, one of which was carbonaceous schisted wall rock, weight 40 pounds, and the other was ore, weight 200 pounds.

*Characteristics of the Samples.*—The wall rock shows no signs of mineralization and carries only a trace of gold.

The gold in the ore is mainly associated with arsenopyrite and the sulphides of iron, which occur in fractures in the quartz gangue. No free gold is visible to the naked eye, but about 80 per cent of it is freed on grinding to 100 mesh.

*Analyses.* The schisted wall rock carried only a trace of gold, the ore assaying as follows:—

| | | | |
|---|---|---|---|
| Gold | 0·93 oz./ton | Copper | 0·02 per cent |
| Silver | 0·13 " | Iron | 4·12 " |
| Lead | 0·03 per cent | Arsenic | 1·02 " |
| Zinc | 0·02 " | Insoluble | 79·51 " |

## EXPERIMENTAL TESTS

### AMALGAMATION

### *Test No. 1*

*Results*—

| Product | Weight, per cent | Assays | Per cent of values |
|---|---|---|---|
| | | Au oz./ton | |
| +35 | 1·2 | 0·27 | 0·8 |
| +48 | 6·9 | 0·30 | 5·2 |
| +65 | 18·9 | 0·50 | 24·0 |
| +100 | 25·2 | 0·52 | 33·2 |
| +200 | 12·2 | 0·49 | 15·2 |
| −200 | 35·6 | 0·42 | 21·6 |

Tailing, gold 0·40 oz./ton. Recovery 56·9 per cent.

### *Test No. 2*

| Product | Weight, Per cent | Assays | Per cent of values |
|---|---|---|---|
| | | Au, oz./ton | |
| +65 | 1·1 | 0·49 | 2·5 |
| +100 | 8·6 | 0·30 | 11·9 |
| +200 | 26·0 | 0·27 | 32·3 |
| −200 | 64·3 | 0·18 | 53·3 |

Tailing, gold 0·22 oz./ton. Recovery 76·3 per cent.

### *Test No. 3*

| Product | Weight, per cent | Assays | Per cent of values |
|---|---|---|---|
| | | Au, oz./ton | |
| +100 | 5·3 | 0·32 | 9·0 |
| +200 | 22·2 | 0·22 | 25·8 |
| −200 | 72·5 | 0·17 | 65·2 |

Tailing, gold 0·19 oz./ton. Recovery 79·6 per cent.

## AMALGAMATION AND FLOTATION OF THE AMALGAMATION TAILING

### *Test No. 4*

*Results—Flotation—*

| Product | Weight, per cent | Assays | Per cent of values |
|---|---|---|---|
| | | Au, oz./ton | |
| Flotation concentrate | 9·0 | 1·78 | 81·5 |
| Tailing + 100 | 6·7 | 0·08 | 2·7 |
| Tailing + 200 | 29·0 | 0·05 | 7·4 |
| Tailing − 200 | 55·3 | 0·03 | 8·4 |

Amalgamation tailing (calculated) gold 0·20 oz./ton.

*Distribution of gold—*

| | | | |
|---|---|---|---|
| Amalgamation | 79·4 per cent | Tailing | 3·8 per cent |
| Flotation concentrate | 16·8 " | Recovery | 96·2 " |

## FLOTATION AND TABLING

### *Test No. 5*

*Results—*

| Product | Weight, per cent | Assays | Per cent of values |
|---|---|---|---|
| | | Au, oz./ton | |
| Flotation concentrate | 8·9 | 10·38 | 84·5 |
| Table concentrate | 4·5 | 1·96 | 8·1 |
| Tailing +100 | 17·4 | 0·28 | 4·4 |
| Tailing −100 | 41·6 | 0·07 | 2·7 |
| Slimes | 27·6 | 0·01 | 0·3 |

### *Test No. 6*

| Product | Weight, per cent | Assays | Per cent of values |
|---|---|---|---|
| | | Au, oz./ton | |
| Flotation concentrate | 8·9 | 9·58 | 83·6 |
| Table concentrate | 5·7 | 2·46 | 13·7 |
| Tailing +100 | 10·7 | 0·06 | 0·6 |
| Tailing +200 | 19·9 | 0·05 | 1·0 |
| Tailing −200 | 25·4 | 0·02 | 0·5 |
| Slimes | 29·4 | 0·02 | 0·0 |

| | *Flotation concentrate—* | *Table concentrate—* |
|---|---|---|
| As | 10·27 per cent | 1·22 per cent |
| Fe | 18·61 " | 9·04 " |
| Cu | 0·25 " | 0·10 " |
| Pb | 0·09 " | 0·06 " |
| Zn | 0·28 " | 0·03 " |
| Insoluble | 36·13 " | 64·31 " |

### *Test No. 7*

| Product | Weight, per cent | Assay | Per cent of values |
|---|---|---|---|
| | | Au, oz./ton | |
| Flotation concentrate | 11·8 | 7·54 | 91·6 |
| Table concentrate | 6·2 | 0·90 | 5·7 |
| Tailing −100 | 5·9 | 0·09 | 0·5 |
| Tailing +200 | 20·6 | 0·06 | 1·3 |
| Tailing −200 | 33·0 | 0·02 | 0·7 |
| Slimes | 22·5 | 0·01 | 0·2 |

### *Summary of Concentration Tests*

| Test No. | Recoveries | | | Ratio of concentration |
|---|---|---|---|---|
| | Flotation | Tabling | Total | |
| 5 | 84·5 | 8·1 | 92·6 | 7·5 : 1 |
| 6 | 83·6 | 13·7 | 97·3 | 6·9 : 1 |
| 7 | 91·6 | 5·7 | 97·3 | 5·6 : 1 |

*Reagents*—

*Test No. 5*—To ball mill: soda ash 3·0 lb./ton, P.T. & T. Co. No. 1580 0·24 lb./ton. To cell: amyl xanthate 0·05 lb./ton, pine oil 0·08 lb./ton.

*Test No. 6*—To ball mill: coal-tar creosote 0·20 lb./ton. To cell: potassium xanthate 0·15 lb./ton, pine oil 0·08 lb./ton.

*Test No. 7*—To ball mill: coal tar creosote 0·16 lb./ton. To cell: amyl xanthate 0·15 lb./ton, sodium sulphide 0·8 lb./ton, pine oil 0·08 lb./ton.

## FLOTATION—TWO-STAGE

### *Test No. 8*

*Procedure.* Ore, 2,000 grammes at −14 mesh ground 10 minutes in ball mill with soda ash 2·0 lb. per ton, P.T. & T. Co. No. 1580 0·12 lb. per ton, Aerofloat No. 25 0·04 lb. per ton. To cell: amyl xanthate 0·05 lb. per ton, pine oil 0·02 lb. per ton. No. 1 concentrate removed. Tailing roughly classified and oversize reground with soda ash 1·5 lb. per ton, No. 1580 0·06 lb. per ton, Aerofloat No. 25 0·06 lb. per ton. To cell: amyl xanthate 0·05 lb. per ton, pine oil 0·03 lb. per ton. No. 2 concentrate removed.

*Results*—

| Product | Weight, per cent | Assay | Per cent of values |
|---|---|---|---|
| | | Au, oz./ton | |
| Concentrate No. 1 | 6·7 | 12·88 | 86·4 |
| Concentrate No. 2 | 4·2 | 2·46 | 10·4 |
| Classifier overflow | 29·8 | 0·037 | 1·1 |
| Tailing | 59·3 | 0·035 | 2·1 |

Average assay of concentrate (calculated) gold 8·83 oz./ton.
Average assay of tailing (calculated) gold /0·036 oz./ton.
Ratio of concentration 9·2 : 1.

A screen analysis of the classifier overflow is as follows:

| Product | Weight, per cent | Assays | Per cent of values |
|---|---|---|---|
| | | Au, oz./ton | |
| +100 | 9·1 | 0·12 | 29·9 |
| +200 | 14·9 | 0·07 | 28·5 |
| −200 | 76·0 | 0·02 | 41·6 |

Assay of classifier overflow (calculated) gold 0·037 oz./ton.

A screen analysis of the final tailing gave the following results:

| Product | Weight, per cent | Assays | Per cent of values |
|---|---|---|---|
| | | Au, oz./ton | |
| +100 | 14·3 | 0·10 | 41·0 |
| +200 | 39·9 | 0·04 | 45·8 |
| −200 | 45·8 | 0·01 | 13·2 |

Assay of final tailing (calculated) gold 0·035 oz./ton.

Results of screen test showing approximate size at which first flotation was made:

| Product | Weight, per cent |
|---|---|
| +28 | 0·2 |
| +35 | 1·0 |
| +48 | 3·9 |
| +65 | 13·0 |
| +100 | 18·5 |
| +200 | 21·9 |
| −200 | 41·5 |

## CYANIDATION

### *Tests Nos. 9 to 13*

Five cyanidation tests were made on samples dry crushed to different sizes.

Agitation in Winchester bottles for 48 hours.

Dilution 1:3 and KCN 0·05 per cent in each test.

Titrations for KCN and CaO were made twice daily.

*Results*—

Head sample: gold 0·90 oz./ton.

| Test No. | Mesh | Tailing* | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | | Au, oz./ton | | KCN | CaO |
| 9 | +65 | 0·081 | 91·0 | 0·38 | 4·3 |
| 10 | +100 | 0·044 | 95·1 | 0·41 | 5·2 |
| 11 | −100 | 0·028 | 96·9 | 0·53 | 5·3 |
| 12 | +150 | 0·014 | 98·4 | 0·72 | 6·5 |
| 13 | −150 | 0·013 | 98·5 | 0·61 | 6·9 |

*Average.

*Results of Screening the Tailings*

| Test No. | Mesh | Weight, per cent | Au, oz./ton | Per cent of values | Tailing, Au, oz./ton |
|---|---|---|---|---|---|
| 9 | +65 | 21·5 | 0·26 | 69·2 | 0·081 |
| | +100 | 19·0 | 0·07 | 16·5 | ............ |
| | +200 | 19·1 | 0·04 | 9·4 | ............ |
| | −200 | 40·4 | 0·01 | 4·9 | ............ |
| 10 | +100 | 26·4 | 0·11 | 65·7 | 0·044 |
| | +200 | 25·7 | 0·04 | 23·4 | ............ |
| | −200 | 47·9 | 0·01 | 10·9 | ............ |
| 11 | +100 | 12·2 | 0·09 | 39·7 | 0·028 |
| | +200 | 26·2 | 0·04 | 37·9 | ............ |
| | −200 | 61·6 | 0·01 | 22·4 | ............ |
| 12 | +200 | 14·3 | 0·04 | 39·9 | 0·014 |
| | −200 | 85·7 | 0·01 | 60·1 | ............ |
| 13 | +200 | 16·3 | 0·03 | 36·8 | 0·013 |
| | −200 | 83·7 | 0·01 | 63·2 | ............ |

## CONCLUSIONS

*Amalgamation.* Amalgamation on wet ground samples gave recoveries of 57 per cent at 8 per cent plus 48 mesh to 79 per cent on ore ground to 5 per cent plus 100 mesh. This latter figure is probably a little higher than that obtainable at this size in actual practice.

*Concentration.* By the use of flotation and tabling the flotation tailings 97·3 per cent of the gold was recovered in a combined concentrate assaying, gold 6·79 oz./ton (refer Test 6). Flotation accounts for the major part of the recovery. Such a concentrate might be sold to a smelter or reground and cyanided in the mill.

Two-stage flotation recovered 96·8 per cent of the gold compared with 97·3 per cent recovery by the combined flotation and tabling method, but it showed an increased ratio of concentration 9·2 : 1 (refer Test 6), to 6·9 : 1 for the combination flow-sheet.

*Cyanidation* gave very good results on the ore when ground to minus 100 mesh. The cyanide and lime consumptions were moderate.

Grinding to 100 mesh is necessary for good results either by concentration or cyanidation.

In view of the fact that a small mill will satisfy the present requirements it is advisable to use a straight cyanidation flow-sheet.

Should the mine develop into a large-scale one a change might be made from cyanidation to concentration followed by cyanidation of the concentrates, in which case the present proposed mill could be used for treatment of the concentrates.

## Report No. 353

### FLOTATION TESTS ON ORE SAMPLE FROM SHERRITT-GORDON MINES, LTD., SHERRIDON, MANITOBA

**G. B. Walker**

*Shipment.* A 50-pound sample of copper-zinc ore has been received, at this laboratory, from Sherritt-Gordon Mines, Ltd., Toronto, Ontario. This sample is said to represent the West End ore-body. A head sample on this ore gave the following analysis:

| | | | |
|---|---|---|---|
| Copper | 2·53 per cent | Insoluble | 21·50 per cent |
| Zinc | 4·98 " | Gold | 0·01 oz./ton |
| Iron | 38·65 " | Silver | 0·51 " |

The copper and zinc values are present as chalcopyrite and sphalerite, and are finely disseminated in pyrrhotite. The ore can be classed as a massive sulphide.

*Purpose of Tests.* The purpose of this investigation was to determine the amenability of this ore sample to differential flotation and to compare the behaviour of this ore with previous samples.

*Preparation of Sample.* The entire sample was stage crushed to −14 mesh in a jaw crusher and rolls. It was then thoroughly mixed by rolling and 1,000-gramme samples were cut out for flotation testing.

*Test Procedure.* A 1,000-gramme charge of ore was ground in a laboratory-size ball mill, at 1 : 1 dilution, with reagents as indicated in the following tests. Tests Nos. 1 to 5 were ground for a 30-minute period which resulted in the following degree of fineness:

*Screen Analysis, Flotation Feed—*

| Mesh | Per cent held | Mesh | Per cent held |
|---|---|---|---|
| 48 | 0·2 | 150 | 6·4 |
| 65 | 2·3 | 200 | 13·6 |
| 100 | 5·9 | −200 | 71·6 |

Tests Nos. 6 to 9 inclusive, were ground for a 40-minute period. A screen analysis is as follows:

| Mesh | Per cent held | Mesh | Per cent held |
|---|---|---|---|
| 48 | .......... | 150 | 6·2 |
| 65 | 0·5 | 200 | 12·3 |
| 100 | 3·1 | −200 | 77·9 |

After grinding was completed the resultant pulp was transferred to a laboratory-size, Ruth, sub-aerated flotation cell where sufficient water was added to make the pulp dilution 4 : 1.

Reagents were added and froth was removed as indicated in the following tests.

Approximately 25 tests were run on this sample but relatively few were saved for assay due to obviously poor results. Following is a brief summary, with results, of the tests assayed.

### *Test No. 1: Effect of Thiocarbanilide*

*Reagents*—

| To ball mill— | | To copper cells— | | To zinc cells— | |
|---|---|---|---|---|---|
| $Na_2CO_3$.. | 0·10lb./ton | Aerofloat No. 25 | 0·05 lb./ton | $CuSO_4$.... | 0·50 lb./ton |
| NaCN... | 0·10 " | Cresylic acid | 0·24 " | | |
| Thiocarbanilide | 0·05 " | | | | |

pH....... 8·4

*Results*—

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu, per cent | Zn, per cent |
| Head.................................... | 100·0 | 2·53 | 5·14 | 100·0 | 100·0 |
| Cu concentrate............................ | 15·3 | 15·24 | 9·74 | 92·5 | 29·0 |
| Zn concentrate............................ | 21·6 | 0·68 | 14·33 | 5·8 | 60·3 |
| Tailing.................................. | 63·1 | 0·07 | 0·87 | 1·7 | 10·7 |

This test appeared to have excessive reagent in the copper float which carried through to the zinc circuit and floated iron. Additional cyanide should tend to depress the iron and allow more thiocarbanilide to be used on the copper.

### *Test No. 2: Effect of High Cyanide and Thiocarbanilide*

*Reagents*—

| To ball mill— | | To copper cells— | | To zinc cells— | |
|---|---|---|---|---|---|
| $Na_2CO_3$.. | 5·0 lb./ton | Cresylic acid | 0·20 lb/ton | $CuSO_4$.... | 0·50 lb./ton |
| NaCN... | 0·20 " | | | Cresylic acid | 0·08 " |
| Thiocarbanilide | 0 10 " | | | | |

pH....... 7·8

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu, per cent | Zn, per cent |
| Head.................................... | 100·0 | 2·58 | 5·18 | 100·0 | 100·0 |
| Cu concentrate............................ | 14·8 | 15·20 | 7·61 | 87·3 | 21·7 |
| Zn concentrate............................ | 31·0 | 0·90 | 11·92 | 10·8 | 71·4 |
| Tailing.................................. | 54·2 | 0·09 | 0·66 | 1·9 | 6·9 |

Zinc reagents were agitated for 5 minutes before zinc flotation.

Zinc flotation was improved by the use of high cyanide and increased thiocarbanilide but copper recovery suffered. Higher copper recovery was made in Test No. 1 with combination of thiocarbanilide and Aerofloat No. 25.

### *Test No. 3: Effect of Reagent No. 305*

*Reagents*—

| To ball mill— | | To copper cells— | | To zinc cells— | |
|---|---|---|---|---|---|
| $Na_2CO_3$.. | 5·0 lb./ton | No. 305... | 0·10 lb./ton | $CuSO_4$.... | 0·50 lb./ton |
| NaCN... | 0·20 " | Cresylic acid | 0·20 " | Cresylic acid | 0·08 " |
| pH....... | 7·8 | | | | |

*Results*—

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu, per cent | Zn, per cent |
| Head | 100·0 | 2·54 | 5·25 | 100·0 | 100·0 |
| Cu concentrate | 12·3 | 17·00 | 9·20 | 83·0 | 21·7 |
| Zn concentrate | 27·7 | 1·40 | 10·83 | 14·2 | 57·1 |
| Tailing | 60·0 | 0·12 | 1·85 | 2·8 | 21·2 |

Reagent No. 305 produced a higher grade copper concentrate but both copper and zinc recoveries were low.

### *Test No. 4: Effect of Potassium Xanthate*

*Reagents*—

| To ball mill— | | To copper cells— | | To zinc cells— | |
|---|---|---|---|---|---|
| $Na_2CO_3$.. | 5·0 lb./ton | Xanthate. | 0·10 lb./ton | $CuSO_4$.... | 0·50 lb./ton |
| NaCN... | 0·20 " | Cresylic acid | 0·20 " | Cresylic acid | 0·08 " |
| pH....... | 7·8 | | | | |

*Results*—

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu, per cent | Zn, per cent |
| Head | 100·0 | 2·49 | 5·32 | 100·0 | 100·0 |
| Cu concentrate | 14·7 | 15·06 | 6·95 | 89·5 | 19·2 |
| Zn concentrate | 30·2 | 0·73 | 12·58 | 8·8 | 71·2 |
| Tailing | 55·1 | 0·08 | 0·93 | 1·7 | 9·6 |

Potassium xanthate caused higher recovery of copper than thiocarbanilide but zinc recovery and grade are low.

### *Test No. 6: Effect of Fine Grinding, Air, and Potassium Xanthate*

*Reagents—*

| To ball mill— | To copper cells— | To zinc cells— |
|---|---|---|
| $Na_2CO_3$.. 12·0 lb./ton | Xanthate. 0·03 lb./ton | $CuSO_4$.... 2·00 lb./ton |
| NaCN... 0·12 " | Cresylic acid 0·40 " | |
| pH...... 8·0 | | |

*Results—*

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu, per cent | Zn, per cent |
| Head | 100·0 | 2·46 | 5·03 | 100·0 | 100·0 |
| Cu concentrate | 13·5 | 16·48 | 5·63 | 90·9 | 15·2 |
| Zn concentrate | 9·4 | 1·25 | 34·35 | 4·8 | 65·2 |
| Zn middling | 11·4 | 0·42 | 2·84 | 1·9 | 6·5 |
| Tailing | 65·7 | 0·09 | 1·00 | 2·4 | 13·1 |

Pulp was agitated with air for 10 minutes before copper float. Zinc reagents were agitated in the pulp for 10 minutes before the zinc float. The rougher zinc concentrate was cleaned at 20:1 dilution for 5 minutes.

### *Test No. 7: Effect of Reagent No. 208*

*Reagents—*

| To ball mill— | To copper cells— | To zinc cells— |
|---|---|---|
| $Na_2CO_3$.. 12·0 lb./ton | No. 208..... 0·03 lb./ton | $CuSO_4$.... 2·00 lb./ton |
| NaCN... 0·12 " | Cresylic acid 0·40 " | |
| pH...... 8·0 | | |

*Results—*

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu, per cent | Zn, per cent |
| Heads | 100·0 | 2·34 | 5·02 | 100·0 | 100·0 |
| Cu concentrate | 17·4 | 12·68 | 5·14 | 94·3 | 17·8 |
| Zn concentrate | 14·4 | 0·54 | 25·16 | 3·3 | 72·0 |
| Zn middling | 22·2 | 0·15 | 1·55 | 1·4 | 6·8 |
| Tailing | 46·0 | 0·05 | 0·38 | 1·0 | 3·4 |

Pulp was agitated before copper and the zinc floats as in Test No. 6. Zinc rougher concentrate was cleaned without additional oil. Excellent copper recovery was obtained in this test. The lower grade zinc concentrate obtained here will compare favourably with the zinc concentrate in Test No. 6 when the difference in recoveries is considered.

### *Test No. 8: Effect of Aerofloat No. 15*

*Reagents*—

| To ball mill— | | To copper cells— | | To zinc cells— | |
|---|---|---|---|---|---|
| $Na_2CO_3$ | 12·0 lb./ton | Cresylic acid | 0·32 lb./ton | $CuSO_4$ | 1·00 lb./ton |
| NaCN | 0·12 " | | | | |
| Aerofloat No. 15 | 0·04 " | | | | |
| pH | 8·0 | | | | |

*Results*—

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu, per cent | Zn, per cent |
| Head | 100·0 | 2·29 | 5·10 | 100·0 | 100·0 |
| Cu concentrate | 13·3 | 15·88 | 6·90 | 91·7 | 18·0 |
| Zn concentrate | 8·4 | 0·93 | 32·58 | 3·4 | 54·0 |
| Zn middling | 7·5 | 0·56 | 6·18 | 1·8 | 9·1 |
| Tailing | 70·8 | 0·11 | 1·36 | 3·1 | 18·9 |

Pulp was agitated with air before copper and zinc floats as in Test No. 6.

Aerofloat made good copper recovery and produced a high-grade zinc concentrate but zinc recovery was low.

### *Test No. 9: Effect of Reagent No. 323*

*Reagents*—

| To ball mill— | | To copper cells— | | To zinc cells— | |
|---|---|---|---|---|---|
| $Na_2CO_3$ | 12·0 lb./ton | No. 323 | 0·03 lb./ton | $CuSO_4$ | 2·00 lb./ton |
| NaCN | 0·12 " | Cresylic acid | 0·40 " | | |
| pH | 8·0 | | | | |

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu, per cent | Zn, per cent |
| Head | 100·0 | 2·46 | 4·93 | 100·0 | 100·0 |
| Cu concentrate | 14·6 | 15·12 | 5·74 | 90·2 | 17·1 |
| Zn concentrate | 11·1 | 1·03 | 28·55 | 4·6 | 64·3 |
| Zn middling | 21·9 | 0·48 | 3·22 | 4·2 | 14·3 |
| Tailing | 52·4 | 0·05 | 0·41 | 1·0 | 4·3 |

The pulp was agitated with air before copper and zinc flotation as in Test No. 6.

Reagent No. 323 has about the same effect as potassium xanthate on this ore.

## OBSERVATIONS

The following observations have been made from tests run but not assayed.

(1) *Reagents found Effective on Sample No. 2.* Several tests were made in which the reagent combination and amounts found most effective on Sample No. 2 were employed, i.e., thiocarbanilide 0·09 pounds per ton; xanthate 0·20 pounds per ton; and TT mixture 0·10 pounds per ton. These reagents proved to be far in excess of the requirements of this sample. In all cases a heavy bulk float was obtained.

(2) *Acidity.* This sample required more soda ash than the previous sample, 3·0 pounds failed to furnish alkalinity. During the course of this investigation the ore oxidized rapidly. This indicates the need for treatment as soon as possible after the ore is broken.

(3) *Lime Alkalinity.* The use of lime in the copper circuit on this sample did not result in a mineralized froth. Its use does not appear feasible in the copper circuit on this ore.

The use of lime in the zinc circuit did not have any marked benefit but its use may be found advantageous in pilot plant operations.

(4) *Soda Ash.* Increased soda ash must be used as the ore oxidizes. A substantial excess should be used in mill operations to provide a margin of safety for sudden ore changes.

(5) *Agitation with Air.* The practice of agitating the pulp with air before the addition of flotation reagents appears to be beneficial.

(6) *Effect of Dichromate.* Potassium dichromate was tried in various amounts in the zinc float but no marked effect was observed. The combination of heat and dichromate should be investigated in pilot plant operations.

## SUMMARY

*Grinding.* Due to the fine dissemination of the minerals in this ore, fine grinding is of prime importance. Grinding to approximately 80 per cent—200 mesh seems advisable.

*Agitation.* Agitation of the pulp with air appears to be of benefit on this ore. Pilot plant operations should investigate this feature.

*Copper Flotation.* Under the conditions employed in these tests it appears reasonable to expect copper recoveries of from 90 to 94 per cent in a 12 to 15 per cent rougher concentrate depending on the reagent used. No attempt was made to clean the rougher copper concentrate in this investigation.

*Zinc Flotation.* The flotation of the sphalerite, in a high-grade concentrate, from this ore presents considerable difficulty due to the fast floating pyrrhotite. Repeated cleaning operations may bring the grade up to the desired 50 per cent zinc. If the rougher concentrates were thickened and reground this difficulty might be reduced to some extent.

## CONCLUSIONS

The ore represented by the sample received at this laboratory is amenable to selective flotation but can be expected to present more difficulties than ore samples which have previously been tested, especially in the zinc flotation.

The most apparent difference between this and previous samples is that very small amounts of reagents are required by this ore, 0·03 pound per ton appears to be sufficient.

## Report No. 354

### EXPERIMENTAL TESTS ON GOLD ORE FROM THE ARROWHEAD CONSOLIDATED MINES, LTD., JOANNES TOWNSHIP, ROUYN AREA, QUEBEC

**J. S. Godard**

*Shipment.* A shipment of 836 pounds of ore was received January 11, 1930, from the Arrowhead Consolidated Mines, per L. Jones, Secretary-Treasurer, 412 New Birks Building, Montreal, Quebec.

*Characteristics of the Ore.* The ore is a gold one. Some fine free gold is visible in small fractures in the quartz which is the chief constituent of the gangue. Ninety-three per cent is amalgamable on grinding to 70 per cent −200 mesh, and about 60 per cent of the remainder is associated with the sulphides, the predominating one of which is iron pyrites.

*Sampling and Analysis.* The shipment was sampled in duplicate. Mr. J. W. Davis, engineer for the company was present during the second sampling.

Four assays were made on each sample with the following results.

| No. 1 Sample | | No. 2 Sample | |
|---|---|---|---|
| No. | Gold, oz./ton | No. | Gold, oz./ton |
| 1 | 1·23 | 1 | 1·34 |
| 2 | 1·37 | 2 | 1·36 |
| 3 | 1·28 | 3 | 1·30 |
| 4 | 1·29 | 4 | 1·34 |
| Average No. 1 | 1·293 | Average | 1·335 |

Average from all assays, gold 1·314 oz./ton.

*Purpose of Experimental Tests.* Experimental tests were made to determine the best metallurgical treatment for this ore.

### Experimental Tests

### CONCENTRATION TESTS

### *Flotation—Tabling the Flotation Tailing*

### *Test No. 1*

*Results—*

| Product | Weight, per cent | Assay Au, oz./ton | Per cent of values |
|---|---|---|---|
| Flotation concentrate | 7·4 | 7·50 | 56·2 |
| Table concentrate | 13·5 | 2·90 | 39·6 |
| Tailing +100 | 23·2 | 0·09 | 2·1 |
| Tailing +200 | 18·5 | 0·04 | 0·8 |
| Tailing −200 | 17·8 | 0·05 | 0·9 |
| Slimes | 19·6 | 0·02 | 0·4 |

### *Test No. 2*

| Product | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| Flotation concentrate | 7·1 | 12·64 | 89·8 |
| Table concentrate | 7·9 | 1·08 | 8·5 |
| Tailing +200 | 23·1 | 0·02 | 0·5 |
| Tailing −200 | 31·0 | 0·02 | 0·6 |
| Slimes | 30·9 | 0·015 | 0·6 |

### *Test No. 3: Two-Stage Flotation*

| Product | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| No. 1 concentrate | 2·0 | 43·02 | 78·4 |
| No. 2 concentrate | 2·6 | 3·80 | 9·0 |
| Classifier overflow +200 | 3·5 | 0·04 | 0·1 |
| Classifier overflow −200 | 19·5 | 0·02 | 0·4 |
| Tailing +100 | 13·7 | 0·84 | 10·5 |
| Tailing +200 | 28·1 | 0·04 | 1·0 |
| Tailing −200 | 30·6 | 0·02 | 0·6 |

The following screen analysis indicates the sizes of the ore particles at the flotation of No. 1 concentrate.

| Size | Weight, per cent | Cumulative, per cent |
|---|---|---|
| +28 +35 | 1·1 | 1·1 |
| +48 | 4·9 | 6·0 |
| +65 | 18·4 | 24·4 |
| +100 | 24·5 | 48·9 |
| +200 | 18·7 | 67·6 |
| −200 | 32·4 | 100·0 |

### *Test No. 4: Amalgamation—Flotation of the Amalgamation Tailing*

| Product | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| Flotation concentrate | 3·6 | 1·35 | 61·0 |
| Tailing +200 | 27·1 | 0·05 | 17·3 |
| Tailing −200 | 69·3 | 0·025 | 21·7 |

Amalgamation tailing (calculated) 0·08 oz./ton.
Head sample, 1·29 oz./ton.
Recovery—Amalgamation 93·8 per cent, Flotation 3·8 per cent, Total, 97·6 per cent.

## AMALGAMATION TESTS

A series of amalgamation tests was made on samples of ore dry crushed to pass given screens.

Screen analyses were made on the amalgamation tailings.

### *Test No. 5: Amalgamation at −18 mesh*

*Results*—

| Size | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| +28 | 21·1 | 1·01 | 41·7 |
| +35 | 17·5 | 0·47 | 16·1 |
| +48 | 12·4 | 0·57 | 13·8 |
| +65 | 12·2 | 0·53 | 12·6 |
| +100 | 11·8 | 0·30 | 6·9 |
| +200 | 8·5 | 0·22 | 3·7 |
| −200 | 16·5 | 0·16 | 5·2 |

Average tailing (calculated) gold 0·51 oz./ton.

### *Test No. 6: Amalgamation at −28 mesh*

| Size | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| +35 | 11·3 | 0·44 | 14·8 |
| +48 | 20·2 | 0·44 | 26·4 |
| +65 | 18·0 | 0·43 | 23·0 |
| +100 | 16·6 | 0·27 | 13·3 |
| +200 | 10·9 | 0·23 | 7·5 |
| −200 | 23·0 | 0·22 | 15·0 |

Average tailing (calculated) gold 0·34 oz./ton.

### *Test No. 7: Amalgamation at −35 mesh*

| Size | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| +48 | 9·6 | 0·44 | 16·7 |
| +65 | 24·3 | 0·33 | 31·9 |
| +100 | 21·9 | 0·24 | 20·7 |
| +200 | 14·6 | 0·18 | 10·8 |
| −200 | 29·6 | 0·17 | 19·9 |

Average tailing (calculated) gold 0·25 oz./ton.

### *Test No. 8: Amalgamation at −48 mesh*

| Size | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| +65 | 19·9 | 0·28 | 29·8 |
| +100 | 24·0 | 0·22 | 28·2 |
| +200 | 21·6 | 0·14 | 16·2 |
| −200 | 34·5 | 0·14 | 25·8 |

Average tailing (calculated) gold 0·19 oz./ton.

### *Test No. 9: Amalgamation at −65 mesh*

| Size | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| +100 | 14·2 | 0·14 | 18·4 |
| +200 | 35·0 | 0·15 | 48·6 |
| −200 | 50·8 | 0·07 | 33·0 |

Average tailing (calculated) gold 0·11 oz./ton.

### *Test No. 10: Amalgamation at −100 mesh*

| Size | Weight, per cent | Assay Au, oz./ton | Per cent of values |
|---|---|---|---|
| +200 | 41·1 | 0·11 | 52·3 |
| −200 | 58·9 | 0·07 | 47·7 |

Average tailing (calculated) gold 0·09 oz./ton.

### *Summary: Amalgamation Tests*

| Test No. | Size | Tailing (average) Au, oz./ton | Recovery, per cent |
|---|---|---|---|
| 5 | − 18 | 0·51 | 60·4 |
| 6 | − 28 | 0·34 | 73·6 |
| 7 | − 35 | 0·25 | 80·6 |
| 8 | − 48 | 0·19 | 85·2 |
| 9 | − 65 | 0·11 | 91·4 |
| 10 | −100 | 0·09 | 92·9 |

## CYANIDATION TESTS

A series of small-scale cyanidation tests was made on samples of ore dry crushed to pass given screens.

Tests were made in Winchester bottles; pulp dilution was 1 : 3; cyanide strength 0·052 per cent KCN; time of agitation 48 hours.

Screen analysis was made on the tailings.

*Results of Screen Analysis*—

### *Test No. 11: Cyanidation at −35 mesh*

| Size | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| + 65 | 30·6 | 0·09 | 68·1 |
| +100 | 25·2 | 0·03 | 18·8 |
| +200 | 17·4 | 0·015 | 6·4 |
| −200 | 26·8 | 0·01 | 6·7 |

Average tailing (calculated) gold 0·04 oz./ton.

### *Test No. 12: Cyanidation at −48 mesh*

| Size | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| +100 | 49·3 | 0·05 | 84·9 |
| +200 | 18·9 | 0·015 | 9·6 |
| −200 | 31·8 | 0·005 | 5·5 |

Average tailing (calculated) gold 0·029 oz./ton.

### *Test No. 13: Cyanidation at −65 mesh*

| Size | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| +100 | 23·7 | 0·13 | 84·8 |
| +200 | 33·8 | 0·01 | 9·4 |
| −200 | 42·5 | 0·005 | 5·8 |

Average tailing (calculated) gold 0·036 oz./ton.

### *Test No. 14: Cyanidation at −100 mesh*

| Size | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| +200 | 37·0 | 0·015 | 47·1 |
| −200 | 63·0 | 0·010 | 52·9 |

Average tailing (calculated) gold 0·012 oz./ton.

### *Test No. 15: Cyanidation at −150 mesh*

| Size | Weight, per cent | Assays Au, oz./ton | Per cent of values |
|---|---|---|---|
| +200 | 18·5 | 0·02 | 31·1 |
| −200 | 81·5 | 0·01 | 68·9 |

Average tailing (calculated) gold 0·012 oz./ton.

### *Summary: Cyanidation Tests*

Head sample, gold 1.21 oz. per ton

| Test No. | Size | Tailing (average) Au, oz./ton | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | | | | KCN | CaO |
| 11 | −35 | 0·04 | 96·6 | 0·35 | 2·53 |
| 12 | −482 | 0·029 | 97·5 | 0·18 | 2·46 |
| 13 | −65 | 0·036 | 97·0 | 0·18 | 2·53 |
| 14 | −100 | 0·012 | 98·9 | 0·42 | 3·67 |
| 15 | −150 | 0·012 | 98·9 | 0·48 | 4·33 |

## CONCLUSIONS

The results of the tests indicate that straight cyanidation of this ore at about 100 mesh with 60 per cent −200 mesh would probably prove to be the best practice.

# Report No. 355

## THE CONCENTRATION OF THE COPPER-NICKEL ORE OF THE FALCONBRIDGE NICKEL MINES, LTD., GARSON, ONTARIO

**C. S. Parsons, G. B. Walker, and A. K. Anderson**

*Shipments.* Fifty tons of ore was received at the Ore Dressing Laboratories, November 12, 1929, shipped by the Falconbridge Nickel Mines, Ltd., Garson, Ontario, E. Craig, Superintendent.

*Characteristics of the Ore.* The ore consisted of sulphides of copper, nickel and iron contained in a basic gangue. A small amount of white quartz also was present.

*Purpose of Experimental Tests.* The object of the investigation was to determine what elimination of gangue could be obtained by concentration at coarse sizes and also to note the effect of this concentration on the recovery of copper and nickel.

*Sampling and Analysis.* No sample of the entire shipment was taken. The feed of each run was sampled separately while the test was in progress.

### EXPERIMENTAL TESTS

Three series of tests were made; first, jigging; second, concentration on a Butchart table; and third, concentration on a Plato table.

Small-scale flotation tests were run on the Butchart table tailing and several on the raw ore.

For the jig tests, 10 tons of the ore was crushed in a jaw crusher and rolls to pass a 1-inch screen. This material was then screened on a Moto-Vibro screen in which the screens are set on an incline.

The sizes at which jigging was done were,—

−1-inch + ¾-inch
−¾-inch + ½-inch
−½-inch + 3 mesh
−3 + 5 mesh
−5 + 10 "

No jigging was done on the −10 mesh product. This was reserved for subsequent table tests.

A James two-compartment suction jig was used in all tests, the tailing from the second compartment being repassed over the jigs, thus making a four-compartment unit. Concentrates were removed from all four.

For the table concentration series of tests, 8 tons of the raw ore was crushed to pass ½ inch and then divided into three lots. The first of these was then crushed to pass 3 mesh, the second 5 mesh, and the third 7 mesh. Each lot was then cut in half. These first halves were passed over a Butchart table and the others over a Plato table. The −10 mesh material from the screening of the lot crushed for jigging was also passed over the tables.

Representative portions of the −5, −7, and −10 mesh Butchart table tailing were taken and reground. Small-scale flotation tests were then made on these.

The detailed tests follow:—

## JIGGING

### *Test No. 1*

Size of feed −1 inch + ¾ inch. Weight = 5,914 pounds.
Head assay: Cu 1·07 per cent, Ni 2·14 per cent, S 15·54 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Concentrate | 59·7 | 1·00 | 2·90 | 20·08 | 52·1 | 82·4 | 80·7 |
| Middling | 32·2 | 1·53 | 1·06 | 8·12 | 43·0 | 16·3 | 17·6 |
| Tailing | 8·1 | 0·70 | 0·34 | 3·19 | 4·9 | 1·3 | 1·7 |
| Heads (calculated) | 100·0 | 1·14 | 2·10 | 14·86 | 100·0 | 100·0 | 100·0 |

Jigging this size of material results in discarding a tailing representing 8·1 per cent of the weight of the feed, a loss of 1·049 pounds copper and 0·556 pound nickel per ton of ore milled. The combined concentrate and middling show a recovery of 95·1 per cent of the copper and 98·7 per cent of the nickel. The concentrate is not much higher in grade than the feed.

### *Test No. 2*

Size of feed −¾ inch + ½ inch. Weight = 3,240 pounds.
Head assay: Cu 1·00 per cent, Ni 2·04 per cent, S 17·26 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Concentrates Nos. 1 and 2 | 59·0 | 1·02 | 3·18 | 15·11 | 54·2 | 83·3 | 75·0 |
| Concentrates Nos. 3 and 4 | 7·0 | 1·44 | 1·92 | 14·85 | 9·1 | 6·0 | 8·1 |
| Tailing | 34·0 | 1·20 | 0·71 | 5·88 | 36·7 | 10·7 | 16·9 |
| Heads (calculated) | 100·0 | 1·11 | 2·25 | 11·88 | 100·0 | 100·0 | 100·0 |

This test shows a larger amount of material discarded as tailing than the preceding one, 34 per cent the weight of feed. With assays as shown, this represents a loss of 7·34 pounds copper and 2·897 pounds nickel per ton of ore milled. A slight increase in the percentage of nickel in the concentrate is noted. The copper sulphides show a tendency to pass out in the tailing.

### *Test No. 3*

Size of feed −½ inch + 3 mesh. Weight = 5,485 pounds.
Head assay: Cu 1·00 per cent, Ni 2·22 per cent, S 15·55 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Concentrate Nos. 1 and 2.......... | 37·6 | 0·80 | 3·05 | 20·79 | 31·5 | 54·5 | 52·7 |
| Concentrate Nos. 3 and 4.......... | 37·9 | 1·10 | 2·15 | 15·35 | 43·6 | 38·7 | 39·2 |
| Tailing.......................... | 24·5 | 0·97 | 0·58 | 4·90 | 24·9 | 6·8 | 8·1 |
| Heads (calculated)................ | 100·0 | 0·95 | 2·10 | 14·83 | 100·0 | 100·0 | 100·0 |

The results of this test show that the copper has a tendency to pass into the tailing. Concentrates Nos. 3 and 4 produced by the jigs treating the tailing from the first pass over the two-compartment jig, show a higher concentration of copper than do concentrates Nos. 1 and 2. The losses in the tailing are high, 4·980 pounds copper and 3·019 pounds nickel per ton of ore milled.

### *Test No. 4*

Size of feed −3 + 5 mesh. Weight = 1,902 pounds.
Head assay: Cu 1·01 per cent, Ni 2·38 per cent, S 17·00 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Concentrates Nos. 1 and 2......... | 54·4 | 0·98 | 3·17 | 23·71 | 53·8 | 73·7 | 76·4 |
| Concentrate No. 3................. | 9·4 | 1·13 | 2·52 | 18·34 | 10·7 | 10·1 | 10·2 |
| Concentrate No. 4................. | 14·0 | 1·38 | 1·98 | 9·05 | 19·5 | 11·9 | 7·5 |
| Tailing.......................... | 22·2 | 0·71 | 0·45 | 4·45 | 16·0 | 4·3 | 5·9 |
| Heads (calculated)................ | 100·0 | 0·99 | 2·34 | 16·88 | 100·0 | 100·0 | 100·0 |

These results again show the tendency of the copper to separate from the nickel and pass out in the tailing. An increase in the copper content and a decrease in the nickel of each succeeding concentrate is noted.

### *Test No. 5*

Size of feed −5+10 mesh. Weight = 1,514 pounds.
Head assay: Cu 0·83 per cent, Ni 2·07 per cent, S 17·60 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Concentrates Nos. 1 and 2......... | 61·4 | 0·93 | 2·82 | 20·65 | 61·9 | 93·4 | 81·9 |
| Concentrate No. 3................. | 3·9 | 1·24 | 1·23 | 11·04 | 5·2 | 2·6 | 2·8 |
| Concentrate No. 4................. | 4·2 | 1·03 | 1·79 | 13·68 | 4·7 | 4·0 | 3·7 |
| Tailing.......................... | 30·5 | 0·85 | Nil | 5·89 | 28·2 | 0·0 | 11·6 |
| Heads (calculated)................ | 100·0 | 0·92 | 1·85 | 15·48 | 100·0 | 100·0 | 100·0 |

This test indicates very definitely that on jigging this ore, the copper does not follow the heavier sulphides of nickel and iron into the concentrates but passes out in the tailing. The assay of 0·85 per cent copper in the discard is equivalent to a loss of 4·681 pounds per ton of ore milled.

## BUTCHART TABLE CONCENTRATION

### *Test No. 1*

Feed, unclassified –3 mesh. Weight = 2,000 pounds.

*Screen Analysis of Feeds*

| Mesh | Per cent |
|---|---|
| – 3+ 10 | 52·0 |
| – 10+ 20 | 15·2 |
| – 10+ 40 | 10·6 |
| – 40+ 65 | 6·1 |
| – 65+100 | 2·6 |
| –100 | 13·5 |
| | 100·0 |

Head assay: Cu 1·23 per cent, Ni 2·25 per cent, S 15·30 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Concentrate | 81·2 | 1·33 | 2·16 | 15·52 | 85·0 | 85·4 | 87·2 |
| Tailing | 18·8 | 1·01 | 1·59 | 9·86 | 15·0 | 14·6 | 12·8 |
| Heads (calculated) | 100·0 | 1·27 | 2·05 | 14·46 | 100·0 | 100·0 | 100·0 |

Tabling a product of this size does not effect a concentration of the copper or nickel. The concentrate is of virtually the same grade as the feed, and the tailing is high. On an unclassified feed, the slimed sulphides float off into the tailing.

### *Test No. 2*

Feed, unclassified –5 mesh. Weight = 1,852 pounds.

After passing over the Butchart table, a sample of tailing was reground and floated to determine what percentage of the metals not saved on the table could be recovered by this method.

*Screen Analysis of Table Feed*

| Mesh | Per cent |
|---|---|
| – 5+ 20 | 54·0 |
| – 20+ 40 | 16·5 |
| – 40+ 65 | 8·2 |
| – 65+100 | 4·5 |
| –100 | 16·0 |
| | 100·0 |

Head assay: Cu 1·18 per cent, Ni 1·77 per cent, S 14·40 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S per cent | Cu | Ni | S |
| Table concentrate | 76·7 | 1·15 | 2·16 | 15·49 | 78·6 | 83·5 | 85·3 |
| Table tailing | 23·3 | 1·03 | 1·41 | 8·78 | 21·4 | 16·5 | 14·7 |
| Heads (calculated) | 100·0 | 1·12 | 1·98 | 13·92 | 100·0 | 100·0 | 100·0 |

The table tailing from the above test was reground. Soda ash at the rate of 10 pounds per ton of feed was added to the grinding circuit. Two flotation concentrates, "A" and "B", were removed. The object aimed at in concentrate "A" was the production of a high copper, low nickel content, and in concentrate "B" the recovery of the remaining nickel.

*Reagents, pounds per ton of flotation feed—*

| Reagents | Grinding | "A" | "B" |
|---|---|---|---|
| Soda ash | 10·0 | | |
| Cresylic acid | | 0·08 | |
| Aerofloat No. 15 | | 0·04 | |
| Aerofloat No. 25 | | | 0·12 |
| Amyl xanthate | | | 0·10 |
| No. 323 | | | 0·10 |
| Pine oil | | | 0·16 |

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Concentrate "A" | 9·4 | 9·86 | 5·00 | 22·24 | 88·4 | 34·7 | 23·4 |
| Concentrate "B" | 26·2 | 0·24 | 2·92 | 24·59 | 6·0 | 56·7 | 72·3 |
| Flotation tailing | 64·4 | 0·09 | 0·18 | 0·59 | 5·6 | 8·6 | 4·3 |
| Heads (calculated) | 100·0 | 1·04 | 1·35 | 8·91 | 100·0 | 100·0 | 100·0 |

The combined table and flotation concentrations show the following:

Head assay: Cu 1·18 per cent, Ni 1·77 per cent, S 14·40 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Table concentrate | 76·7 | 1·15 | 2·16 | 15·49 | 78·6 | 83·5 | 85·3 |
| Flotation concentrate "A" | 2·2 | 9·86 | 5·00 | 22·24 | 18·9 | 5·7 | 3·4 |
| Flotation concentrate "B" | 6·1 | 0·24 | 2·92 | 24·59 | 1·3 | 9·4 | 10·6 |
| Flotation tailing | 15·0 | 0·09 | 0·18 | 0·59 | 1·2 | 1·4 | 0·7 |
| Heads (calculated) | 100·0 | 1·13 | 1·97 | 13·96 | 100·0 | 100·0 | 100·0 |

The results secured in the tabling of this −5 mesh material are similar to those of preceding tests. Copper passes into the tailing. Flotation recovers the greater part of this together with the nickel.

### *Test No. 3*

Feed, unclassified −7 mesh. Weight = 2,041 pounds.

*Screen Analysis of Table Feed*

| Mesh | Per cent |
|---|---|
| − 7+ 20 | 44·6 |
| − 20+ 40 | 19·2 |
| − 40+ 65 | 10·3 |
| − 65+100 | 6·0 |
| −100 | 19·9 |
| | 100·0 |

Head assay: Cu 1·16 per cent, Ni 2·11 per cent, S 16·40 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Table concentrate | 66·2 | 1·16 | 2·50 | 20·60 | 72·1 | 81·9 | 83·2 |
| Table tailing | 33·8 | 0·88 | 1·08 | 8·14 | 27·9 | 18·1 | 16·8 |
| Heads (calculated) | 100·0 | 1·06 | 2·02 | 16·39 | 100·0 | 100·0 | 100·0 |

After tabling, the tailing was reground and floated as in the preceding test.

*Reagents, pounds per ton of flotation feed—*

| Reagents | Grinding | "A" | "B" |
|---|---|---|---|
| Soda ash | 4·5 | | |
| Cresylic acid | | 0·04 | |
| Aerofloat No. 25 | | 0·06 | 0·06 |
| Amyl xanthate | | | 0·05 |
| No. 323 | | | 0·05 |
| Pine oil | | | 0·06 |

*Screen Analysis of Flotation Feed*

| Mesh | Per cent |
|---|---|
| − 65+100 | 0·4 |
| −100+150 | 1·3 |
| −150+200 | 5·0 |
| −200 | 93·3 |
| | 100·0 |

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Flotation concentrate "A" | 19·6 | 2·51 | 2·98 | 21·42 | 88·6 | 52·8 | 52·5 |
| Flotation concentrate "B" | 14·7 | 0·25 | 2·44 | 24·51 | 6·6 | 32·4 | 45·1 |
| Tailing | 65·7 | 0·04 | 0·25 | 0·29 | 4·8 | 14·8 | 2·4 |
| Heads (calculated) | 100·0 | 0·55 | 1·10 | 7·99 | 100·0 | 100·0 | 100·0 |

Calculating the flotation results in terms of original table feed shows:—

Table feed: Cu 1·16 per cent, Ni 2·11 per cent, S 16·40 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Table concentrate | 66·2 | 1·16 | 2·50 | 20·60 | 72·1 | 81·9 | 83·2 |
| Flotation concentrate "A" | 6·6 | 2·51 | 2·98 | 21·42 | 24·7 | 9·5 | 8·8 |
| Flotation concentrate "B" | 5·0 | 0·25 | 2·44 | 24·51 | 1·8 | 5·9 | 7·6 |
| Flotation tailing | 22·2 | 0·04 | 0·25 | 0·29 | 1·4 | 2·7 | 0·4 |
| Heads (calculated) | 100·0 | 0·95 | 2·03 | 16·34 | 100·0 | 100·0 | 100·0 |

These results indicate that tabling alone results in high losses. Flotation of the table tailing results in the recovery of most of these values.

## *Test No. 4*

Feed, unclassified—10 mesh. Weight = 1,627 pounds.

*Screen Analysis of Butchart Table Feed*

| Mesh | Per cent |
|---|---|
| – 10+ 20 | 10·2 |
| – 20+ 40 | 23·2 |
| – 40+ 65 | 15·4 |
| – 65+100 | 10·4 |
| –100 | 40·8 |
| | 100·0 |

Head assay: Cu 1·15 per cent, Ni 2·66 per cent, S 19·66 per cent.

| Product | Weight, per cent | Assay | | | Per cent values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Table concentrate | 57·8 | 1·25 | 3·69 | 29·44 | 62·7 | 82·5 | 81·1 |
| Table tailing | 42·2 | 1·02 | 1·07 | 9·38 | 37·3 | 17·5 | 18·9 |
| Heads (calculated) | 100·0 | 1·15 | 2·58 | 20·97 | 100·0 | 100·0 | 100·0 |

The table tailings were floated using the following reagents:—

*Reagents, pounds per ton of flotation feed—*

| Reagents | Grinding | "A" | "B" |
|---|---|---|---|
| Soda ash | 5·5 | | |
| Cresylic acid | | 0·10 | |
| Cyanide | | 0·22 | |
| No. 323 | | | 0·05 |
| Amyl xanthate | | | 0·05 |
| Aerofloat No. 25 | | | 0·13 |
| Pine oil | | | 0·11 |

*Screen Analysis of Flotation Feed*

| Mesh | Per cent |
|---|---|
| −100+150 | 0·6 |
| −150+200 | 3·5 |
| −200 | 95·9 |
| | 100·0 |

| Product | Weight, per cent | Assay | | | Per cent values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Flotation concentrate "A" | 5·5 | 14·44 | 1·41 | 18·70 | 70·0 | 4·4 | 10·4 |
| Flotation concentrate "B" | 31·5 | 0·80 | 4·25 | 23·40 | 22·2 | 75·7 | 74·8 |
| Flotation tailing | 63·0 | 0·14 | 0·56 | 2·30 | 7·8 | 19·9 | 14·8 |
| Heads (calculated) | 100·0 | 1·13 | 1·77 | 9·85 | 100·0 | 100·0 | 100·0 |

Combining the results secured by table concentration and flotation calculated on the table feed above shows the following:—

Head assay: Cu 1·15 per cent, Ni 2·66 per cent, S 19·66 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Table concentrate | 57·8 | 1·25 | 3·69 | 29·44 | 62·7 | 82·5 | 81·1 |
| Flotation concentrate "A" | 2·3 | 14·44 | 1·41 | 18·70 | 26·1 | 0·8 | 2·0 |
| Flotation concentrate "B" | 13·3 | 0·80 | 4·25 | 23·40 | 8·3 | 13·2 | 14·1 |
| Flotation tailing | 26·6 | 0·14 | 0·56 | 2·30 | 2·9 | 3·5 | 2·8 |
| Heads (calculated) | 100·0 | 1·20 | 2·88 | 21·17 | 100·0 | 100·0 | 2100·0 |

The results obtained by tabling the ore at −10 mesh are very similar to those of preceding tests. High losses of the copper and nickel result. These are recovered by regrinding and floating the table tailing.

## PLATO TABLE CONCENTRATION

### *Test No. 1*

Feed, unclassified—3 mesh. Weight = 1,887 pounds.

*Screen Analysis of Feed*

| Mesh | Per cent |
|---|---|
| − 3+ 10 | 48·0 |
| − 10+ 20 | 13·8 |
| − 20+ 40 | 12·0 |
| − 40+ 65 | 7·1 |
| − 65+100 | 4·2 |
| −100 | 14·9 |
| | 100·0 |

Head assay: Cu 1·29 per cent, Ni 2·37 per cent, S 16·53 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Table concentrate | 81·7 | 1·19 | 2·31 | 17·79 | 82·0 | 87·6 | 89·4 |
| Table tailing | 18·3 | 1·17 | 1·46 | 9·39 | 18·0 | 12·4 | 10·6 |
| Heads (calculated) | 100·0 | 1·19 | 2·15 | 16·25 | 100·0 | 100·0 | 100·0 |

As in previous table concentration tests, high losses of nickel and copper result from concentration of this size of material.

### *Test No. 2*

Feed, unclassified—5 mesh. Weight = 1,848 pounds.

*Screen Analysis of Feed*

| Mesh | Per cent |
|---|---|
| − 5+ 10 | 22·8 |
| − 10+ 20 | 21·0 |
| − 20+ 40 | 17·1 |
| − 40+ 65 | 9·9 |
| − 65+100 | 6·2 |
| −100 | 23·0 |
| | 100·0 |

Head assay: Cu 1·26 per cent, Ni 2·38 per cent, S 16·96 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Table concentrate | 72·6 | 1·24 | 2·64 | 17·23 | 75·8 | 89·7 | 86·2 |
| Table tailing | 27·4 | 1·05 | 0·80 | 7·32 | 24·2 | 10·3 | 13·8 |
| Heads (calculated) | 100·0 | 1·19 | 2·13 | 14·52 | 100·0 | 100·0 | 100·0 |

A slightly higher grade of concentrate is produced by tabling material at this mesh. The copper with considerable nickel, however, passes into the tailing.

### *Test No. 3*

Feed, unclassified—7 mesh. Weight = 1,948 pounds.

