CANADA

DEPARTMENT OF MINES AND RESOURCES

HON. T. A. CRERAR, MINISTER: CHARLES CAMSELL, DEPUTY MINISTER

MINES AND GEOLOGY BRANCH

JOHN McLEISH, DIRECTOR

BUREAU OF MINES

W. B. TIMM, CHIEF

# INVESTIGATIONS IN ORE DRESSING AND METALLURGY

*(Testing and Research Laboratories)*

**January to June, 1937**

OTTAWA
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# INVESTIGATIONS IN ORE DRESSING AND METALLURGY, JANUARY TO JUNE, 1937

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## I

## REVIEW OF INVESTIGATIONS

**C. S. Parsons**
*Chief of Division of Metallic Minerals*

During the half-yearly period ending June 30, 1937, fifty-one investigations were completed and reports thereon were furnished to the parties submitting the ores or other materials for examination and test. In addition, many tests of lesser importance were made. Of the major investigations, the thirteen reports included in Section II were printed as separates. Thirty-eight reports of less general interest are synopsized in Section III.

The investigations were carried out under the direction of C. S. Parsons, Chief of Division. The work on metallic ores was performed by A. K. Anderson, Senior Engineer, W. R. McClelland, J. D. Johnston, H. L. Beer, and W. S. Jenkins. The microscopic and spectrographic work in connexion with the investigations was done by M. H. Haycock. The chemical research was conducted under the direction of R. J. Traill, Senior Engineer, who was assisted by B. P. Coyne. The metallurgical work on iron and steel and non-ferrous alloys, including the mechanical and physical testing, was performed by G. S. Farnham. The chemical work in connexion with all investigations was prepared by H. C. Mabee, Chief Chemist, R. A. Rogers, A. Sadler, R. W. Cornish, J. S. McCree, S. R. M. Badger, A. E. LaRochelle, J. A. Rivington, and the assaying by L. Lutes and C. A. Derry.

Gold ores continued to hold first place and of the fifty-one investigations completed, thirty-two were on gold. Three important investigations were conducted on base metal ores, one on the concentration of a copper-zinc-gold and silver-bearing ore from the Abana mine of the Normetals Corporation, and one on the concentration of the copper-pyrite ore from the Aldermac mine, both mines being in northwestern Quebec. The third was on the possibility of concentrating the chrome ore from the Obonga Lake deposit of the Chromium Mining and Smelting Corporation. An investigation on graphite ore from the Black Donald Graphite mine, Calabogie, Ontario, had as objective the improvement of the recovery and the grade of the graphite products. Tests were also made on one molybdenite ore, one copper-lead ore, two silver-lead ores, two silver ores, and one copper-gold ore. The results of these investigations were used in the designs for ten new milling plants which have been erected or are under construction.

Eighteen investigations of problems in milling and treatment in operating mills were carried out and the results were acknowledged by the respective managements as of great assistance in reducing the operating costs and the metallurgical losses.

The notable contribution made by the microscopic study of the ores and mill products in the solving of ore-dressing problems continues to be demonstrated. The usefulness of this branch of our work has been greatly enhanced by the addition to the mineragraphic laboratory of a new precision microscope and the building of a photographic apparatus for use in conjunction with it.

In the mineragraphic laboratory 665 polished sections were prepared and 27 thin sections were made. From the microscopic examination of these polished sections 39 investigations were completed, of which 28 were of gold ores, 5 of base metal ores, and 6 of miscellaneous ores. Ten special studies, and thirty-nine spectrographic analyses were made.

In the research chemical laboratories three studies were carried out: one on the refractory ores of the Bridge River District, B.C.; one on the plant residues from the Radium Refinery at Port Hope; and one on a gold ore from the property of the Cochenour Willans, Limited. The work on the Minto mine ore from Bridge River district was completed and a summary of all the work done is contained in this report. The laboratory work on certain roasting and calcining problems in connexion with the residue from the Port Hope refinery was also completed. In addition, many studies were made for cyanide solutions and other problems in connexion with ore testing investigations.

Seven samples of rocks and plant products were tested for radioactivity. These comprised plant residues from the Radium Refinery, Port Hope, Ontario; rock samples from South Porcupine, Ontario; three samples of ore from the Venus-Juno Group, Nelson, B.C.; one rock sample from Noranda, Quebec; two rock samples from Yellowknife, Great Slave Lake, N.W.T.; and ilmenite sand, Travancore, India.

The latest design of infrasizer has been added to the research equipment. This apparatus and the super-panner, both of which are the invention of Professor Haultain of Toronto University, form undoubtedly the most useful combination of tools yet developed as aids to the engineer working on ore-treatment problems. As adjuncts to the microscope, these appliances enable information of the greatest practical value to be obtained in research work. They are also to be recommended for use at the mine where they should undoubtedly help to increase profits.

In connexion with the metallurgical investigation of iron and steel and non-ferrous alloys, so far as time permitted work has been done on the effect of various alloying elements on the impact strength of cast iron.

The needs of other departments of the Government occupied about 70 per cent of the time. Data were collected on the iron and steel industry, advice was given on physical metallurgical problems and much test work was undertaken.

Most of the work done for the industry was investigational, rather than routine testing, many of the investigations being for the smaller steel firms.

The following test work was undertaken:

*Heat Treatment:* Fifty-six articles were heat treated for other departments of the Government.

*Melting Tests:* Three melting tests were conducted, two of these being for the industry, about 4½ tons of metal in all being melted.

*Physical Testing:* Ten hardness, thirteen tensile, and three impact tests were made for industrial firms.

*Metallographic Examinations:* Fifteen steels were examined for Government departments and fifteen for industrial firms.

In the chemical laboratories, 5,196 determinations were made, divided as follows:

| | |
|---|---|
| Metallic ore mill products | 2,126 |
| Field samples | 61 |
| Industrial Minerals Division mill products | 113 |
| Iron and steel, etc | 74 |
| Custom assays | 218 |
| Total samples | 2,592 |
| Total determinations | 5,196 |
| Total gold assays | 1,740 |
| Total silver assays | 538 |

# II

# INVESTIGATIONS THE RESULTS OF WHICH ARE RECORDED IN DETAIL

## Ore Dressing and Metallurgical Investigation No. 704

### GOLD ORE FROM THE PARKHILL GOLD MINES, LIMITED, MICHIPICOTEN AREA, ONTARIO

*Shipment.* A shipment of ore, net weight 220 pounds, was received on November 27, 1936. The ore was from the property of the Parkhill Gold Mines, Limited, located in the Michipicoten area of the Sault Ste. Marie mining division, Ontario. The sample was submitted for test work by R. D. Caylor, Mine Manager, Parkhill Gold Mines, Limited, Gold Park P.O., via Wawa, Ontario.

*Purpose of the Tests.* The shipment consisted of ore containing an excess of pyrrhotite. A method was required for overcoming difficulties that have developed in the mill since beginning treatment on this type of ore.

*Characteristics of the Ore.* Six polished sections were prepared and examined microscopically for determining the character of the ore.

The *gangue* is fine-textured, greenish grey to dark grey, highly silicified chloritic schist and light grey vein quartz. Stringers of carbonate are commonly large enough to be seen easily with the naked eye.