*Screen Analysis of Feed*

| Mesh | Per cent |
|---|---|
| − 7+ 10 | 11·6 |
| − 10+ 20 | 29·4 |
| − 20+ 40 | 20·0 |
| − 40+ 65 | 10·6 |
| − 65+100 | 6·3 |
| −100 | 22·1 |
| | 100·0 |

Head assay: Cu 1·12 per cent, Ni 2·50 per cent, S 17·07 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Table concentrate | 70·9 | 1·11 | 2·72 | 21·25 | 56·9 | 85·2 | 87·3 |
| Table tailing | 29·1 | 2·05 | 1·15 | 7·50 | 43·1 | 14·8 | 12·7 |
| Heads (calculated) | 100·0 | 1·38 | 2·26 | 17·25 | 100·0 | 100·0 | 100·0 |

The results secured by tabling material crushed to pass −7 mesh are no improvement over the preceding one.

## *Test No. 4*

Feed, unclassified −10 mesh. Weight = 1,570 pounds.

*Screen Analysis of Feed*

| Mesh | Per cent |
|---|---|
| − 10+ 20 | 9·2 |
| − 20+ 40 | 23·4 |
| − 40+ 65 | 16·4 |
| − 65+100 | 10·6 |
| −100 | 40·4 |
| | 100·0 |

Head assay: Cu 1·15 per cent, Ni 3·79 per cent, S 19·40 per cent.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Table concentrate | 63·4 | 1·42 | 3·44 | 25·33 | 66·7 | 74·8 | 82·6 |
| Table tailing | 36·6 | 1·23 | 2·01 | 9·27 | 33·3 | 25·2 | 17·4 |
| Heads (calculated) | 100·0 | 1·35 | 2·92 | 19·45 | 100·0 | 100·0 | 100·0 |

High losses again result on tabling at this size. As in all previous tests, the fine sulphides are carried off in the tailing.

## FLOTATION

The following small-scale flotation tests were run on 1,000-gramme portions of the ore ground in a porcelain mill containing iron balls.

A screen analysis of the ore after grinding showed:—

| Mesh | Per cent |
|---|---|
| − 45+ 65 | 0·2 |
| − 65+100 | 2·4 |
| −100+150 | 5·0 |
| −150+200 | 17·1 |
| −200 | 75·3 |
| | 100·0 |

## *Test No. 1*

The ore was ground in water together with lime equal to 2·0 pounds per ton of ore.

Two concentrates were made, the first a copper product which was afterwards cleaned, and then by the addition of further reagents, a nickel concentrate.

*Reagents, pounds per ton of ore—*

| Reagents | Grinding | Copper | Nickel |
|---|---|---|---|
| Lime | 2·0 | | |
| Aerofloat No. 25 | | 0·10 | 0·20 |
| Copper sulphate | | | 1·00 |
| No. 208 | | | 0·10 |

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Heads (calculated) | 100·0 | 2·03 | 2·07 | 16·65 | 100·0 | 100·0 | 100·0 |
| Copper concentrate | 6·6 | 21·0 | 9·04 | 32·98 | 69·2 | 29·5 | 13·2 |
| Copper middling | 5·1 | 4·64 | 6·35 | 26·37 | 11·8 | 16·0 | 8·1 |
| Nickel concentrate | 20·4 | 1·64 | 4·06 | 30·38 | 16·6 | 40·8 | 37·3 |
| Tailing | 67·9 | 0·07 | 0·41 | 10·15 | 2·4 | 13·7 | 41·4 |

The combined products show that in a bulk concentrate 97·6 per cent of the copper and 86·3 per cent of the nickel would be recovered in a product assaying 6·11 per cent copper and 5·59 per cent nickel.

## *Test No. 2*

In this test a bulk flotation concentrate was made from ore ground in a natural circuit, no additions of alkaline reagents being made.

*Reagents, pounds per ton of ore—*

| | |
|---|---|
| Aerofloat No. 25 | 0·20 |
| Potassium xanthate | 0·10 |

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Heads | 100·0 | 1·90 | 2·18 | 16·58 | 100·0 | 100·0 | 100·0 |
| Concentrate | 33·5 | 5·24 | 5·59 | 30·17 | 92·3 | 85·7 | 61·0 |
| Tailing | 66·5 | 0·22 | 0·47 | 9·72 | 7·7 | 14·3 | 39·0 |

These results indicate that 66·5 per cent of the total tonnage can be eliminated with a loss of 7·7 per cent of the total copper and 14·3 per cent of the nickel.

### *Test No. 3*

In this test, the ore was ground in a circuit alkaline with lime. A copper concentrate was removed by flotation and the tailing from this operation passed over a Wilfley table.

*Reagents, pounds per ton of ore—*

| Reagents | Grinding | Flotation |
|---|---|---|
| Lime | 2·0 | ........... |
| Aerofloat No. 25 | ........... | 0·10 |

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Heads (calculated) | 100·0 | 1·54 | 2·13 | 17·14 | 100·0 | 100·0 | 100·0 |
| Flotation concentrate | 15·2 | 9·34 | 7·95 | 30·48 | 92·2 | 56·7 | 27·0 |
| Table concentrate | 44·2 | 0·20 | 1·57 | 25·71 | 5·7 | 32·6 | 66·4 |
| Tailing | 40·6 | 0·08 | 0·56 | 2·78 | 2·1 | 10·7 | 6·6 |

Combining the two concentrates from this test gives a product assaying 2·53 per cent copper and 3·20 per cent nickel representing a recovery of 97·9 per cent of the copper and 89·3 per cent of the nickel.

Forty per cent of the total weight of ore milled can be eliminated with a loss of 2·1 per cent of the copper and 10·7 per cent of the nickel.

### *Test No. 4*

Two flotation concentrates were made in this test to note the effects of the reagents used.

*Reagents, pounds per ton of ore—*

| Reagents | Grinding | Flotation | |
|---|---|---|---|
| | | Copper | Nickel |
| Soda ash | 4·0 | ........... | ........... |
| Cresylic acid | ........... | 0·08 | 0·40 |
| Coal-tar creosote | ........... | ........... | 0·20 |
| Acid creosote | ........... | ........... | 0·16 |

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | Cu | Ni |
| Heads (calculated) | 100·0 | 1·74 | 1·98 | 100·0 | 100·0 |
| Copper concentrate | 7·4 | 20·72 | 6·88 | 88·1 | 25·7 |
| Nickel concentrate | 17·6 | 1·06 | 6·25 | 10·6 | 55·4 |
| Tailing | 75·0 | 0·03 | 0·50 | 1·3 | 18·9 |

Combining the two products gives a bulk concentrate assaying 6·99 per cent copper and 6·44 per cent nickel with corresponding recoveries of 98·7 and 81·1 per cent.

The reagents used in the flotation of nickel produced too voluminous a froth, not feasible for mill operation.

### *Test No. 5*

In this test, a study of the effects of the American Cyanamid Reagents Nos. 404, 323, and 303* was made.

A copper concentrate was first removed and then portions of the pulp were floated using these reagents.

*Reagents, pounds per ton of ore—*

| | |
|---|---|
| Grinding circuit | 4·0 soda ash |
| Copper circuit | 0·04 cresylic acid |
| "A" No. 404 | 0·20 |
| Cresylic acid | 0·0 |
| "B" No. 323 | 0·20 |
| Cresylic acid | 0·20 |
| "C" No. 303 | 0·20 |
| Cresylic acid | 0·20 |

"A"

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | Cu | Ni |
| Copper concentrate | 6·1 | 22·50 | 4·71 | 76·2 | 14·3 |
| Nickel concentrate | 40·7 | 1·00 | 4·02 | 22·6 | 80·9 |
| Tailing | 53·2 | 0·04 | 0·18 | 1·2 | 4·8 |

"B"

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | Cu | Ni |
| Copper concentrate | 6·1 | 22·50 | 4·71 | 75·1 | 14·0 |
| Nickel concentrate | 44·6 | 1·00 | 3·74 | 24·1 | 81·6 |
| Tailing | 49·3 | 0·07 | 0·18 | 1·8 | 4·4 |

---

*American Cyanamid No. 303 is potassium ethyl xanthate.

"C"

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | Cu | Ni |
| Copper concentrate | 6·1 | 22·50 | 4·71 | 75·5 | 14·1 |
| Nickel concentrate | 33·0 | 1·20 | 4·77 | 21·8 | 76·5 |
| Tailing | 60·9 | 0·08 | 0·32 | 2·7 | 9·4 |

The rougher copper concentrate is raised by cleaning to 28·60 per cent copper and 3·43 per cent nickel with a reduction in bulk of 26·6 per cent. The middling produced by this operation assays 4·96 per cent copper and 8·39 per cent nickel. This indicates that the nickel in the rougher copper concentrate on cleaning will pass into the nickel circuit, thus raising the percentage recovery of that metal.

A comparison of the three tests "A," "B," and "C," shows no marked difference between "A" and "B". Recoveries of 99 per cent of the copper and 95 per cent of the nickel are indicated in the combined copper and nickel concentrates. Test "C" shows lower recoveries, with a higher grade nickel concentrate of less bulk than either "A" or "B."

### *Test No. 6*

The effect of potassium dichromate was investigated. The nickel concentrate produced was reground and cleaned using cyanide and a further addition of dichromate.

*Reagents, pounds per ton of ore*—

| Reagents | Grinding | Flotation | |
|---|---|---|---|
| | | Copper | Nickel |
| Soda ash | 4·0 | | |
| Cresylic acid | | 0·04 | 0·16 |
| Potassium dichromate | | 0·25 | 0·20 |
| No. 323 | | | 0·20 |
| Aerofloat No. 25 | | | 0·08 |
| Sodium cyanide | | | 0·20 |

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | Cu | Ni |
| Heads (calculated) | 100·0 | 1·72 | 1·99 | 100·0 | 100·0 |
| Copper concentrate | 5·4 | 25·46 | 3·98 | 80·5 | 10·7 |
| Nickel concentrate | 18·7 | 1·32 | 6·76 | 14·5 | 63·2 |
| Nickel middling | 27·6 | 0·22 | 1·58 | 3·6 | 21·9 |
| Tailing | 48·3 | 0·05 | 0·16 | 1·4 | 3·9 |

As a depressant of nickel, potassium dichromate appears to be beneficial in the copper flotation without affecting copper recovery.

### *Test No. 7*

In this test, cyanide was added both to the grinding circuit and to the pulp after removal of a copper concentrate.

*Reagents, pounds per ton of ore—*

| Reagents | Grinding | Flotation | |
|---|---|---|---|
| | | Copper | Nickel |
| Soda ash | 4·0 | | |
| Cyanide | 0·10 | | 0·20 |
| Cresylic acid | | 0·04 | |
| Copper sulphate | | | 0·20 |
| Aerofloat No. 25 | | | 0·04 |

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | Cu | Ni |
| Heads (calculated) | 100·0 | 1·78 | 2·13 | 100·0 | 100·0 |
| Copper concentrate | 5·3 | 25·14 | 2·70 | 74·8 | 6·7 |
| Nickel concentrate | 15·3 | 2·12 | 5·26 | 18·1 | 37·7 |
| Tailing | 79·4 | 0·16 | 1·49 | 7·1 | 55·6 |

Cyanide has the effect of reducing the nickel content in the copper concentrate, but due to its depressing action on all sulphides, lower recoveries of both metals result.

## SUMMARY AND CONCLUSIONS

The results secured in the various jig and table concentration tests are tabulated below.

Average head sample: Cu 1·11 per cent, Ni 2·28 per cent.

| Test | Weight, per cent | Concentrate | | | | Tailing | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | Assay | | Per cent of values | | Assay | | Per cent of values | |
| | | Cu, per cent | Ni, per cent | Cu | Ni | Cu, per cent | Ni, per cent | Cu | Ni |
| Jigging −1″+ ¾″......Mesh | 91·9 | 1·17 | 2·23 | 95·1 | 98·7 | 0·70 | 0·34 | 4·9 | 1·3 |
| " −¾ + ½...... " | 66·0 | 1·06 | 3·05 | 63·3 | 89·3 | 1·20 | 0·71 | 36·7 | 10·7 |
| " −½ + 3...... " | 75·5 | 0·95 | 2·60 | 75·1 | 93·2 | 0·97 | 0·58 | 24·9 | 6·8 |
| " −3 + 5...... " | 77·8 | 1·07 | 2·88 | 84·0 | 95·7 | 0·71 | 0·45 | 16·0 | 4·3 |
| " −5 +10...... " | 69·5 | 0·95 | 2·67 | 71·8 | 100·0 | 0·85 | Nil | 28·2 | Nil |
| Butchart table − 3. " | 81·2 | 1·33 | 2·16 | 85·0 | 85·4 | 1·01 | 1·59 | 15·0 | 14·6 |
| " " − 5. " | 76·7 | 1·15 | 2·16 | 78·6 | 83·5 | 1·03 | 1·41 | 21·4 | 16·5 |
| " " − 7. " | 66·2 | 1·16 | 2·50 | 72·1 | 81·9 | 0·88 | 1·08 | 27·9 | 18·1 |
| " " −10. " | 57·8 | 1·25 | 3·69 | 62·7 | 82·5 | 1·02 | 1·07 | 37·3 | 17·5 |
| Plato table − 3..... " | 81·7 | 1·19 | 2·31 | 82·0 | 87·6 | 1·17 | 1·46 | 18·0 | 12·4 |
| " − 5..... " | 72·6 | 1·24 | 2·64 | 75·8 | 89·7 | 1·05 | 0·80 | 24·2 | 10·3 |
| " − 7..... " | 70·9 | 1·11 | 2·72 | 56·9 | 85·2 | 2·05 | 1·15 | 43·1 | 14·8 |
| " −10..... " | 63·4 | 1·42 | 3·44 | 66·7 | 74·8 | 1·23 | 2·01 | 33·3 | 25·2 |

It appears that there is very little increase in the grade of concentrates produced over that of the average head sample.

A certain amount of gangue material has been eliminated but this is accompanied with a loss of much of the copper and considerable nickel. Apparently the heavier sulphides of iron crowd out the copper minerals, forcing them into the tailing.

Flotation of the table tailing results in the saving of these values.

Fine grinding of the raw ore followed by flotation gives good recoveries of copper and nickel.

Jig or table concentration, unless followed by flotation, results in too great a loss of the valuable constituents and as a method for the economic elimination of worthless gangue is not recommended.

## Report No. 356

### CONCENTRATION TESTS ON ORE SAMPLE FROM SUDBURY OFFSETS, LTD., LEVACK, ONTARIO

**G. B. Walker**

*Shipment.* A 100-pound sample of diamond drill cores was received from Sudbury Offsets, Limited, by Mr. C. T. Young, 1004 Bank of Hamilton Bldg., Toronto, Ontario, on March 18, 1930. It was stated that these samples were taken from drilling operations on the Foy Offset.

*Characteristics of the Ore.* The ore may be described as having a basic norite gangue in which are found occasional feldspar crystals. The sulphide mineralization consists of chalcopyrite and pyrrhotite intimately mixed. No nickeliferous minerals are evident. What free pentlandite is present is too fine to be identified. The bulk of the nickel is probably very closely associated with the pyrrhotite.

*Purpose of Experimental Tests.* The purpose of this investigation was to determine the amenability of the ore sample submitted to concentration by flotation or by tabling and flotation.

*Sampling and Analysis.* The shipment was stage crushed to −10 mesh. It was then thoroughly mixed and a head sample was cut out which yielded the following analysis:

*Head Analysis*

| | Per cent |
|---|---|
| Copper | 0·78 |
| Nickel | 2·43 |
| Sulphur | 26·10 |
| Iron | 24·21 |

### EXPERIMENTAL TESTS

The following procedure was employed in making flotation tests. A 1,000-gramme sample of ore was ground in a laboratory mill with 600 c.c. of water for a 30-minute period. The resultant pulp was transferred to a Ruth flotation machine where additional water was added to reduce the density of the pulp to approximately 22 per cent solids. The reagents which were added to the ball mill and to the flotation machine are indicated in the following tests. Froth was removed from the flotation machine during the periods indicated.

## TABLING AND FLOTATION

### *Test No. 1*

Five thousand grammes of dry 10 mesh ore was tabled for the production of a high sulphide concentrate. The table tailings were ground in a ball mill with 4·0 pounds per ton $Na_2CO_3$ for 45 minutes. The resultant pulp was floated with 0·10 pound per ton potassium xanthate and 0·10 pound per ton of Aerofloat No. 25 for 8 minutes. High recovery and low ratio of concentration are indicated.

| | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Table concentrate | 71·1 | 0·73 | 2·86 | 30·7 | 67·6 | 81·0 | 85·9 |
| Table tailing | 28·9 | 0·86 | 1·65 | 12·4 | 32·4 | 19·0 | 14·1 |
| Flotation concentrate | 12·4 | 1·96 | 3·65 | 27·7 | 31·8 | 18·1 | 13·5 |
| Flotation tailing | 16·5 | 0·03 | 0·15 | 0·93 | 0·6 | 0·9 | 0·6 |
| Head | 100·0 | 0·77 | 2·51 | 25·4 | 100·0 | 100·0 | 100·0 |

Screen analyses of the table concentrate and flotation tailing were as follows:

| Table concentrate | | Flotation tailing | |
|---|---|---|---|
| Mesh | Percentage held | Mesh | Percentage held |
| 10 | ........... | 65 | ........... |
| 14 | 6·6 | 100 | 1·3 |
| 28 | 23·6 | 150 | 10·5 |
| 35 | 17·0 | 200 | 18·0 |
| 48 | 13·9 | −200 | 70·2 |
| 65 | 11·7 | | 100·0 |
| 100 | 10·3 | | |
| −100 | 16·9 | | |
| | 100·0 | | |

### *Test No. 2*

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Flotation concentrate | 44·0 | 1·73 | 4·10 | 30·5 | 97·9 | 76·5 | 56·2 |
| Flotation tailing | 56·0 | 20·03 | 0·99 | 19·8 | 2·1 | 23·5 | 43·8 |
| Head | 100·0 | 0·78 | 2·36 | 23·9 | 100·0 | 100·0 | 100·0 |

*Reagents*—

| To ball mill— | | To cells— | |
|---|---|---|---|
| $Na_2CO_3$ | 4·0 lb./ton | Potassium xanthate | 0·10 lb./ton |
| | | Pine oil | 0·08 " |
| pH | 7·8 | | |

A screen analysis of the flotation tailing is as follows:

| Mesh | Percentage held | Cumulative, per cent |
|---|---|---|
| 65 | | |
| 100 | 1·3 | 1·3 |
| 150 | 5·9 | 7·2 |
| 200 | 10·6 | 17·8 |
| −200 | 82·2 | 100·0 |

### *Test No. 3*

| Products | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Flotation concentrate | 48·0 | 1·60 | 3·75 | 29·9 | 97·4 | 77·3 | 60·2 |
| Flotation tailing | 52·0 | 0·04 | 1·02 | 18·1 | 2·6 | 22·7 | 39·8 |
| Table concentrate | 28·3 | 0·02 | 1·45 | 30·3 | 0·6 | 17·6 | 36·2 |
| Table tailing | 23·7 | 0·07 | 0·50 | 3·6 | 2·0 | 5·1 | 3·6 |
| Head | 100·0 | 0·79 | 2·33 | 23·8 | 100·0 | 100·0 | 100·0 |

*Reagents*—

| | | | |
|---|---|---|---|
| To ball mill— | | To cells— | |
| $Na_2CO_3$ | 4·0 lb./ton | Aerofloat No. 25 | 0·14 lb./ton |
| pH | 7·8 | | |

Flotation tails were thickened and passed over a table. High recovery is indicated but low ratio of concentration.

### *Test No. 4*

| Products | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| "A" concentrate | 16·8 | 3·90 | 7·25 | 33·7 | 92·8 | 53·2 | 22·8 |
| "B" concentrate | 58·1 | 0·06 | 1·75 | 32·4 | 4·8 | 44·4 | 75·7 |
| Tailing | 25·1 | 0·07 | 0·22 | 1·6 | 2·4 | 2·4 | 1·5 |
| Head | 100·0 | 0·71 | 2·30 | 24·8 | 100·0 | 100·0 | 100·0 |

*Reagents*—

| | | | | | |
|---|---|---|---|---|---|
| To ball mill— | | To cells—<br>"A" concentrate | | "B" concentrate | |
| CaO | 1·0 lb./ton | No. 323 | 0·10 lb./ton | $H_2SO_4$ | 7·4 lb./ton |
| | | Pine oil | 0·08 " | Xanthate | 2·20 " |
| | | | | No. 323 | 0·20 " |
| pH | 9·4 | | | Pine oil | 0·12 " |

### *Test No. 5*

| Products | Weight, per cent | Assay |  |  | Per cent of values |  |  |
|---|---|---|---|---|---|---|---|
|  |  | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| "A" concentrate | 13·9 | 5·48 | 8·92 | 31·2 | 96·7 | 51·1 | 16·6 |
| "B" concentrate | 50·6 | 0·03 | 2·05 | 35·4 | 1·9 | 42·7 | 68·5 |
| Tailing | 35·5 | 0·03 | 0·43 | 11·0 | 1·4 | 6·2 | 14·9 |
| Head | 100·0 | 0·79 | 2·43 | 26·2 | 100·0 | 100·0 | 100·0 |

*Reagents*—

To ball mill—
CaO........ 6·0 lb./ton
pH.......... 9·4

To cells—
"A" concentrate
No. 323.......0·10 lb./ton
Cresylic acid........0·08 "

"B" concentrate
$CuSO_4$........ 2·0 lb./ton
No. 323....... 0·10 "
Aerofloat No. 25...... 0·04 "
Cresylic acid. 0·04 "

High lime appears to be very beneficial in producing a high-grade copper-nickel concentrate.

### *Test No. 6*

| Products | Weight, per cent | Assay |  |  | Per cent of values |  |  |
|---|---|---|---|---|---|---|---|
|  |  | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| "A" concentrate | 7·0 | 9·94 | 9·70 | 29·2 | 95·9 | 27·9 | 7·9 |
| "B" concentrate | 20·7 | 0·05 | 4·13 | 31·2 | 1·4 | 35·1 | 24·9 |
| Tailing | 72·3 | 0·03 | 1·25 | 24·2 | 2·7 | 37·0 | 67·2 |
| Head | 100·0 | 0·77 | 2·44 | 26·0 | 100·0 | 100·0 | 100·0 |

*Reagents*—

To ball mill—
CaO........ 8·0 lb./ton
pH.........10·2

To cells—
"A" concentrate
No. 323.....0·10 lb./ton
Cresylic acid.......0·08 "

"B" concentrate
Thiocarbanilide.........0·20 lb./ton
Cresylic acid........0·08 "

Increased lime has resulted in a high-grade concentrate and partial nickel rejection. Thiocarbanilide does not appear effective when added to the second float.

### *Test No. 7—Effect of Aerofloat No. 25*

| Products | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Flotation concentrate | 9·2 | 8·16 | 6·30 | 26·9 | 95·4 | 23·8 | 9·4 |
| Tailing | 90·8 | 0·04 | 2·06 | 26·5 | 4·6 | 76·2 | 90·6 |
| Head | 100·0 | 0·79 | 2·45 | 26·5 | 100·0 | 100·0 | 100·0 |

*Reagents*—

| To ball mill— | | To cells— |
|---|---|---|
| CaO | 4·0 lb./ton | No reagents. |
| Aerofloat No. 25 | 0·10 " | |
| pH | 8·8 | |

### *Test No. 8—Effect of Thiocarbanilide*

| Products | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Flotation concentrate | 10·9 | 7·00 | 7·80 | 25·7 | 96·6 | 33·7 | 9·9 |
| Tailing | 89·1 | 0·03 | 1·88 | 28·6 | 3·4 | 66·3 | 90·1 |
| Head | 100·0 | 0·79 | 2·55 | 28·5 | 100·0 | 100·0 | 100·0 |

*Reagents*—

| To ball mill— | | To cells— | |
|---|---|---|---|
| CaO | 4·0 lb./ton | Cresylic acid | 0·08 lb./ton |
| Thiocarbanilide | 0·10 " | | |
| pH | 8·8 | | |

Flotation tailings from Tests Nos. 10 and 11 were combined, roughly classified, and fed to a table. A high-grade concentrate was attempted. The following results were obtained.

| Products | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Table concentrate | 46·8 | 0·03 | 2·33 | 33·2 | 24·8 | 57·0 | 60·8 |
| Table tailing | 53·2 | 0·08 | 1·55 | 18·8 | 75·2 | 43·0 | 39·2 |
| Head | 100·0 | 0·05 | 1·91 | 25·5 | 100·0 | 100·0 | 100·0 |

Nickel and copper recovery, as well as ratio of concentration, were poor in this operation.

## *Test No. 9—Cyanide to Flotation*

| Products | Weight, per cent | Assay |  |  | Per cent of values |  |  |
|---|---|---|---|---|---|---|---|
|  |  | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| "A" concentrate | 3·8 | 17·20 | 7·65 | 29·3 | 77·0 | 11·8 | 4·4 |
| "B" concentrate | 57·4 | 0·25 | 3·40 | 33·6 | 16·8 | 78·6 | 75·1 |
| Tailing | 38·8 | 0·14 | 0·62 | 13·6 | 6·2 | 9·6 | 20·5 |
| Head | 100·0 | 0·86 | 2·49 | 25·7 | 100·0 | 100·0 | 100·0 |

*Reagents—*

| To ball mill— |  | To cells—<br>"A" concentrate |  | "B" concentrate |  |
|---|---|---|---|---|---|
| CaO | 8·0 lb./ton | NaCN | 0·05 lb./ton | CuSO$_4$ | 4·0 lb./ton |
|  |  | No. 323 | 0·20 " | Amyl xanthate | 0·10 " |
| pH | 10·2 | Cresylic acid | 0·08 " | Pine oil | 0·08 " |

Cyanide, when carefully controlled, has resulted in a substantial increase in the grade of the "A" concentrate, and has also depressed the nickel.

## *Test No. 10—Cyanide to Grinding*

| Products | Weight, per cent | Assay |  |  | Per cent of values |  |  |
|---|---|---|---|---|---|---|---|
|  |  | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| "A" concentrate | 4·2 | 15·16 | 9·92 | 30·9 | 83·0 | 17·2 | 5·0 |
| "B" concentrate | 6·3 | 1·74 | 8·67 | 30·2 | 14·1 | 22·3 | 7·3 |
| "C" concentrate | 62·3 | 0·02 | 2·25 | 35·6 | 1·5 | 56·7 | 84·6 |
| Tailing | 27·2 | 0·04 | 0·35 | 3·0 | 1·4 | 3·8 | 3·1 |
| Head | 100·0 | 0·78 | 2·47 | 26·3 | 100·0 | 100·0 | 100·0 |

| To ball mill— |  | To cells—<br>"A" concentrate |  | "B" concentrate |  | "C" concentrate |  |
|---|---|---|---|---|---|---|---|
| CaO | 6·0 lb./ton | No. 323 | 0·10 lb./ton | Aerofloat No. 25 | 0·06 lb./ton | Na$_2$S | 4·0 lb./ton |
| NaCN | 0·05 " | Cresylic | 0·08 " |  |  | No. 323 | 0·10 " |
|  |  |  |  |  |  | Pine oil No. 5 | 0·08 " |

The addition of the cyanide to the grinding circuit has had less tendency to decrease the copper recovery. A larger amount of nickel was also floated.

### SUMMARY

Tabling followed by flotation has not effected any marked degree of concentration.

Flotation followed by tabling has resulted in high recovery of the values but has also resulted in low-grade concentrates.

Flotation in a circuit made alkaline with soda ash has not produced high-grade concentrates.

The use of lime as a source of alkalinity has effected relatively high-grade concentrates with fair recovery.

Cyanide appears to have a very marked action on this ore. Its use has resulted in partial depression of the nickel and a tendency to retard copper flotation if excessive amounts are used.

Nickel-iron flotation can be effected by the use of sodium sulphide, copper sulphate, or sulphuric acid. A powerful promoting reagent having water-soluble characteristics, such as potassium xanthate or Reagent No. 323, must be used. The latter has been found most effective on this ore.

### CONCLUSIONS

A wide variety of products containing various percentages of copper and nickel can be made from the ore sample submitted by Sudbury Offsets, Limited.

No recommendations as to the best metallurgical practice can be made without reference to the payments and penalties of a custom copper-nickel smelter schedule.

## Report No. 357

### FLOTATION TESTS ON ORE SAMPLE FROM DENISON COPPER MINES, LTD., WORTHINGTON, ONTARIO

**G. B. Walker**

*Shipments.* The first shipment of 2,000 pounds of ore was received from the Denison Copper Mines, Ltd., Worthington, Ontario, on Nov. 21, 1929. Subsequent examination of the ore indicated that it was badly oxidized and hence unfit for flotation testing.

In the latter part of February Mr. John A. Lindsay, 3 Baker Avenue, Toronto, Ontario, delivered about 50 pounds of diamond drill cores which had been cut during development of the company's property in Denison township, Sudbury district.

*Characteristics of the Ore.* The sample of drill cores was clean and unoxidized. The gangue mineral is a basic norite. Chalcopyrite and pyrrhotite, intimately mixed, comprise the sulphide mineralization. No pentlandite or other nickel-bearing mineral was evident. It is likely that most of the nickel is disseminated through the pyrrhotite. Part of the nickel is said to be present in the form of gersdorffite, $NiS_2$, $NiAs_2$.

*Purpose of Experimental Tests.* It was required to determine the amenability of this ore to selective flotation.

It has been stated that a product high in copper and relatively low in nickel could be marketed. A product containing equal amounts of copper and nickel could be marketed if the grade were reasonably high. It was also stated that a market was available for a nickel-iron concentrate if it was low in copper.

*Sampling and Analysis.* The entire lot of ore was stage crushed to −14 mesh. It was then carefully mixed and a head sample was cut out for analysis.

| | |
|---|---|
| Copper | 1·27 per cent |
| Nickel | 1·15 " |
| Sulphur | 5·73 " |

## EXPERIMENTAL TESTS

The procedure was as follows: 1,000 grammes of $-14$ mesh ore was placed in a laboratory mill with 600 c.c. of water and such reagents as indicated in the individual tests. The charge was ground for 20-minute period. The resultant pulp had the following screen analysis:—

| *Mesh* | |
|---|---|
| + 65 | 0·7 per cent |
| +100 | 8·6 " |
| +150 | 13·8 " |
| +200 | 10·3 " |
| −200 | 66·6 " |
| | 100·0 " |

After grinding the pulp was transferred to a laboratory Ruth flotation machine and floated using the following reagents:

### *Test No. 1*

A test was run to show effect of xanthate in a soda ash circuit.

*Reagents*—

| To ball mill— | To copper cells— | To nickel cells— |
|---|---|---|
| $Na_2CO_3$... 4·0 lb./ton | Cresylic..... 0·04 lb./ton | Pot. xan..... 0·15 lb./ton |
| | | Pine oil No. 5 0·08 " |

| Products | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Cu concentrate | 5·8 | 16·84 | 4·97 | 25·7 | 91·6 | 26·8 | 27·6 |
| Ni concentrate | 14·8 | 0·61 | 4·05 | 18·6 | 8·4 | 54·4 | 51·2 |
| Tailing | 79·4 | trace | 0·26 | 1·4 | ........ | 18·8 | 21·2 |
| Head | 100·0 | 1·06 | 1·10 | 5·37 | 100·0 | 100·0 | 100·0 |

### *Test No. 2*

This test was run to show effect of Reagent No. 323 in a soda ash circuit.

*Reagents*—

| To ball mill— | To copper cells— | To nickel cells— |
|---|---|---|
| $Na_2CO_3$... 4·0 lb./ton | Cresylic acid 0·04 lb./ton | No. 323...... 0·15 lb./ton |
| | | Pine oil No. 5 0·08 " |

| Products | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Copper concentrate | 6·4 | 12·48 | 5·42 | 21·5 | 91·0 | 38·6 | 38·6 |
| Nickel concentrate | 13·1 | 0·60 | 3·09 | 17·8 | 9·0 | 45·2 | 55·9 |
| Tailing | 80·5 | trace | 0·18 | 2·8 | ........ | 16·2 | 5·5 |
| Head | 100·0 | 0·87 | 0·90 | 5·98 | 100·0 | 100·0 | 100·0 |

## *Test No. 3*

A test was run to show effect of Reagent No. 208 in a soda ash circuit.

*Reagents*—

To ball mill—
$Na_2CO_3$... 4·0 lb./ton
pH........ 7·8

To copper-nickel cells—
Aerofloat No. 25......0·10 lb./ton

To nickel-iron cells—
No. 208...... 0·20 lb./ton
Pine oil No. 5. 0·04 "
pH.......... 6·4

| Products | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Copper-nickel concentrate......... | 13·5 | 6·32 | 5·19 | 21·1 | 97·2 | 66·6 | 49·0 |
| Nickel-iron concentrate........... | 10·1 | 0·24 | 1·51 | 13·4 | 2·8 | 14·5 | 23·3 |
| Tailing........................... | 76·4 | trace | 0·26 | 2·11 | ........ | 18·9 | 27·7 |
| Head.............................. | 100·0 | 0·88 | 1·05 | 5·80 | 100·0 | 100·0 | 100·0 |

## *Test No. 4*

A bulk flotation test was made.

*Reagents*—

To ball mill—
$Na_2CO_3$.............. 4·0 lb./ton

To cells—
No. 323............... 0·20 lb./ton
Pine oil No. 5......... 0·12 "

| Products | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Concentrate....................... | 13·3 | 8·55 | 3·56 | 25·9 | 94·9 | 64·7 | 63·8 |
| Middling.......................... | 8·0 | 0·38 | 0·94 | 8·4 | 2·5 | 10·4 | 12·6 |
| Tailing........................... | 78·7 | 0·04 | 0·23 | 1·6 | 2·6 | 24·9 | 23·6 |
| Head.............................. | 100·0 | 1·19 | 0·73 | 5·36 | 100·0 | 100·0 | 100·0 |

The rougher concentrates from two tests were combined and cleaned for 7 minutes without additional reagents.

## *Test No. 6*

A test was run to show the effect of thiocarbanilide.

*Reagents*—

To ball mill—
$Na_2CO_3$.............. 4·0 lb./ton
Thiocarbanilide....... 0·20 "

To cells—
Pine oil No. 5......... 0·12 lb./ton

| Products | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Concentrate | 19·5 | 6·65 | 4·09 | 23·8 | 96·4 | 81·8 | 81·9 |
| Tailing | 80·5 | 0·06 | 0·22 | 1·27 | 3·6 | 18·2 | 18·1 |
| Head | 100·0 | 1·34 | 0·97 | 5·64 | 100·0 | 100·0 | 100·0 |

### *Test No. 7*

A test was run using lime instead of soda ash. Two concentrates were made.

*Reagents*—

To ball mill—
CaO....... 4·0 lb./ton

"A" float—
Aerofloat
No. 25......0·08 lb./ton

"B" float—
$H_2SO_4$....... 7·0 lb./ton
No. 323...... 0·20 "
Pine oil No. 5 0·12 "

pH......... 6·0

| Products | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| "A" concentrate | 6·9 | 17·46 | 3·15 | 20·1 | 87·8 | 26·0 | 25·6 |
| "B" concentrate | 16·7 | 0·82 | 2·47 | 21·3 | 10·0 | 49·8 | 66·4 |
| Tailing | 76·4 | 0·04 | 0·26 | 0·57 | 2·2 | 24·2 | 8·0 |
| Head | 100·0 | 1·37 | 0·83 | 5·41 | 100·0 | 100·0 | 100·0 |

Lime has a very marked effect in raising the grade of the copper concentrate.

### *Test No. 8*

This test was run to show effect of cyanide in a lime circuit.

*Reagents*—

To ball mill—
CaO....... 2·0 lb./ton
NaCN.... 0·10 "

To copper cells—
Aerofloat
No. 25......0·08 lb./ton

To nickel cells—
$H_2SO_4$....... 8·0 lb./ton
No. 323...... 0·20 "
Pine oil No. 5 0·12 "

pH........... 5·2

| Products | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Ni, per cent | S, per cent | Cu | Ni | S |
| Copper concentrate | 5·6 | 18·62 | 0·73 | 19·3 | 78·0 | 4·2 | 20·5 |
| Nickel concentrate | 18·1 | 1·42 | 4·59 | 21·8 | 19·2 | 85·0 | 74·9 |
| Tailing | 76·3 | 0·05 | 0·14 | 0·32 | 2·8 | 10·8 | 4·6 |
| Head | 100·0 | 1·34 | 0·98 | 5·29 | 100·0 | 100·0 | 100·0 |

The above reagent combination has effected the cleanest separation obtained on this ore. It is possible that a prolonged rougher float followed by cleaning would result in increased copper recovery to the copper concentrate.

### SUMMARY

In a circuit made alkaline with soda ash, a bulk copper-nickel concentrate can be obtained with cresylic acid. High recovery of copper is indicated. The addition of 0·10 pound per ton of Aerofloat No. 25 caused a considerable increase in the recovery of both copper and nickel in a lower grade product.

For the flotation of a nickel-iron product a water-soluble promoting reagent must be used. Comparative tests, i.e. Tests Nos. 1 and 2, indicate that Reagent No. 323 is more effective than xanthate.

In a lime circuit much higher grade copper concentrates can be produced. The use of cyanide in conjunction with lime has resulted in the most marked separation obtained.

After the nickel-bearing pyrrhotite has been depressed by lime and cyanide, it is more difficult to float. The use of an acid circuit may be found advisable for efficient flotation.

### CONCLUSION

A wide variety of products containing copper and nickel in various proportions can be obtained from the flotation of the ore sample received from Denison Copper Mines, Limited.

However, the most advisable product cannot be determined without more detailed knowledge of the payments and penalties of a custom copper-nickel smelter schedule.

## Report No. 358

### EXPERIMENTAL TESTS ON GOLD ORE FROM THE CENTRAL PATRICIA MINES, LIMITED, SAVANT LAKE AREA, ONTARIO

**J. S. Godard**

*Shipments.* Two shipments of gold ores were received from the Central Patricia Mines, Limited. Shipment No. 1, weighing 8,000 pounds was received March 14, 1930, and shipment No. 2, weighing 1,540 pounds was received April 25, 1930.

*Characteristics of the Ores.* Both were gold ores. The gold is mainly associated with arsenopyrite and iron pyrites, both pyrite and pyrrhotite being present. The gangue is chiefly silica. As the samples were taken at or near the surface both samples were somewhat oxidized.

*Analyses of Head Samples—*

| | |
|---|---|
| No. 1 shipment—Gold | 0·60 oz./ton |
| No. 2 shipment—Gold | 0·82 " |

*Purpose of Experimental Tests.* W. E. Segsworth, 67 Yonge Street, Toronto, consulting engineer for the Central Patricia Mines, Limited, desired that experimental tests be undertaken to determine whether straight cyanidation or concentration and cyanidation of the concentrates were the better method.

## EXPERIMENTAL TESTS

### CONCENTRATION: FLOTATION; TABLING THE FLOTATION TAILING

### *Tests Nos. 1 to 4*

*Test No. 1: Table Tailing Screened—*

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +48 | 1·5 | 0·11 | 3·4 |
| +65 | 3·5 | 0·08 | 5·7 |
| +100 | 9·2 | 0·07 | 13·0 |
| +200 | 12·3 | 0·09 | 22·5 |
| −200 | 41·7 | 0·05 | 42·4 |
| Slimes | 31·8 | 0·02 | 13·0 |

Average table tailing: Au 0·049 oz./ton.

*Concentration—*

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 6·5 | 8·04 | 78·1 |
| Table concentrate | 19·8 | 0·56 | 16·5 |
| Table tailing | 73·7 | 0·049 | 5·4 |

Recovery: 94·6 per cent.

*Test No. 2: Table Tailing Screened—*

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +100 | 2·7 | 0·55 | 33·4 |
| +200 | 9·2 | 0·08 | 16·6 |
| −200 | 46·3 | 0·03 | 31·2 |
| Slimes | 41·8 | 0·02 | 18·8 |

Average table tailing: Au 0·045 oz./ton.

*Concentration—*

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 11·5 | 4·36 | 85·1 |
| Table concentrate | 11·0 | 0·48 | 9·0 |
| Table tailing +100 | 2·1 | 0·55 | 2·0 |
| " +200 | 7·1 | 0·08 | 1·0 |
| " −200 | 35·9 | 0·03 | 1·8 |
| Slimes | 32·4 | 0·02 | 1·1 |

Recovery: 94·1 per cent.

*Test No. 3: Table Tailing Screened—*

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +200 | 6·2 | 0·11 | 12·4 |
| −200 | 52·0 | 0·03 | 69·0 |
| Slimes | 41·8 | 0·01 | 18·6 |

Average table tailing: Au 0·023 oz./ton.

*Concentration—*

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 16·3 | 3·42 | 84·0 |
| Table concentrate | 5·1 | 1·666 | 12·8 |
| Table tailing +200 | 4·9 | 0·11 | 0·8 |
| " −200 | 40·9 | 0·03 | 1·9 |
| Slimes | 32·8 | 0·01 | 0·5 |

Recovery: 96·8 per cent.

*Test No. 4: Table Tailing Screened—*

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +48+65 | 2·3 | 0·09 | 3·7 |
| +100 | 8·0 | 0·14 | 19·9 |
| +200 | 15·6 | 0·08 | 22·2 |
| −200 | 41·3 | 0·05 | 36·8 |
| Slimes | 32·8 | 0·03 | 17·4 |

Average table tailing: Au 0·056 oz./ton.

*Concentration*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 10·7 | 5·00 | 84·4 |
| Table concentrate | 8·1 | 0·66 | 8·4 |
| Table tailing + 48+65 | 1·9 | 0·09 | 0·3 |
| " +100 | 6·5 | 0·14 | 1·4 |
| " +200 | 12·6 | 0·08 | 1·6 |
| " −200 | 33·6 | 0·05 | 2·6 |
| Slimes | 26·6 | 0·03 | 1·3 |

Recovery: 92·8 per cent.

## FLOTATION

### *Tests Nos. 5 to 8*

*Test No. 5: Flotation Tailing Screened*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +100 | 1·8 | 2·42 | 38·6 |
| +200 | 12·6 | 0·21 | 23·5 |
| −200 | 85·6 | 0·05 | 37·9 |

Average flotation tailing: 0·113 oz./ton.

*Flotation*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 11·8 | 4·58 | 84·5 |
| Tailing +100 | 1·6 | 2·42 | 6·0 |
| " +200 | 11·1 | 0·21 | 3·6 |
| " −200 | 75·5 | 0·05 | 5·9 |

*Test No. 6: Flotation Tailing Screened*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +100 | 3·3 | 1·58 | 43·0 |
| +200 | 10·9 | 0·24 | 21·6 |
| −200 | 85·8 | 0·05 | 35·4 |

Average flotation tailing: Au 0·121 oz./ton.

*Flotation*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 10·5 | 4·84 | 82·3 |
| Tailing +100 | 3·0 | 1·58 | 7·7 |
| " +200 | 9·7 | 0·24 | 3·8 |
| " −200 | 76·8 | 0·05 | 6·2 |

*Test No. 7: Flotation Tailing Screened*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +100 | 2·1 | 0·50 | 8·7 |
| +200 | 13·2 | 0·32 | 35·0 |
| −200 | 84·7 | 0·08 | 56·3 |

Average flotation tailing: Au 0·12 oz./ton.

*Flotation*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 11·7 | 4·04 | 81·7 |
| Tailing +100 | 1·9 | 0·50 | 1·6 |
| " +200 | 11·6 | 0·32 | 6·4 |
| " −200 | 74·8 | 0·08 | 10·3 |

*Test No. 8: Flotation Tailing Screened*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +100 | 3·0 | 1·66 | 44·9 |
| +200 | 12·5 | 0·15 | 17·0 |
| −200 | 84·5 | 0·05 | 38·1 |

Average flotation tailing: Au 0·11 oz./ton.

*Flotation*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 14·2 | 3·82 | 85·0 |
| Tailing +100 | 2·6 | 1·66 | 6·8 |
| " +200 | 10·7 | 0·15 | 2·5 |
| " −200 | 72·5 | 0·05 | 5·7 |

## TWO-STAGE FLOTATION TEST

### *Test No. 9*

Ore at −14 mesh, ground 7 minutes with $Na_2CO_3$ 2·0 lb. per ton, No. 34 fuel oil 0·24 lb. per ton, Tarol 0·25 lb. per ton

No. 1 concentrate removed using amyl xanthate 0·08 lb. per ton, pine oil No. 5, 0·02 lb. per ton.

Tailing from No. 1 flotation was deslimed and oversize reground 30 minutes with $Na_2CO_3$ 1·5 lb. per ton, No. 34 fuel oil 0·09 lb. per ton, Tarol 0·10 lb. per ton.

No. 2 concentrate removed using amyl xanthate 0·04 lb. per ton, pine oil No. 5, 0·03 lb. per ton.

*Results*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| No. 1 concentrate | 6·8 | 6·50 | 70·5 |
| No. 2 concentrate | 13·3 | 1·00 | 21·2 |
| Classifier overflow +200 | 1·1 | 0·14 | 0·2 |
| " " −200 | 24·7 | 0·03 | 1·2 |
| Tailing +100 | 1·6 | 1·37 | 3·5 |
| " +200 | 13·0 | 0·07 | 1·5 |
| " −200 | 39·5 | 0·03 | 1·9 |

Recovery: 91·7 per cent.

### *Screen Analyses—First Flotation*

| Mesh | Weight, per cent | Cum., per cent | Mesh | Weight, per cent | Cum., per cent |
|---|---|---|---|---|---|
| +35 | 2·3 | 2·3 | +100 | 14·5 | 33·0 |
| +48 | 4·7 | 7·0 | +200 | 13·1 | 46·1 |
| +65 | 11·5 | 18·5 | −200 | 53·9 | 100·0 |

## Central Patricia No. I

## CYANIDATION

### *Tests Nos. 10 to 18*

A number of small-scale cyanidation tests were made in Winchester bottles, on ore dry crushed to pass given meshes. The cyanide strength in each test was 0·50 per cent KCN and the dilution 1:3. Agitation periods were approximately 48 hours.

## *Screen Analyses of the Cyanidation Tailings*

| Test No. | Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au | Average tailing, Au oz./ton |
|---|---|---|---|---|---|
| 10 | +48 | 7·8 | 0·22 | 13·0 | 0·13 |
| | +65 | 19·9 | 0·37 | 57·2 | |
| | +100 | 12·5 | 0·12 | 11·5 | |
| | +200 | 15·0 | 0·10 | 11·5 | |
| | −200 | 44·8 | 0·02 | 6·8 | |
| 11 | +65 | 9·8 | 0·13 | 18·2 | 0·07 |
| | +100 | 16·4 | 0·13 | 30·6 | |
| | +200 | 19·2 | 0·10 | 27·6 | |
| | −200 | 54·6 | 0·03 | 23·6 | |
| 12 | +100 | 9·3 | 0·10 | 39·4 | 0·024 |
| | +200 | 26·4 | 0·03 | 33·5 | |
| | −200 | 64·3 | 0·01 | 27·1 | |
| 13 | +200 | 24·9 | 0·06 | 49·8 | 0·03 |
| | −200 | 75· | 0·02 | 50·2 | |
| 14 | +200 | 10·8 | 0·04 | 32·6 | 0·013 |
| | −200 | 89·2 | 0·01 | 67·4 | |
| 15 | +65 | 5·3 | 0·09 | 12·4 | 0·039 |
| | +100 | 11·9 | 0·16 | 49·1 | |
| | +200 | 21·9 | 0·04 | 22·7 | |
| | −200 | 60·9 | 0·01 | 15·8 | |
| 16 | +100 | 7·1 | 0·11 | 33·0 | 0·024 |
| | +200 | 21·7 | 0·04 | 36·9 | |
| | −200 | 71·2 | 0·10 | 30·1 | |
| 17 | +200 | 15·6 | 0·06 | 42·5 | 0·022 |
| | −200 | 84·4 | 0·015 | 57·5 | |
| 18 | +200 | 12·5 | 0·06 | 30·0 | 0·025 |
| | −200 | 87·5 | 0·02 | 70·0 | |

*Summary*—

| Test No. | Mesh | Heads, Au oz./ton | Tails (average), Au oz./ton | Extraction, per cent | Reagents, lb./ton | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| 10 | −35 | 0·57 | 0·13 | 77·2 | 2·67 | 14·3 |
| 11 | −48 | 0·57 | 0·07 | 87·7 | 2·83 | 14·5 |
| 12 | −65 | 0·57 | 0·024 | 95·7 | 2·78 | 15·8 |
| 13 | −100 | 0·57 | 0·030 | 94·7 | 2·91 | 16·5 |
| 14 | −150 | 0·57 | 0·013 | 97·6 | 3·08 | 18·7 |
| 15 | −48 | 0·54 | 0·098 | 81·8 | 2·58 | 17·4 |
| 16 | −65 | 0·54 | 0·024 | 95·6 | 2·22 | 23·9 |
| 17 | −100 | 0·54 | 0·022 | 96·0 | 2·46 | 26·9 |
| 18 | −150 | 0·54 | 0·025 | 95·4 | 2·64 | 27·4 |

Note.—In Tests Nos. 15–18 lime equivalent to 14 pounds per ton of ore was agitated for 1½ hours before adding the cyanide.

### *Test No. 19*

### *Cyanidation Test using Continuous Grinding Unit and Pachuca Tank*

In this test the ore at −10 mesh was fed continuously to a small rod mill in closed circuit with a drag classifier. The grinding was done in 1 : 1 pulp 0·03 per cent KCN. The classifier overflow was emptied into a Pachuca tank and agitated for 40 hours. The cyanide strength was 0·05 per cent KCN and the dilution 1 : 2·8.

*Screen Analyses on the Cyanide Tailing—*

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +65 }<br>+100} | 9·6 | 0·16 | 42·2 |
| +200 | 30·2 | 0·03 | 24·9 |
| −200 | 60·2 | 0·02 | 32·9 |

Head sample for test: Au 0·67 ounce per ton. Cyanide tailing (average) Au 0·037 ounce per ton. Extraction 94·6 per cent.

## CONCLUSIONS

Flotation followed by tabling the flotation tailing gave recoveries of 93 to 96·8 per cent, with the increase in recoveries obtained in those tests where finer grinding was practised. The ratios of concentration were approximately 4 and 5 to 1. The tabling operation appears to be essential as when omitted high gold values are found in the coarser sizes of the flotation tailings.

Cyanidation at about 5 to 8 per cent on 100 mesh gave good extractions and were it not for the high consumption of reagents it would appear to be the better metallurgical treatment for this ore. It is believed that the high consumption of reagents is due, at least in part, to the oxidized condition of the ore.

## Central Patricia No. II

## CYANIDATION

### *Tests Nos. 1 to 3*

Three small-scale cyanidation tests were made, in Winchester bottles, on ore dry crushed to pass given meshes. In each test the cyanide strength was 0·05 per cent KCN and the dilution 1: 3. The agitation period was about 48 hours.

| Test No. | Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au | Average tailing, Au oz./ton |
|---|---|---|---|---|---|
| 1 | +100 | 7·4 | 0·20 | 13·0 | 0·114 |
| | +200 | 23·1 | 0·13 | 26·2 | |
| | −200 | 69·5 | 0·10 | 60·8 | |
| 2 | +200 | 20·1 | 0·21 | 27·4 | 0·154 |
| | −200 | 79·9 | 0·14 | 72·6 | |
| 3 | +200 | 12·4 | 0·28 | 18·0 | 0·192 |
| | −200 | 87·6 | 0·18 | 82·0 | |

*Summary—*

| No. | Mesh | Heads, Au oz./ton | Tails, Au oz./ton | Extraction, per cent | Reagents, lb./ton | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| 1.................. | −65 | 0·80 | 0·114 | 85·7 | 5·74 | 12·7 |
| 2.................. | −100 | 0·80 | 0·154 | 80·7 | 5·88 | 13·4 |
| 3.................. | −150 | 0·80 | 0·192 | 76·0 | 6·35 | 15·8 |

## FLOTATION

### *Tests Nos. 4 to 7*

Flotation tests were made at about 10 per cent on 100 mesh.

| Test No. | Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|---|
| 4 | Concentrate.................................. | 15·7 | 4·60 | 84·3 |
| | Tailing.................................. | 84·3 | 0·16 | 15·7 |
| 5 | Concentrate.................................. | 12·9 | 5·18 | 81·8 |
| | Tailing.................................. | 87·1 | 0·17 | 18·2 |
| 6 | Concentrate.................................. | 17·2 | 4·40 | 87·6 |
| | Tailing.................................. | 82·8 | 0·13 | 12·4 |
| 7 | Concentrate.................................. | 26·1 | 3·00 | 93·0 |
| | Tailing.................................. | 73·9 | 0·08 | 7·0 |

### *Test No. 8*

A two-stage flotation test was made, the final flotation tailing being tabled.

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| 1. Flotation concentrate No. 1.................... +100 | 0·1 | 0·97 | 0·1 |
| 2. " " No. 1.................... −100 | 13·6 | 4·74 | 77·3 |
| 3. " " No. 2.......................... | 5·0 | 1·76 | 10·6 |
| 4. Table concentrate.......................... | 7·7 | 0·90 | 8·3 |
| Classifier overflow.......................... +200 | 0·8 | 0·07 | 0·3 |
| " " .......................... −200 | 24·0 | 0·03 | 0·8 |
| Table tailing.......................... +100 | 4·3 | 0·06 | 1·3 |
| " .......................... +200 | 15·8 | 0·04 | 0·3 |
| " .......................... −200 | 21·1 | 0·05 | 0·1 |
| Table slimes.......................... | 7·6 | 0·03 | 0·9 |

Total recovery from products 1-2-3-4 =96·3 per cent.

## CONCENTRATION—FLOTATION AND TABLING

### *Tests Nos. 9 and 10*

Screen analyses on table tailing gave the following results:—

*Concentration*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +100 | 3·5 | 0·02 | 1·9 |
| +200 | 16·9 | 0·04 | 18·4 |
| −200 | 45·0 | 0·05 | 61·0 |
| Slimes | 34·6 | 0·02 | 18·7 |

Average table tailing: Au 0·037 oz./ton.

*Concentration*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 24·5 | 3·16 | 88·8 |
| Table concentrate | 9·1 | 0·81 | 8·4 |
| Table tailings +100 | 2·3 | 0·02 | 0·1 |
| " +200 | 11·2 | 0·04 | 0·5 |
| " −200 | 29·9 | 0·05 | 1·7 |
| Table slimes | 23·0 | 0·02 | 0·5 |

Recovery 88·8+8·4=97·2 per cent.

### *Test No. 10*

*Screen Analyses on Tailing*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +100 | 4·0 | 0·02 | 1·9 |
| +200 | 17·9 | 0·04 | 16·9 |
| −200 | 45·9 | 0·06 | 64·5 |
| Slimes | 19·4 | 0·03 | 13·6 |
| Classifier overflow | 12·8 | 0·01 | 3·1 |

Average tailing: Au 0·043 oz./ton.
NOTE.—In this test the flotation tailing was deslimed before tabling.

*Concentration*—

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 30·1 | 2·50 | 89·1 |
| Table concentrate | 10·0 | 0·66 | 7·8 |
| Table tailings +100 | 2·4 | 0·02 | 0·1 |
| " +200 | 10·7 | 0·04 | 0·5 |
| " −200 | 27·5 | 0·06 | 2·0 |
| Table slimes | 11·6 | 0·03 | 0·4 |
| Classifier overflow | 7·7 | 0·01 | 0·1 |

Recovery 89·1+7·8=96·9 per cent.

## *Test No. 11*

### *Flotation and Tabling—Cyanidation of the Concentrates*

*Concentration—*

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Combined concentrates | 38·4 | 2·28 | 98·1 |
| Tailing +100 | 2·1 | 0·04 | 0·1 |
| " +200 | 10·5 | 0·04 | 0·5 |
| −200 | 33·1 | 0·03 | 1·1 |
| Slimes | 15·9 | 0·01 | 0·2 |

*Cyanidation of the Combined Concentrates—*

| | |
|---|---|
| KCN per cent | 0·15 |
| Dilution | 1 : 3 |
| Time | 65 hours |

*Screen Analyses of the Cyanide Tailing—*

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +200 | 25·2 | 0·09 | 50·3 |
| −200 | 74·8 | 0·03 | 49·7 |

Average cyanide tailing: Au 0·045 oz./ton.