The *metallic minerals* present are: pyrite, pyrrhotite, chalcopyrite, arsenopyrite, and native gold. Pyrite is finely divided and is disseminated along bands in the more schistose portions of the ore; it is only sparingly distributed in the quartz. Pyrrhotite occurs as small, irregular grains disseminated in both quartz and schist; the quantity is not great but is probably enough to interfere with cyanidation. A small amount of chalcopyrite occurs as irregular grains usually associated with pyrrhotite. Rare grains of arsenopyrite are present; in a previous shipment from this company arsenopyrite was the predominant mineral, and this appears to be the great difference between that shipment and the present one.

Only five grains of native gold are visible in the sections. All are somewhat rounded and equi-dimensional, and occur freely in the quartz. They vary in size from a little below 200 mesh to less than 800 mesh. No native gold was seen in the sulphides.

*Sampling and Analysis.* The shipment was crushed and sampled by standard methods and a representative portion was found to contain the following values:

| | | |
|---|---|---|
| Gold | 0·36 | oz./ton |
| Silver | 0·10 | " |
| Copper | 0·04 | per cent |
| Iron | 12·54 | " |
| Pyrrhotite | 1·27 | " |
| Sulphur | 1·03 | " |

## Experimental Tests

The experimental tests consisted of the following:

1. Grinding tests to determine the length of time to reduce the ore to the required size in the experimental apparatus.
2. Cycle Test No. 1, using dry-crushed −150-mesh ore.
3. Cycle Test No. 2, using dry-crushed −150-mesh ore with the addition of a lead salt.
4. Blanket concentration on wet-ground −150-mesh ore followed by cyanidation of duplicate samples with and without the addition of a lead salt.
5. Cyanidation on ore ground in water, filtered, and repulped in cyanide solution.
6. Cyanidation on ore ground in cyanide solution.
7. Cyanidation on ore ground in cyanide solution with the addition of a lead salt.
8. Cyanidation of ore ground in water, aerated 16 hours, filtered, and repulped in cyanide solution.
9. Cyanidation of ore ground in water, filtered, repulped, aerated 16 hours, and cyanided 24 hours.
10. Cyanidation of ore ground in water, filtered, repulped in cyanide solution with and without the addition of a lead salt.

In order to duplicate the grinding practised at the mill the ore had to be ground so that all passed 150 mesh, and at least 92 per cent passed 200 mesh. It was, therefore, necessary to run a series of grinding tests to determine the time required to approximate this grind.

The whole shipment of ore was first crushed to pass a 14-mesh screen. Representative portions were used for the individual tests. The −14-mesh ore was ground in laboratory ball mills in a pulp made up of four parts of ore to three parts of water with a constant weight of ball charge for different lengths of time. The period required to give the following screen test was noted. This grind was used in all subsequent tests:

*Screen Test:*

| Mesh | Weight, per cent |
|---|---|
| −100+150 | 0·3 |
| −150+200 | 4·4 |
| −200 | 95·3 |
| Total | 100·0 |

### CYCLE TESTS

#### *Test No. 2*

A representative portion of −14-mesh ore was crushed dry in a pulverizer to pass a 150-mesh screen. The crushed ore was divided into two lots to be used in two cycle tests, No. 1 and No. 2.

Representative portions of 150-mesh ore were used in seven cycles of 24-hour periods of agitation. The dilution ratio was one part ore to 1·5 parts of solution (1·5:1). The solution was made up to 1·5 pounds of potassium cyanide per ton at the beginning of each cycle. The protective alkali was lime.

At the end of each cycle the cyanide solution was filtered off, the gold was precipitated by zinc dust, and the solution was made up to strength (1·5 pounds of potassium cyanide per ton) and added to the succeeding charge of ore.

The solution from the final cycle was used to determine its reducing power, presence of KCNS and copper.

*Summary:*

| Cycle No. | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| 1 | 0·36 | 0·045 | 87·50 | 1·35 | 9·25 |
| 2 | " | 0·06 | 83·33 | 1·13 | 10·25 |
| 3 | " | 0·075 | 79·17 | 1·20 | 9·90 |
| 4 | " | 0·055 | 84·72 | 1·25 | 10·00 |
| 5 | " | 0·085 | 76·39 | 0·90 | 9·95 |
| 6 | " | 0·045 | 87·50 | 1·05 | 9·95 |
| 7 | " | 0·07 | 80·56 | 0·98 | 9·95 |

Reducing power of final solution: 983 c.c. of $\frac{N}{10}$ $KMnO_4$/litre

KCNS—1·17 grm./litre
Copper—0·208 "

The test shows appreciable fouling and a fairly high tailing.

*Test No. 3*

To determine the effect of adding a lead salt to the cyanide solution, a duplicate test of Cycle Test No. 1 was made.

The ore used was one-half of the representative portion for the first cycle test.

After precipitating the gold from the solution by means of zinc dust, the solution was made up to strength of 1·5 pounds of potassium cyanide per ton, and lead nitrate at the rate of 0·5 pound per ton of solution was added. The cycles were made the same as in Cycle Test No. 1, with the addition of lead nitrate to each charge by adding it to the barren solution.

The solution from the final cycle was used to determine reducing power and KCNS present.

*Summary:*

| Cycle No. | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| 1 | 0·36 | 0·02 | 94·44 | 1·20 | 9·70 |
| 2 | " | 0·015 | 95·83 | 1·43 | 9·80 |
| 3 | " | 0·01 | 97·22 | 1·28 | 9·75 |
| 4 | " | 0·01 | 97·22 | 1·02 | 9·75 |
| 5 | " | 0·02 | 94·44 | 1·05 | 9·80 |
| 6 | " | 0·015 | 93·83 | 1·05 | 9·80 |
| 7 | " | 0·015 | 95·83 | 1·02 | 9·80 |

Reducing power of final solution: 684 c.c. of $\frac{N}{10}$ $KMnO_4$/litre

KCNS—0·975 grm./litre.
Copper—not determined.

The addition of the lead salt improves the extraction of gold. It does not reduce the amount of cyanide consumed.

GRINDING IN WATER; BLANKET CONCENTRATION AND CYANIDATION

*Test No. 4*

To determine the effect of cyaniding a pulp from which the coarse gold had been removed, a representative portion of −14-mesh ore was ground in ball mills, dilution 4:3, to give a product −150 mesh and 94 per cent −200 mesh.

The pulp was concentrated on a corduroy blanket sloping 2·5 inches in 12 inches.

The blanket tailing was filtered and four representative portions used for cyanidation.

The blanket tailing samples were agitated in cyanide solution (1·0 pound of potassium cyanide per ton) for 24 hours at a dilution of 1·5:1. The protective alkali was lime. Lead nitrate was added to two charges at the rate of 0·5 pound per ton of solution.

The blanket tailing was assayed and a screen test made. The value of the blanket concentrate was calculated.