*Summary—*

| Head Au oz./ton | Tailing Au oz./ton | Extraction, per cent | Reagents | | | |
|---|---|---|---|---|---|---|
| | | | Per ton concs. | | Per ton ore | |
| | | | KCN | CaO | KCN | CaO |
| 2·28 | 0·045 | 98·0 | 15·67 | 34·8 | 6·02 | 13·3 |

*Recovery—*

Concentration, 98·1 per cent.
Cyanidation, 98·0 per cent.
Total recovery=96·1 per cent.

## Report No. 359

### THE RECOVERY OF GOLD FROM THE TAILINGS OF THE COUNTY HARBOUR MINES, NOVA SCOTIA

**A. K. Anderson**

*Shipments.* Two bags of sand of approximately 100 pounds each were received at the laboratories, consigned to Mr. W. F. V. Atkinson, 404 Laurier Avenue East, Ottawa, Ontario, and said to be representative samples from two tailing ponds at County Harbour, Nova Scotia.

*Characteristics of the Ore.* Sample No. 2 from the upper pond consisted of damp quartz and schist sand, approximately 20 mesh in size. Ten per cent of the total weight consisted of material remaining on a 10 mesh screen. This coarse material consisted of sharp angular particles of quartz and schist such as would be found remaining within a stamp battery mortar box, and dumped out when the clean-ups were made, small particles of wood with sharp edges, and flat flakes of schist.

Sample No. 1, said to be from pond formed by overflow from the upper pond, was identical in nature to sample No. 2. Fourteen per cent of material remaining on 10 mesh was present.

Both samples contained considerable free mercury and amalgam.

*Purpose of Experimental Tests.* The object of making these tests was to determine what recovery of gold could be obtained. Information concerning the suitability of a proposed electrical amalgamating machine for the recovery of the gold was also asked. As this method or machine had never been demonstrated, it was considered advisable to adhere to known and proved methods of milling practice.

*Sampling and Analysis.* Each lot was sampled separately. This was done by passing through a Jones riffle sampler, discarding one half of the lot at each pass until a portion of approximately 10 pounds was obtained. This part was then ground to pass 65 mesh and again quartered. This quarter was ground to pass 100 mesh and a representative portion secured for assay.

The results secured are as follows—

| | |
|---|---|
| Sample No. 1—Gold—0·75 oz. per ton | $15.00 |
| Arsenic—0·87 per cent. | |
| Sample No. 2—Gold—1·18 oz. per ton | $23.60 |
| Arsenic—1·70 per cent. | |

Owing to the nature and appearance of these samples—both contained coarse crushed material and amalgam—and that the sample said to come from a pond formed by overflow from the other, was of the same nature and only slightly lower in value than the other, the samples were felt not to be representative. The parties interested were advised to re-sample.

Four additional samples were sent in.

Sample No. 1 said to be from about 10 feet distant from the upper end of the upper pond assayed $3.40.

Sample No. 2 taken from points across the pond 30 to 40 feet above the lower end gave a gold content of $3.00 per ton.

Sample No. 3 taken at the upper end of the lower pond was found to contain $1.00 gold per ton, and sample No. 4, taken about the middle of this second pond, $1.20 per ton.

These second lot of samples were of a different nature than those originally furnished being free from amalgam and coarse particles.

## EXPERIMENTAL TESTS

### Shipment No. 1

Tests carried on the first shipment consisted of amalgamation followed by table concentration.

### *Sample No. 1*

*Amalgamation without Regrinding*

| | |
|---|---|
| Tailing—0·21 oz. gold per ton | $4.20 |

Recovery by amalgamation, 72·0 per cent.

*Amalgamation after Regrinding*

| | |
|---|---|
| Tailing—0·21 oz. gold per ton | $4.20 |

Recovery by amalgamation, 72·0 per cent.

Apparently all the values in this sample recoverable by amalgamation are present as amalgam.

*Amalgamation followed by Table Concentration*

| Product | Weight, per cent | Assays Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Table concentrate | 6·9 | 0·84 | 10·8 |
| Table tailing | 93·1 | 0·10 | 17·2 |
| Amalgam | | | 72·0 |

This test shows that after removal of the amalgam present, table concentration recovers an additional 10·8 per cent of the total values in a product assaying 0·84 oz. gold per ton or $16.80. Every 100 tons treated yields 6·9 tons of this concentrate.

### *Sample No. 2*

*Amalgamation without Regrinding*

| | |
|---|---|
| Tailing—0·33 oz. gold per ton | $6.60 |

Recovery by amalgamation, 72·0 per cent.

*Amalgamation after Regrinding*

| | |
|---|---|
| Tailing—0·34 oz. gold per ton | $6.80 |

Recovery by amalgamation, 71·2 per cent.

These results coincide with those obtained in sample No. 1, no additional recovery being obtained by regrinding. It can be assumed that the values recovered by amalgamation are present in the sample in the form of amalgam.

*Amalgamation followed by Table Concentration*

| Product | Weight, per cent | Assay, Au oz./ton | Per cent of values |
|---|---|---|---|
| Table concentrate | 4·7 | 3·26 | 12·5 |
| Table tailing | 95·3 | 0·20 | 15·5 |
| Amalgam | ........... | ........... | 72·0 |

These results indicate that after amalgamation a further recovery of 12·5 per cent of the values are recovered in a product assaying 3·26 oz. gold per ton.

From each 100 tons of feed there are recovered 4·7 tons of this concentrate.

### Shipment No. 2

The second shipment consisting of four samples numbered 1 to 4, as stated before, had an assay of 0·17 oz., 0·15 oz., 0·05 and 0·06 oz. gold per ton.

Samples Nos. 1 and 2 representing the upper dump were tested to see what percentage of the gold could be recovered by amalgamation and table concentration. The lower dump represented by samples Nos. 3 and 4 containing $1.00 and $1.20 per ton are too low to be of value and were not tested.

*Sample No. 1*

One thousand grams of the ore was ground to break up cemented material and amalgamated.

| Product | Weight, per cent | Assay, Au oz./ton | Per cent of values |
|---|---|---|---|
| Amalgam tailing | ........... | 0·13 | *23·2 |
| Table concentrate | 5·0 | 0·70 | 20·6 |
| Table tailing | 95·0 | 0·10 | 56·2 |
| Heads | 100·0 | 0·17 | 100·0 |

*Recovered by amalgamation.

This test indicates that from each ton of sands treated 79 cents is recovered by amalgamation and 70 cents by table concentration.

From each 100 tons of feed there are recovered 5 tons of concentrate assaying 0·70 oz. gold per ton, or $14.00.

With a combined freight and treatment charge of $16.50 per ton, this product containing arsenic is of no value.

This leaves a recovery of 79 cents obtained by amalgamation.

### *Sample No. 2*

This sample was treated in the same way as was sample No. 1.

| Product | Weight, per cent | Assay, Au oz./ton | Per cent of values |
|---|---|---|---|
| Amalgamated tailing | ........... | 0·13 | *9·8 |
| Table concentrate | 4·1 | 0·96 | 26·2 |
| Table tailing | 95·9 | 0·10 | 64·0 |
| Heads | 100·0 | 0·15 | 100·0 |

*Recovered by amalgamation.

This shows that from each ton of feed treated 29 cents is recovered by amalgamation and 79 cents by table concentration.

Each 100 tons of feed yields 4·1 tons of this product assaying $19.20 per ton. This could be shipped to a smelter and a profit realized.

With a ratio of concentration of 24·4 : 1 the profit from the concentrate after deducting freight and treatment at $16.50 per ton would be 1·1 cents per ton of original feed, not sufficient to pay for the cost of concentration. This leaves the gold recovered by amalgamation as the sole values to be considered.

Assuming that sample No. 1, with a value of $3.40 per ton represents the value of this pond containing approximately 6,000 tons, 79 cents a ton is recovered.

To treat this material there would be required a horse-drawn scraper or wagon to place the sand in the ore bin, one amalgamator or mill man at approximately $7.00 a day and one labourer say at $2.50 a day. Operating on a 24-hour basis, wages for three shifts would amount to $28.50 exclusive of the cost of placing the sand in the mill ore bin or wages paid for an engineer to operate the power plant, or supervision. On the basis of 20 tons a day recovering 79 cents a ton this would leave a deficit of $12.70 daily.

A plant to treat this ore, regardless of the gold content, would include a 20 to 30 horse-power engine, a bin large enough to contain approximately 50 tons of sand followed by a reciprocating feeder. This feeder discharges into the ball mill which is followed by a couple of amalgamating plates with a total area of about 80 square feet.

A suitable water supply will be necessary to furnish water to the mill. Tests conducted on the first samples furnished show that no increase in recovery was obtained by grinding. Part of the second shipment, however, contained cemented material. Grinding will be necessary to break up this to ensure an evenly sized product otherwise the plates will be scoured by coarse particles passing over them.

If it is desired to concentrate the material after passing over the amalgamating plates, a concentrating table such as a Wilfley will be required, together with a draining floor or dryer for the concentrates.

The cost of such a plant will depend largely on what suitable second-hand equipment can be secured. Approximately $10,000 will be required to purchase and install it.

## SUMMARY AND CONCLUSIONS

A survey of some fifty tailings dumps in Nova Scotia made by C. S. Parsons of the Mines Branch and published in the Mines Branch Investigations of 1922 did not disclose any of these to average more than $1.00 in gold. This particular dump was not included in this survey. As milling practice was similar in all Nova Scotia gold camps it would seem highly improbable that these two ponds should average $23.60 and $15.00 as indicated by the first samples sent in. This fact, coupled with the nature and appearance of the material, both containing amalgam and coarse particles of ore and country rock would make it appear that the samples were not representative.

The second lot of samples said to be taken from the surface and from one to two feet below, at different places in the dumps are much more characteristic in nature and appearance of tailing-dump residues.

Taking everything into consideration, it is felt that the second samples showing the dumps to contain $3.40 and $1.20 are much nearer the true value than the former.

Before any further expenditures are made, it is recommended that a thorough systematic sampling of these dumps be made by a competent engineer not connected with the enterprise.

# Report No. 360

## EXPERIMENTAL TESTS ON GOLD ORE FROM O'BRIEN MINE, CADILLAC TOWNSHIP, ABITIBI COUNTY, QUEBEC

**J. S. Godard**

*Shipment.* A shipment of 2,800 pounds of ore was received January 22, 1930, from the O'Brien Mines, Limited.

*Characteristics of the Ore.* The sample submitted was that of a high-grade gold ore. The gold is largely in the free state and fairly coarse, though some is associated with sulphides of iron and arsenic. The gangue is chiefly quartz of which there are two varieties, a greyish white one which constitutes the bulk of the gangue and a blackish magnetiferous one. It is in the fractures in the latter one in which the coarser free gold is present.

*Analyses of Head Sample.* The head sample contained, gold 2·77 ounces per ton.

*Purpose of Experimental Tests.* The purpose of the experimental tests was to determine the best metallurgical treatment for this ore.

## EXPERIMENTAL TESTS

### AMALGAMATION

Samples used in these tests were crushed and screened through a 10 mesh screen. The metallics +10 mesh were removed.

Ore at −10 mesh dry crushed to given sizes.

### *Tests Nos. 1 to 6*

Head Sample: Au 2·01 oz. per ton.

| Test No. | Mesh | Size | Weight, per cent | Assays, Au oz./ton | Percentage of values, Au | Average tailing, Au oz./ton |
|---|---|---|---|---|---|---|
| 1.................. | −18 | +28 | 18·1 | *2·09 | 36·5 | 1·03 |
| | | +35 | 18·5 | 1·20 | 21·4 | |
| | | +48 | 11·8 | 1·18 | 13·5 | |
| | | +65 | 10·1 | 1·02 | 10·0 | |
| | | +100 | 8·6 | 0·72 | 6·0 | * |
| | | +200 | 7·3 | 0·52 | 3·7 | |
| | | −200 | 25·6 | 0·36 | 8·9 | |
| 2.................. | −28 | +35 | 11·2 | 1·69 | 21·0 | |
| | | +48 | 18·8 | 1·18 | 24·6 | 0·90 |
| | | +65 | 14·8 | 1·10 | 18·0 | |
| | | +100 | 13·3 | 0·77 | 11·4 | |
| | | +200 | 11·8 | 0·63 | 8·2 | |
| | | −200 | 30·1 | 0·49 | 16·8 | |
| 3.................. | −35 | +48 | 8·9 | 1·13 | 16·8 | 0·60 |
| | | +65 | 20·4 | 0·92 | 31·3 | |
| | | +100 | 16·2 | 0·68 | 16·7 | |
| | | +200 | 14·2 | 0·52 | 12·3 | |
| | | −200 | 40·3 | 0·34 | 22·9 | |
| 4.................. | −48 | +65 | 13·6 | 0·97 | 21·2 | 0·62 |
| | | +100 | 21·7 | 0·90 | 31·3 | |
| | | +200 | 16·1 | 0·72 | 18·6 | |
| | | −200 | 48·6 | 0·37 | 28·9 | |
| 5.................. | −65 | +100 | 26·4 | 1·02 | 53·3 | 0·50 |
| | | +200 | 19·0 | 0·55 | 20·7 | |
| | | −200 | 54·6 | 0·24 | 26·0 | |
| 6.................. | −100 | +200 | 25·1 | 0·75 | 50·2 | 0·38 |
| | | −200 | 74·9 | 0·25 | 49·8 | |

*Sample examined under the microscope showed gold partly coated with mercury, but not freed from quartz gangue.

*Summary*—

| Test No. | Size | Tailing, Au oz./ton | Recovery, per cent |
|---|---|---|---|
| 1..................................................... | −18 | 1·03 | 48·7 |
| 2..................................................... | −28 | 0·90 | 55·2 |
| 3..................................................... | −35 | 0·60 | 70·1 |
| 4..................................................... | −48 | 0·62 | 69·1 |
| 5..................................................... | −65 | 0·50 | 75·1 |
| 6..................................................... | −100 | 0·38 | 81·1 |

## AMALGAMATION, CYANIDATION OF THE AMALGAMATION TAILINGS

### *Tests Nos. 7 to 12*

All samples at −10 mesh, dry crushed to given sizes.

*Results*—

| Test No. | Mesh | Size | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au | Tailing, Au oz./ton |
|---|---|---|---|---|---|---|
| 7 | −28 | +35 | 8·4 | 1·94 | 19·3 | 0·86 |
| | | +48 | 16·3 | 1·00 | 19·0 | |
| | | +65 | 19·2 | 1·00 | 22·3 | |
| | | +100 | 10·8 | 0·84 | 10·6 | |
| | | +200 | 10·4 | 0·77 | 9·3 | |
| | | −200 | 34·9 | 0·48 | 19·5 | |
| 8 | −35 | +48 | 7·4 | 0·73 | 7·4 | 0·73 |
| | | +65 | 16·5 | 1·08 | 24·3 | |
| | | +100 | 18·1 | 1·01 | 24·9 | |
| | | +200 | 12·6 | 0·73 | 12·5 | |
| | | −200 | 45·4 | 0·50 | 30·9 | |
| 9 | −48 | +65 | 16·6 | 1·02 | 22·3 | 0·76 |
| | | +100 | 18·5 | 1·10 | 26·8 | |
| | | +200 | 14·7 | 0·85 | 16·5 | |
| | | −200 | 50·2 | 0·52 | 34·4 | |
| 10 | −65 | +100 | 17·5 | 1·53 | 27·4 | 0·98 |
| | | +200 | 19·3 | 1·16 | 22·9 | |
| | | −200 | 63·2 | 0·77 | 49·7 | |
| 11 | −100 | +200 | 21·8 | 1·01 | 39·6 | 0·56 |
| | | −200 | 78·2 | 0·43 | 60·4 | |

The amalgamation tailings from each of the above tests were cyanided and screened.

| | |
|---|---|
| KCN | 0·05 per cent |
| Dilution | 1 : 3 |
| Time | 42–47 hours |

*Results*—

| Test No. | Mesh | Size | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au | Average tailing, Au oz./ton |
|---|---|---|---|---|---|---|
| 7 | −28 | +35 | 8·8 | 0·35 | 9·8 | 0·32 |
| | | +48 | 15·2 | 0·68 | 32·8 | |
| | | +65 | 15·9 | 0·61 | 30·8 | |
| | | +100 | 13·1 | 0·32 | 13·3 | |
| | | +200 | 9·2 | 0·21 | 6·1 | |
| | | −200 | 37·8 | 0·06 | 7·2 | |
| 8 | −35 | +48 | 7·0 | 1·44 | 33·5 | 0·30 |
| | | +65 | 17·0 | 0·48 | 27·1 | |
| | | +100 | 18·9 | 0·28 | 17·6 | |
| | | +200 | 9·4 | 0·29 | 9·1 | |
| | | −200 | 47·7 | 0·08 | 12·7 | |
| 9 | −48 | +65 | 15·6 | 0·25 | 25·2 | 0·16 |
| | | +100 | 18·2 | 0·41 | 48·1 | |
| | | +200 | 13·5 | 0·19 | 16·5 | |
| | | −200 | 52·7 | 0·03 | 10·2 | |
| 10 | −65 | +100 | 20·8 | 0·59 | 67·0 | 0·18 |
| | | +200 | 17·9 | 0·20 | 19·6 | |
| | | −200 | 6·13 | 0·04 | 13·4 | |
| 11 | −100 | +200 | 26·4 | 0·93 | 89·3 | 0·28 |
| | | −200 | 73·6 | 0·04 | 10·7 | |

### *Summary—Tests Nos. 7 to 11*

Head sample: Au 2·30 oz. per ton.

| Test No. | Amal. tail., Au, oz./ton | Cyan. tail., Au, oz./ton | Recovery, per cent | | Recovery, per cent total gold | | Recovery, per cent total | Reagents, lb./ton | | Time, hours |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | Amal. | Cyan. | Amal. | Cyan. | | KCN | CaO | |
| 7 | 0·86 | 0·32 | 62·5 | 62·8 | 62·5 | 23·6 | 86·1 | 0·74 | 4·1 | 47 |
| 8 | 0·73 | 0·30 | 68·2 | 58·8 | 68·2 | 18·7 | 86·9 | 0·73 | 4·3 | 46 |
| 9 | 0·76 | 0·16 | 67·0 | 79·0 | 67·0 | 26·1 | 93·1 | 0·76 | 4·4 | 45 |
| 10 | 0·98 | 0·18 | 57·4 | 81·6 | 57·4 | 34·7 | 92·1 | 0·70 | 5·5 | 43 |
| 11 | 0·56 | 0·28 | 75·5 | 50·0 | 75·5 | 12·3 | 87·8 | 0·67 | 5·5 | 42 |

## EXPERIMENTAL TESTS IN CONTINUOUS GRINDING UNIT

A number of tests were made in a small continuous grinding circuit, with a capacity of 25 to 30 pounds of ore per hour.

The ore dry crushed to −10 mesh was fed into a rod mill by a continuous belt feeder. The rod mill was in closed circuit with a drag classifier.

For amalgamation, 1,000 grammes of mercury was placed in the well of the classifier and the discharge from the classifier passed over an amalgamation plate 77 inches long with a $\frac{7}{8}$-inch cascade at the centre.

Cyanidation of the amalgamation tailings was done in a Pachuca tank. Capacity 100 pounds of ore at 1 : 3 dilution. The Pachuca tank in addition to being equipped with an air supply was in closed circuit with a 1-inch pump, which assured positive agitation.

Flotation of the amalgamation tailings was done in batch lots in a 2,500-gramme Fahrenwald machine.

Tabling of the amalgamation and flotation tailing was done on a laboratory-size Wilfley table.

### *Test No. 12*

Screen analyses of the plate tailing are as follows:

| Sample No. | Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au | Average plate tailing, Au oz./ton |
|---|---|---|---|---|---|
| 1 | +35<br>+48 | 1·5 | 0·42 | 1·3 | 0·49 |
| | +65 | 8·2 | 0·56 | 9·4 | |
| | +100 | 15·5 | 0·64 | 20·2 | |
| | +200 | 17·7 | 0·62 | 22·4 | |
| | −200 | 57·1 | 0·40 | 46·7 | |
| 2 | +35<br>+48 | 1·4 | 0·62 | 1·7 | 0·52 |
| | +65 | 6·5 | 0·63 | 7·9 | |
| | +100 | 18·7 | 0·54 | 19·4 | |
| | +200 | 17·8 | 0·58 | 19·8 | |
| | −200 | 55·6 | 0·48 | 51·2 | |

*Flotation of the Amalgamation Tailing—*

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 7·1 | 5·76 | 79·4 |
| " tailing | 92·9 | 0·114 | 20·6 |

The flotation concentrate assayed: Fe 20·56 per cent, As 3·53 per cent, Pb nil, Cu 0·0, Zn 1·37 per cent.

*Screen Analyses of the Flotation Tailing—*

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| +48 | 1·9 | 0·38 | 6·3 |
| +65 | 6·0 | 0·22 | 11·5 |
| +100 | 17·8 | 0·19 | 29·5 |
| +200 | 17·3 | 0·15 | 22·8 |
| −200 | 57·0 | 0·06 | 29·9 |

Average flotation tailing: Au 0·114 oz. per ton.

*Tabling of Flotation Tailing—*

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Table concentrate | 10·7 | 0·36 | 31·2 |
| Table middling | 16·5 | 0·15 | 20·0 |
| Table tailing | 72·8 | 0·083 | 48·8 |

*Screen Analyses of the Table Tailing—*

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent, of values, Au |
|---|---|---|---|
| +48 | 0·8 | 0·31 | 3·0 |
| +65 | 3·6 | 0·18 | 7·9 |
| +100 | 7·6 | 0·13 | 11·9 |
| +200 | 15·3 | 0·11 | 20·3 |
| −200 | 26·8 | 0·09 | 29·1 |
| Slimes | 45·9 | 0·05 | 27·8 |

Average table tailing: Au 0·083 oz. per ton.

*Tabling the Amalgamation Tailing—*

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Table concentrate | 5·7 | 4·04 | 42·4 |
| Table middling | 7·6 | 0·53 | 7·4 |
| Table tailing | 86·7 | 0·314 | 50·2 |

*Screen Analyses of the Table Tailing—*

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| +48 | 1·5 | 0·22 | 1·0 |
| +65 | 5·8 | 0·27 | 5·0 |
| +100 | 14·9 | 0·37 | 17·6 |
| +200 | 15·0 | 0·40 | 19·1 |
| −200 | 31·8 | 0·40 | 40·5 |
| Totals | 31·0 | 0·17 | 16·8 |

Average table tailing: Au 0·314 oz. per ton.

*Two-stage Flotation of the Amalgamation Tailing*

To plate tailing, $Na_2CO_3$ 1·45 pounds per ton, Tarol 0·09 pounds per ton, No. 34 fuel oil 0·08 pounds per ton, No. 323 0·14 pounds per ton, pine oil No. 5 0·03 pounds per ton, No. 1 concentrate was removed.

Tailings were then deslimed and oversize reground with $Na_2CO_3$ 2·0 pounds per ton, Tarol 0·08 pounds per ton, No. 34 fuel oil 0·08 pounds per ton. To cell amyl xanthate 0·08 pounds per ton. No. 2 concentrate removed.

*Results—*

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| No. 1 concentrate | 7·8 | 6·36 | 75·2 |
| No. 2 concentrate | 6·4 | 1·36 | 13·2 |
| Slime product | 25·6 | 0·06 | 2·3 |
| Tailing +100 | 4·2 | 0·33 | 2·1 |
| " +200 | 17·2 | 0·12 | 3·1 |
| " −200 | 38·8 | 0·07 | 4·1 |

Slime product contained 0·10 per cent on 200 mesh.
Head sample for test: Au 2·15 oz. per ton.
Unit run 4·25 hours before sampling.
Amalgamation plate cleaned every two hours.

*Summary of Recoveries—*

Amalgamation (average) 76·5 per cent.
Amalgamation and flotation 76·5+18·7 =95·2 per cent.
Amalgamation, flotation and tabling 95·2+1·5 =96·7 per cent.
Amalgamation and tabling 76·5+10·00 =86·5 per cent.
Amalgamation and flotation (two stage) 76·5 +20·8 =97·3 per cent.

## AMALGAMATION—CYANIDATION OF THE AMALGAMATION TAILING

### *Test No. 13*

Screen analyses of the plate tailing gave the following results:

| Sample No. | Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au | Average tailing, Au oz./ton |
|---|---|---|---|---|---|
| 1.............................. | +48 | 2·1 | 0·41 | 1·7 | 0·497 |
| | +65 | 7·4 | 0·50(·50) | 7·5 | |
| | +100 | 18·1 | 0·56 | 20·4 | |
| | +200 | 19·8 | 0·68 | 27·1 | |
| | −200 | 52·6 | 0·41 | 43·3 | |
| 2.............................. | +35 | 0·3 | 0·88 | 0·5 | 0·548 |
| | +48 | 2·0 | 0·60 | 2·2 | |
| | +65 | 7·9 | 0·55 | 7·9 | |
| | +100 | 14·7 | 0·66 | 17·7 | |
| | +200 | 19·0 | 0·70 | 25·4 | |
| | −200 | 55·2 | 0·46 | 46·3 | |

### *Cyanidation of the Amalgamation Tailing*

*Screen Analyses of Cyanide Tailing—*

| Sample No. | Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au | Average tailing, Au oz./ton |
|---|---|---|---|---|---|
| 1.............................. | +48 | 0·8 | 0·13 | 1·3 | 0·079 |
| | +65 | 12·2 | 0·12 | 18·4 | |
| | +100 | 21·2 | 0·11 | 29·3 | |
| | +200 | 18·9 | 0·09 | 21·4 | |
| | −200 | 46·9 | 0·05 | 29·6 | |
| 2... ........................... | +48 | 1·0 | 0·14 | 2·0 | 0·071 |
| | +65 | 17·2 | 0·10 | 24·3 | |
| | +100 | 15·9 | 0·11 | 21·9 | |
| | +200 | 20·7 | 0·09 | 26·3 | |
| | −200 | 45·2 | 0·04 | 25·5 | |

Head sample for test: Au 1·84 oz. per ton.
Unit run 4 hours 5 minutes before sampling.
Pulp density (cyanidation) 1 : 2·6.
KCN per cent 0·076.
Time of agitation 36 hours.

*Summary of Recoveries—*

Amalgamation—average 71·7 per cent.
Cyanidation heads (plate tails Au 0·523 oz./ton) Tails average Au 0·075 oz./ton.
Extraction 85·7 per cent.
Total recovery—amalgamation 71·7 + cyanidation 24·2 =95·9 per cent.

### *Flotation of the Cyanide Tailings*

For the purpose of determining where the gold values remaining after cyanidation were, a sample of the cyanide tailings was floated.

*Results—*

| Product | Weight, per cent | Assay, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate +100 | 0·4 | 0·61 | 3·7 |
| " " −100 | 6·0 | 0·40 | 37·0 |
| Flotation tailing average | 93·6 | 0·041 | 59·3 |

*Screen Analyses of the Flotation Tailing—*

| Product | Weight, per cent | Assay, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +48 | 0·7 | 0·08 | 1·5 |
| +65 | 9·4 | 0·10 | 22·8 |
| +100 | 18·2 | 0·09 | 39·8 |
| +200 | 19·0 | 0·05 | 23·0 |
| −200 | 52·7 | 0·01 | 12·9 |

Average flotation tailing (after cyanidation) Au 0·041 oz. per ton.

From the preceding test it is evident that the sulphides contain considerable values in gold after amalgamation and cyanidation.

### *Test No. 14*

This test was subdivided into four sub-tests namely:

(1) Amalgamation—Cyanidation of the amalgamation tailing.
(2) Amalgamation—Flotation of the amalgamation tailing, and tabling the flotation tailing.
(3) Amalgamation—Flotation of the amalgamation tailing.
(4) Amalgamation—Tabling the amalgamation tailing.

*Amalgamation—*

| Sample No. | Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au | Average tailing, Au oz./ton |
|---|---|---|---|---|---|
| 1 | +65 | 1·2 | 2·25 | 4·2 | 0·64 |
| | +100 | 3·0 | 1·86 | 8·8 | |
| | +200 | 15·6 | 0·92 | 22·6 | |
| | −200 | 80·2 | 0·51 | 64·4 | |
| 2 | +65 | 1·2 | 1·00 | 2·3 | 0·53 |
| | +100 | 3·3 | 1·10 | 6·9 | |
| | +200 | 15·0 | 0·84 | 23·8 | |
| | −200 | 80·5 | 0·44 | 67·0 | |

*Cyanidation of the Amalgamation Tailing—*

| Sample No. | Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au | Average tailing, Au oz./ton |
|---|---|---|---|---|---|
| 1 | +65 | | | | |
| | +100 | 0·9 | 0·03 | 0·8 | 0·036 |
| | +200 | 19·4 | 0·06 | 32·4 | |
| | −200 | 79·7 | 0·03 | 66·8 | |
| 2 | −65 | | | | |
| | +100 | 2·2 | 0·025 | 1·8 | 0·033 |
| | +200 | 16·2 | 0·05 | 24·4 | |
| | −200 | 81·6 | 0·03 | 73·8 | |

*Cyanidation Details—*

Pulp density 1 : 2·5.
KCN per cent, 0·08.
Time of agitation, 38½ hours.
Unit run 4 hours before sampling.

## FLOTATION OF THE CYANIDE TAILINGS

As in Test No. 13 a sample of the cyanide tailings was floated.

*Results—*

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 18·5 | 0·24 | 87·4 |
| Tailing +65 } +100 } | 1·6 | 0·03 | 1·0 |
| " +200 | 13·2 | 0·02 | 5·1 |
| " −200 | 66·7 | 0·005 | 6·5 |

## FLOTATION OF THE AMALGAMATION TAILING, AND TABLING THE FLOTATION TAILING

*Results—*

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 10·4 | 5·48 | 85·9 |
| Table " | 11·5 | 0·38 | 6·6 |
| Table tailing +100 | 0·8 | 0·13 | 0·2 |
| " " +200 | 6·8 | 0·10 | 1·0 |
| " " −200 | 29·8 | 0·07 | 3·2 |
| " slimes | 40·7 | 0·05 | 3·1 |

Recovery, 85·9+6·6=92·5. per cent

*Flotation of the Amalgamation Tailing—*

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 13·6 | 4·12 | 88·6 |
| Tailing +65 | 0·9 | 0·50 | 0·7 |
| " +100 | 4·3 | 0·30 | 2·0 |
| " +200 | 13·2 | 0·16 | 3·3 |
| " −200 | 68·0 | 0·05 | 5·4 |

Recovery, 88·6 per cent.

*Tabling the Amalgamation Tailing—*

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Table concentrate | 11·6 | 3·06 | 66·4 |
| Tailing +100 | 1·2 | 0·39 | 0·9 |
| " +200 | 7·5 | 0·23 | 3·2 |
| " −200 | 38·5 | 0·27 | 19·5 |
| Slimes | 41·2 | 0·13 | 10·0 |

Recovery, 66·4 per cent.

*Summary—*

Head sample: Au 1·94 oz. per ton.
Amalgamation (average), 69·9 per cent.
Amalgamation and cyanidation ...................... 69·9+28·3=98·2 per cent
Amalgamation-flotation and tabling.................. 69·9+27·8=97·7 "
Amalgamation-flotation.............................. 69·9+26·7=96·6 "
Amalgamation-tabling................................ 69·9+20·0=89·9 "

## AMALGAMATION—FLOTATION OF THE AMALGAMATION TAILING AND CYANIDATION OF THE FLOTATION CONCENTRATES

### *Test No. 15*

Head sample: Au 2·02 oz. per ton.

| Sample No. | Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au | Average tailing, Au, oz./ton |
|---|---|---|---|---|---|
| 1 | +65 | 3·6 | 0·62 | 4·7 | 0·47 |
| | +100 | 9·8 | 0·80 | 16·6 | |
| | +200 | 20·2 | 0·66 | 28·2 | |
| | −200 | 66·4 | 0·36 | 50·5 | |
| 2 | +65 | 3·0 | 0·71 | 4·0 | 0·53 |
| | +100 | 10·4 | 0·96 | 18·7 | |
| | +200 | 20·2 | 0·76 | 28·5 | |
| | −200 | 66·6 | 0·39 | 48·8 | |

*Flotation of the Amalgamation Tailing—*

| Sample No. | Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au | Average tailing Au, oz./ton |
|---|---|---|---|---|---|
| 1 | Flotation concentrate.............. | 7·7 | 5·40 | 80·7 | |
| | Tailing +65...................... | 6·5 | 0·30 | 3·8 | |
| | " +100...................... | 14·4 | 0·17 | 4·8 | |
| | " +200...................... | 13·7 | 0·15 | 4·0 | |
| | " −200...................... | 57·7 | 0·06 | 6·7 | |

Flotation concentrate assayed: As 3·95 per cent, Fe 22·30 per cent.

*Cyanidation of the Flotation Concentrates*

Part A—Without regrinding.
Part B—After regrinding.
Part A—Three parts, I-II-III.

*Screening the Cyanide Tailings—*

| Sample No. | Product | Weight, per cent | Assays, Au, oz./ton | Per cent o values, Au | Average tailing, Au, oz./ton |
|---|---|---|---|---|---|
| A-I........................... | +100 | 3·3 | 2·73 | 20·6 | 0·44 |
| | +200 | 6·7 | 1·70 | 26·0 | |
| | −200 | 90·0 | 0·26 | 53·4 | |
| A-II........................... | +100 | 3·0 | 1·60 | 15·1 | 0·32 |
| | +200 | 5·5 | 1·40 | 24·3 | |
| | −200 | 91·5 | 0·21 | 60·6 | |
| A-III........................... | +200 | 6·7 | 1·44 | 27·0 | 0·36 |
| | −200 | 93·3 | 0·28 | 73·0 | |
| B-I........................... | +200 | 2·2 | 1·12 | 9·5 | 0·26 |
| | −200 | 97·8 | 0·24 | 90·5 | |
| B-II........................... | +200 | 4·4 | 1·00 | 15·0 | 0·29 |
| | −200 | 95·6 | 0·26 | 85·0 | |
| B-III........................... | +200 | 3·2 | 1·08 | 11·3 | 0·31 |
| | −200 | 96·8 | 0·28 | 88·7 | |

*Summary—*

| No. | Dil. | KCN, per cent | Time | Heads, Au, oz./ton | Tails, Au, oz./ton | Extr. per cent | Reagents | | | |
|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | Per ton con. | | Per ton ore | |
| | | | | | | | KCN | CaO | KCN | CaO |
| 15-A- I............ | 1 : 3·2 | 0·10 | 48 | 5·40 | 0·44 | 91·8 | 5·5 | 21·8 | 0·42 | 1·7 |
| 15-A- II............ | 1 : 3·4 | 0·20 | 48 | 5·40 | 0·32 | 94·0 | 5·2 | 18·8 | 0·40 | 1·5 |
| 15-A-III............ | 1 : 3·3 | 0·20 | 72 | 5·40 | 0·36 | 93·3 | 7·6 | 21·0 | 0·59 | 1·6 |
| 15-B- I............ | 1 : 3·4 | 0·10 | 48 | 5·70 | 0·26 | 95·4 | 3·9 | 21·2 | 0·28 | 1·5 |
| 15-B- II............ | 1 : 3·4 | 0·20 | 48 | 5·70 | 0·29 | 94·8 | 3·9 | 23·6 | 0·28 | 1·7 |
| 15-B-III............ | 1 : 3·5 | 0·20 | 72 | 5·70 | 0·31 | 94·5 | 5·1 | 21·4 | 0·37 | 1·5 |

*Recoveries—*

| | |
|---|---|
| Amalgamation | 76·4 per cent |
| Flotation | 80·5 " |
| Cyanidation, (average, omitting 15-A-I) | 94·4 " |
| Total net recovery | 94·4 " |

## CONCLUSIONS

Owing to the considerable quantity of coarse free gold present no tests were made in which amalgamation was not the primary step.

In the continuous tests, grinding to 1·5 per cent on 35 and 48 mesh, Test No. 12, gave somewhat better results than in Test No. 14 where the ore was ground to 1·2 per cent on 65 mesh.

Test No. 12 gave a recovery of 76 per cent of the gold, a result which would probably be attained if not bettered in stamp battery crushing to pass 30-mesh, and using both inside and outside amalgamation.

Where cyanidation of the amalgamation tailings was practised a better net recovery was obtained on the more finely ground ore. The lower recovery by amalgamation was more than offset by the higher extraction in cyanidation.

In practice amalgamation followed by concentration and cyanidation of the concentrates would probably prove more economical than the cyanidation of the entire plate tailings. The concentrates cyanided very well and showed a saving in reagents, in pounds per ton of ore, over the cyanidation of the amalgamation tailings.

Results from the flotation of the cyanide tailings showed that the gold remaining is mainly associated with the sulphides of iron and arsenic, and that if cyanidation of the concentrates were to be attempted consideration should be given to the regrinding of the concentrates before cyanidation.

## Report No. 361

## THE CONCENTRATION OF THE COBALT ORES OF THE AGAUNICO COBALT MINES, LIMITED, COBALT, ONTARIO

**A. K. Anderson and D. S. Halford**

*Shipment.* A shipment of 8 bags of ore, gross weight 594 pounds, was received at the Ore Dressing Laboratories, Ottawa, on March, 17, 1920, consigned by D. I. Jemmett of the Agaunico Cobalt Mines, Limited, Cobalt, Ontario.

*Characteristics of the Ore.* The ore consisted of cobalt minerals, cobaltite, smaltite, and erythrite (cobalt bloom) in a diabase gangue with small amounts of calcite. Of the cobalt minerals cobaltite predominates.

*Purpose of Experimental Tests.* The purpose of the tests was to determine the effects of flotation on the recovery of cobalt minerals. The practice at the property is to concentrate on tables with an apparent loss of very fine slimes and oxidized cobalt.

*Sampling and Analysis.* The entire lot was crushed to ¼-inch mesh and passed through a Jones riffle sampler. One-quarter of the lot was stage crushed through finer mesh screens with alternate quartering until a representative sample was secured for assay.

This showed the shipment to contain,—

| | | | |
|---|---|---|---|
| Cobalt | 2·60 per cent | Silver | 0·22 oz./ton |
| Nickel | 0·60 " | Gold | 0·02 " |

All test work was along the lines of concentration by flotation, unless otherwise indicated. The ore was ground 88 per cent through 200 mesh, and floated with reagents as shown in the individual tests. Various combinations of these were tried in order to determine the most suitable. Two thousand gramme representative portions of the material crushed to pass a 14 mesh screen were ground to the required fineness in a Denver ball mill containing iron balls.

## EXPERIMENTAL WORK

### *Test No. 1*

Two thousand grammes of ore −14 mesh, ground in a natural circuit, no reagents being added.

After grinding, the pulp was transferred to a mechanically agitated, flotation machine where 0·05 pound potassium xanthate and 0·12 pound cresylic acid per ton were added.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Heads (calculated) | 100·0 | 2·45 | 0·74 | 100·0 | 100·0 |
| Concentrate | 3·47 | 4·92 | 0·88 | 6·97 | 4·1 |
| Tailing | 96·53 | 2·36 | 0·74 | 93·03 | 95·9 |

Flotation in a natural circuit is not successful. A low-grade cobalt concentrate results and a very low recovery.

### *Test No. 2*

The ore was ground in water containing 8·0 pounds soda ash per ton of ore, and floated with 0·20 pounds per ton potassium xanthate. Pine oil at the rate of 0·08 pound per ton was added to froth.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Heads (calculated) | 100·0 | 2·42 | 0·84 | 100·0 | 100·0 |
| Concentrate | 6·31 | 9·55 | 5·05 | 24·9 | 37·8 |
| Tailing | 93·69 | 1·94 | 0·56 | 75·1 | 62·2 |

Flotation under these conditions yields a low recovery.

### *Test No. 3*

The effect of lime as alkaline reagent was investigated, 5·0 pounds per ton being added to the grinding circuit, and the pulp floated with 0·35 pound potassium xanthate and 0·08 pound pine oil per ton.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Heads (calculated) | 100·0 | 2·55 | 0·75 | 100·0 | 100·0 |
| Concentrate | 4·67 | 7·50 | 2·00 | 13·7 | 12·4 |
| Tailing | 95·33 | 2·31 | 0·69 | 86·3 | 87·6 |

Flotation conditions on this test were poor, the froth was thin and slow floating. Recoveries were very low.

### *Test No. 4*

The ore was ground with 8·0 pounds soda ash per ton. The pulp was then treated with 1 pound per ton sodium sulphide and floated with 0·20 pound per ton potassium xanthate and 0·08 pound pine oil.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Heads (calculated) | 100·0 | 2·40 | 0·77 | 100·0 | 100·0 |
| Concentrate | 7·03 | 13·88 | 4·96 | 40·5 | 44·9 |
| Tailing | 92·97 | 1·54 | 0·46 | 59·5 | 35·1 |

The use of sodium sulphide appears to be of benefit, raising the grade of concentrate produced and increasing the recovery.

### *Test No. 5*

The effect of grinding the ore with 0·15 pounds thiocarbanilide per ton was studied. Soda ash, 8·0 pounds per ton was added to the grinding circuit, and 0·12 pounds cresylic acid used in the flotation cells.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Heads (calculated) | 100·0 | 2·72 | 0·85 | 100·0 | 100·0 |
| Concentrate | 7·60 | 17·20 | 5·60 | 48·0 | 49·5 |
| Tailing | 92·40 | 1·53 | 0·47 | 52·0 | 50·5 |

Thiocarbanilide and cresylic acid appear to be more effective flotation reagents than potassium xanthate and pine oil.

### *Test No. 6*

In this test, the ore was ground with 10 pounds soda ash per ton, treated with 1·0 pound sodium sulphide and floated with 0·20 pound potassium xanthate and 0·08 pound pine oil per ton.

After flotation, the tailing from this process was passed over a Wilfley table and an additional concentrate removed.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ag, oz./ton | Co | Ni |
| Heads (calculated) | 100·0 | 2·68 | 0·19 | 100·0 | 100·0 |
| Flotation concentrate | 6·4 | 22·26 | 1·24 | 53·2 | 41·7 |
| Flotation middling | 9·9 | 7·20 | 0·38 | 26·6 | 19·3 |
| Table concentrate | 1·0 | 6·10 | 0·70 | 2·3 | 3·7 |
| Table tailing | 82·7 | 0·58 | 0·08 | 17·9 | 34·8 |

Flotation under these conditions yields a recovery of 79·8 per cent of the cobalt in the combined concentrate and middling. On cleaning, a product analysing 22·26 per cent cobalt is secured. A low-grade table concentrate is secured representing an additional 2·3 per cent recovery of the cobalt.

### *Test No. 7*

In this test the reagent Flotagen was used for the flotation of the cobalt minerals: 3,000 grammes of the ore at −14 mesh was ground for 30 minutes in the ball mill with 2,000 grammes of water. This resulted in about 80 per cent through 200 mesh. Reagents as listed below were added. After grinding it was floated in a mechanically agitated machine at a dilution of 3 : 1. First froth was removed for 5 minutes when copper sulphate was added and a further froth removed for 2 minutes. The rougher froth was cleaned.

*Reagents*—

| To ball mill— | | To flotation machine— | |
|---|---|---|---|
| Soda ash | 7·0 lb. per ton | Flotagen | 0·10 lb. per ton |
| Flotagen | 0·05 " | Pine oil | 0·25 " |
| | | Copper sulphate | 1·0 " |

| Product | Weight, per cent | Assay | Per cent of values |
|---|---|---|---|
| | | Co, per cent | Co |
| Concentrate | 11·93 | 17·76 | 83·67 |
| Middling | 10·16 | 1·54 | 16·18 |
| Tailing | 77·91 | 0·33 | 10·15 |
| Heads (calculated) | 100·0 | 2·53 | 100·00 |

The use of Flotagen results in much better recoveries of cobalt than reagents used in preceding tests.

The results secured in these tests indicate that much higher recoveries can be obtained by concentration by flotation than by table concentration as practised at the property.

Additional investigations were carried on by D. S. Halford, Canadian Industries, Limited.

## AGAUNICO SHIPMENT No. 2

**D. S. Halford**

The second shipment of ore from the Agaunico mine was received on May 20. It consisted of two separate samples on which test work was to be done to see if they would respond to the same flotation reagents as did the former shipment.

One sample consisted of about 10 pounds of tailing from table concentration. It was made up of a number of small bags of assay rejects. The original tailing had been dried and then pulverized to −200 mesh for assay purposes. It was obviously a poor sample for flotation test work. The sample is designated Lot No. 1, Shipment No. 2.

The second sample consisted of about 25 pounds of comparatively high-grade disseminated ore picked from the bumping table. The individual pieces averaged about 1 inch in size and were crushed to −20 mesh for test purposes. This sample was designated Lot No. 2, Shipment No. 2.

The results of the sample Lot No. 1 were rather disappointing as only 70·1 per cent of the cobalt present was floated in the concentrates and middlings. Since only a trace of cobalt could be discerned in the tails by either tabling or panning it is quite likely a large part is present in the form of oxide, which is to be expected in a sample that was prepared for assaying rather than flotation.

### Experimental Work

### *Lot No. 1*

No further grinding was done on this test: 3,000 grammes of ore was taken and floated at 2 : 1 dilution in the Fahrenwald machine. The following reagents were added: 10·5 grammes of soda ash, 0·1 gramme of Flotagen, 1 gramme of copper sulphate, and 4 drops of pine oil; and the rough concentrates were cleaned.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Concentrate | 4·94 | 11·66 | 1·62 | 55·49 | 52·77 |
| Middling | 15·48 | 0·98 | trace | 14·61 | ........... |
| Tailing | 79·58 | 0·39 | 0·09 | 29·90 | 47·23 |

### *Lot No. 2*

In this test 1,500 grammes of −20 mesh ore was ground in an Abbé jar for 30 minutes with 6 grammes of soda ash, 0·05 gramme of Flotagen, and 1,000 c.c. of water. It was floated in a Ruth machine with the addition of 0·75 gramme of copper sulphate, 2 drops of pine oil, and 0·05 gramme of Flotagen. The flotation concentrates were cleaned and the tailing tabled.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Flotation concentrate............ | 49·30 | 21·72 | 5·08 | 92·24 | 96·03 |
| Flotation middling.............. | 16·11 | 3·67 | 0·57 | 5·09 | 3·52 |
| Table concentrate............... | 1·28 | 6·78 | 0·92 | 0·75 | 0·45 |
| Tailing.......................... | 33·31 | 0·67 | trace | 1·92 | ........... |

The tests incorporated in this section were performed with the assistance of Mr. C. O'Shaughnessy of the Agaunico staff and with the exception of Test No. 1 the samples used were brought by him from the mine in Cobalt. The sample used in Test No. 1 was from Lot 2, Shipment 2, while the others were types of ore and discard products of jigs encountered in the present treatment at the mine.

In Test No. 2 the sample used was reject from the fine jig; in Test No. 3 rejects from the coarse jig; in Test No. 4 hutch and fine screenings from the bumping table; in Test No. 5 a high-grade cobalt-copper sample; in Test No. 6 an ore to be re-treated by bumping tables and jigs; in Test No. 7 high-grade disseminated ore; in Test No. 8 a cobaltite-niccolite sample; and in Test Nos. 9 and 10 a composite sample made from Test Nos. 2, 3, 4, 6, and 7.

As samples were wet upon arrival they were dried in an oven and crushed to −20 mesh. No head samples were cut out. The object of the test being to see if the samples would respond to the same treatment as the original shipment (Lot 1) the same reagents were used in all tests with the exception of Tests Nos. 5, 8, and 10 where differential floats were attempted with the aid of sodium cyanide.

### *Test No. 1*

This test was performed to see how a coarse sample reacted. Excellent results were obtained on the ore of Lot 2, Shipment 2, when ground to approximately 80 per cent through 200 mesh. In this test screening of the tailing showed only 31 per cent through 200 mesh.

1,500 grammes of ore were ground 5 minutes in an Abbé mill with 8 grammes of soda ash, 0·05 gramme of Flotagen, 0·75 gramme copper sulphate, and 1,000 c.c. of water. It was floated in a Ruth machine, 3 drops of pine oil and 0·075 gramme of Flotagen being added at the machine; 0·5 gramme of copper sulphate was also added but this and the last 0·05 gramme of Flotagen had no visible effect. The rough concentrate was cleaned. While results were fair they show clearly the necessity for finer grinding when compared to results of Lot 2, Shipment 2.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Concentrate | 27·88 | 22·1 | 4·38 | 56·52 | 64·08 |
| Middling | 20·77 | 15·16 | 2·06 | 28·88 | 22·45 |
| Tailing | 51·35 | 3·10 | 0·50 | 14·60 | 13·47 |

### *Test No. 2*

The sample consisted of rejects from the fine jig; 1,500 grammes of −20 mesh ore were ground 30 minutes in an Abbé jar with 1,000 c.c. of water, 6 grammes of soda ash, and 0·05 gramme of Flotagen. It was floated in the Ruth machine at a dilution of 2 : 1 with the addition of 0·75 gramme of copper sulphate, 2 drops of pine oil, and 0·025 gramme of Flotagen. The rougher concentrates were cleaned in the Ruth with the addition of 1 drop of pine oil.

A screen test showed the grinding to be:

| | Per cent |
|---|---|
| + 65 mesh | 0·5 |
| +100 mesh | 4·8 |
| +150 mesh | 12·9 |
| +200 mesh | 11·6 |
| −200 mesh | 70·2 |

This analysis will hold approximately true for Tests Nos. 2 to 8 inclusive.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Concentrate | 9·19 | 17·07 | 3·69 | 92·37 | 86·94 |
| Middling | 9·75 | 0·33 | 0·19 | 1·90 | 4·75 |
| Tailing | 81·06 | 0·12 | 0·04 | 5·73 | 8·31 |

### *Test No. 3*

The sample consisted of rejects from the coarse jig; 1,500 grammes of ore was ground 30 minutes in an Abbé jar with 1,000 c.c. of water, 6 grammes of soda ash, 0·05 gramme of Flotagen, and 0·75 gramme of copper sulphate. It was floated in the Ruth machine with a dilution of 2 : 1 with the addition of 2 drops of pine oil, and 0·025 gramme of Flotagen.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Concentrate | 6·67 | 16·08 | 2·52 | 75·88 | 86·16 |
| Middling | 10·81 | 0·20 | 0·25 | 0·76 | 13·84 |
| Tailing | 82·52 | 0·40 | trace | 23·36 | ........... |

### *Test No. 4*

This was a comparatively high-grade sample consisting of hutch and fine screenings from the bumping table, and containing 8·3 per cent cobalt.

A sample of 1,500 grammes was ground 30 minutes in an Abbé jar with 1,000 c.c. water and 0·75 gramme of copper sulphate, 0·05 gramme of Flotagen and 0·75 gramme of copper sulphate. It was floated in the Ruth machine at a dilution of 2 : 1 with the addition of 3 drops of pine oil and 0·025 gramme of Flotagen.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Concentrate | 31·49 | 20·25 | 7·15 | 76·42 | 77·99 |
| Middling | 11·96 | 8·30 | 2·80 | 11·91 | 11·62 |
| Tailing | 56·55 | 1·72 | 0·53 | 11·67 | 10·39 |

### *Test No. 5*

The ore was a mixture of cobaltite and chalcopyrite in intimate association, and could be regarded as high grade. As there was only about 1,000 grammes of this material 500 grammes of reject from the coarse jigs was added. An attempt was made to float first a high-grade copper concentrate low in cobalt, and then a high-grade cobalt concentrate. Difficulty was experienced in the copper assays owing to interfering elements, and a special method of analysis will have to be worked out.

Approximately 1,500 grammes of the ore was ground in an Abbé jar for 30 minutes with 1,000 c.c. of water, 7 grammes of soda ash, and 0·2 gramme of sodium cyanide. It was floated in a Ruth machine, the first froth containing most of the copper being floated with 1 drop of pine oil and 0·01 gramme of Flotagen. This froth was cleaned. An additional 0·75 gramme of copper sulphate and 0·075 gramme of Flotagen was then added and the cobalt concentrate, called Concentrate No. 2, removed.

The cobalt minerals seem to be quite sensitive to cyanide and the copper readily separated by flotation.

| Product | Weight, per cent | Assay | | | Per cent of values | |
|---|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Cu, per cent | Co | Ni |
| Concentrate No. 1 | 13·23 | 3·70 | 1·10 | 25·12 | 5·19 | 4·79 |
| Middling | 8·66 | 13·32 | 4·20 | ........... | 12·17 | 11·99 |
| Concentrate No. 2 | 36·41 | 19·78 | 6·34 | 1·52 | 76·01 | 76·07 |
| Tailing | 41·70 | 1·51 | 0·52 | ........... | 6·65 | 7·15 |

## *Test No. 6*

The sample consisted of ore to be re-treated by bumping tables and jigs. Ground 1,500 grammes in the Abbé jar for 30 minutes with 1,000 c.c. of water, 6 grammes of soda ash, and 0·05 gramme of Flotagen. Added 0·7 gramme of copper sulphate at machine in two portions of 0·35 gramme each, the second portion being added after the froth had been removed for 3 minutes.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni per cent | Co | Ni |
| Concentrate | 17·47 | 16·42 | 9·90 | 71·80 | 92·88 |
| Middling | 11·42 | 5·26 | 1·16 | 15·03 | 7·12 |
| Tailing | 71·11 | 0·74 | trace | 13·17 | ........... |

## *Test No. 7*

The sample consisted of disseminated ore, being much the same as Lot No. 2, Shipment No. 2. Ground 1,500 grammes of ore in an Abbé jar with 1,000 c.c. water, 6 grammes soda ash, 0·75 gramme copper sulphate, and 0·05 gramme Flotagen at the machine. Cleaned the concentrates with addition of 0·01 gramme Flotagen.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Concentrate | 26·90 | 26·00 | 2·40 | 83·03 | 75·79 |
| Middling | 12·38 | 5·12 | 0·93 | 7·53 | 13·51 |
| Tailing | 60·72 | 1·31 | 0·15 | 9·44 | 10·70 |

## *Test No. 8*

In this test an attempt was made to make a separation of cobaltite and niccolite in a sample that ran 6·8 per cent cobalt and 13·3 per cent nickel. Although the concentrates looked bluish grey and the tails had the coppery niccolite appearance, assays showed that very little separation was made.

Ground 1,500 grammes of ore with 7 grammes of soda ash, 0·2 gramme of cyanide, and 1,000 c.c. of water for 30 minutes. At the flotation machine 2 drops of pine oil and 0·75 gramme of copper sulphate were added. This was agitated 15 minutes and 0·05 gramme of Flotagen added. The concentrates were not cleaned.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Concentrate | 71·22 | 7·98 | 23·88 | 83·15 | 68·5 |
| Tailing | 28·78 | 4·00 | 27·08 | 16·85 | 31·5 |

### *Test No. 9*

The ore for this test was obtained by mixing thoroughly 4,000 grammes of coarse jig product, 3,000 grammes of fine jig product, 1,000 grammes of hutch and screen product, 1,000 grammes of bumping table product, and 1,000 grammes of disseminated ore. Tests Nos. 9 and 10 were taken from this mixed product.

Screening of the tailing from Test No. 9 showed 87·3 per cent through 200 mesh. In spite of this, results of test show poorer recovery than was obtained from any of the individual samples going to make it up.

Ground 3,000 grammes of ore for 30 minutes in a Denver mill with 2,000 grammes of water, 10·5 grammes of soda ash, 1·5 grammes copper sulphate, and 0·1 gramme Flotagen.