*Blanket Concentration:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, in products, per cent |
|---|---|---|---|
| Feed | 100·00 | 0·36 | 100·00 |
| Blanket concentrate | 0·39 | 59·10 | 64·03 |
| Blanket tailing | 99·61 | 0·13 | 35·97 |

*Cyanidation of Blanket Tailing:*

| Test No. | $Pb(NO_3)_2$, lb./ton solution | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|
| | | Feed | Tailing | | KCN | CaO |
| 1 | 0·5 | 0·13 | 0·01 | 92·31 | 1·09 | 9·0 |
| 2 | 0·5 | 0·13 | 0·01 | 92·31 | 1·02 | 9·0 |
| 3 | None | 0·13 | 0·01 | 92·31 | 0·87 | 9·0 |
| 4 | None | 0·13 | 0·01 | 92·31 | 0·95 | 9·0 |

Recovery of gold in concentrate, 64·03 per cent.
Gold left in blanket tailing, 35·97 per cent.
Extraction by cyanidation, 92·31 × 35·97 = 33·20 per cent.

As the concentrate was not treated to recover its gold the overall extraction was not determined in the test.

The addition of the lead salt shows an increase in consumption of cyanide with no increase in extraction of gold.

The test shows that when the ore is ground in water and the coarse gold is removed by concentration, the remainder will readily dissolve in cyanide.

### *Test No. 5*

To determine the best method of treatment of the ore by cyanidation, several tests were made using various methods of preparing the ore for agitation.

A representative portion of −14-mesh ore was ground in a ball mill, dilution 4 : 3, to give a −150-mesh product. The pulp was filtered, washed, and repulped with fresh water to give a dilution of 1·5 : 1, and potassium cyanide, 1·0 pound per ton, was added. The protective alkali was lime. Period of agitation, 24 hours.

*Summary:*

| Assay, Au, oz./ton. | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|
| Feed | Tailing | | KCN | CaO |
| 0·36 | 0·03 | 91·66 | 1·08 | 5·46 |

### *Test No. 6*

To determine the effect of grinding in cyanide solution, a representative portion of −14-mesh ore was ground in a ball mill, dilution 4 : 3. The solution used in the ball mill was equivalent to 1·0 pound of potassium cyanide per ton. The protective alkali was lime. After grinding to −150 mesh, the charge was diluted to 1·5 : 1 and the cyanide brought up to 1·0 pound of potassium cyanide per ton. A protective alkali, lime, was added as required. Period of agitation, 24 hours.

*Summary:*

| Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|
| Feed | Tailing | | KCN | CaO |
| 0·36 | 0·065 | 81·94 | 1·40 | 9·34 |

A comparison of the results obtained in Tests Nos. 5 and 6 indicates clearly that fouling is taking place and that a much more rapid solution of the gold takes place if the ore is ground in water and filtered to get rid of soluble salts before the cyanide solution is added.

### *Test No. 7*

This test was a duplicate of Test No. 6 with the addition of lead oxide (PbO) to the mill at the rate of 1·0 pound per ton of ore. Agitation, 24 hours.

*Summary:*

| Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|
| Feed | Tailing | | KCN | CaO |
| 0·36 | 0·015 | 95·83 | 0·78 | 9·0 |

This result is the best extraction obtained from five tests, Nos. 5 to 9 inclusive, and shows the beneficial effect of the addition of the lead salt.

### *Test No. 8*

This was to determine the effect of aeration.

A representative portion of −14-mesh ore was ground in a ball mill, dilution 4:3, to give a −150-mesh product. Lime was added to the mill at the rate of 6·0 pounds per ton of ore. After grinding the pulp was aerated for 16 hours, filtered, washed, and repulped in fresh water. Potassium cyanide, 1·0 pound per ton, was added. The protective alkali was lime. Agitation, 24 hours.

*Summary:*

| Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|
| Feed | Tailing | | KCN | CaO |
| 0·36 | 0·06 | 83·33 | 0·57 | 5·0 in cyanidation; 6·0 in mill. |

### *Test No. 9*

A representative sample of −14-mesh ore was ground in a ball mill, dilution 4:3, to give a product −150 mesh. After grinding, the pulp was filtered, repulped in fresh water, and aerated for 16 hours with lime at the rate of 6·0 pounds per ton of ore. After aeration the pulp was diluted to 1·5:1 and cyanide added to the solution to give 1·0 pound of potassium cyanide per ton of solution. The protective alkali was lime. Agitation period, 24 hours.

*Summary:*

| Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|
| Feed | Tailing | | KCN | CaO |
| 0·36 | 0·055 | 84·72 | 0·51 | 5·3 in cyanidation; 6·0 in aeration. |

*Tests Nos. 10A and 10B*

Duplicate tests were made on ore ground in water, filtered, repulped in cyanide solution with and without the addition of lead nitrate, in order to determine the effect of lead nitrate in the tests.

Two representative portions of −14-mesh ore were ground in ball mills, dilution 4:3, to give a −150-mesh product. The pulp was filtered and washed and repulped in cyanide solution, 1·0 pound of potassium cyanide per ton. The protective alkali was lime. Period of agitation, 24 hours. To one test lead nitrate at the rate of 0·5 pound per ton of solution was added.

*Summary:*

| Test No. | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton ore | | |
|---|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO | Pb $(NO_3)_2$ |
| 10-A | 0·36 | 0·055 | 84·72 | 1·17 | 5·80 | None |
| 10-B | 0·36 | 0·01 | 97·22 | 0·84 | 5·70 | 0·75 |

The use of the lead salt was beneficial in this test. Test No. 10A is a duplicate of Test No. 5 also.

*Summary of Tests Nos. 5 to 10B:*

*Cyanidation:*

| Test No. | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton ore | | |
|---|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO | Lead salts |
| 5 | 0·36 | 0·03 | 91·66 | 1·08 | 5·46 | ........ |
| 6 | " | 0·065 | 81·94 | 1·40 | 9·34 | ........ |
| 7 | " | 0·015 | 95·83 | 0·78 | 9·0 | PbO, 1·0 |
| 8 | " | 0·06 | 83·33 | 0·57 | 11·0 | ........ |
| 9 | " | 0·055 | 84·72 | 0·51 | 11·3 | ........ |
| 10A | " | 0·055 | 84·72 | 1·17 | 5·80 | ........ |
| 10B | " | 0·01 | 97·22 | 0·84 | 5·70 | Pb $(NO_3)_2$, 0·75 |

## SUMMARY AND CONCLUSIONS

Cycle Test No. 1 was made without the use of lead salt. It shows a fairly high tailing and appreciable fouling.

Cycle Test No. 2 was made with the addition of lead nitrate to the barren solution of each cycle and showed a marked improvement in lower tailings and less fouling of solution. The consumption of cyanide remained unchanged in both cycle tests.

In Test No. 4, in which 64 per cent of the total gold was removed by blanket concentration prior to cyanidation, the grinding was done in water; the addition of lead salt showed no improvement.

Cyanidation tests, in which different methods of preparation of the pulp for cyanidation were tried, show the most beneficial results when lead salt is used. The ore may be ground in cyanide solution with the addition of the lead salt to the grinding mill, the pulp diluted and cyanided 24 hours to give a tailing of 0·015 ounce of gold per ton, or the ore may be ground in water, filtered, and repulped in cyanide solution with the addition of lead salt to give a tailing of 0·01 ounce of gold per ton.