It was floated in the Fahrenwald machine for 10 minutes with the addition of 3 drops of pine oil and 0·1 gramme of Flotagen. Concentrates were cleaned in a Fahrenwald machine and recleaned in a Ruth.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Co | Ni |
| Concentrate | 12·53 | 17·48 | 6·12 | 67·40 | 80·71 |
| Middling | 17·39 | 1·98 | 0·53 | 10·60 | 9·70 |
| Tailing | 70·09 | 1·02 | 0·13 | 22·00 | 9·59 |

### *Test No. 10*

In this test an attempt was made to make three finished products, a copper concentrate, a cobalt concentrate, and a nickel concentrate.

A lot, 3,000 grammes of ore, was ground 30 minutes in a Denver mill with 2,000 c.c. of water, 10·5 grammes of soda ash, and 0·3 gramme of cyanide. Copper was removed with 2 drops of pine oil and cleaned in a Ruth machine with the addition of 1 drop of pine oil; 2 grammes of copper sulphate was added to the original float and agitated 5 minutes; 0·1 gramme of Flotagen was added and cobalt concentrate removed for about 5 minutes. By this time the mineral floating had begun to look reddish and was assumed to be niccolite. Apparently this assumption was wrong. Cobalt concentrate was not cleaned. In an attempt to remove the nickel the pulp was made neutral by the addition of 3 c.c. of sulphuric acid. One drop of hardwood creosote and 5 drops of Dupont Reagent No. 89 were added and concentrate called nickel concentrate removed. This was cleaned in the Ruth machine.

Copper assay for the cobalt concentrate was not obtained owing to interfering elements.

| Product | Weight, per cent | Assay | | | Per cent of values | |
|---|---|---|---|---|---|---|
| | | Co, per cent | Ni, per cent | Cu, per cent | Co | Ni |
| Copper concentrate | 0·77 | 1·98 | 0·74 | 24·24 | 0·48 | 0·73 |
| Middling | 1·17 | 2·39 | 0·67 | 1·26 | 0·88 | 1·00 |
| Cobalt concentrate | 12·36 | 16·58 | 4·22 | .......... | 64·34 | 66·85 |
| Nickel concentrate | 1·89 | 14·29 | 3·33 | 0·44 | 8·48 | 8·06 |
| Middling | 10·84 | 2·47 | 0·47 | 0·08 | 8·41 | 6·53 |
| Tailing | 72·97 | 0·76 | 0·18 | 0·13 | 17·41 | 16·83 |

## Report No. 362

## THE CONCENTRATION OF SILVER IN THE ORE OF THE ANIMIKIE MINES, LTD., PORT ARTHUR, ONTARIO

**A. K. Anderson**

*Shipments.* Two shipments of ore, gross weight 390 pounds, the first contained in four sacks and the second in two sacks were received at the Laboratories on March 17, 1930 and April 2, 1930. These came from the Animikie Mines, Ltd., 8 Cumberland Street South, Port Arthur, Ontario; S.W. Ray, Secretary, E. C. Tripp, President.

*Characteristics of the Ore.* The ore consisted of a limestone and calcite gangue carrying a small amount of iron sulphide. The silver is present chiefly as the ruby silver, proustite.

*Purpose of Experimental Tests.* The shipments were made for the purpose of determining the best method of recovering the contained values. The two lots were combined for testing.

*Sampling and Analysis.* The combined lots were stage crushed and quartered by passing through a Jones riffle sampler until a representative portion was secured for analysis. This showed the shipment to contain 46·0 oz. silver and 0·02 oz. gold per ton.

### EXPERIMENTAL TESTS

Most of the tests were made to determine suitable flotation reagents to use for the concentration of the silver in the ore. Some few cyanide tests were also carried out to note what recoveries could be obtained by this process.

All flotation tests were made on 1,000-gramme portions of the ore crushed to pass a 14-mesh screen, ground with water in a porcelain mill containing iron balls to approximately 80 per cent through 200 mesh. The pulp was then floated in a mechanically agitated flotation machine with reagents as noted in the tests.

### *Test No. 1*

*Reagents in Pounds per Ton—*

| | | | |
|---|---|---|---|
| To ball mill— | | To flotation machine— | |
| Soda ash | 4·0 | Potassium ethyl xanthate | 0·14 |
| | | Pine oil | 0·06 |

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Au, oz./ton | Ag, oz./ton | Au | Ag |
| Concentrate | 6·04 | 0·28 | 684·00 | 84·6 | 88·0 |
| Tailing | 93·96 | trace | 6·00 | 15·4 | 12·0 |
| Heads (calculated) | | 0·02 | 46·95 | | |

Potassium xanthate and pine oil in a circuit alkaline with soda ash gave a recovery of 88 per cent of the silver in a rougher concentrate assaying 684·00 ounces of silver per ton. Six ounces of silver remains in the tailing.

### *Test No. 2*

The ore was ground with 4 pounds soda ash per ton and floated first with 0·10 pound American Cyanamid Reagent No. 208 and 0·06 pound pine oil. After removal of this concentrate, 0·10 pound Aerofloat No. 25 was added and a second concentrate removed.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Au, oz./ton | Ag, oz./ton | Au | Ag |
| Concentrate No. 1 | 3·6 | 0·3 | 962·60 | 48·7 | 76·4 |
| " No. 2 | 3·5 | 0·30 | 216·80 | 39·4 | 16·6 |
| Tailing | 92·9 | trace | 3·40 | 11·9 | 7·0 |
| Heads (calculated) | | 0·026 | 45·37 | | |

A screen analysis of the tailing shows:—

| Mesh | Weight, per cent | Assay | |
|---|---|---|---|
| | | Au, oz./ton | Ag, oz./ton |
| +100 | 1·4 | 0·15 | 162·80 |
| −100+150 | 6·5 | 0·02 | 5·23 |
| −150+200 | 8·9 | trace | 2·04 |
| −200 | 83·2 | trace | 0·72 |

The screen analysis indicates the possibility of the presence of native silver as shown by the very high assay of the +100 mesh portion. Fine grinding to at least 150 mesh is apparently required.

### *Test No. 3*

The effect of Aerofloat No. 25 alone was investigated in the test; 5·0 pounds per ton soda ash was added to the grinding mill and 0·20 pound per ton Aerofloat to the flotation machine.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Au, oz./ton | Ag, oz./ton | Au | Ag |
| Concentrate | 4·8 | 0·52 | 884·00 | 100·0 | 94·1 |
| Tailing | 95·2 | trace | 2·80 | ............ | 5·9 |
| Heads (calculated) | ............ | 0·02 | 45·10 | ............ | ............ |

Aerofloat No. 25 appears to be efficient in the flotation of the ore.

### *Test No. 4*

Flotagen was next tried. Soda ash equivalent to 5 pounds per ton of ore was added to the grinding mill and 0·20 pound per ton Flotagen to the flotation machine.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Au, oz./ton | Ag, oz./ton | Au | Ag |
| Concentrate | 6·1 | 0·38 | 698·20 | 100·0 | 92·0 |
| Tailing | 93·9 | trace | 3·92 | ............ | 8·0 |
| Heads (calculated) | ............ | 0·023 | 46·27 | ............ | ............ |

Flotagen added to the cells gives a slightly lower recovery and a lower grade product than that obtained in Test No. 3.

### *Test No. 5*

In this test Flotagen was ground with the ore instead of being introduced into the circuit after grinding.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Au, oz./ton | Ag, oz./ton | Au | Ag |
| Concentrate | 5·3 | 0·32 | 829·00 | 84·8 | 94·8 |
| Tailing | 94·7 | trace | 2·54 | 15·2 | 5·2 |
| Heads (calculated) | ............ | 0·02 | 46·39 | ............ | ............ |

It is apparent that adding this reagent to the grinding circuit gives better results than adding it to the flotation cells. The grade of concentrate and recovery are higher than in the preceding one.

### *Test No. 6*

In this test, the ore was ground finer than in preceding tests, 90 per cent passing 200 mesh. Flotation was made in a circuit alkaline with 5 pounds soda ash per ton. Aerofloat No. 25 was added at the rate of 0·20 pounds per ton. The concentrate was cleaned to note what grade of product could be obtained.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Au, oz./ton | Ag, oz./ton | Au | Ag |
| Concentrate | 2·1 | 0·54 | 1,999·50 | 64·8 | 87·0 |
| Middling | 8·8 | 0·07 | 36·00 | 35·2 | 6·6 |
| Tailing | 89·1 | trace | 3·50 | ........... | 6·4 |
| Heads (calculated) | ........... | 0·017 | 48·28 | ........... | ........... |

The combined middling and concentrate represent a recovery of 93·6 per cent of the silver. Cleaning this gives a 2,000 oz. concentrate. In practice the middling would be returned to the circuit and the contained values recovered in the concentrate.

### *Test No. 7*

In this test table concentration was tried; 1,000 grammes of the ore was ground with water in a ball mill to pass 80 per cent through 200 mesh and then concentrated on a small Wilfley table.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Au, oz./ton | Ag, oz./ton | Au | Ag |
| Concentrate | 2·5 | 0·50 | 1,241·20 | 72·0 | 67·4 |
| Tailing | 97·5 | 0·005 | 15·40 | 28·0 | 32·6 |
| Heads (calculated) | ........... | 0·017 | 46·04 | ........... | ........... |

Table concentration does not give the recoveries secured by flotation.

### *Test No. 8*

To determine the recovery obtained by cyaniding the raw ore, 200-gramme portions were ground to pass 100, and 150 mesh, and then agitated with a 6·3 pound cyanide solution, 3 : 1 dilution, for 48 hours with the following results.

| Mesh | CaO added, lb./ton | Tailing assay | | Recovery, per cent | Reagent consumption, lb./ton | |
|---|---|---|---|---|---|---|
| | | Au, oz./ton | Ag, oz./ton | | KCN | CaO |
| +100 | 3·0 | trace | 5·64 | 87·7 | 6·9 | 2·2 |
| −100 | 6·0 | trace | 3·52 | 92·3 | 6·6 | 4·5 |
| +150 | 3·0 | trace | 12·94 | 71·9 | 6·9 | 2·2 |
| −150 | 6·0 | trace | 3·94 | 91·4 | 6·0 | 4·5 |

These results indicate that ore ground to pass 100 mesh is attacked as readily as when ground to pass 150 mesh.

A recovery of 92·3 per cent of the silver is obtained within 48 hours with a cyanide consumption of 6·6 pounds per ton of ore treated.

### *Test No. 9*

A 1,000-gramme portion of the ore, −14 mesh, was ground in water to pass 90 per cent through 200 mesh. Soda ash at the rate of 5 pounds per ton was added to the mill. A flotation concentrate was then removed by the addition of 0·20 pounds per ton Aerofloat No. 25.

After floating, the tailing was dewatered and cyanided for 48 hours with a 5·0 pounds cyanide solution, 3 : 1 dilution. Lime at the rate of 3 pounds per ton was added.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Au, oz./ton | Ag, oz./ton | Au | Ag |
| Concentrate | 5·41 | 0·40 | 829·00 | 100·0 | 95·7 |
| Tailing | 94·59 | trace | 2·12 | ............ | 4·3 |
| Heads (calculated) | ............ | 0·02 | 46·85 | ............ | ............ |

Cyaniding the above tailing reduces this product from 2·12 to 0·72 ounces silver per ton, a recovery of 66 per cent of the silver not recovered by flotation.

However, this saving amounts to but 1·4 ounces per ton, not sufficient to warrant the expense of cyaniding.

### SUMMARY AND CONCLUSIONS

From the above results it is seen that concentration by flotation can be depended on to yield a recovery of 95 per cent of the silver in the ore.

By refloating the rougher concentrate a product containing 2,000 ounces silver per ton can be obtained.

This product may either be shipped to a smelter or treated hydrometallurgically for extraction of the gold and silver.

Which of these alternatives to adopt can best be determined by the owners as costs will be the deciding factor.

## Report No. 363

## FLOTATION TESTS ON ORE SAMPLE FROM AMULET MINES, LIMITED, ROUYN, QUEBEC

**G. B. Walker**

*Shipment.* A shipment consisting of 500 pounds of copper-zinc ore was received from Amulet Mines, Limited, on July 4, 1930.

*Characteristics.* The ore is a complex sulphide containing chalcopyrite, sphalerite, and pyrite. Relatively fine grinding is required to liberate the sulphides from each other and from the gangue.

*Purpose of Experimental Tests.* The purpose of this investigation was to determine the effect of American Cyanamid Company's Aerofloat reagents on the grade of the zinc concentrate.

*Sampling and Analysis.* The entire sample was stage-crushed in a laboratory breaker and rolls to all −14 mesh. After thorough mixing it was passed through a riffle until a 2-pound sample for head analysis was obtained.

| | |
|---|---|
| Cu | 3·95 per cent |
| Zn | 20·52 " |

## EXPERIMENTAL TESTS

Lots consisting of 1,000 grammes of −14 mesh ore were ground with 700 c.c. of tap water, and reagents as indicated, for a 20-minute period. This grinding period resulted in a pulp having the following screen analysis:

| Mesh | Held, per cent | Cum., per cent |
|---|---|---|
| +65 | ........ | ........ |
| +100 | 2·4 | 2·4 |
| +150 | 6·2 | 8·6 |
| +200 | 11·1 | 19·7 |
| −200 | 80·3 | 100·0 |

The pulp was transferred to a sub-aerated flotation cell of the mechanical type. Reagents were added and froth removed as indicated in the following tests.

### *Test No. 1*

A standard test was made for comparative purposes.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu | Zn |
| Head | 100·0 | 4·24 | 25·50 | 100·0 | 100·0 |
| Copper concentrate | 18·5 | 20·10 | 12·87 | 88·1 | 9·3 |
| Zinc concentrate | 39·3 | 0·70 | 54·40 | 6·5 | 83·9 |
| Zinc middling | 11·9 | 1·33 | 11·71 | 3·7 | 5·5 |
| Tailing | 30·3 | 0·25 | 1·11 | 1·7 | 1·3 |

*Reagents—*

To ball mill—
- $Na_2CO_3$... 0·6 lb./ton
- NaCN.... 0·03 "
- Thiocarbanilide. 0·04 "

pH........ 9·0

To copper cells—
- Xanthate.... 0·02 lb./ton
- Pine oil No. 5 0·08 "

To zinc cells—
- $Na_2CO_3$.......... 6·0 lb./ton
- $CuSO_4$........... 1·75 "
- Xanthate........ 0·20 "
- Aerofloat No. 25. 0·06 "
- Pine oil No. 5.... 0·08 "

Zinc reagents were agitated in the pulp for 3 minutes before zinc flotation.

Rougher zinc concentrates were cleaned for 3 minutes with 0·08 pounds per ton of pine oil No. 5.

### *Test No. 2*

This test was run to show effect of Reagent No. 208 on flotation.

*Reagents*—

| To ball mill— | | To copper cells— | | To zinc cells— | |
|---|---|---|---|---|---|
| $Na_2CO_3$ | 0·60 lb./ton | No. 208 | 0·02 lb./ton | $Na_2CO_3$ | 6·0 lb./ton |
| NaCN | 0·03 " | Pine oil No. 5 | 0·08 " | $CuSO_4$ | 1·75 " |
| Thiocarbanilide | 0·04 " | | | No. 208 | 0·20 " |
| | | | | Aerofloat 25 | 0·06 " |
| | | | | Pine oil No. 5 | 0·08 " |
| pH | 9·0 | | | pH | 10·4 |

Rougher zinc concentrates were cleaned for 3 minutes with 0·04 pounds per ton of pine oil No. 5.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu | Zn |
| Heads | 100·0 | 4·30 | 24·75 | 100·0 | 100·0 |
| Copper concentrate | 19·1 | 20·54 | 12·98 | 91·6 | 10·0 |
| Zinc concentrate | 39·2 | 0·48 | 53·24 | 4·4 | 84·4 |
| Zinc middling | 10·3 | 0·98 | 9·78 | 2·4 | 4·1 |
| Tailing | 31·4 | 0·22 | 1·21 | 1·6 | 1·5 |

### *Test No. 3*

A test was run to show effect of Reagent No. 301 on copper flotation and of Reagent No. 208 on zinc flotation.

*Reagents*—

| To ball mill— | | To copper cells— | | To zinc cells— | |
|---|---|---|---|---|---|
| $Na_2CO_3$ | 0·60 lb./ton | No. 201 | 0·02 lb./ton | $Na_2CO_3$ | 6·0 lb./ton |
| NaCN | 0·03 " | Pine oil No. 5 | 0·08 " | $CuSO_4$ | 1·75 " |
| Thiocarbanilide | 0·04 | | | No. 208 | 0·20 " |
| | | | | Aerofloat 25 | 0·10 " |
| | | | | Pine oil No. 5 | 0·06 " |
| pH | 9·0 | | | pH | 10·4 |

Rougher zinc concentrates were cleaned for 3 minutes with 0·08 pounds per ton NaCN and 0·04 pounds per ton pine oil No. 5.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu | Zn |
| Head | 100·0 | 3·84 | 20·56 | 100·0 | 100·0 |
| Copper concentrate | 23·7 | 15·50 | 16·16 | 95·8 | 18·6 |
| Zinc concentrate | 29·1 | 0·29 | 54·88 | 2·2 | 77·7 |
| Zinc middling | 14·3 | 0·45 | 4·06 | 1·6 | 2·8 |
| Tailing | 32·9 | 0·05 | 0·58 | 0·4 | 0·9 |

### *Test No. 4*

A test was run to show effect of Reagent No. 301 on copper flotation and effect of Sodium Aerofloat on zinc flotation.

*Reagents*—

To ball mill—
$Na_2CO_3$... 0·60 lb./ton
NaCN.... 0·03 "

pH........ 9·0

To copper cells—
No. 301...... 0·06 lb./ton
Pine oil No. 5 0·08 "

To zinc cells—
$Na_2CO_3$..... 6·0 lb./ton
$CuSO_4$....... 1·75 "
Sodium Aerofloat....... 0·20 "
Aerofloat 25. 0·10 "
Pine oil No. 5 0·06 "

pH..........10·4

Rougher zinc concentrates were cleaned for 3 minutes with 0·08 pounds per ton NaCN and 0·04 pounds per ton pine oil No. 5.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu | Zn |
| Heads | 100·0 | 3·92 | 20·79 | 100·0 | 100·0 |
| Copper concentrate | 25·1 | 14·74 | 12·10 | 94·6 | 14·6 |
| Zinc concentrate | 29·6 | 0·30 | 55·95 | 2·3 | 79·7 |
| Zinc middling | 9·6 | 0·70 | 9·68 | 1·7 | 4·5 |
| Tailing | 35·7 | 0·16 | 0·73 | 1·4 | 1·2 |

### *Test No. 5*

*Reagents*—

To ball mill—
$Na_2CO_3$... 0·60 lb./ton
NaCN.... 0·05 "
Thiocarbanilide. 0·04 "

pH........ 8·4

To copper cell—
Xanthate.... 0·02 lb./ton
Pine oil No. 5 0·08 "

To zinc cells—
$Na_2CO_3$..... 6·0 lb./ton
$CuSO_4$....... 1·75 "
Xanthate.... 0·20 "
Aerofloat 25. 0·06 "
Pine oil No. 5 0·04 "

pH..........10·4

Zinc rougher concentrates were cleaned for 3 minutes with 0·08 pounds per ton of pine oil No. 5.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu | Zn |
| Heads | 100·0 | 4·03 | 21·60 | 100·0 | 100·0 |
| Copper concentrate | 28·8 | 13·44 | 19·65 | 96·0 | 26·2 |
| Zinc concentrate | 30·5 | 0·28 | 50·05 | 2·1 | 71·5 |
| Zinc middling | 3·9 | 0·79 | 7·89 | 0·7 | 1·4 |
| Tailing | 36·8 | 0·13 | 0·56 | 1·2 | 0·9 |

### *Test No. 6*

Another test was run to show effect of Reagent No. 208 on zinc flotation.

*Reagents*—

To ball mill—
$Na_2CO_3$... 0·60 lb./ton
NaCN.... 0·05 "
Thiocarbanilide. 0·04 "
pH........ 8·4

To copper cells—
Xanthate.... 0·02 lb./ton
Pine oil No. 5 0·08 "

To zinc cells—
$Na_2CO_3$..... 6·0 lb./ton
$CuSO_4$....... 1·75 "
No. 208...... 0·20 "
Aerofloat 25. 0·06 "
Pine oil No. 5 0·04 "
pH..........10·4

Rougher zinc concentrates were cleaned for 3 minutes with 0·08 pounds per ton pine oil No. 5.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu | Zn |
| Heads.......................... | 100·0 | 3·65 | 20·4 | 100·0 | 100·0 |
| Copper concentrate.............. | 26·1 | 13·44 | 19·84 | 96·1 | 25·3 |
| Zinc concentrate................ | 27·4 | 0·16 | 51·79 | 1·2 | 69·3 |
| Zinc middling................... | 10·5 | 0·67 | 8·71 | 1·9 | 4·5 |
| Tailing......................... | 36·0 | 0·09 | 0·56 | 0·8 | 0·9 |

### *Test No. 7*

A test was run to determine the effect of Reagent No. 208 in a lime circuit for zinc flotation.

*Reagents*—

To ball mill—
$Na_2CO_3$... 0·60 lb/ton
NaCN.... 0·05 "
Thiocarbanilide 0·04 "
pH...... 8·4

To copper cells—
Xanthate.. 0·02 lb./ton
Pine oil No. 5.... 0·08 "

To zinc cells—
CaO......... 0·50 lb./ton
$CuSO_4$....... 1·75 "
No. 208...... 0·20 "
Aerofloat No. 25..... 0·10 "
pH.......... 9·8

Rougher zinc concentrates were cleaned for 3 minutes without additional reagents. The zinc froth was not fast breaking.

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu | Zn |
| Heads.......................... | 100·0 | 3·61 | 21·30 | ........... | ........... |
| Copper concentrate.............. | 22·1 | 15·20 | 19·00 | 93·3 | 19·7 |
| Zinc concentrate................ | 31·1 | 0·34 | 52·56 | 2·9 | 76·8 |
| Zinc middling................... | 9·1 | 0·82 | 6·30 | 2·1 | 2·7 |
| Tailing......................... | 37·7 | 0·17 | 0·48 | ........... | ........... |

### *Test No. 8*

This lot was run to determine the effect of Reagent No. 301 on copper flotation and of Reagent No. 208 on zinc flotation.

*Reagents—*

To ball mill—
$Na_2CO_3$. 2·0 lb./ton
NaCN.. 0·10 "
Thiocarbanilide 0·04 "

pH...... 9·2

To copper cells—
No. 301.... 0·02 lb./ton
Pine oil No. 5.... 0·08 "

To zinc cells—
$Na_2CO_3$ 4·0 lb./ton
$CuSO_4$....... 1·75 "
No. 208...... 0·20 "
Aerofloat No. 25..... 0·10 "
Pine oil No. 5 0·08 "

pH........... 9·8–10·0

| Product | Weight, per cent | Assay | | Per cent of value | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu | Zn |
| Heads........................... | 100·00 | 3·80 | 20·60 | 100·0 | 100·0 |
| Copper concentrate............... | 21·5 | 16·75 | 17·71 | 94·8 | 18·5 |
| Zinc concentrate................. | 27·3 | 0·20 | 56·25 | 1·4 | 74·3 |
| Zinc middling.................... | 13·7 | 0·67 | 9·63 | 2·4 | 6·4 |
| Tailing.......................... | 37·5 | 0·14 | 0·48 | 1·4 | 0·8 |

The rougher zinc concentrates were cleaned for 3 minutes with 0·12 pounds per ton NaCN and 0·08 pounds per ton pine oil No. 5.

### *Test No. 9*

A test was run to show effect of Sodium Aerofloat on zinc flotation.

| Product | Weight, per cent | Assay | | Per cent of value | |
|---|---|---|---|---|---|
| | | Cu, per cent | Zn, per cent | Cu | Zn |
| Heads........................... | 100·0 | 3·29 | 20·60 | 100·0 | 100·0 |
| Copper concentrate............... | 25·3 | 12·68 | 18·00 | 97·5 | 22·1 |
| Zinc concentrate................. | 25·6 | 0·07 | 57·31 | 0·5 | 71·3 |
| Zinc middling.................... | 14·0 | 0·21 | 8·27 | 0·9 | 5·6 |
| Tailing.......................... | 35·1 | 0·10 | 1·58 | 1·1 | 1·0 |

*Reagents—*

To ball mill—
$Na_2CO_3$. 2·0 lb./ton
NaCN... 0·10 "
Thiocarbanilide 0·04 "

pH...... 9·2

To copper cells—
Xanthate.. 0·02 lb./ton
Pine oil No. 5.... 0·08 "

To zinc cells—
$Na_2CO_3$..... 4·0 lb./ton
$CuSO_4$....... 1·75 "
Sodium Aerofloat.. 0·20 "
Aerofloat No. 25..... 0·10 "
Pine oil No. 5 0·04 "

pH.......... 9·8–10·0

The rougher zinc concentrates were cleaned for 3 minutes with 0·12 pounds per ton NaCN and 0·08 pounds per ton pine oil No. 5. A 3-minute agitation period with cyanide was allowed before the cleaner float.

## SUMMARY

A series of tests has been run on a sample of copper-zinc ore from Amulet Mines, Ltd., to determine the effect of Aerofloat reagents on the grade of the zinc concentrates. The effect of No. 208 and No. 301 in the copper circuit as a substitute for xanthate has also been studied.

During the course of the investigation some oxidation of the sample occurred which necessitated increased amounts of cyanide and soda ash to maintain approximately the same degree of separation.

The following table summarizes the results obtained in the copper circuit.

*Reagents and Results of Copper Flotation*

| Test No. | Assay Cu, per cent | Recovery Cu, per cent | Reagents, lb./ton | | | |
|---|---|---|---|---|---|---|
| | | | Thiocarbanilide, per cent | Xanthate | No. 208 | No. 301 |
| 1 | 20·10 | 88·1 | 0·04 | 0·02 | ...... | ...... |
| 2 | 20·54 | 91·6 | 0·04 | ...... | 0·02 | ...... |
| 3 | 15·50 | 95·8 | 0·04 | ...... | ...... | 0·02 |
| 4 | 14·74 | 94·6 | ...... | ...... | ...... | 0·06 |
| 5 | 13·44 | 96·0 | 0·04 | 0·02 | ...... | ...... |
| 6 | 13·44 | 96·1 | 0·04 | 0·02 | ...... | ...... |
| 7 | 15·20 | 93·3 | 0·04 | 0·02 | ...... | ...... |
| 8 | 16·75 | 94·8 | 0·04 | ...... | ...... | 0·02 |
| 9 | 12·68 | 97·5 | 0·04 | ...... | 0·02 | ...... |

As could be expected, the highest recovery was obtained when the lowest grade concentrate was produced. The highest recovery consistent with reasonable concentrate grade appears to have been obtained in Tests Nos. 2, 3, and 8. In Test No. 2 Reagent No. 208 effected higher recovery and a higher grade concentrate than was obtained with xanthate in Test No. 1. Other xanthate tests have yielded higher recoveries but with considerably lower grade concentrates. Tests Nos. 3 and 8, in which a combination of thiocarbanilide and Reagent No. 301 was used, yielded the highest recoveries obtained with a reasonable grade of rougher copper concentrate, i.e. 15·50 per cent to 16·75 per cent Cu.

The following table presents a summary of the results obtained in zinc flotation.

*Reagents and Results in Zinc Flotation*

| Test No. | Assays, Zn, per cent | | | Reagents, lb./ton | | | | |
|---|---|---|---|---|---|---|---|---|
| | Conc. | Midd. | Tail. | Xanthate | No. 208 | Sodium Aerofloat | Aerofloat No. 25 | NaCN |
| 1 | 54·40 | 11·71 | 1·11 | 0·20 | ...... | ...... | 0·06 | ...... |
| 2 | 53·24 | 9·78 | 1·21 | ...... | 0·20 | ...... | 0·06 | ...... |
| 3 | 54·88 | 4·06 | 0·58 | ...... | 0·20 | ...... | 0·10 | 0·08 |
| 4 | 55·95 | 9·68 | 0·73 | ...... | ...... | 0·20 | 0·10 | 0·08 |
| 5 | 50·05 | 7·89 | 0·56 | 0·20 | ...... | ...... | 0·06 | ...... |
| 6 | 51·79 | 8·71 | 0·56 | ...... | 0·20 | ...... | 0·06 | ...... |
| 7* | 52·56 | 6·30 | 0·48 | ...... | 0·20 | ...... | 0·10 | ...... |
| 8 | 56·25 | 9·63 | 0·48 | ...... | 0·20 | ...... | 0·10 | 0·12 |
| 9 | 57·31 | 8·27 | 0·58 | ...... | ...... | 0·20 | 0·10 | 0·12 |

*In Test No. 7 a lime circuit was employed.

Tests Nos. 3, 5, 6, and 8 indicate that an increase in Aerofloat No. 25 is effective for the production of low zinc tailings.

The highest grade zinc concentrate was made when Sodium Aerofloat was used in the rougher float and with cyanide added to the cleaner float. The lowest tailings was obtained in Test No. 8 with Reagent No. 208 The zinc concentrate is the second highest produced on this ore.

The average grades of the zinc concentrates made with various reagents in this series are as follows:

| *Reagents*— | *Per cent of zinc in zinc concentrate* | *Test No.* |
|---|---|---|
| Xanthate | 52·23 | 2 |
| No. 208 | 53·74 | 5 |
| Sodium Aerofloat | 56·63 | 2 |

### CONCLUSIONS

The results obtained in this investigation indicate that the use of Reagent No. 301 in conjunction with thiocarbanilide can be expected to effect higher recovery of copper consistent with concentrate grade than the combination of thiocarbanilide and xanthate.

The use of increased amounts of Aerofloat No. 25 has resulted in lower zinc tailings. The added amount of Aerofloat did not cause any noticeable increase in the volume of the froth.

Sodium Aerofloat has effected the highest grade zinc concentrates, i.e. an average of 56·63 per cent of zinc which included one product that assayed 57·31 per cent zinc. The use of Reagent No. 208 has resulted in higher grade zinc concentrates than were made with xanthate.

The application of either of these reagents to a mill circuit could be done without other changes in the reagent balance. As far as can be ascertained in a test none of these reagents alters the character or volume of the froth.

## Report No. 364

### THE DIFFERENTIAL FLOTATION OF COPPER AND IRON SULPHIDES IN THE ORE OF THE ALDERMAC MINES, LTD., ALDERMAC, QUEBEC

**A. K. Anderson, J. S. Godard, C. S. Parsons**

*Shipment.* A carload of ore weighing 47 tons, consigned from the Aldermac Mines, Limited, Rouyn area, Quebec, was received on July 17, 1930.

*Characteristics of the Ore.* The ore consisted of massive sulphides of iron, pyrite and pyrrhotite intimately associated. Chalcopyrite also was present in small amounts. Very little zinc sulphides were noticeable.

*Purpose of Experimental Tests.* The purpose of the test work was to determine what recovery of copper could be obtained and also what weight and sulphur content of iron pyrite concentrate could be produced.

*Sampling and Analysis.* No head sample of the entire shipment was made. All tests were made on large-scale equipment, the feed for each run being sampled separately. The average analysis of the head samples of the tests showed,—

| | |
|---|---|
| Copper | 1·75 per cent |
| Iron | 28·2 " |
| Sulphur | 39·3 " |

## EXPERIMENTAL TESTS

All test work was conducted on large-scale machines, the feed rate being approximately 1,000 pounds per hour. The flow-sheet unless otherwise stated was as follows.

The ore was crushed in a jaw crusher to approximately ¾-inch and fed to a 4½-foot Hardinge ball mill in closed circuit with a Dorr Simplex classifier. The classifier overflow passed to a mechanically agitated flotation machine where a copper concentrate was removed. The tailing from this machine passed to a two-cell, flat-bottomed, Callow pneumatic flotation machine where an iron pyrite concentrate was removed. This rougher concentrate was cleaned twice in smaller Callow cells.

### *Test No. 1*

| | |
|---|---|
| Feed rate approximately | 1,000 pounds per hour |
| Ball mill discharge | 52 per cent solids |
| Classifier overflow | 40 per cent solids |

*Reagents in Pounds Per Ton Ore—*

| — | To ball mill | To copper cells | To iron cells |
|---|---|---|---|
| Soda ash | 3·0 | | 2·0 |
| Cyanide | 0·02 | | |
| Aerofloat No. 25 | 0·06 | | 0·10 |
| Potassium xanthate | | 0·06 | 0·65 |
| Sodium sulphide | | | 0·5 |
| Copper sulphate | | | 0·5 |
| Pine oil | | 0·03 | |

| Product | Weight, per cent | Assay | | | Per cent of values | |
|---|---|---|---|---|---|---|
| | | Cu, per cent | Fe, per cent | S, per cent | Cu | S |
| Heads | 100·0 | 1·84 | 38·3 | 39·0 | 100·0 | 100·0 |
| Copper concentrate | 15·5 | 11·44 | 38·8 | 40·5 | 96·5 | 16·1 |
| Iron concentrate | 61·2 | 0·06 | 44·9 | 48·8 | 2·0 | 76·6 |
| Tailing | 23·3 | 0·12 | 24·5 | 12·2 | 1·5 | 7·3 |

Under these conditions a copper concentrate assaying 11·44 per cent copper with a recovery of 96·5 per cent is produced. The grade could be raised to over 20 per cent copper by cleaning without materially affecting the recovery.

From each 100 tons of feed 61·2 tons of pyrite concentrate assaying 48·8 per cent sulphur is produced.

### *Test No. 2*

The flow-sheet for this test was the same as the preceding one. No sodium sulphide was used.

*Reagents in Pounds Per Ton Ore—*

| — | To ball mill | To copper cells | To iron cells |
|---|---|---|---|
| Soda ash | 2·3 | | 1·5 |
| Cyanide | 0·015 | | |
| Aerofloat No. 25 | 0·07 | | 0·10 |
| Potassium xanthate | | 0·05 | 0·51 |
| Copper sulphate | | | 0·4 |

| Product | Weight, per cent | Assay | | | Per cent of values | |
|---|---|---|---|---|---|---|
| | | Cu, per cent | Fe, per cent | S, per cent | Cu | S |
| Heads | 100·00 | 1·71 | 37·2 | 39·4 | 100·0 | 100·0 |
| Copper concentrate | 12·81 | 12·56 | 38·9 | 42·5 | 93·9 | 14·5 |
| Iron concentrate | 50·62 | 0·09 | 45·1 | 52·6 | 2·6 | 70·8 |
| Tailing | 36·57 | 0·16 | 23·6 | 15·2 | 3·5 | 14·7 |

A concentrate assaying and containing 12·56 per cent copper, 93·9 per cent of the copper is recovered. By cleaning, the grade would be raised without materially altering the recovery.

From each 100 tons of feed, 50·6 tons of iron pyrite is recovered with a sulphur content of 52·6 per cent.

### *Test No. 3*

In this test the amounts of reagents were slightly increased. Aerofloat and cresylic acid were used for the flotation of copper, and potassium ethyl xanthate and Aerofloat for the iron pyrite flotation.

*Reagents in Pounds Per Ton Feed—*

| — | To ball mill | To copper cells | To iron cells |
|---|---|---|---|
| Soda ash | 3·5 | | 2·0 |
| Cyanide | 0·03 | | |
| Aerofloat No. 25 | 0·033 | | 0·08 |
| Cresylic acid | | 0·04 | |
| Copper sulphate | | | 0·5 |
| Potassium xanthate | | | 0·6 |

| Product | Weight, per cent | Assay | | | Per cent of values | |
|---|---|---|---|---|---|---|
| | | Cu, per cent | Fe, per cent | S, per cent | Cu | S |
| Heads | 100·00 | 1·84 | 37·3 | 38·2 | 100·0 | 100·0 |
| Copper concentrate | 10·32 | 16·52 | 34·5 | 38·0 | 92·6 | 10·3 |
| Iron concentrate | 57·36 | 0·18 | 45·1 | 50·8 | 5·6 | 76·3 |
| Tailing | 32·32 | 0·10 | 21·8 | 15·9 | 1·8 | 13·4 |

A recovery of 92·6 per cent of the copper in a rougher concentrate assaying 16·52 per cent copper is obtained.

An iron pyrite concentrate assaying 50·8 per cent sulphur and comprising 57·4 per cent of the weight of the feed is recovered.

### *Test No. 4*

Owing to a break-down in the mill it was impossible to run the iron flotation cells for this test.

An aeration tank to give approximately ten minutes' contact was placed between the classifier overflow and the copper flotation cells for all the following runs.

*Reagents in Pounds Per Ton Feed—*

| — | To ball mill | To copper cells |
|---|---|---|
| Soda ash | 3·1 | ........... |
| Cyanide | 0·04 | ........... |
| Aerofloat No. 25 | 0·03 | ........... |
| Cresylic acid | ........... | 0·04 |

| Product | Weight, per cent | Assay | | | Per cent of values |
|---|---|---|---|---|---|
| | | Cu, per cent | Fe, per cent | S, per cent | Cu |
| Heads | 100·00 | 1·66 | 38·44 | 38·38 | 100·0 |
| Copper concentrate | 7·81 | 19·36 | 33·46 | 35·27 | 91·1 |
| Tailing | 92·19 | 0·16 | 34·70 | 34·06 | 8·9 |

Aeration of the pulp prior to flotation appears to be somewhat beneficial. A higher grade of concentrate was secured. The recovery however is lower than in preceding tests. Indications are that a reduction in the amount of cyanide used would be of benefit.

### *Test No. 5*

This is a continuation of Test No. 4. The flotation of the iron pyrite was studied to note what effect resulted from aerating the pulp prior to entering the copper circuit.

| | |
|---|---|
| Feed rate | 1,200 pounds per hour |
| Ball mill discharge | 70 per cent solids |
| Classifier overflow | 52 " " |

*Reagents in Pounds Per Ton Feed—*

| — | To ball mill | To copper cells | To iron cells |
|---|---|---|---|
| Soda ash | 3·1 | ........... | 1·7 |
| Cyanide | 0·014 | ........... | ........... |
| Aerofloat No. 25 | 0·04 | ........... | 0·09 |
| Potassium xanthate | ........... | ........... | 0·55 |
| Cresylic acid | ........... | 0·04 | ........... |
| Copper sulphate | ........... | ........... | 0·4 |

| Product | Weight, per cent | Assay | | | Per cent of values | |
|---|---|---|---|---|---|---|
| | | Cu, per cent | Fe, per cent | S, per cent | Cu | S |
| Heads | 100·00 | 1·97 | 38·10 | 39·74 | 100·0 | 100·0 |
| Copper concentrate | 8·49 | 21·08 | 33·06 | 35·81 | 90·9 | 7·5 |
| Copper tailing | ......... | 0·20 | 37·49 | 38·21 | ......... | ......... |
| Iron concentrate | 63·22 | 0·18 | 45·36 | 51·48 | 5·8 | 79·8 |
| Iron tailing | 28·29 | 0·23 | 26·01 | 18·31 | 3·3 | 12·7 |

A sample of the copper concentrate was cleaned in a small flotation machine giving a finished concentrate assaying 25·16 per cent copper, and a middling 6·64 per cent copper; 15·6 per cent of the weight was dropped in the middling.

The test indicates that 90·9 per cent of the copper is recovered in a rougher concentrate assaying 21·08 per cent copper. An iron pyrite concentrate representing 63·2 per cent of the weight of ore milled is also recovered. This has an analysis of 51·48 per cent sulphur.

### *Test No. 6*

This run is a continuation of the previous day's run. The plant was operated from 8 a.m. to 12 p.m. Some time was lost through breakdowns. The results secured by the two shifts are shown below. Due to fluctuating feed, the amounts of reagents added vary slightly.

*A Shift—*

| | |
|---|---|
| Feed rate | 846 pounds per hour |
| Classifier overflow | 39 per cent solids |
| Ball mill discharge | 69 " " |

*Reagents in Pounds Per Ton Feed—*

| — | To ball mill | To copper cells | To iron cells |
|---|---|---|---|
| Soda ash | 3·9 | ........ | 1·9 |
| Cyanide | 0·029 | ........ | ........ |
| Aerofloat No. 25 | 0·06 | ........ | 0·10 |
| Cresylic acid | ........ | 0·035 | ........ |
| Copper sulphate | ........ | ........ | 0·54 |
| Potassium xanthate | ........ | ........ | 0·81 |

| Product | Weight, per cent | Assay |  |  | Per cent of values |  |
|---|---|---|---|---|---|---|
|  |  | Cu, per cent | Fe, per cent | S, per cent | Cu | S |
| Heads | 100·00 | 1·54 | 38·71 | 40·22 | 100·0 | 100·0 |
| Copper concentrate | 6·36 | 21·18 | 34·47 | 36·64 | 87·4 | 5·8 |
| Copper tailing | ........ | 0·23 | 38·91 | 39·26 | ........ | ........ |
| Iron concentrate | 62·29 | 0·22 | 45·96 | 51·46 | 8·9 | 79·7 |
| Iron tailing | 31·35 | 0·18 | 26·21 | 18·62 | 3·7 | 14·5 |

*B Shift—*

| | |
|---|---|
| Ball mill discharge | 73 per cent solids |
| Classifier overflow | 35 " " |

*Reagents in Pounds Per Ton Feed—*

| — | To ball mill | To copper cells | To iron cells |
|---|---|---|---|
| Soda ash | 3·1 | ........ | 1·8 |
| Cyanide | 0·03 | ........ | ........ |
| Aerofloat No. 25 | 0·06 | ........ | 0·10 |
| Cresylic acid | ........ | 0·035 | ........ |
| Copper sulphate | ........ | ........ | 0·47 |
| Potassium xanthate | ........ | ........ | 0·74 |

| Product | Weight, per cent | Assay |  |  | Per cent of values |  |
|---|---|---|---|---|---|---|
|  |  | Cu, per cent | Fe, per cent | S, per cent | Cu | S |
| Heads | 100·00 | 1·78 | 38·91 | 39·24 | 100·0 | 100·0 |
| Copper concentrate | 5·59 | 25·96 | 32·66 | 34·47 | 81·5 | 4·9 |
| Copper tailing | ........ | 0·33 | 37·69 | 37·31 | ........ | ........ |
| Iron concentrate | 58·06 | 0·36 | 45·76 | 50·82 | 11·7 | 75·2 |
| Iron tailing | 36·35 | 0·33 | 25·60 | 21·48 | 6·8 | 19·9 |

It will be noticed that when the grade of rougher copper concentrate is 20 per cent or higher, the recovery is not so high as when the grade is below 20 per cent.

### *Test No. 7*

This run was made under the same conditions and with the same reagents as Test No. 6. It also was carried on for two shifts, A and B.

*A Shift*—

| | |
|---|---|
| Feed rate | 964 pounds per hour |
| Ball mill discharge | 58 per cent solids |
| Classifier overflow | 40 " " |

*Reagents in Pounds Per Ton Ore*—

| — | To ball mill | To copper cells | To iron cells |
|---|---|---|---|
| Soda ash | 3·7 | ........... | 2·1 |
| Cyanide | 0·035 | ........... | ........... |
| Aerofloat No. 25 | 0·06 | ........... | 0·09 |
| Cresylic acid | ........... | 0·036 | ........... |
| Copper sulphate | ........... | ........... | 0·5 |
| Potassium xanthate | ........... | ........... | 0·66 |

| Product | Weight, per cent | Assay | | | Per cent of values | |
|---|---|---|---|---|---|---|
| | | Cu, per cent | Fe, per cent | S, per cent | Cu | S |
| Heads | 100·00 | 1·58 | 38·51 | 40·28 | 100·0 | 100·0 |
| Copper concentrate | 7·14 | 20·72 | 33·67 | 31·70 | 93·6 | 5·7 |
| Copper tailing | ......... | 0·22 | 39·31 | 39·76 | ......... | ......... |
| Iron concentrate | 63·22 | 0·09 | 45·96 | 52·20 | 3·6 | 82·9 |
| Iron tailing | 29·64 | 0·15 | 26·61 | 15·28 | 2·8 | 11·4 |

*B Shift*—

| | |
|---|---|
| Ball mill discharge | 75 per cent solids |
| Classifier overflow | 40 " " |

Reagents as in A shift.

| Product | Weight, per cent | Assay | | | Per cent of values | |
|---|---|---|---|---|---|---|
| | | Cu, per cent | Fe, per cent | S, per cent | Cu | S |
| Heads | 100·00 | 1·84 | 38·51 | 38·71 | 100·0 | 100·0 |
| Copper concentrate | 7·80 | 21·12 | 35·08 | 36·09 | 89·6 | 7·1 |
| Copper tailing | ......... | 0·17 | 37·69 | 37·63 | ......... | ......... |
| Iron concentrate | 57·50 | 0·20 | 45·56 | 50·88 | 6·2 | 74·0 |
| Iron tailing | 34·70 | 0·22 | 27·22 | 21·46 | 4·2 | 18·9 |

The only variation in operating conditions between the two shifts was in the ball mill discharge. "A" shift, where grinding was done at 58 per cent solids, shows a recovery of 93·6 per cent of the copper and 63·2 per cent of the feed recovered as an iron pyrite concentrate assaying 52·2 per cent sulphur.

"B" shift, where the ball mill discharge was 75 per cent solids shows a recovery of 89·6 per cent of the copper and 57·5 per cent of the feed recovered as a pyrite concentrate assaying 50·88 per cent sulphur.

The results would indicate that better results are secured by grinding at 58 per cent solids than at 75 per cent.

### *Test No. 8*

In this test the flotation of the copper was made as in Test No. 7. American Cyanamid Reagent No. 404 was used for flotation of the iron pyrite.

Three sampling periods were taken with varying amounts of Reagent No. 404 added. The first sample from 11·30 to 12·00 was taken when this reagent was being added at the rate of 0·24 pounds per ton. The second sample, after changing the reagent to 0·46 pounds per ton, and the third when 0·40 pounds per ton was added. Other factors were:

| | |
|---|---|
| Ball mill discharge | 58 per cent solids |
| Classifier overflow | 47 " " |

*Reagents in Pounds Per Ton Feed—*

| — | To ball mill | To copper cells | To iron cells |
|---|---|---|---|
| Soda ash | 4·4 | ........... | 2·7 |
| Cyanide | 0·04 | ........... | ........... |
| Aerofloat No. 25 | 0·07 | ........... | ........... |
| Cresylic acid | ........... | 0·04 | ........... |
| Copper sulphate | ........... | ........... | 0·6 |

*Period 11.30 a.m.–12.00 p.m.—*

| Product | Weight, per cent | Assay | | | Per cent of values | |
|---|---|---|---|---|---|---|
| | | Cu, per cent | Fe, per cent | S, per cent | Cu | S |
| Heads | 100·00 | 1·72 | 37·90 | 39·40 | 100·0 | 100·0 |
| Copper concentrate | 7·29 | 20·32 | 34·67 | 32·05 | 86·1 | 5·9 |
| Copper tailing | .......... | 0·13 | 37·69 | 39·04 | .......... | .......... |
| Iron concentrate | 49·78 | 0·28 | 45·76 | 51·96 | 8·1 | 65·6 |
| Iron tailing | 42·93 | 0·23 | 29·23 | 26·08 | 5·8 | 28·5 |

*Period 12.30–1.00 p.m.—*

| | | | | | | |
|---|---|---|---|---|---|---|
| Copper concentrate | 7·71 | 20·32 | 34·67 | 32·05 | 91·3 | 6·3 |
| Iron concentrate | 59·10 | 0·14 | 46·57 | 52·22 | 4·8 | 78·5 |
| Iron tailing | 33·19 | 0·20 | 24·39 | 18·05 | 3·9 | 15·2 |

*Period 1.30–2.00 p.m.—*

| | | | | | | |
|---|---|---|---|---|---|---|
| Copper concentrate | 7·78 | 20·32 | 34·67 | 32·05 | 91·9 | 6·3 |
| Iron concentrate | 59·75 | 0·13 | 47·17 | 51·80 | 4·5 | 78·6 |
| Iron tailing | 32·47 | 0·19 | 24·99 | 18·34 | 3·6 | 15·1 |

Reagent No. 404 although about double the cost of potassium xanthate, does not give so high a recovery of pyrite.

### *Test No. 9*

In this test, a caustic soda solution of Flotagen furnished by Canadian Industries, Limited, was added to both the copper and iron pyrite circuits. It was found that this reagent had very little power to float iron pyrite under the conditions of the test. The small amount of zinc in the ore, however, appeared as a concentrate in the iron pyrite cells. An intensive study of the reagent was not made. It appears to be of some use for the flotation of copper sulphides.

| | |
|---|---|
| Ball mill discharge | 62 per cent solids |
| Classifier overflow | 41 " " |

*Reagents in Pounds Per Ton Ore—*

| — | To ball mill | To copper cells |
|---|---|---|
| Soda ash | 3·4 | ........... |
| Cyanide | 0·03 | ........... |
| Flotagen | 0·05 | ........... |
| Pine oil | ........... | 0·019 |

| Product | Weight, per cent | Assay | | | Per cent of values |
|---|---|---|---|---|---|
| | | Cu, per cent | Fe, per cent | S, per cent | Cu |
| Heads | 100·00 | 1·72 | 37·90 | 39·40 | 100·0 |
| Copper concentrate | 7·14 | 20·96 | 33·87 | 35·82 | 87·0 |
| Tailing | 92·86 | 0·24 | 37·64 | 37·64 | 13·0 |

The test indicates that this solution of Flotagen will float copper. Due to lack of time no further investigation of this reagent was made. Doubtless more suitable combinations and amounts of reagents would yield a higher recovery of copper than indicated in this test.

## SUMMARY AND CONCLUSIONS

The selective flotation of the copper sulphides and iron pyrite in this ore presents no great difficulties. To secure maximum recoveries of copper, Tests Nos. 1 and 2 indicate that it is necessary to produce a comparatively low-grade rougher concentrate, which can then be cleaned to a marketable grade.

The copper minerals are sensitive to cyanide. If over 0·04 pounds cyanide per ton is added, the copper appears to be depressed, resulting in lower recoveries.

The ratio of ore to water in the grinding mill has an effect on the results. The recoveries fall off slightly when grinding exceeds 65 per cent solids. Best results were obtained as shown in Test No. 7, where a ball mill discharge of 58 per cent solids was maintained resulting in a recovery of 93·6 per cent of the copper in a rougher concentrate assaying 20·72 per cent copper, and 63·2 per cent of the mill feed recovered as a pyrite concentrate assaying 52·2 per cent sulphur.

The inclusion of an aeration tank in the circuit before the copper flotation appears to have a stabilizing effect on froth conditions in the cells, and counteracts variations in the density of the ball mill discharge.

A recovery of over 90 per cent of the copper and 60 per cent of the weight of ore milled as a pyrite concentrate with a sulphur content of over 50 per cent, can confidently be expected.

## Report No. 365

### EXPERIMENTAL TESTS ON A SAMPLE OF TAILINGS FROM THE MILL OF THE ARGONAUT GOLD MINES, LIMITED, LARDER LAKE, ONTARIO

**J. S. Godard**

*Shipments.* A shipment of 70 pounds of tailings was received July 23, 1930 from P. M. Fleming, Haileybury Ontario.

*Characteristics of the Tailings.* The ore of the Argonaut Gold Mines, Limited at Larder Lake, Ont., was treated first by cyanidation. Later amalgamation and flotation of the amalgamation tailings was practised. From the low copper content of the tailings sample submitted it is assumed that they were those discharged from the mill when operating with the latter flow-sheet.

The sample contained gold, 0·115 oz. per ton, copper, 0·19 per cent, and iron, both as pyrite and magnetite, 9·93 per cent. The remainder was mainly silica. A screen analysis showed that about 62 per cent was −200 mesh.

A 4,000-gramme sample was screened with the following results.

| Mesh | Weight, per cent | Assays | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Au, oz./ton | Cu, per cent | Fe, per cent | Au | Cu | Fe |
| + 65 | 0·5 | 0·40 | 0·15 | 9·78 | 1·7 | 0·4 | 0·5 |
| +100 | 3·5 | 0·12 | 0·11 | 7·36 | 3·7 | 2·1 | 2·6 |
| +150 | 7·8 | 0·10 | 0·07 | 6·05 | 6·8 | 2·9 | 4·8 |
| +200 | 25·9 | 0·10 | 0·13 | 6·25 | 22·6 | 18·0 | 16·3 |
| −200 | 62·3 | 0·12 | 0·23 | 12·09 | 65·2 | 76·6 | 75·8 |

*Purpose of Experimental Tests.* The purpose of the experimental tests was to determine the best method of recovering the gold in the tailings.

### Experimental Tests

#### CONCENTRATION TESTS

Tests Nos. 1 to 3 consisted of flotation followed by tabling the flotation tailings.

The flotation reagents used were,—

Soda ash 4·0 lb. per ton, coal-tar creosote 0·15 lb. per ton, potassium xanthate 0·30 lb. per ton, pine oil No. 5 0·08 lb. per ton, and copper sulphate 1·0 lb. per ton.

### *Test No. 1*

*Results*—

| Product | Weight, per cent | Assays | Per cent of values, Au |
|---|---|---|---|
| | | Au oz./ton | |
| Flotation concentrate | 4·7 | 1·12 | 36·0 |
| Table concentrate | 8·1 | 0·57 | 31·6 |
| Table tailing +200 | 29·7 | 0·07 | 14·2 |
| " " −200 | 35·8 | 0·05 | 12·2 |
| Slimes | 21·7 | 0·04 | 6·0 |

No regrinding was done before flotation.
Recovery in concentrates 67·6 per cent.

### *Test No. 2*

*Results*—

| Product | Weight, per cent | Assays | Per cent of values, Au |
|---|---|---|---|
| | | Au oz./ton | |
| Flotation concentrate | 4·4 | 1·18 | 44·6 |
| Table concentrate | 8·0 | 0·28 | 19·2 |
| Table tailing +200 | 19·5 | 0·07 | 11·8 |
| " " −200 | 39·8 | 0·05 | 17·1 |
| Slimes | 28·3 | 0·03 | 7·3 |

Sample ground 5 minutes before flotation.
Recovery in concentrates 63·8 per cent.

### *Test No. 3*

*Results*—

| Product | Weight, per cent | Assays | Per cent of values, Au |
|---|---|---|---|
| | | Au oz./ton | |
| Flotation concentrate | 6·3 | 1·12 | 62·0 |
| Table concentrate | 7·0 | 0·18 | 11·1 |
| Table tailing +200 | 7·4 | 0·05 | 3·3 |
| " " −200 | 30·9 | 0·04 | 10·9 |
| Slimes | 48·4 | 0·03 | 12·7 |

Sample ground 20 minutes before flotation.
Recovery in concentrates 73·1 per cent.

## CYANIDATION TEST

Two cyanidation tests were made on the tailings. The cyanide strength was 0·05 per cent KCN and the pulp dilution 3:1, the agitation periods were 24 hours in Test No. 4 and 48 hours in Test No. 5. The cyanidation tailings were screened on 200 mesh.

*Test No. 4*

| Mesh | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au | Average tailing Au oz./ton |
|---|---|---|---|---|
| +200 | 31·9 | 0·03 | 48·4 | 0·02 |
| −200 | 68·1 | 0·015 | 51·6 | ........... |

*Test No. 5*

| | | | | |
|---|---|---|---|---|
| +200 | 36·0 | 0·02 | 52·9 | 0·014 |
| −200 | 64·0 | 0·01 | 47·1 | ........... |

*Summary*

| No. | Head, Au oz./ton | Tails, Au oz./ton | Extraction, per cent | Time, hours | Reagents lb./ton | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| 4 | 0·114 | 0·02 | 82·5 | 24 | 1·1 | 3·3 |
| 5 | 0·114 | 0·014 | 87·7 | 48 | 1·4 | 4·0 |

### CONCLUSIONS

In the treatment of these tailings, cyanidation gives a higher recovery of gold values than flotation.

## Report No. 366

### THE CONCENTRATION OF A LEAD-ZINC-SILVER ORE FROM THE REGAL SILVER MINES, LTD., ALBERT CANYON, BRITISH COLUMBIA

**A. K. Anderson**

*Shipments.* A shipment of ore contained in six bags, gross weight 530 pounds, was received at the Ore Dressing and Metallurgical Laboratories on July 8, 1930, from the Regal Silver Mines, Limited, Albert Canyon, B.C., with head office at 804 Standard Bank Building, Vancouver, B.C., A. S. MacCulloch, Secretary.

The consignment was composed of five samples taken from different parts of the mine.

*Characteristics of the Ore.* The ore consisted of a siliceous gangue containing sulphides of lead, zinc, and iron, together with a small amount of graphitic material.

*Purpose of Experimental Tests.* The shipment was made for the purpose of determining the most suitable method of milling to adopt for the recovery of the contained values.

*Sampling and Analysis.* The five samples were handled separately. These were crushed to pass 10 mesh, quartered and ground to pass the finer sized meshes with intervening quartering through a Jones riffle sampler, until a representative 100-mesh portion was secured for analysis.

The following results were obtained:—

| — | Pb, per cent | Zn, per cent | Ag, oz./ton |
|---|---|---|---|
| Tunnel No. 2—High grade | 11·67 | 3·17 | 10·70 |
| " No. 2— " | 9·35 | 2·43 | 7·60 |
| " No. 3—Drift No. 5 | 4·12 | 1·90 | 4·20 |
| " No. 3— " No. 5, intermediate level | 6·33 | 1·20 | 4·68 |
| " No. 3— " No. 6 | 4·02 | 1·50 | 3·88 |

All samples were examined for tungsten and tin but no indications of these metals were found.

For the experimental tests, a composite sample of four of these lots was made. The high grade from tunnel No. 2 was not included in this. The mixture on which the tests were made had an analysis of Pb 6·30 per cent, Zn 1·85 per cent, Ag 5·22 oz. per ton.

## EXPERIMENTAL TESTS

The nature and appearance of the ore, with sulphides of lead, zinc, and iron intimately associated, indicated that the most suitable method of treatment was flotation, making a selective concentration of the lead followed by one of the zinc. The iron pyrite in the ore is thus discarded in the tailing.

In the following tests, 1,000-gramme samples were ground with 750 grammes water in porcelain mills containing iron balls together with reagents to depress zinc and iron sulphides until 77 per cent of the material passed 200 mesh. After the addition of reagents, a lead concentrate was removed. This was cleaned, raising the grade and producing a middling. The middling in practice would be returned to the circuit without materially affecting the results.

After the lead concentrate was obtained, copper sulphate was added to reactivate the zinc minerals which were next removed.

### *Test No. 1*

*Reagents in Pounds Per Ton of Ore*—

| — | To ball mill | To lead cells | To zinc cells |
|---|---|---|---|
| Soda ash | 6·0 | ........... | ........... |
| Cyanide | 0·10 | ........... | ........... |
| Aerofloat No. 25 | ........... | 0·10 | 0·04 |
| Copper sulphate | ........... | ........... | 1·0 |

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Pb, per cent | Zn, per cent | Ag, oz./ton | Pb | Zn | Ag |
| Heads (calculated) | 100·00 | 5·94 | 1·67 | 5·61 | 100·0 | 100·0 | 100·0 |
| Lead concentrate | 11·15 | 44·80 | 1·04 | 40·00 | 84·1 | 7·0 | 79·5 |
| Lead middling | 8·17 | 7·90 | 1·88 | 6·68 | 10·9 | 9·1 | 9·7 |
| Zinc concentrate | 3·23 | 0·86 | 39·70 | 8·00 | 0·5 | 76·9 | 4·6 |
| Tailing | 77·45 | 0·35 | 0·15 | 0·45 | 4·5 | 7·0 | 6·2 |

These conditions result in a recovery of 95·0 per cent of the lead and 89·2 per cent of the silver in a rougher lead concentrate. By cleaning this, a finished concentrate assaying 44·8 per cent lead and 40·0 oz. silver is obtained; 76·9 per cent of the zinc is recovered in a concentrate assaying 39·7 per cent zinc. The grade of this would be raised by cleaning.

### *Test No. 2*

*Reagents in Pounds Per Ton of Ore—*

| — | To ball mill | To lead cells | To zinc cells |
|---|---|---|---|
| Soda ash | 6·0 | ............ | ............ |
| Cyanide | 0·14 | ............ | ............ |
| Zinc sulphate | 0·20 | ............ | ............ |
| Aerofloat No. 25 | ............ | 0·14 | 0·04 |
| Copper sulphate | ............ | ............ | 1·0 |

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Pb, per cent | Zn, per cent | Ag, oz./ton | Pb | Zn | Ag |
| Heads (calculated) | 100·00 | 5·91 | 1·65 | 5·23 | 100·0 | 100·0 | 100·0 |
| Lead concentrate | 8·86 | 49·60 | 0·49 | 39·60 | 74·3 | 2·6 | 67·1 |
| Lead middling | 8·87 | 13·70 | 1·35 | 12·22 | 20·5 | 7·3 | 20·7 |
| Zinc concentrate | 2·95 | 1·76 | 48·10 | 12·68 | 0·9 | 86·2 | 7·1 |
| Tailing | 79·32 | 0·32 | 0·08 | 0·33 | 4·3 | 3·9 | 5·1 |

The recoveries of lead and silver in the rougher lead concentrate do not show any marked difference compared with those in the preceding test. The effect of zinc sulphate added to the grinding circuit is to increase the grade of lead concentrate by depressing more of the zinc, the silver content remaining approximately the same. A high recovery of zinc is obtained—86·2 per cent in a rougher concentrate assaying 48·1 per cent zinc.

### *Test No. 3*

In this test, potassium ethyl xanthate was used in place of Aerofloat No. 25.

*Reagents in Pounds per Ton of Ore—*

| — | To ball mill | To lead cells | To zinc cells |
|---|---|---|---|
| Soda ash | 6·0 | | |
| Cyanide | 0·14 | | |
| Zinc sulphate | 0·10 | | |
| Potassium xanthate | | 0·15 | 0·06 |
| Pine oil | | 0·04 | 0·02 |

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Pb, per cent | Zn, per cent | Ag, oz./ton | Pb | Zn | Ag |
| Heads (calculated) | 100·00 | 6·11 | 1·70 | 5·55 | 100·0 | 100·0 | 100·0 |
| Lead concentrate | 14·45 | 39·85 | 2·13 | 35·50 | 94·2 | 18·1 | 92·4 |
| Lead middling | 6·17 | 2·21 | 2·63 | 2·36 | 2·2 | 9·5 | 2·6 |
| Zinc concentrate | 2·90 | 0·65 | 41·26 | 1·64 | 0·3 | 70·2 | 0·9 |
| Tailing | 76·48 | 0·26 | 0·05 | 0·30 | 3·3 | 2·2 | 4·1 |

The combination of reagents used in this test gives higher recoveries than those in Tests Nos. 1 and 2. The rougher lead concentrate contains 96·4 per cent of the lead and 95·0 per cent of the silver. The grade of finished concentrate depends largely on the manipulation of the cleaner cells. In this test the finished concentrate assayed 39·85 per cent lead and 35·5 ounces silver per ton.

## SUMMARY AND CONCLUSIONS

These tests indicate that high recoveries of silver and lead are possible by flotation. The finely divided nature of the iron pyrite makes it necessary to grind comparatively fine. Gravity concentration on jigs and tables would not be suitable as a separation of the lead and zinc minerals would not be obtained with the same ease. The flotation process can be depended on to recover over 90 per cent of the lead and silver in a lead concentrate suitable for shipment to a smelter.

For a tonnage of 100 to 250 tons a day, the flow-sheet would include primary breakers feeding ball or rod mills in closed circuit with classifiers. The classifier overflow at approximately 35 to 40 per cent solids passes to lead flotation cells. The tailing from these cells then passes to a conditioning tank where copper sulphate reactivates the zinc minerals before entering the second series of flotation machines where the zinc is recovered.

The rougher concentrate from each unit passes to cleaner cells, the underflow of which may be returned to the head of the rougher cells or other points in the circuit best determined by actual mill operation.

To prepare the concentrates for shipment to a smelter, it will be necessary to dewater, filter and dry them.

This ore presents no exceptional difficulties and high recoveries of the metals may confidently be expected.

## Report No. 367

### EXPERIMENTAL TESTS ON GOLD ORE FROM THE SAN ANTONIO GOLD MINES, LIMITED, BISSETT, MANITOBA

**J. S. Godard**

*Shipment.* A shipment of 1,000 pounds was received August 2, 1930, from San Antonio Gold Mines, Limited, Bissett, Manitoba.

*Characteristics and Analyses of the Ore.* The sample submitted was a gold ore. No free gold was visible in selected hand specimens but some flakes were observed in the table concentrates and on the coarser screen sizes. About 75 per cent can be amalgamated on grinding to 2 per cent on 65 mesh, 45 per cent −200 mesh; the remainder is associated with pyrite which is the predominating sulphide.

Two types of pyrite are present. A fine-grained one, which occurs in patches in the silica, and a much coarser-grained one which occurs in the greenstone portion of the gangue. Crystals in the latter are isolated and are as large as 5 mm. measured along the edges.

The gangue is chiefly silica and apparently represents an intrusion and replacement of the greenstone.

The head sample contained 1·31 ounces per ton in gold.

*Purpose of Experimental Tests.* The purpose of the experimental tests was to determine a suitable method for the recovery of the gold.

### Experimental Tests

### CYANIDATION

Tests Nos. 1 to 4 consisted of cyanidation tests on ore dry crushed through given meshes.

The ore was agitated 48 hours in a 0·05 per cent solution; dilution 3 : 1.

| Test No. | Mesh | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au | Average tailing, Au, oz./ton |
|---|---|---|---|---|---|
| 1 | +65 | 13·7 | 0·18 | 26·4 | 0·094 |
| | +100 | 18·6 | 0·13 | 25·9 | |
| | +200 | 25·0 | 0·11 | 29·4 | |
| | −200 | 42·7 | 0·04 | 18·3 | |
| 2 | +100 | 26·5 | 0·14 | 45·5 | 0·082 |
| | +200 | 25·2 | 0·10 | 30·9 | |
| | −200 | 48·3 | 0·04 | 23·6 | |
| 3 | +200 | 43·2 | 0·08 | 63·5 | 0·055 |
| | −200 | 56·8 | 0·035 | 36·5 | |
| 4 | +200 | 21·7 | 0·08 | 42·5 | 0·041 |
| | −200 | 78·3 | 0·03 | 57·5 | |

*Summary—*

Head sample: Au 1·235 oz./ton.

| No. | Mesh | Tailing, Au, oz./ton | Extraction, per cent | Reagents, lb./ton | |
|---|---|---|---|---|---|
| | | | | KCN | CaO |
| 1 | −48 | 0·094 | 92·5 | 0·36 | 2·36 |
| 2 | −65 | 0·082 | 93·4 | 0·41 | 2·54 |
| 3 | −100 | 0·055 | 95·6 | 0·47 | 2·65 |
| 4 | −150 | 0·041 | 96·7 | 0·47 | 3·80 |

### *Tests Nos. 5 and 6*

The ore was ground in cyanide solution containing KCN 0·05 per cent, CaO 0·10 per cent; dilution 1 : 1.

Agitation dilution was 3 : 1 and the total time of each test was 48 hours.

| Test No. | Mesh | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au | Average tailing, Au, oz./ton |
|---|---|---|---|---|---|
| 5 | +65 | 1·1 | 0·10 | 2·3 | 0·048 |
| | +100 | 8·4 | 0·04 | 7·1 | |
| | +150 | 10·0 | 0·07 | 14·7 | |
| | +200 | 22·9 | 0·07 | 33·5 | |
| | −200 | 57·6 | 0·035 | 42·4 | |
| 6 | +200 | 28·9 | 0·08 | 52·0 | 0·044 |
| | −200 | 71·1 | 0·03 | 48·0 | |

*Summary—*

Head sample: Au 1·30 oz./ton.

| No. | Mesh | Tailing, Au, oz./ton | Extraction, per cent | Reagents, lb./ton | |
|---|---|---|---|---|---|
| | | | | KCN | CaO |
| 5 | 9·5 per cent +100 | 0·048 | 96·4 | 0·40 | 2·70 |
| 6 | 28·9 per cent +200 | 0·044 | 96·7 | 0·40 | 2·84 |

## CONCENTRATION TESTS

### *Tests Nos. 7-9: Flotation, Tabling the Flotation Tailing*

The ore was floated and the tailing tabled.

*Reagents—*

$Na_2CO_3$ 2·0 lb. per ton, coal tar 0·15 lb. per ton, potassium xanthate 0·25 lb. per ton, pine oil 0·04 lb. per ton.

*Results*—

| Test No. | Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values Au |
|---|---|---|---|---|
| 7 | Flotation concentrate | 6·9 | 14·82 | 84·3}96·5 |
| | Table concentrate | 11·8 | 1·25 | 12·2 |
| | Table tailing +48 | 1·0 | 0·42 | 0·3 |
| | " +65 | 4·8 | 0·10 | 0·4 |
| | " +100 | 8·8 | 0·09 | 0·6 |
| | " +200 | 18·1 | 0·06 | 0·9 |
| | " −200 | 14·9 | 0·04 | 0·5 |
| | Slimes | 33·7 | 0·03 | 0·8 |
| 8 | Flotation concentrate | 8·7 | 12·00 | 87·4}96·6 |
| | Table concentrate | 10·3 | 1·07 | 9·2 |
| | Table tailing +100 | 4·2 | 0·20 | 0·7 |
| | " +200 | 19·3 | 0·06 | 1·0 |
| | " −200 | 26·7 | 0·04 | 0·9 |
| | Slimes | 30·8 | 0·03 | 0·8 |
| 9 | Flotation concentrate | 8·6 | 14·36 | 90·6}98·0 |
| | Table concentrate | 4·6 | 2·20 | 7·4 |
| | Table tailing +200 | 14·0 | 0·07 | 0·7 |
| | " −200 | 35·8 | 0·03 | 0·8 |
| | Slimes | 37·0 | 0·02 | 0·5 |

### *Test No. 10: Flotation*

After flotation the tailings were screened.

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 6·1 | 17·96 | 87·6 |
| Tailing +48 | 0·3 | 16·68 | 4·0 |
| " +65 | 2·3 | 1·14 | 2·1 |
| " +100 | 11·6 | 0·28 | 2·6 |
| " +200 | 28·9 | 0·11 | 2·5 |
| " −200 | 50·8 | 0·03 | 1·2 |

### *Test No. 11: Flotation—Amalgamation*

After flotation the tailings were amalgamated.

| Product | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au | Per cent values, Au of Hd. Sample |
|---|---|---|---|---|
| Flotation concentrate | 5·7 | 20·28 | 96·4 | 91·8 |
| Amal. tailing +48 | 0·2 | 0·13 | 0·3 | 0·3 |
| " +65 | 2·2 | 0·15 | | |
| " +100 | 13·0 | 0·10 | 1·1 | 1·0 |
| " +200 | 29·1 | 0·06 | 1·4 | 1·4 |
| " −200 | 49·8 | 0·02 | 0·8 | 0·8 |

Recovery by amalgamation of the flotation tailing—4·7 per cent.
Total recovery=96·5 per cent.

## AMALGAMATION—FLOTATION

### *Test No. 12*

The amalgamation tailings were floated.

| — | Weight, per cent | Assays, Au oz./ton | Per cent of values |
|---|---|---|---|
| Flotation concentrate | 8·6 | 3·20 | 86·3 |
| Flotation tailings+65 | 2·0 | 0·09 | 0·6 |
| " +100 | 11·8 | 0·10 | 3·7 |
| " +200 | 32·4 | 0·05 | 5·1 |
| " −200 | 45·2 | 0·03 | 4·3 |

Amalgamation tailing—Au 0·32 oz. per ton.
Head sample—Au 1·26 oz. per ton.
Recovery by amalgamation—74·5 per cent.
Total recovery, 74·5 per cent + 22·0 per cent=96·5 per cent.

## TWO-STAGE FLOTATION

### *Tests Nos 13 and 14*

| Test No. | — | Weight, per cent | Assays, Au oz./ton | Per cent of values |
|---|---|---|---|---|
| 13 | No. 1 concentrate | 4·5 | 24·30 | 90·3 } 96·2 |
| | No. 2 concentrate | 1·4 | 5·12 | 5·9 |
| | Classifier overflow +200 | 0·7 | 0·06 | } 0·5 |
| | " " −200 | 27·5 | 0·02 | |
| | Tailing +100 | 5·6 | 0·39 | 1·8 |
| | " +200 | 30·0 | 0·04 | 1·0 |
| | " −200 | 30·3 | 0·02 | 0·5 |
| 14 | No. 1 concentrate | 4·8 | 23·12 | 88·9 } 95·8 |
| | No. 2 concentrate | 2·6 | 3·30 | 6·9 |
| | Classifier overflow+200 | 1·7 | 0·03 | } 0·5 |
| | " " −200 | 29·2 | 0·02 | |
| | Tailing +65 | 0·3 | 9·58* | 2·3 |
| | " +100 | 3·7 | 0·24 | 0·7 |
| | " +200 | 29·4 | 0·02 | 0·5 |
| | " −200 | 28·3 | 0·01 | 0·2 |

*8·22 oz. per ton of gold were present as metallics.

### *Approximate Size at which first Flotation was made*

| Mesh | Weight, per cent | Mesh | Weight, per cent |
|---|---|---|---|
| +48 | 1·2 | +200 | 22·3 |
| +65 | 5·9 | −200 | 59·8 |
| +100 | 10·8 | | |

## CONCENTRATION, FLOTATION AND TABLING: CYANIDATION OF THE CONCENTRATES

### *Test No. 15*

Concentration as in Test No. 7. Grinding intermediate between Tests Nos. 7 and 8.

Assay of combined concentrates—Au 13·56 oz/ton.

Ratio of concentration 8·9 : 1

## CYANIDATION OF CONCENTRATES

### *Test No. 15A*

Concentrates ground 5 minutes in ball mill in 1:1 pulp, KCN 0·10 per cent, CaO 0·069 per cent, and CaO 4·0 lb. per ton.

Dilution during agitation period 3 : 1.

Time of agitation 71 hours.

Cyanide strength during agitation, KCN 0·10 per cent.

*Cyanide Tailing Screened—*

| Mesh | Weight, per cent | Assays, Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +65 | 0·1 | 0·58 | 0·1 |
| +100 | 4·2 | 0·74 | 6·9 |
| +200 | 34·5 | 0·65 | 49·7 |
| −200 | 61·2 | 0·32 | 43·3 |

Average tailing—Au 0·452 oz. per ton.

### *Test No. 15B*

Concentrates ground 35 minutes in ball mill in 1:1 pulp, KCN 0·10 per cent, CaO 0·069 per cent, and CaO 6·0 lb. per ton.

Dilution during agitation period 3:1.

Time of agitation 71 hours.

Cyanide strength during agitation, KCN, 0·10 per cent.

*Cyanide Tailing Screened—*

| Mesh | Weight, per cent | Assays Au oz./ton | Per cent of values, Au |
|---|---|---|---|
| +200 | 10·3 | 0·37 | 15·0 |
| −200 | 89·7 | 0·24 | 85·0 |

Average tailing—Au 0·253 oz. per ton.

*Summary.* Recovery by concentration 96·6 per cent.

Head sample to cyanidation—Au 13·56 oz. per ton.

| Test No. | Average tailing, Au, oz./ton | Extraction, per cent | Recovery of total Au | Reagents | | | |
|---|---|---|---|---|---|---|---|
| | | | | Per ton concentrate | | Per ton ore | |
| | | | | KCN | CaO | KCN | CaO |
| 15-A................. | 0·452 | 96·7 | 93·5 | 3·31 | 6·49 | 0·37 | 0·73 |
| 15-B................. | 0·253 | 98·2 | 94·8 | 3·38 | 8·25 | 0·38 | 0·93 |

## CONCLUSIONS

*Cyanidation.* Good extractions were generally obtained in the cyanide tests on this ore. Grinding to 1 per cent +65 mesh and 60 per cent −200 mesh, would probably prove good practice. The consumption of cyanide and lime are very moderate.

*Concentration.* Flotation followed by tabling the flotation tailing gave excellent recoveries of the gold. Invariably some fine free gold was recovered in the table concentrate, indicating that flotation alone is insufficient to secure a high recovery. De-sliming of the flotation tailing before tabling would eliminate about 60 per cent of the flotation tailing without appreciable reduction in the recovery.

In Test No. 10, an attempt was made to supplant tabling of the flotation tailing by substituting a screen to remove the free gold. This was not as effective as tabling.

In Test No. 13, two-stage flotation was practised, but as in the single-stage flotation the coarser tailing was high in gold, though the finer tailing products were satisfactory and might be discarded.

In Test No. 14, the two-stage flotation was followed by a 65-mesh screening of the final tailing; 95·8 per cent of the gold was recovered in the two concentrates and an additional 2·3 per cent on the screen. This latter product was nearly all free gold. A good ratio of concentration, 13 : 1, was made in this test.

*Amalgamation, and Flotation of the Amalgamation Tailing.* By this method 96·5 per cent of the gold was recovered, of which 74·5 per cent was by amalgamation.

An identical total recovery was obtained when the order of the operations was reversed. There is some doubt as to the efficiency if this latter order were practised because of the difficulty of keeping the plates free from the flotation reagents.

*Concentration, Cyanidation of the Concentrates.* Cyanidation of the combined concentrates from flotation and tabling gave extractions approximating those obtained in cyanidation of the ore. The consumption of reagents was moderate, the cyanide consumption is approximately the same when based on a ton of ore. The saving in lime will be more than offset by the cost of the necessary flotation reagents.

When compared with straight cyanidation much may be said for and against this latter method of treatment. Local conditions, proposed tonnage of ore to be milled, possibilities of a change in the nature of the ore, and the predilection of those in charge of operations will no doubt prove to be the deciding factors.

## Report No. 368

### THE RECOVERY OF COPPER AND GOLD FROM THE ORE OF THE TASHOTA GOLD MINES, LTD., TASHOTA, ONTARIO.

**A. K. Anderson**

*Shipment.* Ten sacks of ore containing 765 pounds were received at the Ore Dressing Laboratories on May 2, 1930, from the Tashota Gold Mines, Limited, Tashota, Ontario, with head office at Room 401—1111 Beaver Hall Hill, Montreal, Que. H. S. Denny, Room 423, Brock Building, 200 Bay Street, Toronto 2, Ontario, is consulting engineer.

*Characteristics of the Ore.* The ore consisted of a basic gangue intermixed with white quartz. Chalcopyrite and pyrrhotite were the principal minerals present.

*Purpose of Experimental Tests.* The shipment was made for the purpose of determining what recovery of the values could be obtained.

*Sampling and Analysis.* The lot was crushed to pass ½ inch and quartered. One-quarter was further reduced in size to pass 14 mesh and again quartered. Further reductions in size with intervening cuts through a Jones riffle sampler resulted in a sample passing 100 mesh which on analysis showed the shipment to contain 1·57 per cent copper, 1·85 oz. gold and 1·87 oz. silver per ton.

### Experimental Tests

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

The tests were run on 1,000-gramme representative portions of the ore, crushed to pass 14 mesh ground in porcelain mills containing iron balls to pass 83 per cent through 200 mesh unless otherwise stated. Reagents as indicated in the individual tests were added to the grinding mill.

### FLOTATION

#### *Test No. 1*

In this test a rougher copper concentrate was first taken off and cleaned making a finished concentrate and middling. Copper sulphate was then added together with an additional quantity of reagent and a pyrrhotite concentrate removed.

*Reagents in Pounds per Ton of Ore—*

| — | To ball mill | To copper cells | To pyrrhotite cells |
|---|---|---|---|
| Soda ash | 5·0 | | |
| Cyanide | 0·10 | | |
| Potassium xanthate | | 0·10 | 0·10 |
| Copper sulphate | | | 1·0 |
| Pine oil | | 0·04 | 0·04 |

*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 | 1·54 | 1·68 | 1·75 | 100·0 | 100·0 | 100·0 |
| Copper concentrate | 5·7 | 25·08 | 21·06 | 24·26 | 92·7 | 71·4 | 78·8 |
| Copper middling | 11·7 | 0·40 | 1·66 | 1·46 | 3·0 | 11·5 | 9·7 |
| Pyrrhotite concentrate | 4·5 | 0·94 | 0·48 | 1·00 | 2·7 | 1·3 | 2·6 |
| Tailing | 78·1 | 0·03 | 0·34 | 0·20 | 1·6 | 15·8 | 8·9 |

This test shows a recovery of 95·7 per cent of the copper, 82·9 per cent of the gold and 88·5 per cent of the silver in the rougher copper concentrate. A further recovery of 1·3 per cent of the gold is made by floating the pyrrhotite obtaining a concentrate assaying $9.60 per ton. This leaves $7.20 in the tailing.

A screen analysis of the tailing is of interest.

| Mesh | Weight, per cent | Gold, oz./ton | Silver, oz./ton |
|---|---|---|---|
| +100 | 0·9 | 12·67 | 7·96 |
| −100+150 | 3·6 | 0·95 | 0·55 |
| −150+200 | 12·5 | 0·39 | 0·21 |
| −200 | 83·0 | 0·17 | 0·10 |

These results would seem to indicate that some free gold is present in the ore. Fine grinding apparently is necessary to secure highest recoveries.

### *Test No. 2*

In this test, lime was used in place of soda ash, and hardwood creosote in the flotation of the pyrrhotite. Other conditions were the same as in Test No. 1.

*Reagents in Pounds per Ton of Ore—*

| — | To ball mill | To copper cells | To pyrrhotite cells |
|---|---|---|---|
| Lime | 4·0 | | |
| Potassium xanthate | | 0·10 | 0·10 |
| Copper sulphate | | | 1·0 |
| Hardwood creosote | | | 0·08 |
| Pine oil | | 0·04 | 0·06 |

*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 | 1·53 | 1·77 | 1·87 | 100·0 | 100·0 | 100·0 |
| Copper concentrate | 4·8 | 28·56 | 19·32 | 23·48 | 89·6 | 52·4 | 60·3 |
| Copper middling | 6·0 | 0·96 | 2·38 | 3·06 | 3·8 | 8·1 | 9·8 |
| Pyrrhotite concentrate | 18·5 | 0·43 | 1·60 | 1·76 | 5·2 | 16·7 | 17·4 |
| Tailing | 70·7 | 0·03 | 0·57 | 0·33 | 1·4 | 22·8 | 12·5 |

These results show the depressing action of lime on gold. The rougher copper concentrate contains only 60·5 per cent of the gold in the ore. The pyrrhotite concentrate is of much higher value than that of Test No. 1, and the final tailing has a value of $11.40 in gold.

### *Test No. 3*

In this test the ore was ground with soda ash and cyanide and the copper floated with T.T. mixture. A pyrrhotite concentrate was made as in preceding tests.

*Reagents in Pounds per Ton of Ore—*

| —— | To ball mill | To copper cells | To pyrrhotite cells |
|---|---|---|---|
| Soda ash | 5·0 | | |
| Cyanide | 0·20 | | |
| T. T. mixture | | 0·08 | |
| Hardwood creosote | | | 0·06 |

*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 | 1·55 | 1·43 | 1·40 | 100·0 | 100·0 | 100·0 |
| Copper concentrate | 4·3 | 30·16 | 15·40 | 18·40 | 83·7 | 46·3 | 56·4 |
| Copper middling | 3·6 | 4·56 | 8·90 | 8·26 | 10·6 | 22·4 | 21·2 |
| Pyrrhotite concentrate | 8·8 | 0·81 | 2·06 | 1·68 | 4·6 | 12·7 | 10·5 |
| Tailing | 83·3 | 0·02 | 0·32 | 0·20 | 1·1 | 18·6 | 11·9 |

T.T. mixture apparently is not so effective in the flotation of gold as potassium xanthate, 68·7 per cent of the gold being recovered in the copper float as against 82·9 per cent in Test No. 1.

The pyrrhotite concentrates in preceding tests were low in copper and would have to be either mixed with the copper concentrates or treated separately for the recovery of the gold content.

In the following tests an attempt was made to produce one concentrate containing the copper, gold, and silver.

### *Test No. 4*

A 1,000-gramme sample of the ore was ground as in previous tests with 5·0 pounds soda ash per ton. Amyl xanthate at the rate of 0·10 pounds per ton and 0·04 pounds pine oil were added and a rougher concentrate was removed.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Au, oz./ton | Ag, oz./ton | Cu | Au | Ag |
| Heads | 100·0 | 1·57 | 1·67 | 1·57 | 100·0 | 100·0 | 100·0 |
| Concentrate | 5·2 | 28·68 | 25·40 | 24·00 | 94·7 | 79·2 | 79·8 |
| Middling | 4·5 | 1·44 | 4·31 | 4·03 | 4·1 | 11·6 | 11·6 |
| Tailing | 90·3 | 0·02 | 0·17 | 0·15 | 1·2 | 9·2 | 8·6 |

This shows a recovery of 90·8 per cent of the gold and 91·4 per cent of the silver in the combined concentrate and middling. On cleaning the rougher concentrate a very high-grade concentrate is obtained. The gold, however, shows a tendency to drop out into the middling. The tailing contains $3.40 in gold per ton.

## FLOTATION AND TABLE CONCENTRATION

### *Test No. 5*

This test is the same as Test No. 4 with the exception that the flotation tailing was passed over a Wilfley concentrating 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 | 1·55 | 1·83 | 1·68 | 100·0 | 100·0 | 100·0 |
| Flotation concentrate | 5·1 | 28·68 | 25·40 | 24·00 | 94·1 | 70·7 | 72·8 |
| Flotation middling | 4·5 | 1·44 | 4·31 | 4·03 | 4·2 | 10·6 | 10·8 |
| Table concentrate | 7·0 | 0·20 | 2·40 | 1·82 | 0·9 | 9·2 | 7·6 |
| Table middling | 10·3 | 0·05 | 0·29 | 0·46 | 0·3 | 1·5 | 2·7 |
| Table tailing | 73·1 | 0·01 | 0·20 | 0·14 | 0·5 | 8·0 | 6·1 |

Table concentration of the tailing from the flotation machine results in 7 per cent of the weight of feed being recovered as a concentrate assaying 2·40 oz. gold per ton. A total recovery of gold in the flotation and table concentrate and middlings amounting to 92·0 per cent is obtained.

The presence of gold in the flotation tailing, and its recovery by table concentration indicates the presence of free gold.

### *Test No. 6*

To determine the amount of free gold in the ore, a 1,000-gramme sample was ground 80 per cent −200 mesh and amalgamated. This resulted in a tailing assaying 0·38 ounces gold and 0·94 ounces silver per ton, representing recoveries of 79·5 per cent of the gold and 49·7 per cent of the silver.

This shows that over three-quarters of the gold present in the sample is free.

### FLOTATION AND AMALGAMATION

### *Test No. 7*

A 1,000-gramme sample of the ore was ground as in Test No. 6, amalgamated and then floated using 6·0 pounds soda ash, 0·10 pounds amyl xanthate, and 0·04 pounds pine oil per ton.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Au, oz./ton | Ag, oz./ton | Cu | Au | Ag |
| Heads | 100·00 | 1·44 | 1·85 | 1·87 | 100·0 | 100·0 | 100·0 |
| Amalgamation | ........ | ........ | ........ | ........ | ........ | 76·5 | 48·7 |
| Flotation concentrate | 5·80 | 22·78 | 3·20 | 11·30 | 91·4 | 10·0 | 35·0 |
| Flotation middling | 12·35 | 0·67 | 0·76 | 1·20 | 5·7 | 5·1 | 7·9 |
| Flotation tailing | 81·85 | 0·05 | 0·19 | 0·19 | 2·9 | 8·4 | 8·4 |

These results check those obtained in Test No. 6, 76·5 per cent of the gold is recovered by amalgamation. On floating the tailing after amalgamation 97·1 per cent of the copper, 15·1 per cent of the gold and 42·9 per cent of the silver are recovered in the combined concentrate and middling. As the middling is much lower in grade than the original feed, no trouble may be anticipated by returning the middling to the circuit. Amalgamation followed by flotation results in recoveries of 91·6 per cent of the gold and silver in the ore.

### *Test No. 8*

To note the effect of coarser grinding on recoveries, the ore was ground to the following screen analysis,—

| | |
|---|---|
| + 65 mesh | 0·2 per cent |
| − 65+100 | 3·7 " |
| −100+150 | 8·7 " |
| −150+200 | 20·0 " |
| −200 | 67·4 " |

The sample was then floated and amalgamated.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Au, oz./ton | Ag, oz./ton | Cu | Au | Ag |
| Heads | 100·00 | 1·62 | 1·85 | 1·87 | 100·0 | 100·0 | 100·0 |
| Flotation concentrate | 12·26 | 12·64 | 11·18 | 10·62 | 95·7 | 74·1 | 69·6 |
| Amalgamation | ........ | ........ | ........ | ........ | ........ | 15·0 | 22·4 |
| Amalgamation tailing | 87·74 | 0·08 | 0·23 | 0·17 | 4·3 | 10·9 | 8·0 |

Flotation of this −65 mesh product results in a recovery of 95·7 per cent of the copper as against 97·1 per cent in the preceding test; 89·1 per cent of the gold and 92·0 per cent of the silver are also recovered.

A lower recovery of gold and copper results from coarse grinding.

### *Test No. 9*

In this test a 1,000-gramme sample of the ore was ground even coarser than in Test No. 8. A screen analysis shows the following,—

| | |
|---|---|
| + 65 mesh | 2·7 per cent |
| − 65+100 | 13·9 " |
| −100+150 | 12·1 " |
| −150+200 | 22·5 " |
| −200 | 48·8 " |

This was then floated and amalgamated as in Test No. 8.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Au, oz./ton | Ag, oz./ton | Cu | Au | Ag |
| Heads | 100·00 | 1·65 | 1·85 | 1·87 | 100·0 | 100·0 | 100·0 |
| Flotation concentrate | 17·07 | 9·20 | 7·79 | 7·56 | 95·5 | 71·9 | 69·0 |
| Amalgamation | ........ | ........ | ........ | ........ | ........ | 16·5 | 23·5 |
| Tailing | 82·93 | 0·09 | 0·26 | 0·17 | 4·5 | 11·6 | 7·5 |

While the recovery of copper is approximately the same as in Test No. 8, the grade of concentrate is much lower. To secure a high-grade concentrate regrinding of this rougher concentrate would have to be practised. The recovery of gold also is lower than in preceding tests.

## FLOTATION AND CYANIDATION

### *Test No. 10*

As flotation and amalgamation leave an appreciable amount of gold in the tailing, cyanide tests were made on the residues after flotation. In order to have a product not too low in gold for cyaniding, flotation was not made with the object of securing as low grade a tailing as possible. Grinding was such as to yield 87·4 per cent −200 mesh.

*Reagents in Pounds per Ton Ore—*

| | |
|---|---|
| Soda ash | 5·0 |
| Aerofloat No. 25 | 0·08 |

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Au, oz./ton | Ag, oz./ton | Cu | Au | Ag |
| Heads | 100·00 | 1·40 | 1·70 | 1·59 | 100·0 | 100·0 | 100·0 |
| Concentrate | 4·96 | 21·72 | 27·30 | 25·40 | 76·9 | 79·6 | 79·0 |
| Middling | 5·75 | 3·14 | 2·60 | 2·40 | 12·9 | 8·8 | 8·6 |
| Tailing | 89·29 | 0·16 | 0·22 | 0·22 | 10·2 | 11·6 | 12·4 |

A sample of the flotation tailing was amalgamated, leaving a residue assaying 0·12 ounce gold and 0·16 ounce silver per ton. This represents an additional recovery of 5·3 per cent of the gold and 3·4 per cent of the silver in the feed, or a total recovery of 93·7 per cent and 91·0 per cent of the gold and silver by flotation and amalgamation. This establishes the fact shown in the preceding tests that fine grinding yields highest recoveries.

Samples of the flotation tailing before amalgamation, with an analysis of 0·16 per cent copper, 0·22 ounce gold and 0·22 ounce silver were cyanided for 48 hours, 3 : 1 dilution with a 3 pound KCN solution and 6 pounds lime per ton of ore. This treatment left a tailing containing 0·05 ounce gold per ton.

An extraction of 77·3 per cent of the gold remaining in the flotation tailing is thus obtained, or a total recovery of 97·4 per cent by flotation and cyanidation.

There is an indicated consumption of 6·0 pounds cyanide and 5·3 pounds lime per ton of tailing treated. This high cyanide consumption is doubtless due to the 0·16 per cent of copper remaining in the flotation tailing.

### *Test No. 11*

The object of this test was to secure a flotation tailing as low in copper as possible to establish the fact that this metal is responsible for the high cyanide losses found in Test No. 10.

After removing a copper concentrate, the pyrrhotite was floated with hardwood creosote. The tailing was then amalgamated and after removing amalgam, was agitated for 48 hours, 3 : 1 dilution with a 2-pound KCN solution. Lime at the rate of 7 pounds per ton was added.

| Product | Weight, per cent | Assay | | | Per cent of values | | |
|---|---|---|---|---|---|---|---|
| | | Cu, per cent | Au, oz./ton | Ag, oz./ton | Cu | Au | Ag |
| Heads | 100·00 | 1·69 | 1·85 | 1·87 | 100·0 | 100·0 | 100·0 |
| Copper concentrate | 14·85 | 10·88 | 9·40 | 9·08 | 95·7 | 75·4 | 72·1 |
| Pyrrhotite concentrate | 10·52 | 0·33 | 0·30 | 0·34 | 2·1 | 1·7 | 1·9 |
| Tailing before amalgamation | 74·63 | 0·05 | 0·50 | | | | |
| Tailing after amalgamation | | | 0·07 | 0·07 | 2·2 | 2·8 | 2·8 |
| Recovered by amalgamation | | | | | | 20·1 | 23·2 |

The tailing from flotation after being amalgamated gave a product assaying 0·05 per cent copper, 0·07 ounce gold and 0·07 ounce silver per ton. This was cyanided leaving a residue containing 0·01 ounce gold and 0·01 ounce silver, or an additional recovery of 2·4 per cent calculated on the original feed. Cyanide consumed was 1·2 pounds per ton of material treated, much lower than in Test No. 10 where the material cyanided contained 0·16 per cent copper; 4·9 pounds lime per ton were required.

The pyrrhotite concentrate produced is too low grade to be added to the copper concentrate and contains too much copper to be cyanided economically. The gold it contains represents 1·7 per cent of the total with a value of $6 per ton.

The recovery in the copper concentrate plus that by amalgamation amounts to 95·7 per cent of the copper, 95·5 per cent of the gold, and 95·3 per cent of the silver.

The object of the test—to determine if a flotation tailing low in copper could be cyanided with a reasonable reagent cost—is clearly demonstrated. In Test No. 10 where the copper content of the feed to cyanidation contained 0·16 per cent copper, the cyanide consumed was 6 pounds per ton, while a 0·05 per cent copper content gives 1·2 pounds.

## SUMMARY AND CONCLUSIONS

All tests indicate that recoveries of from 90 to 95 per cent of the copper can be easily obtained. Due to the presence of free gold, flotation alone does not yield maximum recoveries of this metal.

Flotation followed by table concentration as in Test No. 5 yields a recovery of 92·0 per cent of the gold in concentrates and middlings.

Tests Nos. 6 and 7 show that 75 to 80 per cent of the gold is free and can be recovered by amalgamation.

Amalgamation followed by flotation as shown in Test No. 7 gives a recovery of 91·6 per cent of the gold.

Flotation followed by amalgamation gives 93·7 per cent recovery of the gold, and when augmented by cyanidation, 97·4 per cent.

It will be noticed that in the earlier tests when the ore was fresh there was no difficulty in producing a flotation tailing under 0·05 per cent copper. As the ore aged, however, flotation had to be forced to reduce it to 0·05 per cent. In Test No. 12, all the sulphides including pyrrhotite had to be floated. As copper sulphides are strong cyanicides it is necessary to have a low copper content in the feed to the cyanide annex or high-cyanide consumption and high refining costs on the low-grade bullion produced will result.

A combination of flotation, amalgamation, and cyanidation yields maximum recoveries. However, due to the strong possibility that the ore from the mine may oxidize before being milled, it is advised that the cyanide annex be considered very carefully before it is erected.

The process indicated is one employing a combination of amalgamation and flotation. To secure a high-grade copper concentrate, grinding to 80 per cent −200 mesh will yield best results. Amalgamation between the grinding mill and classifier will do away with the certainty of gold being held in the classifier and discharged at odd intervals. Equally good recoveries doubtless would be obtained by placing the amalgamation plates after the flotation machines if these are of the best type for the flotation of gold with no dead spots in which the gold can concentrate.

Amalgamation before flotation will recover 75 per cent of the gold. If following flotation, the bulk of the gold will be in the copper concentrate and recovered by the copper smelter.

The process, therefore, is amalgamation followed by flotation. The flotation tailing should be impounded until sufficient data have been collected from the operation of the mill to determine the nature of the mill tailing before considering the feasibility of a cyanide annex.

## Report No. 369

## THE RECOVERY OF GOLD FROM THE ORE OF THE NIGHT HAWK LAKE MINING CO., LTD., NIGHT HAWK LAKE, ONTARIO

**A. K. Anderson**

*Shipment.* A shipment contained in 36 bags, gross weight 3,590 pounds, was received by freight at the Ore Dressing and Metallurgical Laboratories on September 6, 1930, consigned by James G. Cameron from Connaught Station, Ontario. The Callinan-Flin Flon Mines, Limited, Mount Julian P.O., Ont., John W. Callinan, President, are also interested. The shipment was made up of two lots, No. 1 and No. 2, of approximately equal weights.

*Characteristics of the Ore.* The material consisted of a red porphyry containing small crystals of iron pyrite free from oxidation.

*Purpose of Experimental Tests.* The shipment was made for the purpose of determining the value of the ore, the amount of free gold present, and the best method of recovering the contained values.

*Sampling and Analysis.* Each lot was crushed to pass ¾-inch mesh and quartered. One-quarter was then crushed to pass 6 mesh and again quartered. By further grinding to finer sizes and cutting through a Jones riffle sampler, a representative portion passing 100 mesh was finally secured. Fire assay showed the following,—

| | | | |
|---|---|---|---|
| Lot No. 1—Gold........ | 0·17 oz./ton | Silver.................... | 0·07 oz./ton |
| Lot No. 2—Gold........ | 0·14 " | Silver.................... | 0·03 " |

### Experimental Tests

Due to the similarity of the two lots, all tests were made on Lot No. 1. These included amalgamation; amalgamation and concentration by flotation; flotation; cyanidation and flotation and cyanidation of the flotation concentrate.

The investigation discloses that approximately 80 per cent of the gold is free with a probable recovery of 58 per cent by standard amalgamation practice. Flotation gives a recovery of 94 per cent with fine grinding. Amalgamation followed by flotation recovers 96·7 per cent of the gold. On cyaniding these concentrates, the indicated recovery drops to 81 per cent. Cyanidation of the raw ore gives an extraction of 97 per cent of the gold.

The tests unless otherwise stated were made on representative 1,000-gramme portions of the ore ground in porcelain mills containing iron balls.

### AMALGAMATION

### *Test No. 1*

A 1,000-gramme sample of the ore −14 mesh was ground to pass approximately 94 per cent −200 mesh and amalgamated.

| | | |
|---|---|---|
| Heads.......................................................... | 0·17 | oz. gold |
| Amalgamation tailing.............................................. | 0·035 | " |
| Recovery....................................................... | 79·4 | per cent |

This indicates that over three-quarters of the gold in the sample is free.

## AMALGAMATION AND FLOTATION

### *Test No. 2*

A 1,000-gramme sample of the ore was ground in water with 4·0 pounds soda ash per ton to pass 94 per cent through 200 mesh and then amalgamated. After removing the amalgam, the pulp was transferred to a laboratory-size mechanically agitated flotation machine. A concentrate was removed after adding 0·10 pound potassium xanthate and 0·04 pound pine oil.

| Product | Weight, per cent | Assay, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Heads | 100·00 | 0·17 | 100·0 |
| Flotation concentrate | 3·13 | 0·48 | 8·8 |
| Flotation tailing | 96·87 | 0·01 | 5·7 |
| Recovered by amalgamation | ............ | ............ | 85·5 |

These results show that 85·5 per cent of the gold is recovered by amalgamation at this fineness of grinding, and an additional 8·8 per cent in a flotation concentrate assaying $9.60 a ton, or a total recovery of 94·3 per cent of the gold. Each 100 tons of feed produces 3·13 tons of flotation concentrate.

## FLOTATION

### *Test No. 3*

In this test concentration without amalgamation was applied.

A representative 1,000-gramme portion of the ore −14 mesh was ground in water containing soda ash equivalent to 4·0 pounds per ton of ore. After grinding to 94 per cent −200 mesh, the pulp was transferred to the flotation machine where 0·10 pound potassium xanthate and 0·04 pound pine oil per ton were added and a concentrate removed.

| Product | Weight, per cent | Assay, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Heads | 100·00 | 0·168 | 100·0 |
| Flotation concentrate | 3·58 | 4·42 | 94·2 |
| Flotation tailing | 96·42 | 0·01 | 5·8 |

Flotation at this fineness of grinding gives a recovery of 94·2 per cent of the gold in a concentrate assaying 4·42 oz. gold per ton.

A sizing test on the tailing shows:—

| | |
|---|---|
| +100 mesh | 0·2 per cent |
| −100+150 | 1·9 " |
| −150+200 | 3·8 " |
| −200 | 94·1 " |

### *Test No. 4*

Flotation of coarser ground material was then tried. Grinding in this test was:

| | |
|---|---|
| +65 mesh | 1·7 per cent |
| −65+100 | 8·0 " |
| −100+150 | 10·7 " |
| −150+200 | 19·0 " |
| −200 | 60·6 " |

Flotation was conducted as in Test No. 3.

| Product | Weight, per cent | Assay, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Heads | 100·00 | 0·164 | 100·0 |
| Flotation concentrate | 3·18 | 4·10 | 79·4 |
| Flotation tailing | 96·82 | 0·035 | 20·6 |

On this size of material, 60 per cent −200 mesh, the recovery of gold is 79·4 per cent as against 94·2 per cent as in Test No. 3.

### *Test No. 5*

This test was made on material even coarser than in Test No. 4.

| Product | Weight, per cent | Assay, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Heads | 100·00 | 0·188 | 100·0 |
| Flotation concentrate | 3·40 | 4·4 | 79·5 |
| Flotation tailing | 96·60 | 0·04 | 20·5 |

A screen analysis of the tailing was made to note the whereabouts of the unrecovered gold.

| Mesh | Weight, per cent | Assay, Au, oz./ton |
|---|---|---|
| +65 | 10·5 | 0·19 |
| −65+100 | 25·1 | 0·04 |
| −100+150 | 6·5 | 0·07 |
| −150+200 | 14·8 | 0·02 |
| −200 | 43·1 | 0·01 |

This screen analysis of the tailing shows that the +150 mesh portion is of much higher value than the −150 mesh part. To secure maximum recoveries by flotation it is apparent that the ore should be ground to pass 150 mesh.

## CYANIDATION

### *Test No. 6*

In this test, the ore was ground to pass 100 mesh and agitated for 48 hours, 1 : 3 dilution with a 2·0 pound KCN solution. Lime at the rate of 2 pounds per ton of ore was added.

The tailing after this treatment had a gold content of 0·005 ounce per ton representing an extraction of 97·0 per cent of the gold. Reagent consumption was 0·54 pound cyanide and 1·7 pounds lime per ton of ore milled.

### *Test No. 7*

This test was made on ore ground to pass 150 mesh treated in the same manner as in Test No. 6.

| | | |
|---|---|---|
| Tailing | 0·005 | oz. gold per ton |
| Extraction | 97·0 | per cent |
| Consumption—KCN | 0·60 | lb. per ton |
| CaO | 1·7 | lb. per ton |

These tests show that the raw ore is amenable to cyanidation. No benefit is to be derived by grinding finer than 100 mesh.

## CONCENTRATION AND CYANIDATION

### *Test No. 8.*

In this test, a flotation concentrate was made as in previous tests. The concentrate was then reground and cyanided.

The ore was ground with water and soda ash equivalent to 4·0 pounds of ore. A concentrate was then removed by the addition of 0·10 pounds potassium xanthate and 0·06 pounds pine oil per ton.

| Product | Weight, per cent | Assay, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Heads | 100·00 | 0·17 | 100·0 |
| Flotation concentrate | 6·83 | 2·08 | 83·6 |
| Tailing | 93·17 | 0·03 | 16·4 |

This concentrate with an assay value of 2·08 oz. gold per ton was reground and then agitated for 48 hours, 1 : 3 dilution with a 2·0 pound KCN solution and 4 pounds lime per ton of ore.

This treatment left a residue containing 0·05 oz. gold per ton representing an extraction of 97·6 per cent of the gold in the concentrate or a recovery of 81·6 per cent on the original feed. A cyanide consumption of 9·0 pounds KCN per ton of concentrate is indicated.

A sample of the concentrate was first amalgamated and then cyanided as above.

By amalgamation the gold in the concentrate was reduced from 2·08 oz. to 0·38 oz. per ton. The residue after cyaniding for 48 hours had a gold content of 0·02 oz. This represents a recovery of 68·3 per cent of the gold in the original feed by amalgamation with an additional 14·4 per cent by cyanidation, a total recovery of 82·7 per cent by this method.

### *Test No. 9*

In this test approximately 700 pounds of the ore was fed at the rate of 100 pounds per hour to a small rod mill, the discharge of which passed over an amalgamating plate. The pulp after passing the plate was elevated to a classifier the oversize from which was returned to the ball mill. The classifier overflow at approximately 35 per cent solids flowed to a conditioning tank where about 20 minutes' contact was obtained and thence to a 6-cell, Denver Sub-A flotation machine. Cells No. 2 and No. 3 were used as rougher cells, the concentrate from these flowing to cell No. 1 where the finished concentrate was removed. Cells No. 4, No. 5, and No. 6 were maintained as scavenger cells, the concentrate from these being returned to cell No. 2 together with the middling from cell No. 1.

Soda ash at the rate of 3 pounds per ton was added to the grinding mill, 0·10 pounds amyl xanthate to the conditioning tank, and 0·04 pounds pine oil to the flotation machine.

The concentrate was then cyanided in a small-scale apparatus.

| Product | Weight, per cent | Assay, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Heads | 100·00 | 0·17 | 100·0 |
| Amalgam tailing | ........... | 0·07 | 58·8* |
| Flotation concentrate | 7·96 | 0·81 | 37·9 |
| Flotation tailing | 92·04 | 0·006 | 3·3 |

*Recovery in amalgam.

These results show that amalgamation followed by flotation results in a recovery of 96·7 per cent of the gold in the ore. A concentrate containing 0·81 oz. gold per ton is obtained in a product equal to 7·96 per cent of the weight of ore milled. This concentrate was rather pure pyrite containing 43·8 per cent iron, and 46·8 per cent sulphur.

The following screen tests made on the plate tailing and flotation tailing or classifier overflow are of interest.

| Mesh | Plate tailing, per cent | Classifier overflow, per cent |
|---|---|---|
| +65 | 8·6 | 0·0 |
| −65+100 | 17·7 | 2·1 |
| −100+150 | 10·3 | 1·7 |
| −150+200 | 22·1 | 6·7 |
| −200 | 41·3 | 89·5 |
|  | 100·0 | 100·0 |

The flotation tailing was filtered and representative 200-gramme portions were reground and agitated for 48 hours, 1 : 3 dilution with a 3-pound per ton KCN solution and lime equivalent to 5 pounds per ton of ore. It was found that the solutions became foul from the presence of soluble sulphides. The addition of 4 pounds lead acetate per ton corrected this condition. Very poor results were obtained, however, only 20 per cent of the gold being extracted. It was felt that, owing to oxidation of the concentrate prior to cyaniding and to the possible contamination with carbon from other operations being carried on, these results are not conclusive, especially as in Test No. 8 no trouble was experienced in securing a good extraction from the concentrates. It will be necessary, therefore, to conduct further large-scale flotation tests to secure a concentrate for additional cyanide investigations.

### SUMMARY AND CONCLUSIONS

The greater part of the gold in the ore is present as free gold. When ground very fine as in Test No. 1, approximately 80 per cent of the gold is recovered by amalgamation. However, as these conditions would not be duplicated in practice, the result shown in Test No. 9, 58 per cent recovered by amalgamation is more nearly correct.

Concentration by flotation can be depended on to yield a recovery of well over 90 per cent of the gold in the ore. Fine grinding is necessary to secure this. Amalgamation and flotation as shown in Test No. 9 gives a recovery of 96·7 per cent of the gold.

Cyanidation of the ore ground to pass 100 mesh results in an extraction of 97 per cent of the gold with a cyanide consumption of 0·60 pounds KCN per ton of ore milled.

Flotation followed by cyanidation of the concentrate results in a recovery of 81·6 per cent of the gold as indicated in Test No. 8. With a ratio of concentration of 14·6 : 1, the cyanide consumption of 9 pounds per ton of concentrate figures out to 0·62 pounds per ton of original ore, closely approaching that indicated in the cyanide tests on the raw ore.

Cyanidation of the raw ore gives highest recoveries. However, due to the low gold content of the ore, $3.40 per ton, and the ease with which it is concentrated, the deciding factor in the adoption of the method to apply will be largely one of costs, the initial cost of installation and subsequent operation. These can be determined only by a detailed study of each type of mill, cyanide or amalgamation and flotation to determine if a lower final return by flotation will offset the higher costs of a cyanide plant.

## Report No. 370

### EXPERIMENTAL TESTS ON GOLD ORE FROM THE BROOKFIELD MINE OF THE UNITED GOLDFIELDS OF NOVA SCOTIA, LTD., LIVERPOOL, NOVA SCOTIA

**J. S. Godard**

*Shipment.* A shipment of 875 pounds of ore was received May 27, 1930. This ore was taken from Brookfield mine, Queens county, Nova Scotia, and was consigned by Mr. J. P. Messervey, Deputy Inspector of Mines for Nova Scotia, on behalf of Mr. E. H. Henderson, Engineer for the United Goldfields of Nova Scotia, Limited, of Liverpool, N.S.

*Characteristics and Analysis of the Ore.* The ore is a low-grade gold one. About 40 per cent of the gold is freed and amalgamable at −30 mesh. The main portion is associated with arsenopyrite and pyrite, which occur as fine-grained crystals in a hard, siliceous gangue.

The head sample contained 0·21 ounce gold per ton.

*Purpose of Experimental Tests.* Stamp-battery amalgamation has been practised on this ore crushed to pass 30-mesh screens and the amalgamation tailing concentrated on a Wilfley table.

About 40 per cent of the gold is recovered by amalgamation and an additional 30 per cent is collected in the table concentrate, which, however, has not been treated for the recovery of the gold therein.

The following experiments were conducted for the purpose of checking the amalgamation recovery and to determine a method of treatment for this ore that will substantially increase the recovery obtained with the existing flow-sheet.

## Experimental Tests

### AMALGAMATION

### *Tests Nos. 1 to 6*

The ore was crushed dry to pass given screens and amalgamated.