The other tests, including water washing without removal of free gold by concentration, aeration, and grinding in cyanide solution without the use of lead salt, gave tailings varying from 0·03 to 0·065 ounce of gold per ton.

The cyanide consumptions in Tests Nos. 5 to 10B should also be noted.

The beneficial effect of the addition of lead salt is demonstrated by the results of the various tests. They not only prevent fouling but give a decided increase in the rate of dissolution of the gold, as indicated in Test No. 10B and others.

Grinding in water and concentration methods such as are at present practised in the mill to remove the bulk of the free gold also overcomes the difficulty due to fouling of the solution by the pyrrhotite, as indicated in Test No. 4, and the addition of lead salt is not required.

## RECOMMENDATIONS

From the results of these tests it is apparent that two methods of procedure may be adopted to overcome the difficulty due to the fouling of the cyanide solution by pyrrhotite at present experienced in the mill.

The first and easiest method is to add a lead salt to the ore. This can be done by adding PbO to the ball mill or by adding lead nitrate to the barren solution going back to the grinding circuit. The use of an excessive amount of lead salt should be avoided, as it accumulates in the circuit and its full effect may not show up for some days after it is first added.

The second method is to change over to grinding in water. This would involve the use of an additional filter. The present mill circuit would all be in water up to discharge of the first thickener. A filter would be used on the thickener product, and the filter cake repulped in cyanide solution. The rest of the process would be straight cyanide practice with the exception that a quantity of barren solution would have to be discharged periodically to equalize the moisture left in the filter cake.

It is recommended that the addition of lead salts be tried, and should they fail it will then be necessary to use the water grind.

## Ore Dressing and Metallurgical Investigation No. 705

GOLD ORE FROM PRESTON EAST DOME MINES, LIMITED, SOUTH PORCUPINE, ONTARIO

*Shipment.* A shipment of five sacks of ore, net weight 302 pounds, was received on January 4, 1937. The sample was submitted by D. G. H. Wright, Consulting Engineer, Room 301, 331 Bay Street, Toronto, Ontario.

The property is located in Tisdale township, Porcupine mining division, Timiskaming county, Ontario.

*Characteristics of the Ore.* The *gangue* is somewhat variable in character. It consists largely of a light greenish grey material resembling a porphyry with grey to milky quartz. A small quantity of carbonate is present.

The ore contains pyrite and a small quantity of limonite. Pyrite is sparingly disseminated as medium to small grains and crystals. Some grains show replacement by limonite. No native gold or other *metallic minerals* could be seen in the sections examined.

*Sampling and Assaying.* The shipment was crushed and a sample cut out for assay. The sample assayed:

| | |
|---|---|
| Gold | 0·195 oz./ton |
| Silver | 0·06 " |
| Iron | 3·26 per cent |
| Sulphur | 0·92 " |

### Experimental Tests

Cyanidation and amalgamation tests were conducted on the ore as well as flotation tests with cyanidation of the concentrates.

An extraction of 95 to 96 per cent of the gold can be made from the ore in 24 hours by straight cyanidation, but as the pulp is rather slow in settling some difficulty may be experienced in the use of this process. Sixty-five per cent of the gold can be extracted by barrel amalgamation, but when the amalgamation tailing is treated by cyanidation the overall extraction is the same as when the ore is treated by cyanidation.

When the ore is ground 85 per cent through 200 mesh, 97·5 per cent of the gold can be recovered in a flotation concentrate with a high ratio of concentration. Ninety-eight per cent of the gold can be extracted from the concentrate by regrinding and cyanidation, but care must be taken in the selection of flotation reagents to prevent serious frothing in the agitators.

The tests are described in detail as follows:

## CYANIDATION

### *Tests Nos. 1 to 8*

Samples of the ore were ground in water, approximately 79·2, 85·9, 88·6, and 92·8 per cent through 200 mesh in ball mills. The pulps were transferred to bottles without filtering, lime and cyanide added, and the pulps agitated for periods of 24 and 48 hours at a dilution ratio of 1·5 : 1. Solutions were kept at the equivalent of 1 pound of potassium cyanide per ton.

*Summary:* Feed: gold, 0·195 oz./ton

| Test No. | Grinding, per cent −200 mesh | Agitation, hours | Tailing assay, Au, oz./ton | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| 1 | 79·2 | 24 | 0·015 | 92·4 | 0·07 | 3·47 |
| 2 | 85·9 | 24 | 0·01 | 94·9 | 0·11 | 3·49 |
| 3 | 88·6 | 24 | 0·01 | 94·9 | 0·11 | 3·55 |
| 4 | 92·8 | 24 | 0·01 | 94·9 | 0·11 | 3·56 |
| 5 | 79·2 | 48 | 0·01 | 94·9 | 0·13 | 3·52 |
| 6 | 85·9 | 48 | 0·01 | 94·9 | 0·16 | 3·58 |
| 7 | 88·6 | 48 | 0·01 | 94·9 | 0·16 | 3·61 |
| 8 | 92·8 | 48 | 0·01 | 94·9 | 0·14 | 3·64 |

### *Tests Nos. 9 to 12*

Samples of ore were ground in cyanide solution approximately 79·2, 85·9, 88·6, and 92·8 per cent through 200 mesh in ball mills. The pulps were transferred to cyanide agitators and agitated for 24 hours at 1·5 : 1 dilution. Solutions were kept at the equivalent of 1·0 pound of potassium cyanide per ton.

*Summary:* Feed: gold, 0·195 oz./ton

| Test No. | Grinding, per cent −200 mesh | Agitation, hours | Tailing assay, Au, oz./ton | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| 9 | 79·2 | 24 | 0·015 | 92·4 | 0·10 | 2·88 |
| 10 | 85·9 | 24 | 0·0075 | 96·2 | 0·16 | 2·95 |
| 11 | 88·6 | 24 | 0·01 | 94·9 | 0·14 | 2·95 |
| 12 | 92·8 | 24 | 0·0075 | 96·2 | 0·21 | 3·08 |

## BARREL AMALGAMATION AND CYANIDATION

### *Tests Nos. 13 and 14*

Samples of the ore were ground in water approximately 79·2 and 88·6 per cent through 200 mesh. The pulps were amalgamated with mercury in jar mills for one hour. The amalgamation tailings were sampled and assayed and portions of each agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 24 hours at 1·5 : 1 dilution. The cyanide tailings were assayed for gold.