*Results*—

| Test No. | Mesh | Weight, per cent | Assay, Au, oz./ton | Per cent of values, Au | Average tailing, Au, oz./ton |
|---|---|---|---|---|---|
| 1................................ | +28 | 5·8 | 0·14 | 6·0 | 0·134 |
| (Ore at −20 mesh) | +35 | 16·3 | 0·17 | 20·7 | |
| | +48 | 14·6 | 0·13 | 14·2 | |
| | +65 | 16·6 | 0·14 | 17·3 | |
| | +100 | 11·6 | 0·15 | 13·0 | |
| | +200 | 10·7 | 0·11 | 8·8 | |
| | −200 | 24·4 | 0·11 | 20·0 | |
| 2................................ | +35 | 8·2 | 0·15 | 10·1 | 0·122 |
| (Ore at −28 mesh) | +48 | 20·8 | 0·13 | 22·2 | |
| | +65 | 18·1 | 0·11 | 16·4 | |
| | +100 | 14·0 | 0·13 | 15·0 | |
| | +200 | 12·9 | 0·12 | 12·8 | |
| | −200 | 26·0 | 0·11 | 23·5 | |
| 3................................ | +48 | 15·6 | 0·12 | 15·6 | 0·12 |
| (Ore at −35 mesh) | +65 | 24·4 | 0·12 | 24·4 | |
| | +100 | 15·4 | 0·12 | 15·4 | |
| | +200 | 15·1 | 0·14 | 17·6 | |
| | −200 | 29·5 | 0·11 | 27·0 | |
| 4................................ | +65 | 15·7 | 0·11 | 13·8 | 0·125 |
| (Ore at −48 mesh) | +100 | 19·6 | 0·11 | 17·2 | |
| | +200 | 23·1 | 0·14 | 25·8 | |
| | −200 | 41·6 | 0·13 | 43·2 | |
| 5................................ | +100 | 22·8 | 0·13 | 25·9 | 0·115 |
| (Ore at −65 mesh) | +200 | 28·4 | 0·11 | 27·2 | |
| | −200 | 48·8 | 0·11 | 46·9 | |
| 6................................ | +200 | 30·1 | 0·10 | 30·1 | 0·10 |
| (Ore at −100 mesh) | −200 | 69·9 | 0·10 | 69·9 | |

*Summary—*

Head sample: Au 0·21 oz./ton.

| No. | Mesh | Average tailing, Au, oz./ton | Recovery, per cent |
|---|---|---|---|
| 1 | −20 | 0·134 | 36·2 |
| 2 | −28 | 0·122 | 42·3 |
| 3 | −35 | 0·12 | 42·8 |
| 4 | −48 | 0·125 | 40·5 |
| 5 | −65 | 0·115 | 45·2 |
| 6 | −100 | 0·10 | 52·4 |

### *Tests Nos. 7 to 9*

### CYANIDATION

Ore ground from −14 mesh in cyanide solution, KCN 0·05 per cent, CaO 0·091 per cent. Dilution during grinding 1 : 1, during agitation 3 : 1.

Total time of tests 48 hours.

| Test No. | Mesh | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au | Average tailing, Au, oz./ton | Time of grinding, min. |
|---|---|---|---|---|---|---|
| 7 | +65 | 1·6 | 1·60* | 46·4 | 0·055 | 20 |
| | +100 | 15·5 | 0·03 | 8·5 | ........... | |
| | +150 | 10·0 | 0·03 | 5·4 | ........... | |
| | +200 | 22·7 | 0·03 | 12·3 | ........... | |
| | −200 | 50·2 | 0·03 | 27·4 | ........... | |
| 8 | +200 | 35·0 | 0·03 | 35·0 | 0·03 | 30 |
| | −200 | 65·0 | 0·03 | 65·0 | ........... | |
| 9 | +200 | 22·9 | 0·025 | 19·8 | 0·029 | 50 |
| | −200 | 77·1 | 0·030 | 80·2 | ........... | |

*Probably free gold.

### SUMMARY OF CYANIDATION TESTS

Head sample—Au 0·21 oz./ton.

| No. | Average tailing, Au, oz./ton | Extraction, per cent | Reagents, lb./ton | |
|---|---|---|---|---|
| | | | KCN | CaO |
| 7 | 0·055 | 73·8 | 0·68 | 3·4 |
| 8 | 0·030 | 85·7 | 0·74 | 3·8 |
| 9 | 0·029 | 86·2 | 1·02 | 3·1 |

## FLOTATION, TABLING THE FLOTATION TAILINGS

### *Test No. 10*

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 6·2 | 2·78 | 87·2 |
| Table concentrate | 14·4 | 0·12 | 8·7 |
| Table tailing +200 | 20·8 | 0·01 | 1·1 |
| Table tailing −200 | 29·6 | 0·01 | 1·5 |
| Slimes | 29·0 | 0·01 | 1·5 |

Recovery, 95·9 per cent.

*Reagents*—

To ball mill, No. 34 fuel oil 0·20 lb. per ton, potassium xanthate 0·05 lb. per ton.
To cell, American Cyanamid Co. No. 404 0·15 lb. per ton, Aerofloat No. 25 0·04 lb. per ton, pine oil No. 5 0·04 lb. per ton.

### *Test No. 11*

| — | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 6·9 | 3·00 | 83·3 |
| Table concentrate | 8·5 | 0·40 | 13·7 |
| Table tailing +100 | 4·3 | 0·01 | 0·2 |
| Table tailing +200 | 23·2 | 0·005 | 0·5 |
| Table tailing −200 | 27·8 | 0·01 | 1·1 |
| Slimes | 29·3 | 0·01 | 1·2 |

Recovery, 97 per cent.

*Reagents*—

To ball mill, $Na_2CO_3$ 2·0 lb. per ton, coal tar 0·20 lb. per ton, pine oil No. 5 0·04 lb. per ton.
To cell, amyl xanthate 0·20 lb. per ton, pine oil No. 5 0·04 lb. per ton.

## TWO-STAGE FLOTATION

### *Test No. 12*

| — | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| No. 1 concentrate | 7·0 | 2·20 | 72·2 |
| No. 2 concentrate | 3·8 | 1·02 | 18·2 |
| Tailing +100 | 16·3 | 0·08 | 6·1 |
| " +200 | 22·8 | 0·01 | 1·1 |
| " −200 | 33·6 | 0·01 | 1·6 |
| Slimes | 16·5 | 0·01 | 0·8 |

Recovery, 90·4 per cent.

*Reagents—*

*No. 1 Flotation—*

To ball mill, $Na_2CO_3$ 1·0 lb. per ton, No. 1580 0·16 lb. per ton, potassium xanthate 0·05 lb. per ton, pine oil No. 5 0·02 lb. per ton.

*No. 2 Flotation—*

To ball mill, $Na_2S$ 0·60 lb. per ton, No. 1580 0·09 lb. per ton, potassium xanthate 0·03 lb. per ton.

To cell, amyl xanthate 0·06 lb. per ton, Aerofloat No. 25 0·05 lb. per ton, pine oil No. 5 0·03 lb. per ton.

The following screen analysis indicates approximately the size of the ore particles at the first flotation.

| Mesh | Weight, per cent | Cumulative, per cent |
|---|---|---|
| +35 | 1·5 | 1·5 |
| +48 | 4·9 | 6·4 |
| +65 | 18·5 | 24·9 |
| +100 | 21·6 | 46·5 |
| +200 | 16·6 | 63·1 |
| −200 | 36·9 | 100·0 |

## FLOTATION, TABLING THE FLOTATION TAILING AND TREATMENT OF THE CONCENTRATES

### *Test No. 13*

Concentration as in Test No. 11.

*Treatment of the Concentrates—*

I. CYANIDATION

Cyanide strength 0·15 per cent KCN; dilution 3 : 1; time of agitation, 65 hours.

*Screen Analysis on the Cyanidation Tailing*

| Mesh | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| +100 | 20·2 | 0·44 | 28·3 |
| +150 | 16·5 | 0·13 | 6·8 |
| +200 | 20·4 | 0·20 | 13·0 |
| −200 | 42·9 | 0·38 | 51·9 |

Ratio of concentration 6 : 1.
Average tailing, Au 0·314 oz. per ton.
Head sample to cyanidation, Au 2·0 oz. per ton.
Extraction=84·3 per cent, net recovery 81·7 per cent.

*Reagent Consumption, Pound per Ton Concentrate—*

KCN 3·72, CaO 12·3.

*Reagent Consumption, Pound per Ton Ore—*

KCN 0·62, CaO 2·1.

## II. ROASTING AND CYANIDATION OF THE ROASTED CONCENTRATES

A sample of the concentrates was roasted to drive off most of the arsenic and sulphur. No attempt was made to make the roast thorough.

Loss in weight during roasting 9·4 per cent.

Assay of roasted concentrate, gold 2·04 ounces per ton.

The roasted concentrates were cyanided for 67 hours in 3 : 1 pulp with 0·15 per cent KCN.

*Screen Analysis on the Cyanide Tailing*

| Mesh | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| +100 | 20·2 | 0·23 | 40·8 |
| +150 | 22·8 | 0·07 | 14·0 |
| +200 | 17·8 | 0·07 | 10·9 |
| −200 | 39·2 | 0·10 | 34·3 |

Average tailing, Au 0·114 oz. per ton.
Head sample to cyanidation, Au 2·04 oz. per ton.
Extraction, 94·4 per cent.
Net recovery, 91·6 per cent.

*Reagent Consumption in Pound per Ton—*

| Reagent | Roasted concentrates | Concentrates | Ore |
|---|---|---|---|
| KCN | 1·30 | 1·18 | 0·20 |
| CaO | 33·9 | 30·7 | 5·1 |

## FLOTATION AND TABLING, AND AMALGAMATION

### *Test No. 14*

Concentration as in Test No. 13.

*Amalgamation—*

A lot of 685 grammes of concentrates was ground in a porcelain pebble mill for 45 minutes with the following charge: Water 600 c.c., caustic soda 5 grammes, mercury 100 grammes, 1-inch chilled iron balls 8¾ pounds.

The amalgamation tailing assayed, Au 0·69 ounce per ton.

Head sample to amalgamation, Au 2·01 ounces per ton.

Recovery, 65·7 per cent.

Net recovery, 63·7 per cent.

## FLOTATION AND TABLING—CYANIDATION OF THE CONCENTRATES

### *Test No. 15*

Concentration as in Test No. 13.

Cyanidation done in four parts, A, B, C, and D.

Concentrates were ground 25 minutes in ball mill in 1 : 1 pulp, KCN 0·10 per cent, CaO 0·10 per cent, then diluted to 3 : 1 and agitated for 48 hours. Cyanide strength was maintained at 0·10 per cent by additions made twice daily.

At the end of the 48-hour agitation period the following additions were made and agitation continued for another 17 hours.

| | |
|---|---|
| To B—$Na_2O_2$ | 2·0 lb. per ton |
| To C—BrCN | 2·9 " |
| To D—BrCN | 5·8 " |

Agitation was continued on A without any additions. The protective alkalinity of C and D was reduced to 0·0068 per cent CaO, by the addition of $H_2SO_4$, before adding the bromocyanide.

*Summary—*

Head sample: Gold 1·60 oz./ton.

| No. | Tailing, Au, oz./ton | Extraction, per cent | Net recovery, per cent | Reagents, lb./ton of concentrates | | | |
|---|---|---|---|---|---|---|---|
| | | | | KCN | CaO | $Na_2O_2$ | BrCN |
| 15-A | 0·21 | 86·8 | 84·3 | 3·52 | 13·9 | ........ | ........ |
| 15-B | 0·22 | 86·2 | 83·5 | 3·67 | 13·3 | 2·0 | ........ |
| 15-C | 0·21 | 86·8 | 84·3 | 3·94 | 17·7 | ........ | 2·9 |
| 15-D | 0·23 | 85·6 | 83·0 | 4·06 | 17·9 | ........ | 5·8 |

Net recovery is based on 97 per cent recovery of gold in concentrates.

## FLOTATION AND TABLING—ROASTING THE CONCENTRATES AND TREATMENT OF THE ROASTED CONCENTRATES

### *Test No. 16*

*Concentration—*

Nine lots of ore each weighing 1,000 grammes were floated in a Ruth machine. Rougher concentrates from lots 1 to 4 were combined and cleaned. Middling returned to lot 5. Rougher concentrates from lots 5 to 8 were combined and cleaned. Middling returned to lot 9. Rougher from lot 9 combined with other concentrates.

The flotation tailings were deslimed and the coarse product tabled. The table concentrates were combined with the flotation concentrates.

Assay of combined concentrates, Au 2·18 ounces per ton.

*Roasting*—

The concentrates were roasted in a muffle furnace. During the roasting the temperature exceeded that desired and some sintering occurred especially along a crack which developed in the bottom of the muffle.

Loss in weight during roasting was 17·8 per cent.

Assay of roasted concentrates, Au 2·52 ounces per ton.

*Treatment of the Roasted Concentrates*—

I. CYANIDATION

Part A—Roasted concentrates were ground 40 minutes in ball mill in 1 : 1 pulp, KCN 0·10 per cent, CaO 0·078 per cent and with CaO 4·0 pounds per ton, then diluted to 4 : 1 and agitated 69 hours in 0·10 per cent KCN solution.

Part B—Roasted concentrates were ground 20 minutes in ball mill 1 : 1 pulp, KCN 0·10 per cent, CaO 0·067 per cent and with CaO 10 pounds per ton, then diluted 3 : 1 and agitated 70 hours in 0·10 per cent KCN solution.

RESULTS OF SCREENING THE CYANIDE TAILINGS

*Test No. 16A*

| Mesh | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au | Average tailing, Au, oz./ton |
|---|---|---|---|---|
| +200 | 4·3 | 0·12 | 2·0 | 0·254 |
| −200 | 95·7 | 0·26 | 98·0 | |

*Test No. 16B*

| Mesh | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au | Average tailing, Au, oz./ton |
|---|---|---|---|---|
| +200 | 24·2 | 0·23 | 22·7 | 0·245 |
| −200 | 75·8 | 0·25 | 77·3 | |

*Summary*

| No. | Head, Au, oz./ton | Tailing, Au, oz./ton | Extraction, per cent | Reagents, lb./ton roasted concentrates | |
|---|---|---|---|---|---|
| | | | | KCN | CaO |
| 16-A | 2·52 | 0·254 | 90·0 | 4·15 | 9·2 |
| 16-B | 2·52 | 0·245 | 90·2 | 2·20 | 7·6 |

II. AMALGAMATION OF ROASTED CONCENTRATES

Roasted concentrates 440 grammes, mercury 50 grammes, water 440 c.c., iron balls 8¾ pounds, amalgamated 45 minutes in ball mill.

*Amalgamation Tailing Screened on 200 Mesh—*

| Mesh | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au | Average tailing, Au, oz./ton |
|---|---|---|---|---|
| +100 | 2·1 | 1·09 | 1·5 | 1·503 |
| +200 | 22·6 | 0·95 | 14·3 | |
| −200 | 75·3 | 1·68 | 84·2 | |

Recovery, 40·4 per cent.

## CONCENTRATION—FLOTATION AND TABLING—ROASTING AND CYANIDATION OF THE ROASTED CONCENTRATES

### *Test No. 17*

### I. Concentration

Ten lots of ore, each of 2,000 grammes, were floated in a Fahrenwald machine and all the concentrates were combined. The flotation tailings were deslimed and tabled on a laboratory Wilfley table.

The flotation and table concentrates were combined. The combined concentrates assayed,

| | |
|---|---|
| Gold | 1·92 oz/ton |
| Arsenic | 11·71 per cent |
| Sulphur | 14·36 " |

The ratio of concentration was 10·8 : 1.

### II. Roasting the Concentrates

Lime equivalent to 9·4 pounds per ton of concentrates was mixed with the concentrates which were then roasted in a gas-fired muffle furnace. The maximum temperature during the roasting was about 550°C. and the elimination of all the arsenic and sulphur was not attempted.

The loss of weight during the roasting was 18·9 per cent.

The roasted concentrates assayed,

| | |
|---|---|
| Gold | 2·58 oz/ton |
| Arsenic | 1·03 per cent |
| Sulphur | 1·30 " |

### III. Cyanidation of the Roasted Concentrates

Two cyanidation tests were made on the roasted concentrates namely, A and B.

In A, no regrinding was practised previous to cyanidation, but regrinding in cyanide solution was done in B.

### *Test A*

Cyanide strength 0·10 per cent KCN, dilution 3 : 1, time of agitation 69 hours.

*Screen Analysis on Cyanidation Tailing—*

| Mesh | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au | Average tailing, Au, oz./ton |
|---|---|---|---|---|
| +100 | 6·0 | 0·08 | 3·8 | 0·127 |
| +200 | 28·2 | 0·13 | 28·9 | |
| −200 | 65·8 | 0·13 | 67·3 | |

### *Test B*

Concentrates were reground 25 minutes in ball mill in 1 : 1 pulp, KCN 0·10 per cent, CaO 0·097 per cent, and lime equivalent to 20 pounds per ton. Dilution during agitation was 3 : 1, cyanide strength was 0·10 per cent KCN, and the time of agitation was 70 hours.

*Screen Analysis on the Cyanidation Tailing—*

| Mesh | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au | Average tailing, Au, oz./ton |
|---|---|---|---|---|
| +200 | 7·1 | 0·13 | 7·6 | 0·121 |
| −200 | 92·9 | 0·12 | 92·4 | |

### *Summary—Test No. 17—Cyanidation*

| No. | Extraction, per cent | Net recovery, per cent | Reagent consumption, lb./ton | | | | | |
|---|---|---|---|---|---|---|---|---|
| | | | Roasted concentrates | | Concentrates | | Ore | |
| | | | KCN | CaO | KCN | CaO | KCN | CaO |
| A | 95·0 | 91·1 | 9·6 | 39·0 | 8·1 | 31·6 | 0·75 | 2·9 |
| B | 95·2 | 91·3 | 8·2 | 46·3 | 6·7 | 37·6 | 0·62 | 3·5 |

Note—Net recovery based on 96 per cent recovery of gold in concentrates.

Lime consumption includes that mixed with concentrates before roasting.

## SUMMARY

*Amalgamation—*

Amalgamation tests showed recoveries of 36 per cent at −20 mesh to 52 per cent at −100 mesh. At −28 mesh the recovery was 42 per cent, which approximates that obtained by the stamp-battery.

*Cyanidation—*

Cyanidation tests showed extractions of 74 to 86 per cent of the gold. In Test No. 7, where the minimum extraction was obtained, namely 74 per cent, the high gold value on the 65 mesh tailing was probably due to free gold. The gold values on the other screens were practically the same throughout.

The reagents consumption was moderate, about ¾ lb. per ton of cyanide, KCN, and 3½ lb. per ton of lime were sufficient.

*Concentration*—

*(a) Flotation and Tabling.* 96 to 97 per cent of the gold was recovered in the form of combined flotation and table concentrates. A ratio of concentration of not less than 9 : 1 might reasonably be expected.

By following the flotation with tabling free gold that was not collected in the flotation was caught on the table as well as a small quantity of sulphides, particularly arsenopyrite, that escaped flotation.

*(b) Two-Stage Flotation.* A two-stage flotation test, No. 12, gave a fair recovery, 90 per cent. The coarse screen size following the second flotation contained high gold values mainly as free gold.

This method of concentration might prove effective if it were followed by a coarse vibrating screen, about 65 mesh, and the oversize on this screen either returned to the head of the grinding circuit or combined with the concentrates.

### *Treatment of Concentrates*

I. *Without Roasting*—

*(a) Amalgamation.* Barrel type amalgamation gave a recovery of 66 per cent of the gold contained in the concentrates.

*(b) Cyanidation.* The concentrates proved fairly refractory to cyanidation and the best extraction obtained was about 87 per cent. The reagents consumption when based on per ton of ore was moderate as in the straight cyanidation tests.

II. *After Roasting*—

*(a) Amalgamation.* Amalgamation of the roasted concentrates gave a recovery of only 40 per cent of the gold. This recovery might possibly be increased to a certain extent by varying the conditions of roasting, but it is very doubtful if it could, under the best of roasting conditions, be made to produce good results.

*(b) Cyanidation.* Cyanidation of the roasted concentrates gave the best extraction, 95 per cent, and a net overall recovery of 91 per cent, when based on 96 per cent recovery by concentration.

## CONCLUSIONS

From the results of the test work it is shown that amalgamation is not an effective means of recovering the gold.

The two main methods of treating this ore are by concentration, flotation and tabling, and cyanidation of the concentrates; and by straight cyanidation of the ore, with a difference in recovery of about 2 per cent in favour of the latter method, which on the head sample amounts to about nine cents saving per ton of ore.

Roasting of the concentrates previous to cyanidation gave the best metallurgical results, but when consideration is given to the grade of the ore it does not appear to be a sound economic step.

Whether concentration is practised previous to cyanidation or not is largely a matter of preference.

## Report No. 371

# FLOTATION TESTS ON A SAMPLE OF CYANIDE TAILINGS FROM THE KIRKLAND LAKE GOLD MINES, LIMITED, KIRKLAND LAKE, ONTARIO

**J. S. Godard**

*Shipments.* A shipment of approximately 1,200 pounds of cyanide tailings from the mill of the Kirkland Lake Gold Mines, Limited, Kirkland Lake, Ontario, was received September 2, 1930.

*Characteristics of the Tailings.* The tailings contained 0·062 oz. per ton in gold, chiefly associated with a fine iron pyrite which constitutes about 2 per cent of the weight. They were ground to 92 per cent −200 mesh and when received contained 23 per cent moisture, a trace of cyanide, and 0·009 per cent water-soluble lime.

*Purpose of Experimental Tests.* The purpose of the experimental tests was to determine what recovery of the gold remaining in the cyanide tailings could be obtained by flotation, as well as to secure sufficient concentrates to allow a series of cyanidation tests to be made on the concentrates by the staff of the Kirkland Lake Gold Mines, Limited, at Kirkland Lake, Ontario.

## EXPERIMENTAL TESTS

As the tailings contained a small amount of water-soluble lime, four small-scale preliminary tests were made to ascertain the effect, if any, of this lime on the recovery of gold and the grade of concentrates produced.

Two tests, Nos. 1 and 2, were made by diluting the tailings to 1 : 1 then agitating with the reagents before flotation.

In Tests Nos. 3 and 4, the tailings were first agitated in 1 : 1 pulp, filtered to remove the water-soluble lime, and rediluted before adding the flotation reagents.

### *Test No. 1*

No previous washing was done.

*Results*—

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 15·3 | 0·26 | 64·5 |
| Tailing +200 | 8·3 | 0·08 | 10·7 |
| " −200 | 76·4 | 0·02 | 24·8 |

Head sample from previous products, Au 0·062 oz. per ton.

### *Test No. 2*

No previous washing. In this test the rougher concentrates from four tests were combined and cleaned.

The flotation tailings were screened on 200 mesh with the following results:—

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| +200 | 8·8 | 0·07 | 31·2 |
| −200 | 91·2 | 0·015 | 68·8 |

Flotation tailing from products, Au 0·02 oz. per ton.

*Concentration*—

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 3·8 | 1·02 | 65·9 |
| Middling | 4·6 | 0·04 | 3·1 |
| Tailing | 91·6 | 0·02 | 31·0 |

Head sample from products, Au 0·059 oz. per ton.

### *Test No. 3*

The cyanide tailings were water-washed before flotation.

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 11·7 | 0·34 | 63·4 |
| Tailing +200 | 7·5 | 0·09 | 10·8 |
| " −200 | 80·8 | 0·02 | 25·8 |

Head sample from products, Au 0·063 oz. per ton.

### *Test No. 4*

The cyanide tailings were water-washed before flotation and the rougher concentrates from four tests were combined and cleaned.

The flotation tailings were screened on 200 mesh with the following results:—

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| +200 | 7·2 | 0·08 | 23·8 |
| −200 | 92·8 | 0·02 | 76·2 |

Flotation tailing from products, Au 0·024 oz. per ton.

*Concentration—*

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 3·1 | 1·28 | 62·8 |
| Middling | 4·2 | 0·03 | 2·1 |
| Tailing | 92·7 | 0·024 | 35·1 |

Head sample from products, Au 0·063 oz. per ton.

*Conclusions.* A higher recovery was made in those tests where the cyanide tailing was taken direct to flotation, but the grade of concentrates was lower and better froth conditions prevailed in those tests where washing was practised before flotation.

## CONTINUOUS FLOTATION TEST IN 100 POUND PER HOUR UNIT

### *Test No. 5*

The cyanide tailings were not washed before flotation. The flow-sheet for this test being as follows.

The tailings were put through a ¾-inch perforated plate to break up the lumps, then fed by a continuous belt feeder into a small rod mill containing only two rods for the purpose of further reducing the lumps.

It was found necessary to add all the water with the feed, otherwise owing to the clayey nature of the tailings difficulty was experienced in keeping the feed rate constant.

The rod mill discharge emptied into a contact tank adjusted for approximately 20 minutes' contact. From the contact tank the pulp flowed into No. 2 cell of a 6-cell Denver machine. The rougher concentrates from cells 2 and 3 were returned by means of a pump to the head of No. 1 cell from which the final concentrates were taken. Cells 4, 5, and 6 were scavenger cells and the concentrates from these three cells were pumped back to No. 2 cell.

Samples of the feed, concentrates and tailings were taken at 15-minute intervals during the test.

The reagents used were as follows,—

$Na_2CO_3$ 3·0 lb. per ton fed in solution to the rod mill. $CuSO_4$ 0·50 lb. per ton, amyl xanthate 0·20 lb. per ton were fed into the contact tank and pine oil No. 5, 0·01 lb. per ton, to the head of No. 2 cell.

The dilution for flotation was about 3·5 : 1.

Once the unit steadied down no difficulties were experienced.

The results of the test are tabulated as follows,—

## *No. 1 Series of Samples*

### HEAD SAMPLE SCREENED ON 200 MESH

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| +200 | 8·2 | 0·08 | 10·7 |
| −200 | 91·8 | 0·06 | 89·3 |

Head sample No. 1 from products, Au 0·062 oz./ton.
No. 1 concentrate assayed, Au 1·28 oz./ton.

### No. 1 TAILING SAMPLE SCREENED ON 200 MESH

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| +200 | 8·1 | 0·07 | 23·6 |
| −200 | 91·9 | 0·02 | 76·4 |

Average tailing sample from products, Au 0·024 oz./ton.

## *No. 2 Series of Samples*

### HEAD SAMPLE SCREENED ON 200 MESH

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| +200 | 8·8 | 0·08 | 11·3 |
| −200 | 91·2 | 0·06 | 88·7 |

Head sample No. 2 from products, Au 0·062 oz./ton.
No. 2 concentrate assayed, Au 1·36 oz./ton.

### No. 2 TAILING SAMPLE SCREENED ON 200 MESH

| Product | Weight, per cent | Assays, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| +200 | 8·1 | 0·07 | 23·6 |
| −200 | 91·9 | 0·02 | 76·4 |

Average tailing sample from products, Au 0·024 oz./ton.

*Summary*—

| No. | Head sample, Au, oz./ton | Concentrate, Au, oz./ton | Tailing, Au, oz./ton | Recovery, per cent | Ratio of concentrate |
|---|---|---|---|---|---|
| Sample No. 1 | 0·062 | 1·28 | 0·024 | 62·5 | 33 : 1 |
| Sample No. 2 | 0·062 | 1·36 | 0·024 | 62·4 | 35 : 1 |

### CONCLUSIONS

No difficulty was experienced in the flotation of these tailings—62·5 per cent of the gold being recovered in the flotation concentrates, which assayed as high as 1·36 ounces per ton. A ratio of concentration of 35 : 1 was secured.

In the flotation of these tailings the recovery obtained is largely from the −200 mesh material, the +200 mesh flotation tailing showed only a reduction of 20 cents per ton from the head sample.

Whether cyanide tailings of this grade may be profitably concentrated and the concentrates re-treated for the gold they contain rests mainly with the success and cost of re-treatment.

## Report No. 372

### THE RECOVERY OF GOLD FROM THE ORE OF THE CASEY MOUNTAIN OPERATING SYNDICATE, SUMMIT LAKE, PATRICIA PORTION, ONTARIO

**A. K. Anderson**

*Shipment.* Four bags containing 485 pounds of ore were received at the laboratories of the Ore Dressing and Metallurgical Division of the Department of Mines, Ottawa, on September 19, 1930. This material came from claim No. K.R.L., 9681, Summit lake, Patricia Portion, Ontario, shipped by the Casey Mountain Operating Syndicate, Limited, 411 McCallum-Hill Building, Regina, Sask., R. G. Williamson, President and General Manager.

*Characteristics of the Ore.* The ore consisted of dark magnetiferous quartz carrying gold and small amounts of mispickel, iron pyrite, and traces of galena and chalcopyrite. Very little if any free gold is visible.

*Purpose of Experimental Tests.* The shipment was made for the purpose of determining the most suitable method to apply for the recovery of the gold.

These tests show the shipment to contain 0·72 ounce gold per ton; 90 per cent of the gold is free at 200 mesh grinding and can be recovered by amalgamation.

Grinding through 48 mesh with 36 per cent through 200 shows that 82 per cent of the gold can be amalgamated.

Concentration is not applicable due to the very small amount of sulphides present; 75 per cent of the gold can be recovered by this method in a concentrate containing 23·50 ounces gold per ton, with a ratio of concentration of 46·5 : 1.

Cyaniding the ore ground to pass 48 mesh with 53 per cent through 200 gives an extraction of 96·5 per cent of the gold. Finer grinding does not materially increase the extraction.

*Sampling and Analysis.* The entire shipment contained in the four boxes was crushed to pass ½-inch screen and quartered. One-quarter was then crushed to pass 6 mesh and again quartered. After grinding to succeeding finer sizes with intervening cuts through a Jones riffle sampler, a representative 100 mesh portion was obtained for analysis.

This sample showed the shipment to contain 0·72 ounce gold and 0·11 ounce silver per ton. A small amount of arsenic, 0·42 per cent, is present.

## EXPERIMENTAL TESTS

The investigation included tests by amalgamation, flotation, and cyanidation.

### AMALGAMATION

### *Test No. 1*

A representative 1,000-gramme portion of the ore −14 mesh was ground with 750 grammes of water in a porcelain mill containing iron balls. After grinding and removing the balls, the pulp was amalgamated with mercury. The mercury was then removed and the residue assayed.

| | |
|---|---|
| Heads | 0·72 oz. gold per ton |
| Amalgamation tailing | 0·07 " " |
| Recovery | 90·3 per cent |

A screen analysis of the tailing shows the following.

| Mesh | Weight, per cent | Assays, Au, oz./ton | Per cent of values |
|---|---|---|---|
| +100 | 4·6 | 0·61 | 37·7 |
| −100+150 | 4·6 | 0·14 | 8·7 |
| −150+200 | 17·1 | 0·06 | 13·9 |
| −200 | 73·7 | 0·04 | 39·7 |

To secure highest recovery, fine grinding is necessary, as indicated by the above analysis.

### *Test No. 2*

As it is not practical to amalgamate gold ore ground to the fineness shown in the preceding test, a 1,000-gramme portion of the ore was ground dry to pass 48 mesh, mixed with an equal weight of water and amalgamated.

| | |
|---|---|
| Heads | 0·72 oz. gold per ton |
| Amalgamation tailing | 0·13 " " |
| Recovery | 81·9 per cent |

*Screen Analysis of Tailing*

| Mesh | Weight, per cent | Assay, Au, oz./ton | Per cent of values |
|---|---|---|---|
| −48+65 | 9·03 | 0·15 | 10·4 |
| −65+100 | 24·92 | 0·17 | 32·7 |
| −100+150 | 14·00 | 0·07 | 7·6 |
| −150+200 | 15·69 | 0·06 | 7·3 |
| −200 | 36·36 | 0·15 | 42·0 |

This test shows that amalgamation at a size that would be produced in practice gives a recovery of 81·9 per cent of the gold.

## FLOTATION

### *Test No. 3*

The ore was concentrated by flotation to note what recovery of gold would be obtained by this method.

A 1,000-gramme portion of the ore −14 mesh was ground with 750 grammes of water and soda ash equivalent to 3 pounds per ton of ore. Grinding was such that 73 per cent passed 200 mesh.

After grinding, the pulp was transferred to a laboratory-size, mechanically agitated flotation machine where, after the addition of 0·10 pounds amyl xanthate and 0·06 pounds pine oil per ton, a concentrate was removed.

| Product | Weight, per cent | Assay, Au, oz./ton | Per cent of values |
|---|---|---|---|
| Heads (calculated) | 100·00 | 0·67 | 100·0 |
| Flotation concentrate | 2·15 | 23·50 | 75·2 |
| Flotation tailing | 97·85 | 0·17 | 24·8 |

These results indicate that flotation is not applicable. The tailing contains $3.40 a ton, being too high-grade to be discarded. This would mean a cyanide annex to recover the values. In the flotation concentrate which contains 23·50 ounces gold per ton, 75·2 per cent of the gold is recovered. A very high ratio of concentration is obtained, 46·5 : 1.

## CYANIDATION

### *Test No. 4*

Representative portions of the ore ground to pass 48 mesh were cyanided for 48 hours, 1 : 3 dilution with a 1·0 pound KCN solution. Lime at the rate of 3 pounds per ton of ore was added.

| | |
|---|---|
| Heads | 0·72 oz. gold per ton |
| Cyanide tailing | 0·025 " " |
| Extraction | 96·5 per cent |
| Reagent consumption | KCN—0·24 lb./ton ore.<br>CaO — 1·7 " |

A screen analysis of the tailings shows the following:

| Mesh | Weight, per cent | Assay, Au, oz./ton |
|---|---|---|
| −48+65 | 14·4 | 0·025 |
| −65+100 | 24·3 | 0·035 |
| −100+150 | 7·4 | 0·02 |
| −150+200 | 1·1 | 0·87 |
| −200 | 52·8 | 0·01 |

### *Test No. 5*

In this test the ore was ground to pass 65 mesh and cyanided as in the preceding test.

| | |
|---|---|
| Heads | 0·72 oz. gold/ton. |
| Cyanide tailing | 0·02 " |
| Extraction | 97·2 per cent. |
| Reagent consumption | KCN—0·24 lb./ton ore.<br>CaO — 2·1 lb. " |

A screen analysis of the tailing shows the following:

| Mesh | Weight, per cent | Assay, Au, oz./ton |
|---|---|---|
| −65+100 | 9·0 | 0·03 |
| −100+150 | 7·2 | 0·02 |
| −150+200 | 1·6 | 0·08 |
| −200 | 82·2 | 0·015 |

### *Test No. 6*

The ore was ground to pass 100 mesh.

| | |
|---|---|
| Heads | 0·72 oz. gold/ton |
| Cyanide tailing | 0·025 " |
| Extraction | 96·5 per cent. |
| Reagent consumption | KCN—0·24 lb./ton ore.<br>CaO — 2·1 lb. " |

A screen analysis of the tailing is as follows:

| Mesh | Weight, per cent | Assay Au, oz./ton |
|---|---|---|
| −100+150 | 16·1 | 0·04 |
| −150+200 | 4·4 | 0·035 |
| −200 | 79·5 | 0·02 |

### *Test No. 7*

The ore was ground to pass 150 mesh.

| | |
|---|---|
| Heads | 0·72 oz. gold/ton. |
| Cyanide tailing | 0·02 " |
| Extraction | 97·2 per cent. |
| Reagent consumption | KCN—0·48 lb./ton ore.<br>CaO — 2·48 lb. " |

A screen analysis of the tailing is as follows:

| Mesh | Weight, per cent | Assay, Au, oz./ton |
|---|---|---|
| −150+200 | 4·6 | 0·06 |
| −200 | 95·4 | 0·02 |

### *Test No. 8*

The ore was ground to pass 200 mesh.

| | |
|---|---|
| Heads | 0·72 oz. gold/ton |
| Cyanide tailing | 0·02 " |
| Extraction | 97·2 per cent. |
| Reagent consumption | KCN—0·48 lb./ton ore.<br>CaO — 3·62 " |

*Summary of the Tests*—

| Test No. | Mesh grinding | Tailing, Au, oz./ton | Extraction, per cent | Consumption | |
|---|---|---|---|---|---|
| | | | | KCN | CaO |
| 4 | 48 | 0·025 | 96·5 | 0·24 | 1·7 |
| 5 | 65 | 0·02 | 97·2 | 0·24 | 2·1 |
| 6 | 100 | 0·025 | 96·5 | 0·24 | 2·1 |
| 7 | 150 | 0·02 | 97·2 | 0·48 | 2·48 |
| 8 | 200 | 0·02 | 97·2 | 0·48 | 3·62 |

It is apparent that maximum extraction can be obtained by grinding to pass 65 mesh. The tests indicate that the tailing after cyaniding at this mesh is but 10 cents lower than the tailing after cyaniding at 48 mesh. Little increased extraction is to be expected from cyaniding at finer meshes.

### SUMMARY AND CONCLUSIONS

Tests Nos. 1 and 2 show that a large percentage of the gold is free. The ore coarsely ground to give a high duty in the grinding mill can be successfully amalgamated to yield a recovery of 82 per cent of the gold.

Flotation results in the recovery of a concentrate containing 75 per cent of the gold.

Cyanidation of the −48 mesh material shows an extraction of 96·5 per cent of the gold. Little increase is to be expected from finer grinding.

As the ore responds to cyanidation with such ease, straight cyanidation of the ore ground to 48 mesh is undoubtedly the proper method to employ for the recovery of gold from this class of ore.

## Report No. 373

## THE RECOVERY OF GOLD FROM THE ORE OF THE PARKHILL GOLD MINES, LIMITED, WAWA, ONTARIO

**A. K. Anderson**

*Shipment.* A shipment of 24 bags of gold ore, gross weight 2,700 pounds, was received at the Ore Dressing and Metallurgical Laboratories on September 25, 1930. This was forwarded by the Parkhill Gold Mines, Limited, Wawa, Michipicoten district, Ontario. The Canadian Enterprises, Limited, Keefer Building, Montreal, G. S. Turner, General Manager, is interested in the property.

This consignment contained six lots of ore, of four bags each, taken from various points in the mine workings.

*Characteristics of the Ore.* The ore was of a light-coloured siliceous nature somewhat sugary in appearance. A very small amount of iron pyrite was present.

*Purpose of Experimental Tests.* The shipment was made primarily for the purpose of obtaining the assay value of the different lots comprising the shipment. Later, it was requested that tests be made to determine the metallurgical process most suitable for the recovery of the contained gold.

*Sampling and Analysis.* Each of the six lots was handled separately. They were crushed to pass a ½-inch screen and quartered. After grinding to succeeding finer sizes with intervening cuts through a Jones riffle sampler, a representative portion −100 mesh was secured for analysis.

The following are the assays of the six lots.

| | |
|---|---|
| Lot No. 1—from ore pile | 1·26 oz. gold/ton. |
| Lot No. 2—from stock pile | 0·88 " |
| Lot No. 3—station 104 East, 1st level east | 1·45 " |
| Lot No. 4—from 1st level west | 0·39 " |
| Lot No. 5—from east level | 0·11 " |
| Lot No. 6—from 2nd level west | 1·90 " |

A composite sample was made from all six lots for test purposes. This had a gold content of 0·99 ounce gold per ton.

## Experimental Tests

The investigation was made on the composite sample having an assay value of 0·99 ounce gold per ton. This work included tests by amalgamation, cyanidation and flotation. The results showed that when the ore was ground to pass 48 mesh, 77·7 per cent of the gold was recovered by amalgamation. Cyanidation of the unground tailing from this treatment gave an additional extraction of 16·4 per cent.

Flotation of the finely ground ore gave a recovery of 88 per cent of the gold, with a ratio of concentration of 61 : 1.

Straight cyanidation of the ore ground to pass 48 mesh gave an extraction of 93 per cent, while 200 mesh grinding gave 98 per cent.

## AMALGAMATION

### *Test No. 1*

A representative 1,000 gramme portion of the ore was ground dry to pass 48 mesh, mixed with an equal weight of water and amalgamated with mercury. The amalgam was then removed and the residue sampled.

| | |
|---|---|
| Heads | 0·99 oz. gold/ton. |
| Amalgamation tailing | 0·20 " |
| Recovery | 77·7 per cent. |

A portion of the amalgamation tailing was then agitated for 48 hours 1 : 3 dilution with a 1·0 pound per ton cyanide solution and 3 pounds of lime per ton of ore.

*Screen Test of Cyanide Tailing*

| Mesh | Weight, per cent | Assay, Au, oz./ton |
|---|---|---|
| −48+65 | 8·41 | 0·15 |
| −65+100 | 12·61 | 0·24 |
| −100+150 | 14·49 | 0·12 |
| −150+200 | 19·47 | 0·06 |
| −200 | 45·02 | 0·05 |
| Average assay |  | 0·09 |

This cyanide test shows an extraction of 79·3 per cent of the gold in the amalgamation tailing, or 16·4 per cent of the gold in the original ore.

This test shows a total recovery of 94·1 per cent of the gold by amalgamation followed by cyanidation of the tailing.

The screen analysis of the residue after cyanidation shows that much of the gold is not freed above 200 mesh.

## FLOTATION

### *Test No. 2*

In this test, a representative 1,000 gramme portion of the ore −14 mesh was ground in a porcelain mill containing iron balls, 750 grammes water and soda ash equivalent to 6·0 pounds per ton of ore. The time of grinding was such that 81·4 per cent passed a 200 mesh screen.

After grinding and removing the balls, the pulp was floated in a mechanically agitated flotation machine using 0·10 pound amyl xanthate and 0·04 pound pine oil per ton.

| Product | Weight, per cent | Assay, Au, oz./ton | Per cent of values, |
|---|---|---|---|
| Heads (calculated) | 100·0 | 0·82 | 100·0 |
| Flotation concentrate | 1·64 | 44·20 | 88·0 |
| Flotation tailing | 98·36 | 0·10 | 12·0 |

Flotation gives a recovery of 88 per cent of the gold in a concentrate assaying 44·20 ounces gold per ton. The ratio of concentration is very high, each 100 tons of ore milled producing 1·64 tons of concentrate. The tailing contains $2.00 in gold per ton. To recover this, a further treatment by cyanide will be necessary.

## CYANIDATION

### *Test No. 3*

Representative portions of the ore were ground dry to pass 48, 65, 100, 150, and 200 mesh. Each size was then agitated for 48 hours, 1 : 3 dilution, with a 1·0 pound per ton cyanide solution and 3 to 4 pounds lime per ton.

### *48 Mesh Grinding*

| | |
|---|---|
| Heads | 0·99 oz./ton |
| Cyanide tailing | 0·07 “ |
| Extraction | 93·0 per cent |
| Consumption, lb./ton ore—KCN | 0·60 |
| CaO | 2·25 |

### *65 Mesh Grinding*

| | |
|---|---|
| Heads | 0·99 oz./ton |
| Cyanide tailing | 0·04 “ |
| Extraction | 95·0 per cent |
| Consumption, lb./ton ore—KCN | 0·60 |
| CaO | 2·25 |

A screen analysis of the tailing shows:

| Mesh | Weight, per cent | Assay, Au, oz./ton |
|---|---|---|
| −65+100 | 6·28 | 0·11 |
| −100+150 | 12·21 | 0·07 |
| −150+200 | 26·13 | 0·05 |
| −200 | 55·38 | 0·02 |

### *100 Mesh Grinding*

| | |
|---|---|
| Heads | 0·99 oz. per ton |
| Cyanide tailing | 0·03 “ |
| Extraction | 97·0 per cent |
| Consumption, lb./ton ore—KCN | 0·60 |
| CaO | 2·95 |

### *150 Mesh Grinding*

| | |
|---|---|
| Heads | 0·99 oz. per ton |
| Cyanide tailing | 0·06 “ |
| Extraction | 94·0 per cent |
| Consumption, lb./ton ore—KCN | 1·5 |
| CaO | 3·4 |

### *200 Mesh Grinding*

| | |
|---|---|
| Heads | 0·99 oz. per ton |
| Cyanide tailing | 0·02 “ |
| Extraction | 98·0 per cent |
| Consumption, lb./ton ore—KCN | 1·2 |
| CaO | 3·55 |

For comparison, the results secured at different meshes are tabulated.

| Mesh | Tailing assay, Au, oz./ton | Extraction, per cent | Consumption, lb./ton | |
|---|---|---|---|---|
| | | | KCN | CaO |
| −48 | 0·07 | 93·0 | 0·60 | 2·25 |
| −65 | 0·04 | 95·0 | 0·60 | 2·25 |
| −100 | 0·03 | 97·0 | 0·60 | 2·95 |
| −150 | 0·06 | 94·0 | 1·50 | 3·40 |
| −200 | 0·02 | 98·0 | 1·20 | 3·55 |

### SUMMARY AND CONCLUSIONS

Amalgamation of 48 mesh material gives a recovery of 77·7 per cent. Followed by cyanidation, a combined recovery of 94·1 per cent is obtained.

Flotation gives a recovery of 88 per cent. Due to the very small bulk of concentrate, flotation is not applicable. Cyanidation of material ground to pass 200 mesh gives a recovery of 98 per cent.

These results indicate that cyanidation is the best method to adopt for the recovery of the gold in this ore.

## Report No. 374

### EXPERIMENTAL TESTS ON GOLD ORE FROM THE SISCOE GOLD MINES, LTD., AMOS, QUEBEC

**J. S. Godard**

*Shipments.* A shipment of 160 pounds of ore was received October 24, 1930, from the Siscoe Gold Mines, Ltd., Amos, Que.

*Characteristics and Analysis of the Ore.* The ore is a medium grade gold ore. The gold is mainly free milling though a small amount is associated with crystals of iron pyrites which are scattered through the gangue. The gangue is chiefly quartz, though some crystals of tourmaline are also present.

Analyses of the ore showed it to contain, gold 0·53 ounce per ton, and sulphur 0·44 per cent.

*Purpose of Experimental Tests.* Mr. J. M. Forbes, consulting engineer for the Siscoe Gold Mines, Ltd., requested that a few experimental tests be made to determine the relationship of the free gold to that contained in the iron pyrites.

### EXPERIMENTAL TESTS

### *Test No. 1*

### CONCENTRATION, FLOTATION AND TABLING

This test was made for the purpose of becoming familiar with the ore and to act as an indicator.

*Results*—

| Product | Weight, per cent | Assay, Au, oz./ton | Per cent of values, Au |
|---|---|---|---|
| Flotation concentrate | 6·2 | 5·86 | 63·7 |
| Table concentrate | 3·6 | 3·68 | 23·2 |
| Tailing +200 | 7·4 | 0·93 | 12·0 |
| " −200 | 45·5 | 0·01 | 0·8 |
| Slimes | 37·3 | 0·005 | 0·3 |

### *Tests Nos. 2 and 3*

In order to determine the relationship of the free gold to that associated with the iron pyrites the free gold was first amalgamated and the amalgamation tailings floated to collect the pyrite.

Tests Nos. 2 and 3 were made on ore ground to different sizes.

*Results—Test No. 2—*

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Au, oz./ton | S, per cent | Au | S |
| Flotation concentrate | 3·0 | 1·20 | 11·96 | 69·4 | 78·5 |
| Tailing +200 | 20·5 | 0·04 | 0·07 | 15·8 | 3·1 |
| " −200 | 76·5 | 0·01 | 0·11 | 14·8 | 18·4 |

Amalgamation tailing, gold 0·052 oz. per ton.

*Recovery—*

| | |
|---|---|
| Amalgamation | 90·2 per cent |
| Flotation | 6·8 " |
| Total | 97·0 " |

*Test No. 3—*

| Product | Weight, per cent | Assay | | Per cent of values | |
|---|---|---|---|---|---|
| | | Cu, oz./ton | S, per cent | Au | S |
| Flotation concentrate | 3·5 | 0·84 | 9·18 | 76·8 | 81·6 |
| Tailing +200 | 15·1 | 0·005 | 0·05 | 2·1 | 1·9 |
| " −200 | 81·4 | 0·01 | 0·08 | 21·1 | 16·5 |

Amalgamation tailing, gold 0·038 oz. per ton.

*Recovery—*

| | |
|---|---|
| Amalgamation | 92·8 per cent |
| Flotation | 5·5 " |
| Total | 98·3 " |

## CONCLUSIONS

From the tests, 90 to 93 per cent of the gold is in the free state when ground to the sizes indicated in Tests Nos. 2 and 3, the remainder is associated with the pyrite.

The results obtained from the amalgamation phase of the testing approaches the theoretical free gold when ground as indicated and will be somewhat higher than can be secured in general amalgamation practice.

## Report No. 375

### EXPERIMENTAL FLOTATION TESTS ON COBALT ORE FROM THE YORKSHIRE COBALT MINES, LTD., COBALT, ONT.

**J. S. Godard**

*Shipment.* A shipment of 250 pounds, consisting of two lots of somewhat similar ore, was received November 3, 1930, from Mr. W. E. McCready of the Yorkshire Cobalt Mine, Box 888, Cobalt, Ontario.

*Characteristics of the Ore.* The ore is a cobalt one in which the principal mineral is cobaltite, which occurs in very fine crystals disseminated through a gangue material about diabase in composition. Fine grinding is necessary to free the mineral from the gangue.

There is very little difference in the composition of both lots, one being a sample of the ore and the other lot a sample taken across a 4-foot width.

*Analyses of Head Samples.* The analyses of the head samples of each lot were as follows,—

| Lot No. | Co, per cent | Ni, per cent | As, per cent | Ag, oz./ton |
|---|---|---|---|---|
| 1 | 2·19 | 0·04 | 2·85 | 1·18 |
| 2 | 2·70 | 0·03 | 3·55 | 0·46 |

*Purpose of Experimental Tests.* Mr. McCready requested that experimental concentration tests be made on each lot of ore.

### Experimental Tests

### Lot No. 1

### FLOTATION—TABLING THE FLOTATION TAILING

### *Tests Nos 1, 2, and 3*

The variable factor in these tests was the fineness of grinding.

| Test No. | Product | Weight, per cent | Assay | Per cent of values |
|---|---|---|---|---|
| | | | Co, per cent | Co |
| 1 | Flotation concentrate | 8·1 | 16·67 | 62·8 |
| | Table concentrate | 3·7 | 9·70 | 16·7 |
| | Tailing +100 | 13·2 | 0·42 | 2·6 |
| | " +200 | 25·3 | 0·40 | 4·7 |
| | " −200 | 18·7 | 0·52 | 4·5 |
| | Slimes | 31·0 | 0·60 | 8·7 |
| 2 | Flotation concentrate | 11·3 | 15·68 | 81·0 |
| | Table concentrate | 3·8 | 3·34 | 5·8 |
| | Tailing +200 | 13·7 | 0·13 | 0·8 |
| | " −200 | 21·9 | 0·29 | 2·9 |
| | Slimes | 49·3 | 0·42 | 9·5 |
| 3 | Flotation concentrate | 10·9 | 15·30 | 78·9 |
| | Table concentrate | 2·5 | 3·90 | 4·6 |
| | Tailing +200 | 4·8 | 0·21 | 0·5 |
| | " −200 | 28·8 | 0·31 | 4·2 |
| | Slimes | 53·0 | 0·47 | 11·8 |

*Reagents*—

Soda ash 3·0 lb./ton, coal tar 0·20 lb./ton—added to ball mill; amyl xanthate 0·15 lb./ton, pine oil No. 5 0·08 lb./ton, Aerofloat No. 25 0·04 lb./ton—added to the cell.

Grinding as in Test No. 2 is the most satisfactory, and additional tests were done at about this fineness.

### FLOTATION TESTS

#### *Tests Nos 4, 5, and 6*

*Results*—

| Test No. | Product | Weight, per cent | Assay | Per cent of value |
|---|---|---|---|---|
| | | | Co, per cent | Co |
| 4 | Concentrate | 9·9 | 15·80 | 69·0 |
| | Tailing | 90·1 | 0·78 | 31·0 |
| 5 | Concentrate | 12·9 | 11·14 | 76·0 |
| | Tailing | 87·1 | 0·52 | 24·0 |
| 6 | Concentrate | 13·0 | 11·10 | 75·4 |
| | Tailing | 87·0 | 0·54 | 24·6 |

*Reagents*—

*Test No. 4.* Soda ash 5·0 lb./ton, coal tar 0·20 lb./ton, Flotagen 0·16 lb./ton—added to ball mill.
To cell—amyl xanthate 0·05 lb./ton, pine oil No. 5 0·12 lb./ton.

*Test No. 5.* Soda ash 5·0 lb./ton, coal tar 0·20 lb./ton, Flotagen 0·16 lb./ton—added to ball mill.
To cell—amyl xanthate 0·05 lb./ton, copper sulphate 1·0 lb./ton, pine oil No. 5 0·12 lb./ton.

*Test No. 6.* Soda ash 5·0 lb./ton, sodium sulphide 1·0 lb./ton, Flotagen 0·16 lb./ton, coal tar 0·20 lb./ton—added to ball mill.
To cell—amyl xanthate 0·05 lb./ton, pine oil No. 5 0·12 lb./ton.

### FLOTATION TEST

#### *Test No. 7*

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

*Results*—

| Product | Weight, per cent | Assays | Per cent of values |
|---|---|---|---|
| | | Co, per cent | Co |
| Concentrate | 7·7 | 20·95 | 75·3 |
| Middling | 5·7 | 2·12 | 5·7 |
| Tailing | 86·6 | 0·47 | 19·0 |

*Reagents—*

Soda ash 5·0 lb./ton, coal tar 0·24 lb./ton, American Cyanamid Co. No. 404, 0·40 lb./ton, sodium sulphide 2·0 lb./ton—added to ball mill.

To cell—amyl xanthate 0·10 lb./ton, Aerofloat No. 25, 0·12 lb./ton. Rougher concentrates were cleaned without any additional reagents.

The concentrate in Test No. 7 assayed: cobalt 20·95 per cent; nickel 0·07 per cent; arsenic 28·32 per cent; silver 6·98 ounces per ton; and gold 0·08 ounce per ton.

## Lot No. 2

### FLOTATION TESTS

### *Tests Nos. 1, 2, 3, and 4*

*Results—*

| Test No. | Product | Weight, per cent | Assays | Per cent of values |
|---|---|---|---|---|
| | | | Co, per cent | Co |
| 1 | Concentrate | 13·5 | 14·75 | 76·7 |
| | Tailing | 86·5 | 0·70 | 23·3 |
| 2 | Concentrate | 13·6 | 15·90 | 79·4 |
| | Tailing | 86·4 | 0·65 | 20·6 |
| 3 | Concentrate | 16·3 | 12·60 | 79·1 |
| | Tailing | 83·7 | 0·65 | 20·9 |
| 4 | Concentrate | 15·3 | 14·65 | 83·3 |
| | Tailing | 84·7 | 0·53 | 16·7 |

*Reagents—*

*Test No. 1.* Soda ash 3·0 lb./ton, coal tar 0·24 lb./ton—added to ball mill.
To cell—amyl xanthate 0·20 lb./ton, pine oil No. 5 0·08 lb./ton, Aerofloat No. 28, 0·08 lb./ton.

*Test No. 2.* Soda ash 4·0 lb./ton, coal tar 0·24 lb./ton, Flotagen 0·16 lb./ton—added to ball mill.
To cell—amyl xanthate 0·075 lb./ton, pine oil No. 5 0·04 lb./ton, Aerofloat No. 25, 0·12 lb./ton, copper sulphate 1·0 lb./ton.

*Test No. 3.* Soda ash 6·0 lb./ton, coal tar 0·24 lb./ton, Flotagen 0·20 lb./ton, sodium sulphide 3·0 lb./ton—added to ball mill.
To cell—amyl xanthate 0·075 lb. /ton, Aerofloat No. 25 0·15 lb./ton.

*Test No. 4.* Soda ash 5·0 lb./ton, coal tar 0·24 lb./ton, American Cyanamid Co. No. 404, 0·40 lb./ton, sodium sulphide 2·0 lb./ton—added to ball mill.
To cell—amyl xanthate 0·10 lb./ton, Aerofloat No. 25 0·12 lb./ton.