*Summary:*

Feed: gold, 0·195 oz./ton

| Test No. | Grinding, per cent −200 mesh | Amalgamation tailing assay, Au, oz./ton | Cyanide tailing assay, Au, oz./ton | Total extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| 13 | 79·2 | 0·095 | 0·015 | 92·4 | 0·16 | 3·46 |
| 14 | 88·6 | 0·07 | 0·01 | 94·9 | 0·10 | 3·61 |

SETTLING TEST

*Test No. 15*

A sample of the ore was ground 80 per cent through 200 mesh in a ball mill and agitated in cyanide solution for 24 hours. The pulp was transferred to a settling tube and allowed to settle for periods of one hour at various dilutions. Pulp level readings were taken at 5-minute intervals. The results of this test may be summarized as follows:

| Time | Height of Pulp Level, in Feet | | |
|---|---|---|---|
| Start | 2·56 | 3·255 | 3·96 |
| 5 minutes | 2·53 | 3·195 | 3·77 |
| 10 " | 2·50 | 3·135 | 3·55 |
| 15 " | 2·47 | 3·08 | 3·335 |
| 20 " | 2·44 | 3·025 | 3·20 |
| 25 " | 2·41 | 2·975 | 3·10 |
| 30 " | 2·38 | 2·93 | 3·015 |
| 35 " | 2·36 | 2·885 | 2·945 |
| 40 " | 2·33 | 2·84 | 2·88 |
| 45 " | 2·305 | 2·795 | 2·82 |
| 50 " | 2·28 | 2·75 | 2·77 |
| 55 " | 2·255 | 2·71 | 2·715 |
| 1 hour | 2·23 | 2·67 | 2·665 |
| Drop in pulp level for one hour | 0·33 | 0·585 | 1·295 |
| Potassium cyanide, lb./ton sol'n | 0·92 | 1·05 | 1·05 |
| Lime, " " | 0·70 | 0·75 | 0·75 |
| Dilution ratio | 1·5 : 1 | 2 : 1 | 2·5 : 1 |

CYCLE CYANIDATION TEST

Five lots of ore were treated successively with one batch of solution to determine whether the solution would become inactive after repeated use owing to fouling properties the ore might have. At the end of the fifth cycle the tailing assays indicated that no fouling of the solution had taken place. A litre of the final solution required 53 c.c. of $\frac{N}{10}$ $KMnO_4$ to destroy its reducing power.

The work indicates that the slow settling rate would rule out cyanidation of the ore as a means of treatment by reason of the amount of settling capacity required to handle it.

## FLOTATION

A series of flotation tests was conducted, first to determine the grinding necessary to produce a low flotation tailing, and second to find a suitable reagent combination to prevent troublesome frothing in the cyanide agitators. A description of these tests follows:

### *Test No. 16*

A sample of the ore was ground 88·6 per cent through 200 mesh in a ball mill with 1·0 pound of soda ash. The pulp was conditioned with 0·10 pound of potassium amyl xanthate and floated with 0·05 pound of pine oil per ton. The products were assayed for gold.

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, per cent |
|---|---|---|---|
| Concentrate | 3·4 | 4·80 | 94·4 |
| Tailing | 96·6 | 0·01 | 5·6 |
| Feed (cal.) | 100·0 | 0·173 | 100·0 |

Ratio of concentration: 29·4 : 1

### *Test No. 17*

A sample of the ore was ground 71 per cent through 200 mesh in a ball mill with 1·0 pound of soda ash and 0·10 pound of potassium amyl xanthate added to the charge. A concentrate was floated off with 0·05 pound of pine oil per ton.

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, per cent |
|---|---|---|---|
| Concentrate | 5·3 | 3·30 | 94·9 |
| Tailing | 94·7 | 0·01 | 5·1 |
| Feed (cal.) | 100·0 | 0·184 | 100·0 |

Ratio of concentration: 18·85 : 1

### *Test No. 18*

This was a duplicate of Test No. 17 except that 0·50 pound per ton of copper sulphate was added to the cell near the end of the frothing period.

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, per cent |
|---|---|---|---|
| Concentrate | 5·7 | 3·12 | 95·2 |
| Tailing | 94·3 | 0·01 | 4·8 |
| Feed (cal.) | 100·0 | 0·187 | 100·0 |

Ratio of concentration: 17·5 : 1

*Test No. 19*

A sample of the ore was ground 79·2 per cent through 200 mesh in a ball mill with 1·0 pound of soda ash and 0·10 pound of potassium amyl xanthate added to the charge. A concentrate was floated off with 0·05 pound of pine oil per ton and 0·50 pound of copper sulphate per ton.

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, per cent |
|---|---|---|---|
| Concentrate | 5·24 | 3·56 | 97·5 |
| Tailing | 94·76 | 0·005 | 2·5 |
| Feed (cal.) | 100·00 | 0·191 | 100·0 |

Ratio of concentration: 19·1 : 1

FLOTATION AND CYANIDATION

*Test No. 20*

A large sample of ore, sufficient to produce about 1,000 grammes of concentrate, was treated as in Test No. 19. The concentrate was filtered, washed, and reground in cyanide solution 98·5 per cent through 325 mesh and agitated in a bottle for 48 hours. Quite a depth of froth was noticed on top of the pulp when the bottle was allowed to settle. At the end of the agitation period the sample was filtered and washed and an assay sample cut from the cake, the remainder of it was repulped in fresh cyanide solution and agitated for 24 hours in a Denver super-agitator. This time no froth formed on the pulp. The first and second samples of cyanide tailing were assayed for gold.

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, per cent | Extraction, per cent total gold | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| Concentrate | 5·5 | 3·04 | 97·25 | | | |
| Tailing | 94·5 | 0·005 | 2·75 | | | |
| Feed (cal.) | 100·0 | 0·172 | 100·00 | | | |
| 1st cyanide tailing | | 0·15 | | | | |
| 2nd cyanide tailing | | 0·04 | | 96·0 | 0·24 | 1·42 |

Ratio of concentration: 18·2 : 1

*Test No. 21*

Another large sample of ore was ground 88·6 per cent through 200 mesh with soda ash and xanthate, and floated under the same conditions as in Test No. 20.

The concentrate was filtered, washed, and reground in cyanide solution nearly all through 325 mesh, and agitated for 48 hours in a Denver super-agitator. It was thought that this type of agitator would handle the froth more efficiently than the bottle. At the end of 48 hours the cyanide tailing was filtered and washed and a sample cut out for assay. The remainder of the cake was repulped in fresh cyanide solution and agitated for another 24 hours. No frothing occurred in the repulped tailing. The first and second samples of cyanide tailing were assayed for gold.

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, per cent | Extraction, per cent total gold | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| Concentrate......... | 5·2 | 3·42 | 97·4 | ............ | ............ | ............ |
| Tailing............. | 94·8 | 0·005 | 2·6 | ............ | ............ | ............ |
| Feed (cal.).......... | 100·0 | 0·183 | 100·0 | ............ | ............ | ............ |
| 1st cyanide tailing... | ............ | 0·48 | ............ | ............ | ............ | ............ |
| 2nd cyanide tailing.. | ............ | 0·08 | ............ | 95·1 | 0·25 | 0·96 |

Ratio of concentration: 19·2 : 1

## *Test No. 22*

The formation of froth in the agitators coupled with the fact that no froth formed in the washed and repulped tailing led to the belief that the pine oil had not been washed out of the concentrate before agitation and that talc or carbonates in the concentrate may have been the cause. It was therefore decided to try and float without copper sulphate so as not to bring up so much talc and carbonates. The quantity of pine oil was increased to 0·10 pound per ton to compensate for the copper sulphate and otherwise the test was conducted the same as Test No. 21, with the ore ground 79·2 per cent through 200 mesh. The concentrate produced filtered much more readily but, although the frothing in the agitators was greatly reduced, there was still enough to be troublesome.