### FLOTATION

### *Test No. 5*

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

*Results*—

| Product | Weight, per cent | Assay | Per cent of values |
|---|---|---|---|
| | | Co, per cent | Co |
| Concentrate | 10·1 | 20·27 | 79·1 |
| Middling | 8·4 | 2·26 | 7·4 |
| Tailing | 81·5 | 0·43 | 13·5 |

*Reagents*—

Soda ash 5·0 lb./ton, coal tar 0·24 lb./ton, American Cyanamid Co. No. 404, 0·40 lb./ton, sodium sulphide 2·0 lb./ton—added to ball mill.
To cell—amyl xanthate 0·10 lb./ton, Aerofloat No. 25 0·15 lb./ton.

Rougher concentrates were cleaned without any additional reagents.

The flotation concentrate assayed, cobalt 20·27 per cent; nickel 0·05 per cent; arsenic 26·92 per cent; silver 6 ounces per ton; gold 0·04 ounce per ton.

## CONCLUSIONS

*Grinding.* Grinding to −100 mesh, with 75 per cent −200 mesh, would probably prove to be good practice.

*Flotation.* Flotation on this ore recovered about 78 per cent of the cobalt in Lot No. 1 and about 83 per cent in Lot No. 2. The ratios of concentration are respectively 13 : 1 and 10 : 1. The grade of concentrate was about 20 per cent cobalt on each lot.

As the ore is somewhat slow to respond to flotation the cells should be from 25 to 35 per cent oversize from that usually designed for the average ore of this grade.

While the number and quantities of reagents used in the last test on each lot appear to be rather formidable, other reagent combinations probably applied in lesser quantities may produce equally good results.

*Tabling.* Tabling of the flotation tailing is a doubtful economic procedure, because of the low grade of the concentrate so produced. Should this step be thought advisable, desliming of the flotation tailing in a classifier or cone previous to tabling is suggested. It will increase the efficiency of the tables because a more uniform feed would be provided and decrease the number of tables required, without involving much decrease in recovery, as the finer mineral particles that have escaped flotation will not be collected in the table concentrate.

The tables would serve as an indicator of the performance of the flotation cells.

No difficulties should be experienced in the flotation of this ore.

## Report No. 376

# THE TREATMENT OF GOLD ORES FROM THE ENGLISH BROOK AREA, RICE LAKE DISTRICT, MANITOBA

**C. S. Parsons**

*Shipment.* A shipment consisting of 24 bags of ore was received at the laboratories of the Ore Dressing and Metallurgical Division of the Mines Branch, Department of Mines, Ottawa, on November 6, 1930. This shipment was made from the English Brook area of the Rice Lake district and contained three lots of ore; Lot No. 1, 10 bags of ore, weighing 550 pounds from the Lotus Claim; Lot No. 2, 10 bags of ore, weighing 550 pounds from the Denver Claim; and Lot No. 3, 4 bags of ore, weighing 250 pounds from the Regina Claim; and was shipped by J. A. Bradley, Room 207, Boyd Building, Winnipeg, Manitoba.

*Characteristics of the Ore.* The ore is siliceous and contains a small amount of sulphide. The gold occurs principally in the native state, only five per cent or less was found associated with the sulphide when the ore was crushed to pass 100 mesh. No flakes of gold larger than 20 mesh were observed in any of the samples. Pyrite is the predominating sulphide and the presence of a small amount of molybdenite is suspected.

The Lotus and Regina ores were very similar and contained more sulphides than the Denver. There was also more coarse gold in the Denver than in either of the other two.

*Sampling and Analysis.* Each lot of ore was carefully sampled. Two separate samples were cut from each and check assays were made on each.

The assays in ounces per ton of gold are as follows:—

| — | (1) | (2) | (3) | (4) | Average |
|---|---|---|---|---|---|
| Lotus | 0·81 | 0·81 | 0·82 | 0·79 | 0·81 |
| Denver | 0·73 | 0·77 | 0·83 | 0·81 | 0·77 |
| Regina | 0·30 | 0·27 | 0·29 | 0·27 | 0·28 |

The ore was difficult to sample on account of the presence of the large proportion of native gold which was freed when crushing the samples.

*Note.* Only 0·06 ounce of silver was found.

*Purpose of Experimental Tests.* The shipment was made for the purpose of designing a small 20- to 30-ton mill. The following tests were made to determine the process for recovering the gold which could be best adapted to such a small mill and still be consistent with good practice and recovery.

*Summary of Experimental Tests.* The first tests conducted were plain amalgamation tests.

Amalgamating at −35 mesh gave recoveries as follows: Lotus ore 71·6 per cent; Denver 92·2 per cent; Regina 67·9 per cent.

The second tests conducted were flotation tests, grinding the ore to −65 mesh. The results were as follows: Lotus ore 70·7 per cent recovery, grade of concentrate 4·56 ounces per ton, ratio of concentration 15·6: 1; Denver ore 85·1 per cent recovery, grade of concentrate 5·6 ounces per ton, ratio of concentration 33·3 : 1; Regina ore 71·8 per cent recovery, grade of concentrate 3·56 ounces per ton, ratio of concentration 13·7 : 1.

The third series of tests was by cyanidation, grinding the ore −35 mesh. The results were as follows: Lotus ore 96·3 per cent extraction with a cyanide consumption of 2·2 pounds per ton of ore; Denver ore 96·1 per cent extraction with a cyanide consumption of 0·2 pound per ton of ore; Regina ore 96·5 per cent extraction with a cyanide consumption of 0·2 pound per ton of ore.

The fourth series was the larger scale tests made on the remainder of the ore samples in which the ore was ground in a small ball mill and amalgamated on plates situated so that the circulating load between the ball mill and classifier, which operated in closed circuit with ball mill, passed over the plates. The final overflow of the classifier was practically −100 mesh. This overflow was the tailing from the amalgamation plates and was concentrated by flotation. The results were as follows: Lotus ore, recovery by amalgamation 91·4 per cent, additional recovery by flotation 6·3 per cent, grade of flotation concentrate 1·2 ounces per ton and ratio of concentration 23·6 : 1. Denver and Regina samples mixed and run together, 90·8 per cent recovery by amalgamation, additional recovery by flotation 3·1 per cent, grade of flotation concentrate 1·2 ounces per ton, and ratio of concentration 58 : 1.

The fifth of the series of tests made was the cyanidation of the flotation concentrates obtained from the preceding test. The concentrates without further grinding were mixed together and cyanided. The following is an average of three tests. Lotus, Denver, and Regina concentrates cyanided 52 hours in a 0·2 KCN solution, in a 3 : 1 pulp showed 92 per cent extraction with a cyanide consumption of 9·3 pounds per ton of ore.

*Process Recommended.* The process recommended for the 20-30 ton mill to treat these ores is briefly as follows. Crush the ore to ½ inch or finer as feed to a ball mill. Grind the ore in a ball mill to pass −65 mesh and approximately 70 per cent −200 mesh, pass the discharge of the mill directly over amalgamation plates either placing a screen at the mill discharge or reducing the size of the feed to the ball mill in order to prevent coarse material from scouring the plates. The plates must be set on a steep slope. The tailing from the plate should pass over a mercury trap and discharge into a Dorr type classifier which will operate in closed circuit with the ball mill. The overflow of the classifier, which is the amalgamation tailing, can then be either again passed over a second series of plates and go to waste or be floated in a small mechanical flotation machine, the concentrate being accumulated and either shipped to a smelter or cyanided at the mine.

The practice just recommended has been found to be quite successful and is at present being used in the mill of the Granada Rouyn mine near Noranda, Quebec.

## EXPERIMENTAL TESTS

### AMALGAMATION

A representative sample, 2,000 grammes −14 mesh ore, was cut from each of the three lots of ore. Each lot was crushed dry in a Braun pulverizer to −35 mesh and amalgamated in a jar with water and mercury. The mercury was removed and the residue assayed.

| — | Assay heads | Assay tailing | Extraction, per cent |
|---|---|---|---|
| Lotus | 0·81 | 0·23 | 71·6 |
| Denver | 0·77 | 0·06 | 92·2 |
| Regina | 0·28 | 0·09 | 67·9 |

### FLOTATION

A sample, 2,000 grammes −14 mesh, was cut from each lot of ore. This was floated after grinding to approximately −65 mesh in a laboratory ball mill. The reagents used were 0·08 pound per ton of Aerofloat No. 25 and 0·01 pound per ton amyl xanthate. A small amount of pine oil was added to cell.

| Products | Weight, per cent | Assay, Au oz./ton | Recoveries, per cent |
|---|---|---|---|
| Lotus calculated heads* | 100·00 | 0·42 | 100·00 |
| Lotus concentrate | 6·4 | 4·56 | 70·70 |
| Lotus tailing | 93·6 | 0·13 | 29·30 |
| Denver calculated heads* | 100·0 | 0·20 | 100·0 |
| Denver concentrate | 3·0 | 5·60 | 85·1 |
| Denver tailing | 97·0 | 0·03 | 14·9 |
| Regina calculated heads* | 100·0 | 0·36 | 100·0 |
| Regina concentrate | 7·3 | 3·56 | 71·8 |
| Regina tailing | 92·7 | 0·11 | 28·2 |

*The calculated heads assays obtained from the product of the tests do not check with the assay of the complete shipment. This is due to there being so much free native gold in the ore. The native gold is so heavy that it works into the cracks and hollows in the ball mill and cell and is difficult to work out. These tests are given in the report simply to show that flotation alone, and not supplemented by amalgamation, could not be used to recover the gold in the ore.

### CYANIDATION

A sample of each lot of ore was carefully cut from a large lot of the ore crushed to pass −35 mesh in a Braun pulverizer.

Each lot was agitated for 36 hours in a 3 : 1 pulp with a 0·054 per cent KCN solution and with the equivalent of 4 pounds of lime per ton of ore.

| Name of lot | Assay heads, oz./ton | Assay tailing, oz./ton | Extraction, per cent | KCN consumption per ton ore | Lime consumption per ton ore |
|---|---|---|---|---|---|
| Lotus | 0·81 | 0·03 | 96·3 | 2·2 | 3·94 |
| Denver | 0·77 | 0·03 | 96·1 | 0·2 | 3·72 |
| Regina | 0·28 | 0·01 | 96·5 | 0·2 | 3·88 |

## CONTINUOUS TEST USING AMALGAMATION AND FLOTATION

A continuous test was made on the balance of each lot of ore remaining after the head sample had been cut. The amounts of ore used in this test were approximately as follows:

Lotus, 450 pounds; Denver, 400 pounds; Regina, 100 pounds. The Denver and Regina lots were dry mixed and run as one sample. The flow-sheet used was to crush each lot to −6 mesh. This 6 mesh material was fed to a small ball mill operating in closed circuit with an Akins classifier. Amalgamation plates were placed between the ball mill discharge and the classifier so that the circulating load in the ball mill classifier circuit passed over the plates. The dilution of the classifier overflow was set at 3·5 : 1 and this product was the amalgamation tailing. A small 6-cell mechanical flotation machine was used to float the amalgamation tailing passing over the classifier weir. A concentrate and a tailing were made by flotation. The ore was fed at the rate of 125 pounds per hour. Starting with the Lotus ore samples were cut every five minutes at the following points: (1) plate discharge, (2) amalgamation tailing represented by the classifier overflow and feed to the flotation cells, and (3) tailing, and (4) concentrate discharge of the flotation cells.

The reagents used were, amyl xanthate 0·10 pound per ton ore added to the ball mill, and Aerofloat No. 25, 0·08 pounds per ton added direct to the flotation cells.

| Products | Assay heads, Au, oz./ton | Recovery, per cent | Ratio of concentration |
|---|---|---|---|
| Lotus heads | 0·81 | 100·0 | Using factor of 0·02 ounce obtained from clean-up actual recovery by amalgamation is 89 per cent 23·6 : 1. |
| Amalgamation tailing | 0·07 | 91·4 | |
| Flotation concentrate | 1·20 | 6·3 | |
| Flotation tailing | 0·02 | 2·3 | |
| Denver and Regina heads | 0·65 | 100·0 | Using factor of 0·02 ounce obtained from clean-up actual recovery by amalgamation is 87·8 per cent 58 : 1. |
| Amalgamation tailing | 0·06 | 90·8 | |
| Flotation concentrate | 1·20 | 3·1 | |
| Flotation tailing | 0·04 | 6·1 | |

A screen analysis was made of the amalgamation tailing from both lots of ore and also of the tailing from the clean-up of the ball mill and classifier after it had been collected and run over the plates. The analysis of the clean-up was made to determine whether or not any quantity of gold remained locked up in the circuit.

*Lotus Amalgamation Tailing*

| Mesh | Weight, grammes | Weight, per cent | Assay, Au, oz./ton |
|---|---|---|---|
| −65+100 | 7·2 | 2·4 | 0·12 |
| −100+150 | 20·0 | 6·7 | 0·06 |
| −150+200 | 71·4 | 23·8 | 0·05 |
| −200 | 201·4 | 67·1 | 0·08 |
| Totals | 300·0 | 100·0 | 0·07 |

*Denver and Regina Amalgamation Tailing*

| Mesh | Weight, grammes | Weight, per cent | Assay, Au, oz./ton |
|---|---|---|---|
| −65+100 | 12·7 | 3·2 | 0·11 |
| −100+150 | 32·9 | 8·2 | 0·07 |
| −150+200 | 80·0 | 20·0 | 0·04 |
| −200 | 274·4 | 68·6 | 0·08 |
| Totals | 400·0 | 1·000 | 0·07 |

*Amalgamation Tailing from Clean-up of Circuit*

| Mesh | Weight, grammes | Weight, per cent | Assay, Au, oz./ton |
|---|---|---|---|
| −65+100 | 1·0 | 0·3 | 0·15 |
| −100+150 | 6·3 | 2·1 | 0·33 |
| −150+200 | 50·1 | 16·7 | 0·17 |
| −200 | 242·6 | 80·9 | 0·22 |
| Totals | 300·0 | 100·0 | 0·21 |

The total weight of the material obtained from cleaning up the ball mill-classifier circuit was 100 pounds. After passing over the plates the tailing assayed 0·21 ounce per ton, Au. This is considerably higher than the tailing samples taken during the run and therefore when calculating the actual recovery by amalgamation this should be taken into consideration. If this is done the tailing from the amalgamation should be increased by 0·02 ounce per ton. The recovery is then reduced from 91·4 to 89 per cent in the Lotus and from 90·8 to 87·8 per cent in the Denver-Regina.

The screen tests on the amalgamation tailing indicate the degree of crushing for amalgamation and flotation.

### *Cyanidation of Flotation Concentrates*

Three tests were made on mixed concentrates obtained from the preceding continuous test. These concentrates were cyanided without being reground.

| | | |
|---|---|---|
| Heads | 1·20 | ounces per ton Au |
| Cyanide tailing | 0·10 | " " |
| Extraction | 92·0 | per cent |
| Reagent consumption—KCN | 9·3 | pounds per ton of ore |
| CaO | 10·00 | " " |

The above is an average of three tests.

## SUMMARY AND CONCLUSIONS

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

(1) A large proportion of the gold in all three lots of ore can be recovered by amalgamation.

(2) The ore cyanides very readily and a high extraction is obtained at −35 mesh with normal cyanide consumption.

(3) Flotation alone was not a successful means of concentrating the gold in these ores.

(4) That reasonably fine grinding followed by amalgamation and flotation of tailing will recover over 90 per cent of the gold in the ores.

(5) The flotation concentrate obtained in (4) can be readily cyanided with a high recovery and normal consumption of reagents without further grinding.

The conclusion drawn from results of the experimental test, keeping in mind the size of the mill to be erected, is that a combination method of amalgamation followed by flotation of the amalgamation tailing is the best method of treatment being much cheaper to install and much simpler to operate. The recovery by this method should exceed 90 per cent of the gold, 85 per cent of which should be recovered by amalgamation when the ore is crushed to the size indicated. These recommendations are based on the assumption that the samples received represented the ore which will be mined and put through the future mill, and that the mill will be properly designed and efficiently operated.

# III

# REPORTS OF INVESTIGATIONS: NON-METALLIC MINERALS SECTION

## Report No. 377

### ANALYSES AND EXPERIMENTAL TESTS ON A RADIUM-BEARING ORE FROM THE GREAT BEAR LAKE DISTRICT, N.W.T.

**B. P. Coyne and J. S. Godard**

*Shipments.* Two small samples marked Nos. 13 and 14 weighing 3,085 and 1,991 grammes respectively, were received December 1, 1930.

The samples were submitted by H. S. Spence of the Mineral Resources Division of the Department of Mines, on behalf of Mr. Gilbert Labine, of the Eldorado Gold Mining Company, Limited.

The samples were taken by Mr. Labine from two pitchblende veins on the property of the Eldorado Gold Mining Company, Limited, at Echo bay, Great Bear lake, N.W.T.

*Characteristics of the Ore.* While the samples were taken from two different veins a superficial examination showed them to be very similar. Both were high-grade pitchblende ores. The veins containing the pitchblende are cut by very minute thread-like veinlets, which are composed of a number of finely-crystalline sulphide minerals. Some erythrite is present and the presence of a small quantity of galena is suspected in sample No. 13.

Both samples contained some silver and while sample No. 13 contained an appreciable quantity of gold, sample No. 14 contained only a trace of this metal.

It was thought that some magnetite might be present, but a magnetic separation test failed to reveal its presence at least in such a quantity that it could be definitely detected.

The gangue material is of a siliceous nature and the veins occur in a pillow lava.

*Sampling.* The entire samples were crushed to pass 48 mesh. No. 13 was cut twice at this size and No. 14 once. The cuts were crushed to pass 200 mesh and were sampled in duplicate.

*Analyses.* The following is the analyses of the two samples.

| Element | No. 13 | No. 14 |
|---|---|---|
| $U_3O_8$ | 56·91 per cent | 63·94 per cent |
| Th | Nil | Nil |
| $V_2O_5$ | Trace | Nil |
| Cu | 0·70 per cent | 0·66 per cent |
| Ni | Trace | Trace |
| Co | 0·13 per cent | 0·10 per cent |
| PbO | 12·00 " | 12·13 " |
| Bi | 0·18 " | Trace |
| Ag | 1·41 oz./ton | 1·72 oz./ton |
| Au | 0·19 " | Trace |
| As | 0·15 per cent | 0·14 per cent |
| Fe | 1·01 " | 0·77 " |
| S | 2·08 " | 0·90 " |
| Ra | 144·51 mg./ton | 162·39 mg./ton |

## EXPERIMENTAL TESTS

Three small-scale experimental tests were made on the ore for the purpose of obtaining some additional information about the ore and to collect, if possible, the sulphides to permit their further examination.

### MAGNETIC SEPARATION TESTS

A magnetic separation test was made on each sample, for the purpose of determining the presence, or absence, of magnetite, by passing the ore ground to −200 mesh through a Dings magnetic machine.

Tests on both samples failed to reveal the presence of magnetite.

### FLOTATION

### *Test on Sample 13*

A small-scale flotation test was made, and the flotation concentrate was analysed with the following results.

| | | | |
|---|---|---|---|
| $U_3O_8$ | 49·40 per cent | S | 10·70 |
| Au | 0·96 oz./ton | Cu | 3·57 |
| Ag | 5·54 " | As | 0·12 |
| PbO | 12·71 per cent | $MoS_2$ | 0·30 |

Ratio of concentration 8·7 : 1.

The results of the flotation test was somewhat disappointing, principally because of the tendency of the finely ground pitchblende to float. This pitchblende masked the sulphide minerals, and the succeeding microscopic examination failed to reveal much definite information.

The flotation concentrate contained roughly five times the head assay in the following elements, gold, sulphur, and copper, and four times the head assay in silver.

It was from the results of this test, however, that the presence of molybdenite was first established and the presence of galena first suspected.

A comparison of the relationship of lead to $U_3O_8$ in the head sample to that in the flotation concentrate may be shown as follows:

$$\text{In the head sample } \frac{\text{PbO}}{U_3O_8} = \frac{12 \cdot 00}{56 \cdot 91}$$

$$\text{In the flotation concentrate } \frac{\text{PbO}}{U_3O_8} = \frac{12 \cdot 71}{49 \cdot 40}$$

If there were no additional lead other than that relating to the pitchblende the relationship in the flotation concentrate would be:

$$\frac{\text{PbO}}{U_3O_8}\left(\frac{12 \cdot 00}{56 \cdot 91}\right) = \frac{X}{49 \cdot 40} \text{ X being equal to } 10 \cdot 4 \text{ per cent PbO.}$$

In place of the calculated value of 10·4 per cent we have by analyses found 12·71 per cent PbO, an increase of 2·30 per cent. It was on this basis of reasoning plus the presence of other sulphides that the presence of galena is suspected. Also, the relationship of lead to $U_3O_8$ in both samples is higher than usually credited to pitchblende.

### CONCLUSIONS

Both samples were very high-grade radium-bearing ores. Their value is enhanced by the absence of the element thorium and the analyses of the $U_3O_8$ much simplified by the low iron content of the samples.

## Report No. 378

# THE GRINDING OF SCRAP MICA FROM THE BLACKBURN MICA MINE, TEMPLETON TOWNSHIP, PAPINEAU COUNTY, QUEBEC

**R. K. Carnochan**

*Purpose of Experimental Tests.* Blackburn Brothers, Limited, of Ottawa, who operate the Blackburn mica mine, Templeton township, Papineau county, Quebec, have a trimming and splitting plant at Ottawa. A certain amount of scrap mica is produced both at the mine and the plant. Mr. H. L. Forbes, engineer for Blackburn Brothers, Limited, desired that tests be made to determine a suitable method of grinding the scrap mica.

*Shipments.* Two lots of scrap mica were sent in by Blackburn Brothers, Limited, from their Ottawa plant. One was received November 21, 1929, and the other January 24, 1930. Each shipment contained 10 bags and weighed 1,000 pounds.

*Characteristics of the Shipments.* The shipments consisted of phlogopite mica ranging in size from very small flakes up to pieces 3 inches by 5 inches and ½ inch thick. Very little gangue was present in the mica.

## Experimental Tests

### GRINDING

As the mica was slightly damp when received it was dried by placing the bags on steam coils. The net weight of dry mica in the first lot was 914 pounds, and in the second lot 978 pounds.

When phlogopite mica is ground the product is a fawn colour and because of this is unsuitable for the use in making wall-paper or paints. It was decided to grind the mica dry, as dry grinding is cheaper and wet grinding is necessary only when grinding mica for wall-paper or paints, as it preserves the sheen and gives a brighter product. It is not essential to keep the sheen on mica that is used for other purposes.

### *Grinding with Hammer Mill*

A number of tests were made on the dried mica using a No. 00 Sturtevant hammer mill for grinding and a 3-foot by 5-foot Hummer vibrating screen for separating out the −20 material, the oversize being returned to the mill. Different grates and hammers were used in the mill to see which would give the best results. The method of making a test was to run 50 pounds of mica through the mill and over the Hummer screen. Unground mica was added to the +20 to bring it up to 50 pounds and it was then ground and screened again. Some more unground mica was again added to the +20 and it was ground and screened for the third time. After the third grinding, conditions would be in equilibrium, that is about the same amount of −20 would be obtained from the second and third pass and, therefore, the third pass could be used to figure capacity and a screen test could be made on the −20 with confidence that it would be secured in continuous operation. Sometimes four passes were made but usually three were sufficient.

The different grates used on the hammer mill were, $\frac{1}{2}$ inch, $\frac{1}{4}$ inch, $\frac{1}{8}$ inch, and $\frac{1}{16}$ inch. A $\frac{1}{16}$-inch round hole screen was also used. U hammers and straight hammers were used.

Table I gives the results of the tests made with the hammer mill.

### *Grinding with KEK Mill*

A quantity of the scrap mica was ground in the hammer mill, fitted with a $\frac{1}{16}$-inch grate and U hammers, and then run over the Hummer screen fitted with a 20 mesh wire cloth. Fifty pounds of the +20 mesh material were ground in a 13-inch KEK mill and screened on 20 mesh with the Hummer screen. To the oversize enough +20 from the hammer mill was added to bring it up to 50 pounds, and this was ground in the KEK mill and screened. The oversize was again brought up to 50 pounds and ground and screened. This made the third time that the KEK mill had been used and the grinding was in equilibrium; that is the same amount of −20 was being obtained from each run.

Table II gives the results of this work.

### *Grinding with Raymond Pulverizer*

Four different tests were made with the Raymond pulverizer.

The first test was made by grinding the scrap mica in the hammer mill, fitted with a $\frac{1}{16}$-inch grate and U hammers, and then screening on 20 mesh with the Hummer screen. Twenty-five pounds of the +20 material was then put through the Raymond No. 0000 pulverizer with the regulator set at $\frac{1}{8}$ inch and screened on 20 mesh. The oversize was brought up to 25 pounds by adding +20 material from the hammer mill and run through the Raymond pulverizer. This procedure was kept up using different feed rates until a rate was obtained that loaded the Raymond pulverizer to full capacity but did not overload it.

The second test was the same as the first except that the regulator was set at $\frac{3}{8}$ inch, this giving a greater capacity but not so much fine material in the −20.

The third test was the same as the first except that the scrap mica was fed directly to the Raymond pulverizer, the regulator being set at $\frac{1}{8}$ inch.

The fourth test was the same as the first except that the scrap mica was fed directly to the Raymond pulverizer and the regulator set at $\frac{3}{8}$ inch.

Table III gives the data obtained from these tests.

### *Summary of Grinding Tests*

The largest capacity obtained from the hammer mill where the +20 was returned to the mill was 127 pounds of −20 per hour and this was obtained by using a $\frac{1}{16}$-inch grate and either U or straight hammers. The straight hammers would be best to use in practice as they are less costly.

The KEK mill gave the largest capacity of any machine used, 340 pounds of −20 per hour, but this machine is no longer being manufactured.

The Raymond pulverizer gave fair capacities, when grinding the +20 from the hammer mill the capacity was 107 pounds of −20 per hour with the regulator set at $\frac{1}{8}$ inch and 173 pounds of −20 per hour with the regulator set at $\frac{3}{8}$ inch, when grinding the scrap mica the capacity was 70 pounds of −20 with the regulator set at $\frac{1}{8}$ inch and 261 pounds of −20 per hour with the regulator set at $\frac{3}{8}$ inch. The Raymond pulverizer has an advantage in that it can be regulated so as to give a −20 product containing a large amount of fines if desired. It will take the scrap mica without any preliminary crushing but in commercial operation it would be best to crush the mica in a hammer mill, screen on 20 mesh and feed the +20 to the Raymond pulverizer. This would do away with any possibility of the feed blocking the Raymond pulverizer, and it would also give a more regular feed.

## SCREENING

### *Screening with Rotex Screen*

All the −20 mica produced in the grinding tests was screened on a No. 7 (40 inches by 84 inches) Rotex screen fitted with a 65 mesh silk bolting cloth. This screen was used because tests made in our laboratory on mica show that the Rotex screen has a larger capacity per square foot and gives a cleaner oversize than an inclined vibrating screen when screening on 20 or 65 mesh.

Different slopes were tried on the Rotex screen. A slope on the screen surface of about 4 inches in its length was found to give the best results.

Suction was tried on the screen to make a better separation, but it was found that the suction held the mica flakes against the screen and so blocked it. The best results were obtained without suction.

The capacity of the No. 7 Rotex screen varies according to the product being fed to it. At a fixed rate to the Rotex screen the finer the feed the greater the amount of throughs and the greater the amount of fines left in the oversize. Hence it is necessary on fine feeds to feed slower.

Table IV shows the results obtained by the Rotex screen.

### *Summary of Tests on Rotex Screen*

A study of the table shows that for any product screened the faster the feed the more throughs per given time and the more fines in the oversize.

Hence it is necessary in determining the capacity to select a test in which the fines in the oversize are not excessive. Considering this it seems that the tests from which capacities should be estimated are the third one on "hammer mill discharge," the fifth one on "−20 from hammer mill," and the second one on "−20 from Raymond pulverizer."

## CONCLUSIONS

(1) It would be unnecessary to grind this mica wet as due to its colour it cannot be used for wall-paper or paints and it is only where mica is ground for those purposes that wet grinding is needed to preserve the sheen, besides dry grinding gives much larger capacities for the same horse-power and equipment value.

(2) A hammer mill or Raymond pulverizer would be suitable for dry grinding. The Raymond pulverizer having the advantage of giving a product containing a lot of fines if desired. The hammer mill has a larger capacity for the horse-power consumed and the value of the machine.

(3) A Rotex screen would be suitable for screening the ground mica. A three-surface screen could be used to make an oversize to return to the grinder and three different sizes of finished products.

(4) Due to the presence in the mica of scrap iron some form of magnetic pulley should be used to remove this iron before grinding the mica.

## TABLE I

### Tests made with Hammer Mill

| Grate and hammers used | Feed, pounds per min. | +20, pounds per min. | −20, pounds per min. | Feed, pounds per hr. | +20, pounds per hr. | −20, pounds per hr. | Screen test on −20 | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | Per cent, −20 +28 | Per cent, −28 +35 | Per cent, −35 +48 | Per cent, −48 +65 | Per cent, −65 +100 | Per cent, −100 +150 | Per cent, −150 +200 | Per cent, −200 +325 | Per cent, −325 |
| ½-inch grate, U hammers | 8·33 | 7·16 | 1·17 | 500 | 430 | 70 | 6·5 | 24·2 | 18·9 | 12·4 | 11·9 | 8·8 | 4·7 | 12·6 | ...... |
| ¼-inch grate, U hammers | 7·14 | 5·86 | 1·28 | 428 | 351 | 77 | 12·4 | 29·8 | 15·9 | 9·4 | 9·1 | 6·7 | 3·7 | 13·0 | ...... |
| ⅛-inch grate, U hammers | 8·33 | 6·25 | 2·08 | 500 | 375 | 125 | 16·5 | 23·6 | 14·3 | 9·1 | 10·7 | 7·3 | 4·2 | 14·3 | ...... |
| $\frac{1}{16}$-inch grate, U hammers | 5·50 | 3·39 | 2·11 | 330 | 203 | 127 | 11·3 | 21·5 | 12·4 | 8·8 | 11·0 | 9·1 | 5·6 | 20·3 | ...... |
| $\frac{1}{16}$-inch grate, straight hammers | 5·56 | 3·45 | 2·11 | 334 | 207 | 127 | 9·7 | 24·2 | 14·8 | 11·0 | 10·7 | 9·0 | 4·7 | 10·2 | 5·7 |
| $\frac{1}{16}$-inch round hole screen, U hammers | 2·17 | 0·28 | 1·89 | 130 | 17 | 113 | 1·1 | 12·8 | 14·5 | 12·7 | 14·1 | 12·8 | 6·7 | 13·4 | 11·9 |
| $\frac{1}{16}$-inch round hole screen, straight hammers | 2·08 | 0·77 | 1·31 | 125 | 46 | 79 | 0·2 | 8·4 | 14·7 | 13·5 | 14·9 | 13·5 | 7·6 | 16·4 | 10·8 |

## TABLE II

### Tests made with KEK Mill

| Machine used | Feed, pounds per min. | +20, pounds per min. | −20, pounds per min. | Feed, pounds per hr. | +20, pounds per hr. | −20, pounds per hr. | Screen test on −20 | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | | | Per cent, −20 +28 | Per cent, −28 +35 | Per cent, −35 +48 | Per cent, −48 +65 | Per cent, −65 +100 | Per cent, −100 +150 | Per cent, −150 +200 | Per cent, −200 |
| Hammer mill, $\frac{1}{16}$-inch grates, U hammers, running on scrap, no +20 returned.............. | 7·87 | 3·68 | 4·19 | 472 | 221 | 251 | 2·8 | 17·5 | 17·5 | 13·9 | 13·8 | 10·7 | 5·4 | 18·4 |
| KEK mill, running on +20 from hammer mill, +20 returned, third run...................... | 11·78 | 6·11 | 5·67 | 707 | 367 | 340 | 9·8 | 20·5 | 14·4 | 10·7 | 10·7 | 9·0 | 5·1 | 19·8 |

## TABLE III

### Tests made with Raymond Pulverizer

| Method used | Feed, pounds per min. | +20, pounds per min. | −20, pounds per min. | Feed, pounds per hr. | +20, pounds per hr. | −20, pounds per hr. | Screen test on −20 |  |  |  |  |  |  |  |  |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
|  |  |  |  |  |  |  | Per cent, −20 +28 | Per cent, −28 +35 | Per cent, −35 +48 | Per cent, −48 +65 | Per cent, −65 +100 | Per cent, −100 +150 | Per cent, −150 +200 | Per cent, −200 +325 | Per cent, −325 |
| Hammer mill, 1/16-inch grate, U hammers, running on scrap, no +20 returned.......... | 8·72 | 3·99 | 4·73 | 523 | 239 | 284 | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... | ...... |
| Raymond pulverizer on +20 from hammer mill, +20 returned, regulator at 1/8 inch..... | 1·97 | 0·18 | 1·79 | 118 | 11 | 107 | 0·4 | 5·0 | 7·2 | 8·6 | 10·6 | 15·7 | 10·7 | 32·9 | 8·9 |
| Raymond pulverizer running on +20 from hammer mill, +20 returned regulator at 3/8 inch..... | 4·28 | 1·40 | 2·88 | 257 | 84 | 173 | 1·5 | 18·1 | 16·2 | 12·3 | 13·2 | 10·9 | 5·6 | 6·2 | 16·0 |
| Raymond pulverizer running on scrap, +20 returned, regulator set at 1/8 inch.............. | 1·24 | 0·06 | 1·18 | 74 | 4 | 70 | 0·1 | 1·5 | 3·4 | 5·5 | 10·9 | 16·6 | 13·2 | 15·5 | 33·3 |
| Raymond pulverizer running on scrap, +20 returned, regulator set at 3/8 inch.............. | 7·50 | 3·15 | 4·35 | 450 | 189 | 261 | 0·6 | 14·7 | 16·7 | 12·6 | 13·1 | 11·6 | 6·3 | 12·4 | 12·0 |

# TABLE IV

## Tests made with Rotex Screen

| Feed | Screen used, mesh | Feed, lb./hr. | Oversize, lb./hr. | Fines, lb./hr. | Screen test on oversize, per cent | | | | Screen test on fines, per cent | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | +20 | −20+28 | −28+35 | −35+20 | −20+28 | −28+35 | −35+48 | −48+65 | −65+100 | −100+150 | −150+200 | −200+325 | −325 |
| Hammer mill.... | 20 | 333 | 113 | 220 | .... | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ...... |
| Discharge........ | ........ | ........ | ........ | ........ | .... | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ...... |
| “ ........ | 20 | 437 | 188 | 249 | .... | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ........ | ...... |
| “ ........ | 20 | 606 | 300 | 306 | 92·0 | 6·2 | 0·7 | 1·1 1·3 | 19·3 | 18·9 | 13·7 | 10·5 | 9·9 | 7·2 | 4·9 | 8·9 | 5·4 |
| “ ........ | 20 | 980 | 530 | 450 | 87·8 | 8·2 | 1·7 | 2·3 1·6 | 18·0 | 19·4 | 13·7 | 10·5 | 9·8 | 8·0 | 4·3 | 6·8 | 7·9 |
| “ ........ | 20 | 1,500 | 900 | 600 | 86·0 | 8·5 | 2·1 | 3·4 2·5 | 21·7 | 18·9 | 13·0 | 9·5 | 8·6 | 7·3 | 3·9 | 9·1 | 5·5 |

| Feed | Screen used, mesh | Feed, lb./hr. | Oversize, lb./hr. | Fines lb./hr. | Screen test on oversize, per cent | | | | Screen test on fines, per cent | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | | | | | +65 | −65+80 | −80+100 | −100 | −65 | −65+80 | −80+100 | −100 |
| −20 from hammer mill.................................... | 65 | 450 | 335 | 115 | 79·2 | 7·8 | 6·2 | 6·8 | 0·01 | 0·2 | 4·8 | 94·99 |
| “ “ .................................... | 65 | 337 | 247 | 90 | 82·0 | 8·0 | 5·6 | 4·4 | 0·03 | 0·2 | 7·4 | 92·37 |
| “ “ .................................... | 65 | 276 | 198 | 78 | 82·0 | 8·2 | 5·8 | 4·0 | 0·2 | 0·4 | 8·2 | 91·2 |
| “ “ .................................... | 65 | 230 | 160 | 70 | 81·8 | 8·4 | 5·6 | 4·2 | 0·2 | 0·4 | 7·4 | 92·0 |
| “ “ .................................... | 65 | 197 | 135 | 62 | 84·0 | 7·6 | 5·4 | 3·0 | 0·4 | 0·2 | 7·8 | 91·6 |
| “ “ .................................... | 65 | 136 | 94 | 42 | 84·6 | 8·6 | 5·2 | 1·6 | 0·2 | 0·2 | 6·6 | 93·0 |
| “ “ .................................... | 65 | 110 | 75 | 35 | 85·2 | 8·2 | 5·0 | 1·6 | 0·2 | 0·4 | 8·8 | 90·6 |
| “ “ .................................... | 65 | 89 | 61 | 28 | 84·8 | 8·6 | 5·0 | 1·6 | 0·2 | 0·2 | 8·4 | 91·2 |
| −20 from Raymond pulverizer.............................. | 65 | 270 | 87 | 183 | 66·4 | 12·7 | 13·3 | 7·6 | ........ | ........ | 5·9 | 94·1 |
| “ “ .............................. | 65 | 176 | 52 | 124 | 74·4 | 12·9 | 10·7 | 2·0 | 0·02 | 0·2 | 14·3 | 85·48 |
| “ “ .............................. | 65 | 133 | 40 | 93 | 73·4 | 13·8 | 10·6 | 2·2 | 0·01 | 0·03 | 9·1 | 90·86 |

## Report No. 379

# EXPERIMENTAL TESTS ON THE POSSIBLE USE OF AN OCCURRENCE OF DOLOMITIC LIMESTONE NEAR THE HOWEY GOLD MINE, IN THE CYANIDE TREATMENT OF HOWEY GOLD ORE

**M. F. Goudge and J. S. Godard**

*Introduction.* Owing to the high transportation charges from railhead to the Howey gold mine in the Red Lake area of Ontario and the proximity of a body of dolomitic limestone to the mine, the possibility of burning this limestone and utilizing, in the cyanide process, the lime so obtained has received the serious consideration of Mr. H. G. Young, Superintendent of the Howey Gold Mines, Limited.

Mr. Young submitted five samples of this limestone to the Department of Mines with the request that they be investigated as to burning, then tested metallurgically.

## ANALYSES AND BURNING OF THE LIMESTONE

**M. F. Goudge**

The first four samples submitted by Mr. Young were very low-grade dolomitic limestones, high in silica, and when burned for two hours at 1,200°C. yielded a product that would slake only after being in contact with water for six yours. They appeared generally unsuited for the intended purpose.

The fifth sample, received February 25, 1930, was a better grade of limestone than the previous samples.

Analyses showed it to contain:

| | | | |
|---|---|---|---|
| Insoluble | 7·12 per cent | $R_2O_3$ | 4·47 per cent |
| $CaCO_3$ | 50·84 " | $MgCO_3$ | 36·75 " |

This limestone was calcined for a period of two hours at an average temperature of 900°C. A brownish-coloured lime was obtained which would slake in cold water in five minutes.

Analyses of the lime:

| | | | |
|---|---|---|---|
| Insoluble | 14·25 per cent | $Fe_2O_3$ | 6·78 per cent |
| CaO | 46·20 " | MgO | 28·15 " |
| $CO_2$ | 1·50 " | $SO_3$ | Nil |

Apparently this latter stone must be burned at a temperature not exceeding 1,000°C. in order to produce a lime of satisfactory slaking propensities.

There is present a very considerable amount of free quartz and material that will not slake.

This lime was submitted to the Ore Dressing Division for metallurgical testing.

# COMPARATIVE TESTS OF A BURNED DOLOMITIC LIMESTONE WITH THAT OF A COMMERCIAL LIME, IN THE CYANIDE PROCESS, FOR THE HOWEY GOLD MINES, LTD., RED LAKE DISTRICT, ONTARIO

**J. S. Godard**

A small sample of burned dolomitic limestone was received March 4, 1930, from Mr. M. F. Goudge, Mineral Resources Division.

It was requested that this lime be compared with an ordinary commercial lime in the cyanide process using gold ore from Howey Gold Mines, Limited.

*Analyses of the Limes Used—*

| — | Insol. | $Fe_2O_3$ | CaO | MgO | $CO_2$ | $SO_3$ |
|---|---|---|---|---|---|---|
| Commercial | 5·84 | ............ | 64·46 | 0·23 | 15·00 | ............ |
| Howey | 14·25 | 6·78 | 46·20 | 28·15 | 1·50 | ............ |

Comparative tests were made to ascertain any difference in the six main points involved, namely:

1. Available CaO.
2. Alkalinity.
3. Lime consumption.
4. Effect on extraction of the gold.
5. Effect on cyanide consumption.
6. Settling rate.

### *Tests to Determine the Available CaO*

### *(Standard Sugar Solution Test)*

| | |
|---|---|
| Cane sugar solution, 2 per cent | 2 litres |
| Lime, commercial | 4 grammes |
| Agitation period | 2 hours |

Done in duplicate.
Average per cent CaO in solution, 0·0311 per cent.

Test repeated in duplicate, substituting Howey lime for commercial.

Average per cent CaO in solution, 0·0373 per cent.

Per cent available alkalinity in lime.

| | |
|---|---|
| Commercial | 15·55 per cent |
| Howey | 18·65 " |

Difference in favour of Howey lime, 3·10 per cent.

### *Cyanide Tests*

Comparative cyanide tests were made on ore ground to various sizes. Commercial lime used in odd number tests. Howey lime used in even number tests.

| | |
|---|---|
| Cyanide, Cassels | 96·7 per cent NaCN |
| Strength of solution | 0·05 " KCN |
| Dilution | 1 : 3 |
| Time | 45 hours |

*Screen Analyses on the Cyanide Tailings—*

Ore ground −65 mesh.

| Test No. | Mesh | Weight, per cent | Assay, Au, oz./ton | Per cent of values, | Average tailing, Au, oz./ton |
|---|---|---|---|---|---|
| 1 | +100 | 24·4 | 0·04 | 48·5 | 0·02 |
| | +200 | 28·7 | 0·02 | 28·2 | |
| | −200 | 46·9 | 0·01 | 23·3 | |
| 2 | +100 | 24·0 | 0·03 | 40·0 | 0·018 |
| | +200 | 31·8 | 0·02 | 35·5 | |
| | −200 | 44·2 | 0·01 | 24·5 | |

Ore ground −100 mesh.

| | | | | | |
|---|---|---|---|---|---|
| 3 | +200 | 27·0 | 0·02 | 59·4 | 0·009 |
| | −200 | 73·0 | 0·005 | 40·6 | |
| 4 | +200 | 30·3 | 0·02 | 63·6 | 0·010 |
| | −200 | 69·7 | 0·005 | 36·4 | |

Ore ground −150 mesh.

| | | | | | |
|---|---|---|---|---|---|
| 5 | +200 | 14·1 | 0·015 | 32·8 | 0·006 |
| | −200 | 85·9 | 0·005 | 67·2 | |
| 6 | +200 | 15·3 | 0·02 | 42·4 | 0·007 |
| | −200 | 84·7 | 0·005 | 57·6 | |

*Summary—Cyanide Tests*

| Test No. | Mesh | Head, Au, oz./ton | Tailing, Au, oz./ton | Extraction, per cent | Reagents, lb./ton | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| 1 | −65 | 0·70 | 0·02 | 97·2 | 0·13 | 3·7 C. |
| 2 | −65 | ......... | 0·018 | 97·4 | 0·12 | 3·4 H. |
| 3 | −100 | 0·71 | 0·009 | 98·7 | 0·01 | 4·6 C. |
| 4 | −100 | ......... | 0·010 | 98·6 | 0·12 | 4·4 H. |
| 5 | −150 | 0·73 | 0·006 | 99·2 | 0·12 | 5·3 C. |
| 6 | −150 | ......... | 0·007 | 99·0 | 0·25 | 5·1 H. |

*Settling Tests*

Comparative settling tests were made on each cyanide pulp at the end of the agitation period.

## CONCLUSIONS

1. *Available CaO.* Standard sugar solution tests indicated that Howey lime yielded 3·10 per cent more available CaO than the commercial lime.

2. *Alkalinity.* In the cyanide tests where Howey lime was compared with equal quantities of the commercial lime a higher protective alkalinity was obtained where Howey lime was used.

3. *Lime Consumption.* The lime consumption, measured in pounds per ton of ore, was very similar for each lime in corresponding tests.

4. *Effect on Extraction of Gold.* No appreciable differences were found in the extractions obtained in corresponding tests.

5. *Effect on the Cyanide Consumption.* The cyanide consumptions in Tests Nos. 1 and 2 were almost identical. In Tests Nos. 3 and 4, and 5 and 6, less cyanide was consumed in those tests where commercial lime was used.

The differences measured in per cent are very large, but when due consideration is given to the fact that the titrations were made on 25 c.c. samples and any errors are multiplied by $\frac{20}{3}$, the differences may be attributed to error of experiment rather than any cyanicidal effect of the Howey lime. About $\frac{1}{10}$ c.c. of silver nitrate would account for the differences involved. No. 3 is without doubt too low.

Again the variations in the cyanide consumptions, omitting No. 3, are no greater than those in the cyanide tests, conducted on Howey ore during March 1929, where the lime used was the same throughout. At that time indicated cyanide consumptions were from 0·10 to 0·35 pound KCN per ton of ore.

*Settling Rates.* Tests to determine any differences in the settling rates were made on the cyanide pulps at the end of the agitation period.

## GENERAL CONCLUSIONS

The literature on the subject of the use of magnesian lime in the cyanide process is meagre and the opinions expressed antithetic.

Chemically magnesia, MgO, is sparingly soluble in water and that present in the Howey lime should theoretically pass out of the circuit with the tailings. On the other hand magnesia, MgO, reacts with sulphates in solution forming the soluble salt $MgSO_4$, which is said to be more or less of a cyanicide.

Past work on the cyanidation of Howey ore shows it to be a fairly low consumer of lime and a low consumer of cyanide. The low lime consumption indicates the absence of soluble sulphates in any appreciable quantities. Were these soluble sulphates present considerable lime would be required to hold the protective alkalinity as the lime would be largely consumed in forming the insoluble calcium sulphate and only the excess available to give the necessary OH reaction.

Low cyanide consumption indicates the absence of formation of thiocyanates in appreciable quantities.

Were this ore a heavy consumer of lime or cyanide, or both, the writers would hesitate before recommending the use of such a high magnesian lime in the cyanide circuit. However, taking into consideration the transportation charges on lime secured from outside sources and the fact that Howey ore is a fairly low consumer of both lime and cyanide, tends us to commit ourselves to the experiment of burning the dolomitic limestone and the use of this lime in the mill; providing, of course, that the dolomitic limestone approximates in analysis that of the last sample submitted and that it is burned as recommended in this report.

## Report No. 380

### THE EXAMINATION OF ANHYDRITE FROM CANADIAN DEPOSITS

**R. A. Rogers**

*Shipments.* Thirteen samples of anhydrite, total weight 2,637 pounds, have been received from L. H. Cole, Mines Branch, Ottawa. These were collected by him during the summers of 1927, 1928, 1929, and 1930 from various Canadian occurrences, distributed as follows:—

| | | | |
|---|---|---|---|
| Nova Scotia............. | 8 samples | Manitoba................. | 1 sample |
| New Brunswick........... | 3 " | British Columbia......... | 1 " |

*Purpose of Tests.* To determine the possibility of utilizing anhydrite as a plaster-making material.

*Sampling and Analysis.* The anhydrite as received was in lumps weighing 1½ to 3 pounds. Each sample was first crushed to 1 inch in a gyratory crusher. Five samples were then ground in pebble mills to 100 mesh. One sample was reduced to 100 mesh in hammer and burr mills. The other seven samples were passed through a small jaw crusher, small rolls, disk pulverizer, and 20 mesh screen.

The ground material in each case was cut with a Jones riffle and a sample obtained for chemical analysis. The analyses with one exception were made by the writer. The following components were determined:—

| | | | |
|---|---|---|---|
| Insoluble........................... ....... | | Magnesia.................... | ($MgO$) |
| Ferric oxide.................. | ($Fe_2O_3$) | Sulphur trioxide.............. | ($SO_3$) |
| Alumina...................... | ($Al_2O_3$) | Water........................ | ($H_2O$) |
| Lime......................... | ($CaO$) | Carbon dioxide............... | ($CO_2$) |

The combined water in the samples varied from 0·06 per cent to 8·55 per cent showing that different amounts of gypsum are present.

### EXPERIMENTAL TESTS

Many patents have been filed in Canada, the United States, and Europe, in which anhydrite forms the base for the manufacture of commercial plasters.

The Annual Report on the Mines, 1929, Part I, of the Department of Public Works and Mines of Nova Scotia, contains a report, "Anhydrite Plasters and Cements," which gives the results of an investigation on anhydrite undertaken by A. E. Flynn, A.R.S.M., Professor of Mining Engineering, Nova Scotia Technical College, Halifax, Nova Scotia.

*Method of Procedure.* About 250 small-scale tests have been made.

*Grinding.* Lots of 2,000 grammes of the 20 mesh or 100 mesh material were ground in a pebble jar for 4 hours. Screen analyses made after grinding showed that most of the product would pass through 325 mesh.

*Catalysers.* Varying percentages of agents from 0·5 to 5·0 per cent and in one test, 10 per cent, were added either to the sample in the pebble jar or else to the water in which the sample was mixed. The following agents consisting almost entirely of inorganic chemical salts were used:—

| | | |
|---|---|---|
| 1. | Aluminium sodium sulphate | $Al_2\ Na_2\ (SO_4)_4.\ 24H_2O$ |
| 2. | Aluminium sulphate | $Al_2(SO_4)_3.\ 18H_2O$ |
| 3. | Calcium chloride | $CaCl_2$ |
| 4. | Magnesium chloride | $MgCl_2.\ \ 6H_2O$ |
| 5. | Magnesium sulphate | $MgSO_4.\ \ 7H_2O$ |
| 6. | Sodium sulphate $Na_2SO_4$ (prepared by drying the salt from western Canada.) | |
| 7. | Sodium pyrosulphate | $Na_2S_2O_7$ |
| 8. | Ammonium sulphate | $(NH_4)_2\ SO_4$ |
| 9. | Ammonium acid sulphate | $NH_4HSO_4$ |
| 10. | Volcanic ash | |
| 11. | Sodium tetraborate | $Na_2B_4O_7$ |
| 12. | Sodium pyrosulphate<br>Sodium sulphate | |
| 13. | Sodium carbonate | $Na_2CO_3$ |
| 14. | Zinc sulphate<br>Potassium sulphate | |
| 15. | Zinc sulphate<br>Sodium sulphate | |
| 16. | Zinc sulphate<br>Sodium pyrosulphate | |
| 17. | Zinc sulphate<br>Calcium hydroxide | |
| 18. | Zinc sulphate | $ZnSO_4.\ 7H_2O$ |
| 19. | Potassium sulphate | $K_2SO_4$ |

*Consistency.* Distilled water was used for all the tests. Sufficient water was mixed with the sample so that when a perpendicular face was made after thorough mixing, the plaster would not slump. The amount of water required varied from 19 to 50 cubic centimetres per 100 grammes of anhydrite.

*Time of Setting.* The time of setting was determined by the glass plate method, and is the elapsed time from when the sample is first added to the water, mixed, and spread on a thin glass plate, till it will leave this plate when the latter is flexed by the hands.

*Briquettes.* In this investigation, only the tensile strength of the plasters was determined. Five briquettes were made in standard moulds for each test. These were dried at room temperature for at least 7 days, but to constant weight.

*Tensile Strength.* The briquettes were broken in an Olsen-Boyd automatic cement tester. Plasters have been made with tensile strengths exceeding 500 pounds per square inch.

*Water in Plaster.* The water in the air-dried briquettes was determined.

*Other Characteristics.* In a large percentage of the tests, an efflorescence of varying amount formed on the surface of the plaster. This was caused by the chemical added to the anhydrite.

In nearly every case, shrinkage to a greater or less extent was observed, although in some tests swelling occurred and cellular plasters were obtained.

## SUMMARY OF TESTS

A complete summary of the tests and chemical analyses of the samples will be contained in the report, "Anhydrite, Its Occurrence, Properties, and Possible Utilization" to be published by the Mines Branch.

## CONCLUSIONS

The tests show that plasters may be made from anhydrite by the addition of small amounts of certain chemical salts. These plasters will set and as a rule have a tensile strength comparable with and in many cases exceeding that of ordinary plaster of Paris (calcined gypsum). The various anhydrites give different results with the same salt. Objectionable features are the shrinkage and efflorescence which occur in many cases, but these difficulties may be almost entirely overcome by using zinc sulphate and potassium sulphate together as the catalyser. This is the mixture particularly recommended by Victor Lefebure in his Canadian Patent No. 293,763, October 8, 1929, for "Process for the Manufacture of Plasters from Anhydrite."

## IV

# REPORT OF INVESTIGATIONS: SECTION OF FERROUS METALLURGY

**T. W. Hardy**

During most of the year the efforts of this section, as well as those of the section of Hydrometallurgy, were largely directed toward the making of tests in connexion with the demonstration of the Musso process for the semi-direct production of steel from iron ore. As was mentioned in the last Summary Report, the tests to demonstrate the technical and commercial possibilities of the Musso process were undertaken by the Department in co-operation with a group of Canadian financiers interested in the process. For these tests a pilot plant designed by the inventor and financed by the group interested, was installed in the pyrometallurgical laboratory. The tests were conducted under the general supervision of the inventor.

A report of the tests on the Musso process will be found on pages 189 to 195 in this report. It is a matter of considerable regret that the mechanical difficulties that presented themselves throughout were so great as to make it impossible to get any data on the metallurgical features of the process.

The tests have, however, allowed an opportunity of becoming more familiar with the general problem of the low-temperature reduction of iron ores and have made it clear that, even with a practical and cheap method of carrying out this reduction, it is by no means assured that by converting them into sponge iron, the commercial utilization of Canada's low-grade iron ores is made possible.

In its relation to the steel industry, sponge iron must be regarded as a base for melting, that is to say, as a substitute for muck bars and other special melting bases in the manufacture of quality steels, or as a substitute for steel scrap in the manufacture of common steels. It is of course obvious that where it is used as a substitute for muck bar and other special melting bases it commands a higher price, and can stand more cost in its production than when it is used in competition with steel scrap and pig iron for the manufacture of common steels.

For use as a material that is to be melted in a steel-making furnace sponge iron must be high in metallic iron and low in those impurities that must be fluxed off, since it is even less economical to melt materials high in gangue in a steel-melting furnace than it is in the blast furnace. It is, therefore, fair to assume that sponge iron for conversion into steel should contain at least 90 per cent iron and not over 10 per cent gangue, although it is probable that an iron content as low as 80 per cent might be acceptable provided the remainder is more or less a self-fluxing combination.

In view of these considerations, it seems clear that in order for an ore to be suitable for manufacture of sponge iron for subsequent conversion into steel, it must be of such a mineralogical nature that its gangue may be liberated by grinding to a reasonable size and removed by concentration; that is, the ore must respond to a high degree to beneficiation methods, either before or after the low-temperature reduction process, and its value as a raw material for the manufacture of sponge iron depends to a large extent upon the degree to which it responds to concentration.

In the case of magnetic iron ores, concentration may be carried out either before or after the metallizing operation, depending on the physical characteristics of the ores and the type of equipment used for reduction. In the case of hematites and hematite-magnetite mixtures, gravity concentration is not feasible, particularly where fine grinding is necessary and magnetic concentration is impossible. The non-magnetic hematite of such ores must be converted to a magnetite by a reducing roast or the iron content completely reduced to sponge iron before magnetic concentration methods can be applied.