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, per cent | Extraction, per cent total gold | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| Concentrate......... | 4·7 | 3·90 | 97·5 | ............ | ............ | ............ |
| Tailing............. | 95·3 | 0·005 | 2·5 | ............ | ............ | ............ |
| Feed (cal.).......... | 100·0 | 0·188 | 100·0 | ............ | ............ | ............ |
| Cyanide tailing from concentrate........ | ............ | 0·105 | ............ | 94·87 | 0·29 | 0·74 |

Ratio of concentration: 21·3 : 1

## *Test No. 23*

A sample of the ore was ground 79·2 per cent through 200 mesh in a ball mill with 1·0 pound of soda ash and 0·10 pound of potassium amyl xanthate added to the charge. A concentrate was floated off with 0·25 pound of cresylic acid per ton. The concentrate was filtered and washed and then reground 99·5 per cent through 325 mesh in cyanide solution.

The pulp was agitated 48 hours at 2·5:1 dilution in cyanide solution containing the equivalent of 1·0 pound of potassium cyanide per ton. The flotation tailing and the cyanide tailing from the reground concentrate were assayed for gold.

*Summary:* Feed: gold, 0·195 oz./ton

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, per cent | Extraction, per cent total gold | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| Flotation tailing..... | 95·0 | 0·005 | 2·5 | ........... | ........... | ........... |
| Cyanide tailing from concentrates....... | 5·0 | 0·06 | 1·6 | ........... | ........... | ........... |
| Average tailing (cal.) | 100·0 | 0·008 | ........... | 95·9 | 0·24 | 0·81 |

Ratio of concentration: 20 : 1

## FLOTATION

### *Test No. 24*

A sample of the ore was ground 79·2 per cent through 200 mesh in a ball mill with 1·0 pound of soda ash and 0·10 pound of potassium amyl xanthate added to the charge. A concentrate was floated off using Crocetol "X", 0·25 pound per ton, as a frother. This test was solely for finding what tailing could be produced by using this frothing reagent. The tailing produced assayed 0·005 ounce per ton in gold and the ratio of concentration was low at 11·8 : 1.

## CONCLUSIONS

Straight cyanidation of the ore is not to be recommended owing to the slow settling rate.

The gold can be concentrated by flotation with a high ratio of concentration, 20 : 1 or better, and after regrinding the concentrate can be treated by cyanidation giving as good an overall extraction as can be obtained by straight cyanidation of the ore. This process is therefore to be recommended as it avoids the settling problem and reduces the size of cyanide plant required.

The test work has indicated, however, that if pine oil is used as a frother, trouble arises in the form of frothing in the cyanide agitators, resulting in low extraction of gold from the concentrate, and this is aggravated if copper sulphate is added to the reagent combination, as it seems to promote the flotation of talc and carbonates, which in turn absorb the pine oil and increase the frothing in the cyanide agitators.

The frothing in the cyanide agitators disappeared when cresylic acid was used as a frother and good extraction was obtained. (*See* Test No. 23.)

A good flow-sheet would be to grind the ore in a ball mill classifier circuit and to place a unit flotation cell equipped with trap between them. Although the results of the work did not show that this was necessary, it would still be advisable to put the trap there to take care of any coarse gold in the ore that might otherwise escape in the flotation tailing.

The unit-cell tailing would go to the classifier, which would overflow to a battery of flotation machines where the final concentrate would be taken off. The flotation tailing would go to waste as a finished product and the concentrate should be filtered and washed to get rid of flotation reagents and then reground in cyanide solution nearly all through 325 mesh and agitated as long as may be found necessary.

The small-scale work indicated that the ore should be ground about 80 per cent through 200 mesh in order to produce the minimum flotation tailing. It might be possible, however, to reduce this figure somewhat in practice.

Following the above method 95 per cent or more of the total gold in the ore should be recovered as bullion.

## Ore Dressing and Metallurgical Investigation No. 706

GOLD ORE FROM SISCOE GOLD MINES, LIMITED, SISCOE, QUEBEC

*Shipment.* Four boxes of ore, total weight 254 pounds, were received on February 16, 1937, from C. O. Stee, Mine Manager, Siscoe Gold Mines, Limited, Siscoe, Quebec.

Sample A, consisting of two boxes and weighing 96 pounds, was a representative sample from the Main ore zone; and Sample B, weighing 158 pounds, was a representative sample from the "K" ore zone.

The property is situated on Siscoe island, in the Harricanaw area, Quebec, 40 miles south of Amos.

*Sampling and Analysis.* After cutting, crushing, and grinding by standard methods, a sample of each lot of ore was obtained which assayed as follows:

| | Sample A | Sample B |
|---|---|---|
| Gold | 0·375 oz./ton | 0·82 oz./ton |
| Silver | 0·14 " | 0·16 " |
| Acid insoluble | 76·24 per cent | 60·82 per cent |
| Silica | 70·28 " | 48·70 " |
| Lime | 3·55 " | 7·31 " |
| Magnesia | 3·58 " | 15·75 " |
| Alumina | 8·86 " | 8·00 " |
| Iron | 5·70 " | 7·06 " |
| Sulphur | 0·17 " | 0·27 " |
| Tellurium | Nil | Nil |
| Lead | Nil | Nil |
| Graphitic carbon | 0·04 " | 0·03 " |
| Carbon dioxide | 2·40 " | 5·12 " |
| Copper | 0·08 " | 0·08 " |
| Molybdenum sulphide | 0·13 " | 0·05 " |

*Characteristics of the Ore.* Six polished sections taken from Sample B ore, which is said to come from the new "K" ore-body, were prepared and examined microscopically for the purpose of determining the character of the ore.

The *gangue* is a medium- to fine-textured, dark grey siliceous rock which contains a considerable quantity of carbonate, some of which, from its brown colour, appears to be ferruginous. Patches of translucent quartz are present, and there is probably a quantity of talc.

The only common *metallic mineral* is pyrite, occurring as sparingly disseminated medium to fine grains. A small quantity of chalcopyrite and occasional grains of magnetite are disseminated in the gangue. Six grains of native gold were seen, all in gangue and they vary in size from about 1100 mesh to smaller than 2300 mesh. It is probable that they are not representative of the gold in the ore, and that grains of considerably larger size are also present.

Six sections from the old ore, of which Sample A is representative, were polished and examined superficially to compare them with those from the "K" ore-body (Sample B). Both appear to be very similar and the few grains of gold seen in the old ore were in gangue; they were, however, considerably larger in size than those seen in the ore from the "K" ore-body.

## Experimental Tests

The test work was conducted principally on Sample B, and consisted of amalgamation and cyanidation. A series of settling tests was also made on both the A and B samples. Amalgamation and cyanidation on Sample B, from the "K" ore zone, gave an overall recovery of over 99 per cent of the gold.

### AMALGAMATION

*Tests Nos. 1, 2, and 3*

The ore at −14 mesh was ground in a ball mill to pass 62 per cent through 200 mesh in Test No. 1, 72·7 per cent in Test No. 2, and 86·9 per cent in Test No. 3. The pulps were amalgamated with mercury for 1 hour in a jar mill. The amalgamation tailing was assayed for gold.