The magnetic concentration of finely divided sponge iron is not simple. In the dry separation, the operation is accompanied by considerable dusting and the separation of a comparatively large amount of magnetics from a small amount of non-magnetic gangue is hindered by mechanical entanglement. Incidentally, it has been found that finely divided sponge iron is much more difficult to briquet than coarser material. Wet magnetic separation of sponge iron is said to be in use in an experimental way in Sweden, but while it is quite probable that in the continuous operation of a commercial plant this procedure could be carried out without oxidizing the iron, oxidation is difficult to avoid in small-scale work.

Whether the magnetic or other method of concentration is carried out before or after the metallizing operation, it is clear that the degree to which the ore responds to concentration largely determines its values for the manufacture of sponge iron. This being the case it is equally clear that a study of the ore from the beneficiation standpoint might well precede the study of its response to low-temperature reduction. In the case of magnetic ores such studies are readily carried out on the ores in their natural state, while with non-magnetic and mixed ores, by subjecting them to a magnetic roast and thus converting them to an artificial magnetite, the same methods may be adopted.

In view of these considerations it has been decided that before beginning any comprehensive work on the production of sponge iron, a detailed study will be made of Canadian ores and the conditions under which they may be beneficiated. The preliminary tests will be made with the Davis magnetic tube, a laboratory machine used in making magnetic assays of ore and determining the amount of grinding for the good separation of iron mineral from gangue. As has already been stated, hematites and other non-magnetic ores will first be converted into magnetics by a reducing roast before making these tests.

The results obtained from these laboratory tests with the Davis magnetic tube should serve to indicate the degree of fineness to which a given ore must be ground, before or after metallization, in order to produce a magnetically concentrated sponge iron of a desired grade. They should also serve to indicate the best possible grade of concentrated sponge iron procurable from a given ore. From these two important pieces of information, it should be possible to form a definite conclusion regarding the suitability of a particular ore for the manufacture of sponge iron.

Reports on such tests on two ores used in Canada will be found in this report. Neither of these ores is strictly a Canadian ore, but since Wabana ore is and will continue to be the basis of the Nova Scotia steel industry, and Bell ore is one of the typical Lake Superior ores used extensively by the Algoma Steel Corporation, a study of them was deemed profitable.

## Report No. 381

## THE MUSSO PROCESS FOR THE SEMI-DIRECT PRODUCTION OF STEEL FROM IRON ORE

**T. W. Hardy**

*Object of Investigation.* The object of this investigation was to assist a group of interested Canadian financiers in arriving at a clear idea of the technical merit and economic possibilities of the Musso process for the semi-direct production of steel from iron ore.

The work was a co-operative effort on the part of the financial group on one hand and the Department of Mines on the other. Under the terms of the agreement entered into by the two parties, the Department provided laboratory space and facilities, technical staff, and operating labour, and also prepared the ores, coals, and other raw materials. The financial group provided and installed all special apparatus and machinery, ores, coals, and fluxes. It also provided a qualified engineer and all extra labour beyond that supplied by the Department.

The terms of the agreement provided that the tests were to be carried out under the immediate supervision of the inventor. It was also provided that upon the completion of the tests the special apparatus and equipment installed by the group interested should become the property of the Department.

### *Nature and Scope of the Musso Process*

The Musso process provides for the manufacture of steel ingots from iron ore without the use of the blast furnace. As outlined by the inventor, Alfred Musso, the process consists of four main steps.

1. The production of sponge iron in an externally heated metallic retort from a mixture of finely divided ore and bituminous coal. The maximum temperature within the retort is limited to 950° C so that no fusion takes place. The fuel for heating the retort, consisting chiefly of carbon monoxide, is continuously generated by the action of the carbon on the ore within the retort from which it is continuously evacuated. This gas is burned under surface combustion conditions, in combustion rings that surround the retort.

2. The magnetic separation of the reduced iron from the gangue of the ore.
3. The continuous melting of the concentrated sponge iron in a special melting furnace, in which mechanically adhering particles of gangue, together with other impurities including sulphur and phosphorus are separated from the metal which is then molten and substantially pure iron.
   The fuel for this operation is also the gas generated within and evacuated from the reduction retort.
4. The alloying and otherwise finishing into steel of this molten iron in a steel-making furnace, the molten iron being drawn from the sponge melting furnace at regular intervals for this batch type steel-making operation.

### *Description of the Process*

Iron ore and bituminous coal, finely ground and properly mixed, are introduced into a rotary retort. The retort is externally heated and sealed so that it is substantially air-tight, and may or may not be divided into compartments determined according to certain selected temperatures. If partitions are used to separate the compartments, they will be designed so that the solid materials being treated can pass continuously from one compartment to the other, while the gases generated in each compartment will not be allowed to intermingle, but will be rapidly removed from the compartment in which they are generated by means of pumps. The retort may be operated as a single reducing chamber, in which case the gases generated in the various sections may be withdrawn through one or more pipes by means of exhaust pumps.

The rapid removal of the gases and the consequent prevention of the building up of pressure in the retort are claimed by the inventor as features that greatly accelerate the rate of reduction of the iron ore.

The removed gases constitute the fuel which is used to heat the retort and to supply all the energy necessary for the conversion of the sponge iron into steel.

The retort is heated externally by burning the gases evacuated from it, after proper purification, in combustion rings which surround the retort. These rings are made of silicon carbide, a porous refractory material, and the gas is introduced into them mixed with the theoretical proportion of air necessary for complete combustion. This mixture reaches the combustion rings under pressure and is burned according to the well-known method of catalytic combustion. In this way the combustion is flameless, the refractory material composing the combustion rings is brought up to incandescence, and heat is radiated to the wall of the rotating retort and thence by conduction and radiation to the charge.

The rotation of the retort mixes the particles comprising the charge in a continuous fashion, and thus exposes new charge surfaces to the direct action of the heat; and in this way the transfer of heat to the body of the charge is rapid. This is one of the reasons for the adoption of a rotary retort.

As the raw materials pass from the charging to the discharging end of the retort they are brought up to the reduction temperature and maintained at that temperature until reduction has taken place to the desired extent. It is not economically or even technically desirable to carry reduction to completion in the retort as it is more satisfactory to complete the reduction in the melting operation by means of the residual carbon in the sponge since the excess carbon from the coal is never completely removed in the magnetic concentration. For this reason a metallization of 90 per cent is considered quite satisfactory.

From the retort the crude sponge iron passes continuously to a cooler-conveyer through a sealed pipe that excludes the air. From the cooler, the crude sponge iron passes to a storage bin from which it is fed to a magnetic separator.

This magnetic separator is intended to separate the reduced iron from the excess carbon coal ash and such gangue material as may be mechanically free. From the magnetic concentrator the sponge iron passes to a storage bin.

For the conversion of the concentrated sponge iron into steel, the process provides for a two-stage melting operation. In the first furnace the sponge is to be melted and the residual gangue and other impurities fluxed off. From this first furnace the molten metal, assumed to be substantially pure iron, is to be drawn off at intervals and poured into a second furnace in which it is converted into steel by suitable additions.

As to the type of furnace to be used for these melting operations the inventor is not very definite. He provides that they may be electric furnaces, more or less of the standard type, or they may be furnaces heated by the surface combustion of the gas produced as a by-product from the reduction retort. He favours the use of furnaces heated by the catalytic combustion of gas, since the cost of the plant would be lower than if electric furnaces were used and since it is his belief that there is an ample supply of a suitable gas produced in the reduction retort.

### *Principal Features of Construction of Pilot Plant*

The pilot plant was designed on the basis of a productive capacity of 4 tons of metallic iron per day. The retort, which is 24 inches in inside diameter by 12 feet long, is made of Rezistal No. 4 plate, welded. The interior fittings of the retort, the head of the discharge end and the supporting rolls are also built of heat-resisting metal of the nickel-chromium-iron type. The retort is enclosed in a stationary brick furnace structure and the ends of the retort register with two end plates which seal the space between the retort and the brick structure, so that no air from the outside can penetrate and convert the more or less neutral atmosphere therein into an oxidizing one which would lower the life of the retort.

The stationary heads carry sliding rings which have wearing surfaces in contact with flanges on the ends of the rotating retort. The sliding rings are held against the flanges of the retort by coiled springs located outside the stationary head in a cool atmosphere. Graphite lubricant is forced against the wearing surface of the sliding ring through hollow rods which support the springs. The wearing surface of the sliding ring is water-cooled.

Due to the continuous generation of gases in the retort the pressure of the atmosphere tends to build up, but exhaust pumps serve the double purpose of keeping this pressure down to about atmospheric, and making the gases available for combustion around the outside of the retort and elsewhere.

The combustion rings are distributed along the retort body and concentric with it so as to ensure a certain temperature gradient of from 200°C. at the charging end to a maximum of 950°C. at the other. In the pilot plant, the temperature and the pressure used in the retort were regulated manually, but there should be no difficulty in making these controls automatic.

For the purpose of cooling and purifying the gases exhausted from the retort, certain apparatus was installed between the retort and the gas holder in three parallel circuits. The gas exhausted from the intermediate and final zones of the retort passed in separate circuits successively through (1) a heat exchanger, (2) a water-tube condenser (24 inches by 8 feet), and (3) an exhaust pump. The circuit through which the gases from the first zone of the retort were drawn, was similar to the others except that the heat exchanger was omitted. From the pumps, the gases from the three zones were mixed and passed through the tar washer and thence to a gas holder.

For use as a fuel in the combustion rings the gas is drawn from the gas holder by a gas compressor which boosts the pressure up to 10 pounds per square inch. At the burners the gas is mixed with the proper amount of air also at 10 pounds pressure. These burners are provided for each combustion ring, one being situated at the bottom and the others at the sides of the retort.

For melting and steel making, the inventor provided two special crucibles for this pilot plant. These were shaped like inverted cones and were 20 inches in diameter at the top and 25 inches deep. These crucibles were of silicon carbide lined with chromite. It was originally intended that these should be of zirconia, or at least lined with zirconia, but some trouble was encountered in the fabrication of this material and the idea was temporarily abandoned. The crucibles were housed in a brick furnace, and, like the rotary retort, were designed to be fired by surface combustion of the retort gas.

### *Trial Runs and Difficulties Encountered*

Actual experimental work with the pilot plant was begun on June 26, 1930, and during the period ending August 29, 1930, six attempts were made to operate the plant long enough to get some data on the metallurgical characteristics of the process, without success.

The tests to date have demonstrated that the plant, as originally designed, is inoperative and that the mechanical facilities provided by the inventor are inadequate to permit even a technical demonstration of the soundness of his theories.

The mechanical difficulties encountered may be classified under three heads:

1. Difficulties in cleaning and purifying the gas.
2. Difficulties in getting the material to move positively and smoothly through the retort.
3. Difficulty in getting the material to discharge freely from the retort into the cooler.

It should be stated that the difficulties grouped under (2) and (3), namely, those concerned with the movement of the material through and out of the retort were anticipated by many who had had experience in handling hot materials. That the metallurgical advantages to be gained, if any, by dividing the retort longitudinally into three zones, mechanically partitioned from one another would be insignificant compared with the mechanical difficulties in the way of obtaining a free flow of material from one compartment to the other, was predicted, and the tests have demonstrated the soundness of the principle that such a retort should be of simple construction, free from internal fittings and complications.

It should also be pointed out, however, that the mechanical complexities that were incorporated into the design of the retort for the pilot plant are not vital to the process and could readily be eliminated with advantage. There appears to be no reason why a retort could not be designed that would permit a free passage of the material through and out of the retort.

While the difficulties encountered in the handling and cleaning of the gas might have been anticipated, they apparently were not. Two hours after the start of the first trial, trouble was encountered in the form of clogging of the suction pumps with dust and tar. However, after beginning the second trial, three tanks containing water, through which the gas was made to bubble, were installed between the pumps and the condensers and these were effective in protecting the pumps.

That the gas handling and cleaning apparatus provided was totally unsuitable, was abundantly clear by the time the third attempt to operate had been made. It was found that the pipes leading from the retort were almost filled with dust and that the heat exchangers were closed tight with dust and tar, making it impossible to pull any gas out of the retort and this in turn making it impossible to operate the system.

It was decided to discard the heat exchangers and condensers and to install a simple spray washer close to the retort for the purpose of washing the tar and dust from the gas as soon as possible so as to prevent their deposition in the pipes and pumps. By this time the partitions had been removed from the retort which was now being operated as a single reducing chamber. Before beginning the fourth trial, changes were also made in the piping for removing the gas from the retort, the small diameter pipes originally installed being replaced by one 6-inch pipe. This 6-inch pipe served to convey the gases from the retort to the spray washer. In order to prevent the collection of dust in this pipe it was kept as short as possible and means were provided for cleaning it continuously.

These changes were decidedly helpful and throughout the three trials that followed, no serious trouble was experienced in handling the gas that was generated. In each of these three latter tests, however, serious trouble was encountered due to the failure of the discharging mechanism to function. This failure to discharge naturally caused the material to build up in the retort and ultimately led to stalling. Such minor changes in the discharging mechanism as could be made were carried out without materially improving the situation and as it had become obvious that substantial changes in the design of the retort would have to be made before it would function, the tests were discontinued, pending a decision from those interested as to their intentions.

Following the cessation of the reduction tests, experiments were commenced with the crucible furnaces. As had been expected, it was found impossible to attain steel-melting temperatures in these crucibles, even when empty.

### *General Considerations*

Since it purports to be a steel process and not merely a sponge iron process, the Musso process must be regarded in that light. It is, therefore, convenient in discussing the advantages and disadvantages of the process to consider first the sponge-making phase and then the steel-making phase.

While the trial runs with the pilot plant have demonstrated that the plant, as originally designed, is a mechanical failure, they, together with the work independently carried out in these laboratories on the production of sponge iron from ore-coal mixtures (see report No. 384), have indicated that sponge iron can be produced by the Musso method, provided the proper mechanical facilities can be worked out.

That the grade of sponge iron so produced may not be at all suitable for conversion into steel does not appear to have been appreciated by the inventor, who apparently assumed that the bulk of the impurities present in the crude sponge iron would be rejected in magnetic concentration and that any not so rejected would be readily fluxed off in the sponge-melting furnaces.

The two most serious difficulties in the way of making sponge iron by the Musso process suitable for conversion into steel are: (1) the fact that the separation of gangue and other impurities in magnetic concentration to an extent sufficient to permit of the production of a concentrate containing 90 per cent iron, is possible only with certain ores; (2) the fact that a large proportion of the sulphur (and often phosphorus) contained in an ore-coal mixture is found in the magnetically concentrated sponge iron, and that the removal of this high sulphur content from the sponge iron in melting is technically difficult and economically out of the question. The Musso process, as outlined by the inventor assumes that the sulphur contained in the concentrated sponge iron, together with any phosphorus and gangue present, can be fluxed off in the first or sponge melting operation. The feasibility of this is extremely doubtful, and, in any case no data have been presented to show that it has been or can be done.

Quite apart from the Musso demonstration, the possibility that it might be feasible to combine the sulphur content of the ore-coal mixture with lime, in the retort during the reducing operation, has engaged some attention, and the results obtained to date have been most encouraging. The indications are that by charging lime with the ore-coal mixture into the retort and carrying out the reduction under suitable conditions, the sulphur present is largely, if not entirely, fixed as calcium sulphide, which being non-magnetic should be rejected in the magnetic separation.

The extent to which this calcium sulphide and other finely divided non-magnetic materials are rejected in magnetic concentration depends upon the efficiency of the separator and it is clear that there is a demand for a more efficient separator for the dry concentration of sponge iron. Recent work done in these laboratories indicates that by charging lime with the ore and coal, practically all the sulphur may be converted into calcium sulphide, since by wet magnetic concentration, a concentrate containing but 0·04 to 0·10 per cent of sulphur is obtained. Dry separation of this same material, however, yields a concentrate containing 0·15 to 0·20 per cent sulphur, the difference between the results obtained by the two methods being of course attributable to the relative inefficiency of the dry separator. While the wet concentration of sponge iron may be feasible under certain conditions, the liability to oxidation and the necessity of drying before the sponge iron can be melted render such an operation less desirable than dry separation.

### *Conclusions*

The tests carried out in these laboratories have failed to demonstrate that the Musso process, in its present state of development has any technical or commercial value. This failure is attributable to the inadequacy of the mechanical means provided by the inventor to carry out his ideas and does not necessarily mean that the ideas themselves are unsound.

As a result of our experience with this pilot plant and after a careful study of the data available, we are of the opinion that the steel-making features of the Musso process are weak and impracticable, and that the process must, therefore, be regarded simply as a sponge iron process.

As an undeveloped sponge iron process it appears to possess possibilities. There is reason for believing that it is feasible to design and construct a metallic retort that would function satisfactorily in the production of sponge iron from an ore-coal mixture, using Musso's principle of heating the retort externally with the gases evacuated from it. The suitability of this sponge iron for the manufacture of steel depends upon the characteristics of the ore from which it was produced and also upon the perfection of a method for eliminating the sulphur and gangue contents of the sponge iron before melting.

## Report No. 382

## LABORATORY CONCENTRATION OF WABANA IRON ORE

**T. W. Hardy and H. H. Bleakney**

*Object of Investigation.* To determine the degree to which Wabana iron ore may be beneficiated, with particular reference to its adaptability to the manufacture of sponge iron.

### *General Considerations*

An ore to be suitable for the manufacture of sponge iron for subsequent conversion into steel, must be of such a mineralogical nature that the iron mineral may be liberated from the gangue material by grinding to a reasonable size and separated from it by concentration; in other words the ore must respond to a high degree to beneficiation methods, either before or after the low-temperature reduction process, and its value as a raw material for the manufacture of sponge iron depends to a large extent upon the degree to which it responds to concentration.

Wabana ore is mined in Newfoundland and is therefore not a Canadian ore. The Wabana deposits, however, are controlled by the British Empire Steel Corporation and form the basis upon which the steel industry of Nova Scotia has been built. A considerable tonnage of the ore is also shipped abroad.

With the rapid depletion of the better grades of Lake Superior ores, the vast reserves of Wabana ore in Newfoundland assume a position of increasing importance. In a recent paper entitled "Some Aspects of the Iron Ore Situation," published in the September 1930 issue of "Mining & Metallurgy," F. B. Richards points out the importance of the Wabana deposits as a reserve, but qualifies his remarks by stating "Furthermore, the Wabana ore, which is mined in Newfoundland, is not of the best quality. It does not lend itself to the very excellent practice we get from the use of Lake Superior ore."

There is no doubt that Wabana ore would be metallurgically more desirable if its phosphorus and silica contents were materially lower, and while under present conditions, beneficiation of this ore may be economically out of the question, such a procedure, if it can be carried out to any considerable degree, may be found desirable in the future when beneficiation will be more generally applied because of the exhaustion of high-grade ores.

That Wabana ore is of such a mineralogical nature that its beneficiation to any appreciable extent will be difficult, is indicated by a study of Memoir 78 of the Geological Survey of Canada, "Wabana Iron Ore of Newfoundland" by A. O. Hayes. However, it was deemed advisable to carry out a series of tests on the concentration of the Wabana ore with the primary object of determining the best grade of sponge iron capable of being produced from this material and at the same time to determine whether it is possible to effect a removal of phosphorus from this ore to a value comparable to the phosphorus content of the average Lake Superior ore. For this work it was decided to follow the general method adopted by Lee, Gandrud, and De Vaney in their work on the magnetic concentration of iron ores of Alabama and described by them in U.S. Bureau of Mines Bulletin 278. This method, in brief, involves first the conversion of hematite into magnetite by a reducing roast, followed by crushing to various degrees of fineness and the magnetic concentration of these various sized samples in the Davis magnetic tube.

### *Experimental Method*

For these tests a representative sample of 50 pounds of Wabana ore, crushed to pass a 20 mesh Tyler screen, was roasted to the magnetic state in an electrically heated rotary retort which is described in detail on page 201 of this report.

As the sole object was to convert the hematite into magnetite, no attempt was made to study the details of the reducing roast or to find out how this might be done most efficiently. The procedure adopted was as follows: Fifty pounds of 20 mesh ore was charged into the retort which was then brought up to a temperature of 1,000° F. (538° C). Twenty minutes after the retort had reached the temperature, a circulation of city gas through the retort was begun. Two hours later, the power was cut off and the charge was allowed to cool down to 200° F. (93° C) in the furnace, the circulation of gas through the retort being continued during the cooling period in order to exclude atmospheric oxygen which would re-oxidize the $Fe_3O_4$ to $Fe_2O_3$.

The magnetic ore was then carefully sampled to provide a head sample, and then small quantities were ground to pass 20, 80, 100, 150, 200, and 250 mesh. Each of these six samples was then concentrated in the Davis magnetic tube and the products therefrom were weighed and analysed chemically.

The Davis magnetic tube is described in several publications of the U.S. Bureau of Mines, including Bulletin 278, "Magnetic Concentration of Iron Ore of Alabama" from which the following quotation is made:

> . . . . . . . the Davis Magnetic Tube, a laboratory testing machine used in making magnetic assays of the ore and in determining the amount of grinding necessary for good separation of iron mineral from gangue; it consists of a strong C-shaped electro-magnet between the poles of which is suspended a mechanically operated glass tube. The tube moves up and down between the poles of the magnet and at the same time rotates through a small angle. In making tube tests a known weight of ore is placed in the tube which is full of water. The top of the tube is closed and water is allowed to flow into it near the top and out through the bottom. The tube is then placed in an inclined position and set in motion and the agitation thus produced causes the ore to move downward. The magnetic particles are held over the points of the magnetic while the non-magnetic material is washed out by the water. After about 15 minutes the water below the magnet becomes clear. The magnetic concentrate is then removed, dried and weighed. The tailing which is collected at the discharge end is also dried and weighed.

In the tests on Wabana ore, 45 grammes of each of the six sizes was concentrated in each test.

| — | Size of ore fed to magnetic tube (mesh) | | | | | |
|---|---|---|---|---|---|---|
| | −20 | −80 | −100 | −150 | −200 | −250 |
| *Tube Feed—* | | | | | | |
| Iron..................per cent | 51·70 | 51·70 | 51·70 | 51·70 | 51·70 | 51·70 |
| Insoluble............. “ | 14·30 | 14·30 | 14·30 | 14·30 | 14·30 | 14·30 |
| Phosphorus........... “ | 1·00 | 1·00 | 1·00 | 1·00 | 1·00 | 1·00 |
| *Tube Concentrate—* | | | | | | |
| Weight...............per cent | 88·9 | 85·6 | 82·0 | 80·0 | 79·3 | 78·9 |
| Iron.................. “ | 55·10 | 56·60 | 58·10 | 58·70 | 59·50 | 59·70 |
| Insoluble............. “ | 12·0 | 10·6 | 9·0 | 8·1 | 7·5 | 7·2 |
| Phosphorus........... “ | 0·81 | 0·73 | 0·65 | 0·60 | 0·54 | 0·53 |
| *Tube Tailing—* | | | | | | |
| Iron..................per cent | 24·50 | 22·20 | 22·30 | 23·10 | 21·80 | 21·30 |
| Insoluble............. “ | 33·0 | 36·2 | 38·60 | 39·0 | 39·8 | 40·5 |
| Phosphorus........... “ | 2·30 | 2·45 | 2·50 | 2·50 | 2·65 | 2·70 |
| *Ratio of Concentration*............ | 1·12 : 1 | 1·17 : 1 | 1·22 : 1 | 1·25 : 1 | 1·26 : 1 | 1·27 : 1 |
| *Iron recovered*............per cent | 94·7 | 93·7 | 92·2 | 90·8 | 91·3 | 91·1 |
| *Insoluble Rejected*........ “ | 25·4 | 36·6 | 48·4 | 54·7 | 58·4 | 60·3 |
| *Phosphorus Rejected*...... “ | 28·0 | 37·5 | 46·7 | 52·0 | 57·2 | 58·2 |

### CONCLUSIONS

A study of the figures given in the tabulation of results obtained in these concentration tests, makes it abundantly clear that the gangue material present in Wabana ore is so intimately associated with the iron that its liberation to any considerable extent is not effected even by grinding to −200 mesh.

The results show that even after grinding to −200 mesh, about 41·6 per cent of the insoluble matter and about 42·8 per cent of the phosphorus contained in the ore are still locked up with the iron and are retained in concentration. It is, therefore, quite apparent that the economy effected in blast furnace and open-hearth practice by the use of ore beneficiated to the degree indicated would not warrant the expense involved.

It is equally clear that the degree of beneficiation obtainable with Wabana ore is too low to make this material suitable for the manufacture of sponge iron. If it is assumed that the iron content of the −200 mesh concentrate exists as $Fe_3O_4$, it may be calculated that if these concentrates were reduced to sponge iron, 100 per cent metallized, the sponge iron would contain but 77·4 per cent iron and would contain 0·70 per cent phosphorus. Actually the content of metallic iron in the sponge iron would be less than 77 per cent, probably not over 70 per cent as the metallization obtained in commercial practice rarely exceeds 90 to 95 per cent; moreover, in commercial operation the efficiency of concentration would not be so high as that obtainable in these laboratory tests.

To summarize: The results obtained in these tests indicate that even with fine grinding Wabana ore will not produce a concentrate high enough in iron and low enough in phosphorus to make a sponge iron suitable for conversion into steel.

## Report No. 383

### THE LABORATORY CONCENTRATION OF BELL IRON ORE

**T. W. Hardy and H. H. Bleakney**

It was desired to determine the degree to which Bell iron ore may be beneficiated, with particular reference to its adaptability to the manufacture of sponge iron.

### *General Considerations*

It has become clear that, to be suitable for the manufacture of sponge iron for subsequent conversion into steel, an ore must be of such a mineralogical nature that the iron mineral may be separated from the gangue minerals by grinding to a reasonable size and separated from it by concentration; in other words the ore must respond to a high degree to beneficiation methods either before or after the low-temperature reduction process, and its value as a raw material for the manufacture of sponge iron depends to a large extent upon the degree to which it responds to concentration.

Bell ore is not a Canadian ore, but is mined in the Mesabi range of the Lake Superior district in Minnesota. It is, however, one of the ores used by the Algoma Steel Corporation, and also it was the ore chosen for the demonstration of the Musso process for the semi-direct production of steel from iron ore. Therefore, it seemed desirable to determine under ideal laboratory conditions just how completely the iron mineral might be separated from the gangue, thus making it possible to pre-determine the grade of sponge iron that might be hoped for in practice.

## *Experimental Method*

For this work it was most convenient to use the method described in detail in Report No. 382, pages 195 to 198. This method, in brief, involves first the conversion of the hematite into an artificial magnetite by a reducing roast, followed by crushing the magnetite to various degrees of fineness and the wet magnetic concentration of these different sized samples in the Davis magnetic tube.

For these tests a representative sample of Bell ore, crushed to pass a 20 mesh Tyler screen, was roasted to the magnetic state in an electrically heated rotating retort, the retort being held at a temperature of 1,000° F. (538° C.), while a circulation of city gas through the retort was maintained during the heating and cooling of the ore. No attempt was made to study the details of the reducing roast; the sole object being to obtain the ore in the form of magnetite so that by means of magnetic separation the effect of fineness of grinding on the concentration of the ore might be studied.

The artificial magnetite was then carefully sampled to provide a head sample, and then small quantities were ground to pass 20, 80, 100, 150, and 200 mesh Tyler screens. Each of the five samples were then concentrated in a Davis magnetic tube. In each case the magnetic concentrate and the non-magnetic tailing were weighed and assayed.

| | Size of ore fed to magnetic tube (mesh) | | | | |
|---|---|---|---|---|---|
| | −20 | −80 | −100 | −150 | −200 |
| *Tube Feed—* | | | | | |
| Iron.............................per cent | 57·90 | 57·90 | 57·90 | 57·90 | 57·90 |
| Insoluble......................... " | 14·30 | 14·30 | 14·30 | 14·30 | 14·30 |
| Phosphorus..................... " | 0·10 | 0·10 | 0·10 | 0·10 | 0·10 |
| *Tube Concentrate—* | | | | | |
| Weight...........................per cent | 88·0 | 87·6 | 86·9 | 86·4 | 86·2 |
| Iron.............................. " | 64·04 | 64·86 | 65·25 | 65·69 | 65·98 |
| Insoluble......................... " | 6·50 | 5·70 | 5·20 | 4·80 | 4·40 |
| Phosphorus..................... " | 0·079 | 0·076 | 0·076 | 0·075 | 0·075 |
| *Tube Tailing—* | | | | | |
| Iron.............................per cent | 11·82 | 8·53 | 8·50 | 9·05 | 7·81 |
| Insoluble......................... " | 71·50 | 74·94 | 75·50 | 74·30 | 77·00 |
| Phosphorus..................... " | 0·25 | 0·25 | 0·26 | 0·26 | 0·25 |
| *Ratio of Concentration*...................... | 1·14 : 1 | 1·14 : 1 | 1·15 : 1 | 1·16 : 1 | 1·16 : 1 |
| *Insoluble Rejected*..................per cent | 60·0 | 65·1 | 68·4 | 71·0 | 73·5 |
| *Phosphorus Rejected*................ " | 30·5 | 33·4 | 34·0 | 35·2 | 35·4 |
| *Calculated iron content of sponge iron theoretically producible from this concentrate**.. | 85·2 | 86·7 | 87·4 | 88·2 | 88·7 |

*Iron assumed to be completely metallized.

### CONCLUSIONS

The results tabulated show that Bell ore responds to concentration to a considerable degree; 60 per cent of the insoluble being rejected in the −20 mesh material and 77 per cent in the −200 mesh.

The degree of beneficiation obtained with −20 mesh material makes it theoretically possible to produce a sponge iron containing 85 per cent iron while the corresponding figure for the −200 mesh material is 88·7 per cent. Since reduction in practice would not be 100 per cent, as assumed for these calculated values, and as the results of magnetic concentration in commercial practice would probably not be so good as those obtained in the laboratory experiments, it is probable that in commercial practice Bell ore would not produce a concentrated sponge iron containing more than 80 to 85 per cent iron.

## Report No. 384

## THE RATE OF REDUCTION TO SPONGE IRON OF A TYPICAL LAKE SUPERIOR ORE UNDER CERTAIN CONDITIONS IN AN EXTERNALLY HEATED ROTATING RETORT

**T. W. Hardy and W. S. Jenkins**

*Object of Investigation.* In the Musso process for the semi-direct production of steel from iron ore, the production of sponge iron from a mixture of iron ore and bituminous coal in an externally heated rotating metallic retort is the first and most important step. Since the commercial success of such a process depends to a large extent upon the productive capacity of the retort, it seems advisable to determine the rate of reduction when conditions are kept as closely as possible to those proposed by the inventor of the Musso process.

*General Considerations.* Among the more important factors governing the rate of reduction of iron ore particles are: (1) size of particle, (2) physical structure of the oxide, (3) temperature of reduction, (4) type and amount of reducing agent, and (5) heat input. In addition to these, it is the claim of the inventor of the Musso process that the rate of reduction is materially accelerated by exhausting the reaction gases from the retort as fast as they are formed, and by maintaining a pressure somewhat below atmospheric within the retort. The type of ore, reducing agent, particle size, and reduction temperature have been specified by the inventor and are easily duplicated. The rate and quantity of heat input for the proposed pilot plant, is, however, not definitely known, and in any case, cannot readily be duplicated in a laboratory test. The evacuation of the reaction gases from the retort was also found difficult to carry out, on account of the clogging of the pump with dust and tar.

It was, however, thought that the determination of the percentage metallization of a definite weight of the ore-coal mixture heated to a definite temperature in an externally heated retort for various periods of time under otherwise comparable conditions would be of value in enabling us to form some idea of the rate of reduction and that by keeping such variables as the proportions of ore and coal in the mixture, the particle size, and the reduction temperature in agreement with the values specified by Musso, it should be possible to form at least an approximate idea of the time required for reduction in the Musso pilot plant.

### *Experimental Method*

In these experiments, reductions were carried out in an externally heated, rotating retort which forms the heating chamber of a 60 kw. electric furnace. The alloy retort, which has an inside diameter of 14¾ inches and an effective loading length of 48 inches, extends 9 inches out of the charging end of the furnace and is supported on rollers provided for this purpose on the outside of the furnace shell. At the opposite end of the furnace, an alloy sleeve or hub extension of the alloy retort extends about one foot beyond the furnace shell and provides means of rotating the retort and for supporting this end of the retort outside of the furnace on two bearings of the roller type. The retort is equipped at the charging end with a cast plug, heavily insulated, and provision is made for tight closing by means of clamps. An alloy exhaust tube is provided in this plug to relieve any pressure caused by gases generated within the retort.

The furnace is equipped with nickel-chromium heating elements located on the side walls on the furnace roof and also under the retort. The thermocouple, which in conjunction with a Leeds and Northrup indicating controller serves to automatically control the temperature of the furnace, enters through the roof, and is so placed that the hot junction is almost touching the rotating retort. The furnace is supported on trunnions so that it may be tilted forward or backward as desired.

In each of the experiments herein recorded, the following procedure was observed. The furnace was first brought to a temperature of 1,700°F. A charge consisting of an intimate mixture of 73 pounds of Bell ore and 27 pounds of Fairmont coal, both ground to 20 mesh, was quickly introduced into the retort. This charge was kept in the retort, which was rotating at a rate of 1 r.p.m., for a definite period of time. The weight charged, 100 pounds, was chosen because it gave a maximum thickness of bed of 4 inches, which thickness is the same proportion to the retort diameter, as the proposed bed thickness in the Musso retort is to its diameter. After being kept in the rotating retort for the predetermined period of time, the sponge was discharged into a container provided with a self-sealing cover. When cold, the sponge iron was carefully sampled and analysed for total iron and metallic iron. The percentage metallization was calculated from these two results. It should be noted that these analyses are of the crude sponge before magnetic concentration.

The introduction of the cold charge into the hot retort always caused a drop in temperature of about 20°F. The indicating temperature controller, however, showed that the outside of the retort reached the operating temperature again in about 15 minutes. In the results given herein, the time given for each run includes this fifteen minutes during which the retort was below the operating temperature of 1,700°F.

*Analyses of Materials Charged*

| Bell iron ore<br>(Mesabi Range, Lake Superior)<br>dried at 212°F. | per cent |
|---|---|
| Iron | 56·50 |
| Silica | 11·90 |
| Phosphorus | 0·062 |
| Sulphur | 0·04 |
| Alumina | 0·69 |
| Lime | 0·28 |
| Magnesia | 0·21 |
| Manganese | 0·51 |
| Loss on ignition | 5·47 |

| Fairmont coal<br>(Fairmont, Pa.)<br>as charged | per cent |
|---|---|
| Moisture | 2·00 |
| Volatile matter | 35·20 |
| Fixed carbon | 54·60 |
| Ash | 8·20 |
| Sulphur | 1·40 |

*Metallization Obtained in Various Periods of Time*

| Time | Weight charged | Depth of bed | Temperature | Analyses | | Metallization |
|---|---|---|---|---|---|---|
| | | | | Total iron | Metallic iron | |
| hr. min | lb. | in. | °F. | per cent | per cent | per cent |
| 0 50 | 100 | 4 | 1,700 | 51·17 | 8·00 | 15·6 |
| 1 5 | 100 | 4 | 1,700 | 52·64 | 13·78 | 26·2 |
| 1 45 | 100 | 4 | 1,700 | 57·56 | 32·10 | 55·8 |
| 2 15 | 100 | 4 | 1,700 | 57·59 | 45·06 | 78·2 |
| 2 45 | 100 | 4 | 1,700 | 60·65 | 58·12 | 95·8 |
| 3 15 | 100 | 4 | 1,700 | 63·60 | 61·70 | 97·0 |
| 4 15 | 100 | 4 | 1,700 | 64·86 | 62·12 | 95·7 |

### CONCLUSIONS

The tabulated results of the tests show that in order to obtain a metallization of 90 per cent or better, under the conditions obtained in these experiments a total time of about 2¾ hours is required.

If conditions in this batch-type laboratory retort were strictly analogous to conditions in the Musso retort, we would be justified in stating that an ore particle would require 2½ to 2¾ hours in the Musso retort for a 90 per cent reduction to metallic iron. As has been pointed out, however, conditions with respect to rate and quantity of heat input are probably not analogous and this together with the absence in these tests of the evacuation of the reaction gases, which operation is claimed by the inventor to speed up the rate of reduction materially, makes it unwise to draw this conclusion.

The tests are of value however in that they indicate that the time for heating and reduction to a 90 per cent metallized sponge in the Musso retort need not exceed 2¾ hours.

## Report No. 385

### SINTERING AND SPONGE IRON TESTS ON PYRITE RESIDUES

**T. W. Hardy and W. S. Jenkins**

*Shipment.* A shipment of 500 pounds of pyrite residues was received from the Sulphide Research Corporation, c/o Canada Power and Paper Corporation, St. Maurice Division, Cap de la Madeleine, P.Q.

*Nature of Sample and Object of Tests.* This material is the residue or by-product resulting from the Freeman process for the utilization of pyrites in pulp and acid manufacture. This process is fully described by Mr. Horace Freeman in a paper entitled "The Utilization of Pyrites in Pulp and Acid Manufacture" which was read at the Annual Meeting of the Canadian Institute of Mining and Metallurgy in March 1930. Briefly, the Freeman process is designed to displace the use of imported elemental sulphur by native pyrite concentrates in the manufacture of pulp and acid. It involves the combustion of the sulphur content of the finely divided concentrates, under controlled conditions, to sulphur dioxide, with the accompanying formation of oxide of iron as a by-product. It is the commercial utilization of this iron oxide residue that is the problem involved in the tests herein recorded.

*Screen Analysis.* As received the shipment consisted largely of very finely divided material, with a few clinkers as large as 1½ inches in diameter. Screening through a 10 mesh screen gave the following results.

| | | |
|---|---|---|
| Through 10 mesh | 467 pounds or | 93·4 per cent |
| 10 mesh | 33 " | 6·6 " |

The 33 pounds of oversize material were crushed to pass a 10 mesh screen and then thoroughly mixed with the bulk of the shipment. A screen analysis of the shipment as a whole at this stage gave the following results,—

| | | | | | |
|---|---|---|---|---|---|
| On 20 mesh | | 1·35 per cent | −65 | +100 | 5·70 per cent |
| −20 | +28 | 0·75 " | 100 | 150 | 15·55 " |
| 28 | 35 | 0·75 " | 150 | 200 | 26·30 " |
| 35 | 48 | 0·95 " | −200 | | 45·80 " |
| 48 | 65 | 2·85 " | | | |

*Chemical Analysis of Shipment.* The shipment was carefully sampled and a head sample obtained which on chemical analysis gave the following results,—

| | |
|---|---|
| Iron (Fe) | 63·00 per cent |
| Sulphur (S) | 0·65 " |
| Insoluble | 6·92 " |

## *Sintering Tests*

With the object of agglomerating and lowering the sulphur content of the finely divided, high-sulphur material and thus making it suitable for blast furnace use, a number of sintering tests were made.

The tests were carried out in a down-draught sintering pan of the laboratory type, the size of the pan grate being 12 by 18 inches.

The results of a number of typical sintering tests, in which the proportions of fuel, moisture and other ingredients of the charge were varied, are shown in the accompanying table.

## Sinter Tests

| Mix No. | Charge—pounds | | | | | Thickness of bed, in. | Time, mins. | Weight sinter, lb. | Weight, fines, lb. | Analyses, sinter | | Character of sinter |
|---|---|---|---|---|---|---|---|---|---|---|---|---|
| | Pyrite residues | Coke breeze | Soft coal | Water | Returned fine sinter | | | | | Fe, per cent | S, per cent | |
| P 8 | 40·0 | 2·4 | 2·4 | 4·5 | Nil | 4·5 | 40·0 | 20·0 | 19·0 | 64·86 | 0·25 | Well fused, weak, brittle. |
| P 9 | 34·0 | 2·0 | 2·0 | 4·4 | 6·0 | 4·5 | 37·0 | 23·0 | 16·0 | 64·02 | 0·50 | Well fused, stronger than P 8. |
| P 10* | 34·0 | 2·0 | 2·0 | 4·4 | 6·0 | 4·5 | 40·0 | 22·0 | 16·0 | 60·65 | 0·14 | Well fused, similar to P 9. |
| P 11 | 30·0 | 2·1 | 0·9 | 3·9 | 6·0 | 3·5 | 35·0 | 21·5 | 11·0 | 64·02 | 0·18 | Well fused, stronger than P 10. |
| P 12 | 30·0 | 2·1 | 0·9 | 4·9 | 6·0 | 3·5 | 30·0 | 22·5 | 10·0 | 64·02 | 0·11 | Well fused, fairly strong. |
| P 13 | 30·0 | 0·9 | 0·6 | 5·6 | 6·0 | 3·0 | 30·0 | 26·5 | 6·0 | 63·82 | 0·14 | Well fused, fairly strong. |

*Charge also contained 1·2 pounds sand and 1·2 pounds lime.

In each test, the general procedure adopted was as follows. The various ingredients making up the charge to be sintered were thoroughly mixed and distributed evenly over the grate through a $\frac{1}{4}$-inch screen. The depth of the bed varied from 3 to $4\frac{1}{2}$ inches from charge to charge. To facilitate ignition, a layer of charcoal was spread over the surface of the bed and the whole ignited by means of a movable gas burner, air being drawn down through the charge by a suction pump throughout the entire operation. The suction in the line connecting the pan with the pump was usually about 12 inches of water at the start of the run, and gradually diminished as the charge sintered and became porous. When the sintering action appeared completed, the charge was dumped and the fines separated from the sinter by means of a 10 mesh screen. The analyses shown in the table are those of the sinter, the fines being discarded. These discarded fines always contained a large proportion of small particles of sinter.

The results of the tests indicate that a product suitable for blast furnace use can readily be obtained by sintering the pyrite residues. The sinter obtained while satisfactory is rather weak and brittle, due probably to the finely divided nature of the residues and their comparative freedom from gangue, and it seems probable that a stronger sinter could be obtained by mixing the residues with fine iron ore. The sulphur content of the residue can be reduced to 0·15 per cent by sintering, making it acceptable to the blast furnace.

### *Sponge Iron Tests*

The high iron content of the sintered material and its rather fragile nature suggested that it might be more desirable for the manufacture of sponge iron than for use in the blast furnace. It was, therefore, decided to reduce the sintered residues to sponge iron in an electrically heated, batch type, rotary retort, using city gas as the reducing agent; and to determine the suitability of this sponge iron for the production of steel in electric or open-hearth furnaces.

The results of a test in which some of this sinter was metallized are summarized herein. Reduction was effected by passing a stream of city gas through the retort during the heating operation.

These tests, which should be confirmed on a larger scale, indicate that by sintering and metallizing these residues, using a process such as the Wiberg or the Norsk Staal for converting the sinter into sponge iron, a product is obtained suitable for conversion into steel in electric or open-hearth furnaces. The iron content of the concentrated sponge iron should preferably be higher, and further tests are desirable to determine whether or not the above results can be improved upon.

## CONCLUSIONS

The tests indicate that this material may be made chemically and mechanically suitable for blast furnace use by sintering.

They also indicate that it may readily be converted into a comparatively low-sulphur sponge iron suitable for direct conversion into steel. Further experimentation on the conversion of these residues into sponge iron is desirable.

### Sponge Iron Tests*

*Results of Metallization Test*

| Materials, pounds | | | Analyses, per cent | | | Contents, pounds | | |
|---|---|---|---|---|---|---|---|---|
| — | Weight | Per cent | Total iron | Met. iron | Sulphur | Total iron | Met. iron | Sulphur |
| Sintered pyrite residue | 41·5 | 83·9 | 66·96 | Nil | 0·11 | 27·79 | Nil | 0·046 |
| Lime | 8·0 | 16·1 | ......... | ......... | ......... | ......... | ......... | ......... |
| Sponge iron (crude) | 41·8 | 84·4 | 67·18 | 62·85 | 0·12 | 28·08 | 26·27 | 0·050 |

Metallization, 93·5 per cent.

---

*City gas used as reducing agent.

*Results of Magnetic Concentration Test*

| Materials, pounds | | | Analyses, per cent | | | Contents, pounds | | |
|---|---|---|---|---|---|---|---|---|
| — | Weight | Per cent | Total iron | Met. iron | Sulphur | Total iron | Met. iron | Sulphur |
| Sponge iron (crude) | 38·6 | 100·0 | 67·18 | 62·85 | 0·12 | 25·98 | 24·26 | 0·046 |
| Magnetic concentrate | 32·2 | 84·4 | 80·24 | 76·20 | 0·06 | 25·84 | 24·54 | 0·019 |
| Tailing | 6·4 | 15·6 | 5·93 | 0·75 | 0·50 | 0·38 | 0·05 | 0·032 |

## Report No. 386

## SINTERING TESTS ON SIDERITE ORE FROM THE HELEN MINE, MICHIPICOTEN, ONTARIO

**T. W. Hardy and W. S. Jenkins**

*Shipment.* A shipment of 1,000 pounds of siderite ore from the Helen mine was received from the Algoma Steel Corporation, Sault Ste. Marie, Ontario.

*Nature of Sample and Object of Tests.* The material supplied consisted of the raw ore, as mined, without any preliminary treatment. Sintering tests were made in order to learn the quality of sinter obtainable, the extent to which the sulphur could be eliminated, and the best practice to obtain the desired results.

*Chemical Analysis of Shipment.* The shipment was crushed to pass a 3 mesh screen and carefully sampled. The head sample obtained gave the following results on chemical analysis:

| | | |
|---|---|---|
| Iron (Fe) | 36·8 | per cent |
| Sulphur (S) | 1·34 | " |

## *Sintering Tests.*

Various proportions of ore, coke, and water were used and various suctions and bed depths were tried with a view to obtaining a combination which would give the best result as regards both the quality of the sintered product and the degree of desulphurization. Tests were conducted on material crushed to pass a 3 mesh screen and on the same material crushed to pass a 10 mesh screen in order to obtain some information on the effect of particle size.

The tests were carried out on a 12 by 18-inch laboratory type, down-draught sintering pan. Bed depths varied from 3 to 4·3 inches.

Table IA shows the results obtained on sintering raw ore crushed to pass a 3 mesh screen. Excellent sinters were obtained, as evidenced by the low percentages of fines, but the outstanding characteristic of the material was its great fusibility. In all tests there was a strong tendency for the sinter to stick to the grates and in the early tests even to plug the holes in the grates. To minimize that trouble a layer of returned fine sinter was laid on the grates before the charge, which helped materially. Possibly due to the highly fusible nature of the ore, desulphurization was not so complete as had been hoped for. About 0·40 per cent appeared to be the lowest figure consistently obtainable. In these and following tests, it was found necessary to decrease the suction during the run from an initial value of about 20 inches of water to a final figure of about 5 inches owing to the excessive heat generated causing the sides of the pan to become red hot. It was also found that the thicker bed depths, i. e. over 4 inches, gave better results than the thinner ones. A charge containing as low as 3 per cent of coke was found to yield very good results, but about 4 per cent gave the greatest amount of sinter and the least fines.

Table IB shows the results obtained on material crushed to pass a 10 mesh screen. The product obtained on this material was also very good, and there was less tendency for the sinter to stick to the grates. However, more coke was required in order to avoid excessive quantities of fines, and no improvement over the 3 mesh material was obtained in desulphurization.

Tables IIB and C show the results obtained on pre-calcined ore crushed to pass 3 and 10 mesh screens respectively. These results are comparable with those obtained on the raw ore and indicate that no significant improvement is obtained by pre-calcining the material.

## SUMMARY

The following are the significant features revealed by the above investigation.

(1) The siderite ore which was subjected to these tests possesses excellent sintering qualities but owing to its very fusible nature precautions must be observed to prevent the sinter from sticking to the pan and clogging the grates.

(2) Desulphurization is very incomplete, about 0·40 per cent being the lowest value practically obtainable in these tests.

(3) Pre-calcining the ore does not assist in sulphur reduction nor improve the quality of the sinter.

## TABLE I

## Sinter Tests

### *A. Sinter Tests on Raw Ore Crushed to 3 Mesh*

| Mix No. | Charge—pounds | | | | Thickness of bed, in. | Time, mins. | Weight, of sinter in pounds | Weight of fines in pounds | Analysis, sinter | | Character of sinter |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Ore | Coke breeze | Water | Returned fine sinter | | | | | Fe, per cent | S, per cent | |
| 1 | 40 | 3·2 | 4·3 | 3 | 3·0 | 35 | 13·5 | 15·5 | 54·33 | 0·43 | All sinters hard and strong. Material is very fusible and tends to clinker and stick to grates. |
| 2 | 50 | 3·0 | 2·7 | 3 | 4·0 | 35 | 25·5 | 9·5 | 54·12 | 0·67 | |
| 3 | 60 | 2·4 | 3·1 | 11 | 4·3 | 35 | 40·0 | 10·0 | 53·91 | 0·40 | |
| 7 | 52 | 1·8 | 3·1 | 15 | 4·3 | 40 | 34·5 | 14·0 | 53·91 | 0·42 | |
| 8 | 42 | 1·8 | 3·1 | 25 | 4·3 | 50 | 36·5 | 15·5 | 53·99 | 0·40 | |

### *B. Sinter Tests on Raw Ore Crushed to 10 Mesh*

| | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 4 | 42 | 2 | 2·6 | 15 | 4·0 | 35 | 21·5 | 20·5 | 52·86 | 0·29 | All sinters hard and strong. Tendency to stick to grates not so great as with 3 mesh |
| 5 | 50 | 4 | 2·7 | 12 | 4·0 | 35 | 35·0 | 9·5 | 53·49 | 0·57 | |
| 6 | 40 | 3·2 | 2·2 | 8 | 3·5 | 25 | 23·5 | 10·0 | 52·86 | 0·46 | |

## TABLE II

### A. *Calcination Test on Raw Ore Crushed to 3 Mesh*

| Charge | | | Product | | |
|---|---|---|---|---|---|
| Weight | Iron, per cent | Sulphur, per cent | Weight | Iron, per cent | Sulphur, per cent |
| 124·0 | 36·86 | 1·34 | 81·5 | 54·12 | 1·88 |
| 115·0 | 36·86 | 1·34 | 75·5 | 54·12 | 1·92 |

### B. *Sinter Tests on Calcined 3 Mesh Siderite*

| Mix No. | Charge—pounds | | | | Thickness of bed, in. | Time, mins. | Weight, of sinter in pounds | Weight, of fines in pounds | Analysis, sinter | | Character of sinter |
|---|---|---|---|---|---|---|---|---|---|---|---|
| | Cal. | Coke breeze | Water | Returned fine sinter | | | | | Fe, per cent | S, per cent | |
| 9 | 50 | 2·0 | 5·2 | .......... | 4·0 | 30 | 33 | 16·0 | 54·12 | 0·29 | Good hard sinter. |
| 10 | 50 | 3·0 | 5·3 | .......... | 4·0 | 30 | 38 | 11·5 | 54·12 | 0·47 | |
| 12 | 65 | 2·6 | 6·8 | 3 | 4·25 | 40 | 49 | 19·0 | .......... | 0·35 | Sticks to grate. |

### C. *Sinter Tests on Calcined 10 Mesh Siderite*

| | | | | | | | | | | | |
|---|---|---|---|---|---|---|---|---|---|---|---|
| 11 | 43 | 3·4 | 3·7 | 3·5 | 4·0 | 38 | 30 | 15·5 | 53·32 | 0·54 | Good hard sinter. |

(4) Material crushed to pass a 10 mesh screen has no advantage over material crushed to pass a 3 mesh screen.

## Report No. 388

### THE CHEMICAL AND METALLURGICAL CHARACTERISTICS OF THREE SAMPLES OF CHILLED SHOT

**T. W. Hardy**

*Shipment.* Three samples of chilled shot marked No. 1, No. 2, and No. 3, respectively, were received in July 1930 from Mr. L. H. Cole of the Mineral Resources Division of the Mines Branch.

*Origin of Samples and Object of Tests.* The samples were accompanied by a memorandum from Mr. Cole in which he stated that in connexion with his investigation of the building stone industry of Canada, it had come to his attention that there is a considerable tonnage of chilled shot used in the gangsaws for cutting granite and that at the present time all this material is being imported. His object in submitting the samples was to determine the nature of the metal of which these shot are made with a view to finding out if this material could be manufactured in Canada.

*Chemical Composition.* The chemical composition of the three samples was found to be as follows:—

| Sample No. | 1 | 2 | 3 |
|---|---|---|---|
| Sample marked origin | "Globe Shot" United States | Airgrie, Glasgow, Scotland | Aberdeen, Scotland |
| | per cent | per cent | per cent |
| Total carbon | 3·35 | 3·15 | 3·26 |
| Manganese | 0·45 | 0·52 | 0·49 |
| Phosphorus | 0·47 | 1·08 | 1·04 |
| Sulphur | 0·110 | 0·110 | 0·156 |
| Silicon | 1·56 | 2·20 | 1·86 |
| Nickel | Nil | Nil | Nil |
| Chromium | Nil | Trace | 0·17 |

It will be seen from the above analyses that these three samples of shot are of cast iron and that there is no difference of any importance, as far as composition is concerned between the three samples. The two Scottish samples are higher in phosphorus than the other sample, but all three are of the general character ordinarily found in cast iron castings. It is important to note, however, that carbon content in these shot is largely if not entirely present in the combined form, due to the drastic quenching involved in the shotting of the molten iron.

*Summary.* It has been found by chemical analysis, supplemented by metallographic examination, that these three samples are of chilled cast iron, and that there is no noteworthy difference between them as far as chemical composition is concerned.

There does not appear to be any reason why these shot should not be made in Canada by any iron foundry, provided the necessary mechanical arrangements for shotting the metal and grading the shot are available.

## V

# REPORT OF THE CHEMICAL LABORATORY SECTION

*Chief Chemist*

**H. C. Mabee**

During the year, 2,884 samples of ores and products from the Ore Dressing and Metallurgical Laboratories were received, analysed, and reported on. Twenty-four samples of mineralized rock were received for identification and valuation, making a total of 2,908 samples which passed through the chemical laboratories. Approximately 10,000 determinations were made on these samples, representing an increase over the previous year of 34 per cent and almost double each of the three preceding years.

In addition to the regular routine work on samples received from the other sections of the division, the services of the chemical staff and the facilities of the laboratories were again extended to the Base Metals Extraction Company in connexion with their experimental work on the treatment of Canadian base metal ores; also to the Cassels Cyanide Company, and the American Cyanamid Company of Niagara Falls, Ontario, in determining to what extent their various flotation reagents could be applied to the successful treatment and recovery of metals from Canadian complex ores.

The services of R. A. Rogers, of the chemical staff, was placed at the disposal of the non-metallic minerals section for the investigation of gypsum and anhydrite from Canadian occurrences. The anhydrite investigation was still in progress at the close of the year.

The analytical work was performed on a wide variety of ores, ore dressing and metallurgical products. The chemical laboratories are well equipped for the analysis of inorganic substances and inorganic chemical research.

The creation of the section of ferrous metallurgy increased the work considerably. Two junior chemists were added to the staff and additional equipment was purchased. During the summer months the services of two temporary student assistants were used, one in the assay laboratory and the other in the main chemical laboratory.

In the following table is given a list of the ores, concentration products, metals and furnace products upon which reports of analyses were issued during the year.

| | Samples |
|---|---|
| Antimony ore | 5 |
| Clays and shales | 13 |
| Cobalt | 80 |
| Copper | 20 |
| Dolomite | 1 |
| Copper-gold | 6 |
| Gold-silver | 84 |
| Gold ores | 1,321 |

| | Samples |
|---|---|
| Gypsum | 29 |
| Identification and valuation | 24 |
| Iron and metallurgical products | 544 |
| Lead-silver-barite | 47 |
| Lead-zinc-silver | 272 |
| Magnesite | 10 |
| Manganese | 8 |
| Mercury ore | 9 |
| Molybdenite | 29 |
| Nickel-copper ore | 301 |
| Platinum | 1 |
| Rock (feldspar) | 5 |
| Silica sand | 85 |
| Uraninite | 14 |

# INDEX

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