*Results:*

| Test No. | Grinding, per cent through 200 mesh | Assay, Au, oz./ton | | Recovery, per cent |
|---|---|---|---|---|
| | | Feed | Tailing | |
| 1 | 62·0 | 0·82 | 0·165 | 79·9 |
| 2 | 72·7 | 0·82 | 0·115 | 86·0 |
| 3 | 86·9 | 0·82 | 0·085 | 89·6 |

The above results indicate the amount of free gold in the ore at the different degrees of comminution.

### CYANIDATION

*Tests Nos. 4 to 7*

The ore at −14 mesh was ground in ball mills in cyanide solutions of 1 pound per ton strength. Five pounds of lime per ton of ore was added to maintain protective alkalinity. After grinding, the pulps were agitated in the same solutions for periods of 24 and 48 hours. The cyanide tailings were assayed for gold.

Screen tests showed the grinding as follows:

| Mesh | Weight, per cent | | | |
|---|---|---|---|---|
| | Test No. 4 | Test No. 5 | Test No. 6 | Test No. 7 |
| − 48 + 65 | 0·8 | 0·1 | ........... | ........... |
| − 65 + 100 | 7·1 | 2·9 | 0·9 | 0·2 |
| −100 + 150 | 15·9 | 9·6 | 5·6 | 5·0 |
| −150 + 200 | 13·2 | 14·5 | 12·1 | 7·4 |
| −200 | 61·9 | 72·7 | 80·2 | 86·9 |

*Cyanidation:*

Feed: gold, 0·82 oz./ton

| Test No. | Agitation, hours | Assay of tailing, Au, oz./ton | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | | | | KCN | CaO |
| 4 | 24 | 0·10 | 87·8 | 0·93 | 3·48 |
| 4 | 48 | 0·16 | 80·5 | 0·94 | 3·60 |
| 5 | 24 | 0·20 | 75·6 | 0·85 | 3·40 |
| 5 | 48 | 0·145 | 82·3 | 0·90 | 3·44 |
| 6 | 24 | 0·17 | 79·3 | 0·82 | 3·40 |
| 6 | 48 | 0·285 | 65·2 | 1·02 | 3·60 |
| 7 | 24 | 0·10 | 87·8 | 1·16 | 3·76 |
| 7 | 48 | 0·03 | 96·3 | 1·32 | 3·76 |

The above tests show the futility of straight cyanidation on an ore like this. The percentage of extraction varies according to the size of the individual particles of free gold in the particular sample.

## HYDRAULIC CLASSIFICATION AND BLANKET CONCENTRATION

### *Test No. 8*

The ore at −14 mesh was ground in a ball mill to pass 80·2 per cent through 200 mesh. The pulp was passed through a hydraulic classifier, or trap, and a trap concentrate recovered. The trap tailing was passed over a corduroy blanket, set at a slope of 2·5 inches to the foot, and a blanket concentrate secured. The two concentrates were combined and assayed.

*Results:*

*Hydraulic Classification:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| Feed | 100·0 | 0·82 | 100·0 | |
| Hydraulic concentrate | 0·2 | 195·50* | 47·7 | 500 : 1 |
| Hydraulic tailing | 99·8 | 0·43 | 52·3 | |

*Blanket Concentration:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| Feed | 100·0 | 0·43 | 100·0 | |
| Blanket concentrate | 4·7 | 7·02* | 76·7 | 21·3 :1 |
| Blanket tailing | 95·3 | 0·105 | 23·3 | |

*Calculated.

The assay of the combined trap and blanket concentrates resulted as follows:

| | |
|---|---|
| Gold | 12·24 oz./ton |
| Silver | 1·56 " |
| Iron | 7·20 per cent |
| Sulphur | 1·14 " |
| Lead | Nil |
| Tellurium | Nil |
| Copper | 0·24 per cent |
| Molybdenum sulphide | 0·05 " |
| Lime | 6·60 " |

## CONCENTRATION, AMALGAMATION AND CYANIDATION

### *Tests Nos. 9 and 10*

The ore at −14 mesh was ground in ball mills to pass 76·3 per cent through 200 mesh in Test No. 9 and 90·2 per cent in Test No. 10. The pulps were passed through a hydraulic trap and a trap concentrate recovered. The trap tailing was passed over a corduroy blanket and a blanket concentrate obtained. The combined concentrates were amalgamated with mercury. The amalgam residue was added to the blanket tailing and agitated in cyanide solution of 1 pound per ton strength for 24- and 48-hour periods.

*Results:*

*Hydraulic Classification:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| *Test No. 9* | | | | |
| Feed | 100·00 | 0·82 | 100·0 | |
| Hydraulic concentrate | 0·35 | 114·72* | 49·0 | 285 : 1 |
| Tailing | 99·65 | 0·42 | 51·0 | |
| *Test No. 10* | | | | |
| Feed | 100·00 | 0·82 | 100·0 | |
| Hydraulic concentrate | 0·16 | 281·56* | 54·9 | 625 : 1 |
| Tailing | 99·84 | 0·37 | 45·1 | |

*Blanket Concentration:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| *Test No. 9* | | | | |
| Feed | 100·0 | 0·42 | 100·0 | |
| Blanket concentrate | 2·0 | 14·62* | 69·6 | 50 : 1 |
| Tailing | 98·0 | 0·13 | 30·4 | |
| *Test No. 10* | | | | |
| Feed | 100·0 | 0·37 | 100·0 | |
| Blanket concentrate | 1·2 | 18·50* | 60·0 | 83 : 1 |
| Tailing | 98·8 | 0·150 | 40·0 | |

*Calculated.

The combined trap and blanket concentrates were amalgamated and the amalgam residue added to the blanket tailing. This product assayed 0·17 ounce of gold per ton in Test No. 9 and 0·19 ounce of gold per ton in Test No. 10.

*Cyanidation of Blanket Tailing and Amalgam Residues:*

| Test No. | Agitation, hours | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|
| | | Feed | Tailing | | KCN | CaO |
| 9 | 24 | 0·17 | 0·01 | 94·1 | 0·50 | 3·0 |
| 9 | 48 | 0·17 | 0·005 | 97·1 | 0·90 | 3·8 |
| 10 | 24 | 0·19 | 0·005 | 97·4 | 0·70 | 3·2 |
| 10 | 48 | 0·19 | 0·005 | 97·4 | 1·30 | 4·0 |

*Summary of Tests Nos. 9 and 10:*

| | Test No. 9, per cent | Test No. 10, per cent |
|---|---|---|
| Gold recovered in trap concentrate | 49·0 | 54·9 |
| Gold recovered in blanket concentrate | 35·5 | 32·9 |
| Bullion recovered by amalgamation | 79·3 | 76·8 |
| Bullion recovered by cyanidation (48 hours) | 20·1 | 22·6 |
| Overall recovery | 99·4 | 99·4 |

### *Tests Nos. 11 and 12*

An attempt was made to duplicate the flow-sheet of the Siscoe mill in so far as was feasible in small-test work.

The ore at −14 mesh was ground in a ball mill in cyanide solution of a strength of 1 pound of potassium cyanide per ton, 5 pounds of lime per ton of ore being added to maintain protective alkalinity. In Test No. 11 the ore was ground to pass 66·2 per cent through 200 mesh and in Test No. 12, 85·4 per cent. The pulps were sampled and passed through a hydraulic trap. The trap tailing was passed over a corduroy blanket and a sample taken of the blanket tailing. The trap concentrate and the blanket concentrate were combined and amalgamated. The amalgamation residue was added to the blanket tailing and this product sampled and agitated in cyanide solutions of 1 pound per ton strength for periods of 24 and 48 hours.

A screen test on the cyanide tailing resulted as follows:

*Screen Test:*

| Mesh | Weight, per cent | |
|---|---|---|
| | Test No. 11 | Test No. 12 |
| − 48 + 65 | 0·4 | |
| − 65 + 100 | 4·0 | 0·3 |
| −100 + 150 | 11·7 | 4·7 |
| −150 + 200 | 17·4 | 9·4 |
| −200 | 66·2 | 85·4 |

After grinding in cyanide solution, the feed to the hydraulic trap assayed 0·63 ounce of gold per ton in Test No. 11 and 0·64 ounce of gold per ton in Test No. 12.

*Results:*

*Hydraulic Classification:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| *Test No. 11* | | | | |
| Feed | 100·00 | 0·63 | 100·0 | |
| Hydraulic concentrate | 0·42 | 40·95* | 27·3 | 238 : 1 |
| Tailing | 99·58 | 0·46 | 72·7 | |
| *Test No. 12* | | | | |
| Feed | 100·0 | 0·64 | 100·0 | |
| Hydraulic concentrate | 0·1 | 155·50* | 24·3 | 1000 : 1 |
| Tailing | 99·9 | 0·485 | 75·7 | |

*Blanket Concentration:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| *Test No. 11* | | | | |
| Feed | 100·00 | 0·46 | 100·0 | |
| Blanket concentrate | 3·97 | 9·41* | 81·2 | 25 : 1 |
| Tailing | 96·03 | 0·09 | 18·8 | |
| *Test No. 12* | | | | |
| Feed | 100·00 | 0·485 | 100·0 | |
| Blanket concentrate | 2·65 | 14·08* | 76·9 | 38 : 1 |
| Tailing | 97·35 | 0·115 | 23·1 | |

*Calculated.

The trap and blanket concentrates were combined and amalgamated. The amalgam residue was added to the blanket tailing, this product assaying 0·145 ounce of gold per ton in Test No. 11 and 0·125 ounce of gold per ton in Test No. 12.

*Cyanidation of Blanket Tailing and Amalgam Residue:*

| Test No. | Agitation, hours | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|
| | | Feed | Tailing | | KCN | CaO |
| 11 | 24 | 0·145 | 0·005 | 96·6 | 0·75 | 2·60 |
| 11 | 48 | 0·145 | 0·005 | 96·6 | 0·80 | 2·60 |
| 12 | 24 | 0·125 | 0·005 | 96·0 | 0·65 | 2·80 |
| 12 | 48 | 0·125 | 0·005 | 96·0 | 0·87 | 2·80 |

*Summary:*

| | Test No. 11, per cent | Test No. 12, per cent |
|---|---|---|
| Gold recovered in trap concentrate | 20·9 | 18·9 |
| Gold recovered in blanket concentrate | 45·4 | 45·4 |
| Bullion recovered by amalgamation | 59·1 | 62·8 |
| Gold extracted by cyanidation in ball mill | 23·2 | 22·0 |
| Gold recovered by cyanidation of blanket tailing and amalgam residue | 17·1 | 14·6 |
| Bullion recovered by cyanidation | 40·3 | 36·6 |
| Overall recovery | 99·4 | 99·4 |

AMALGAMATION AND CYANIDATION

*(Cycle Test)*

*Test No. 13*

This was to determine whether any noticeable fouling of cyanide solution occurred during the agitation periods. Any increase in the gold content of the tailing would indicate a degree of fouling.

The ore at −14 mesh was ground in cyanide solution of 1 pound of potassium cyanide per ton strength to pass 72·5 per cent through 200 mesh. The pulp was passed over a corduroy blanket, the blanket concentrate amalgamated, and the amalgam residue added to the blanket tailing. Portions of this product, assaying 0·12 ounce of gold per ton, were agitated in the primary cyanide grinding solution for a period of 24 hours. After agitation the cyanide tailings were filtered, washed, and assayed, and a fresh portion of the product agitated in the same cyanide solution. The process was repeated for four different cycles.

*Results:*

Feed: gold, 0·12 oz./ton

| Cycle No. | Assay of cyanide tailing, Au, oz./ton | Reagents consumed, lb./ton | |
|---|---|---|---|
| | | KCN | CaO |
| 1 | 0·005 | 0·69 | 4·40 |
| 2 | 0·007 | 0·50 | 4·48 |
| 3 | 0·005 | 0·54 | 4·40 |
| 4 | 0·005 | 0·70 | 4·60 |

At the end of the test the cyanide solution assayed as follows:

Reducing power 117 c.c. $\frac{N}{10}$ $KMnO_4$/litre.

| | |
|---|---|
| KCNS | 0·146 grm./litre |
| Copper | 0·076 " |
| Iron | 0·005 " |

It can be seen from this test that any slight fouling of the cyanide solution has no effect on the extraction. The value of the tailing in the first cycle, 0·005 ounce of gold per ton, was duplicated in the fourth cycle.

*Settling Test*

*Test No. 14*

Owing to the comparatively large amount of talc in Sample B from the "K" ore zone, it was decided to run a short series of settling tests on this ore and at the same time a similar series on Sample A from the main ore zone.

The ores at −14 mesh were ground in a ball mill to pass 67·4 per cent through 200 mesh in Sample A and 73·9 per cent through 200 mesh in Sample B. One pound of potassium cyanide per ton of solution and 5 pounds of lime per ton of ore were added in each case. The pulps were made up to dilution ratios of 1·5 : 1 and 2 : 1 and placed in a tall glass tube of 2-inch diameter. The rate of settling was carefully observed for a 1-hour period.

*Results:*

| Sample No. | Ratio of liquid to solid | Rate of settling, ft./hour | Alkalinity of solution at end of test, pound CaO per ton of solution |
|---|---|---|---|
| B.. | 1·5 : 1 | 0·20 | 0·55 |
| A. | 1·5 : 1 | 0·66 | 0·55 |
| B. | 2 : 1 | 0·31 | 0·45 |
| A. | 2 : 1 | 1·22 | 0·40 |

The above results indicate that the pulp of Sample B from the "K" ore zone settles much more slowly than does Sample A.

## SUMMARY

The work on Sample B, from the "K" ore zone, indicates that 59 per cent of the gold can be recovered by amalgamation and 40 per cent by cyanidation at a grind of 66 per cent through 200 mesh, these recoveries being obtained by grinding in cyanide solution, amalgamating the trap and blanket concentrates, adding the amalgam residue to the blanket tailing, and agitating this product for a 24-hour period.

Some difficulty may be experienced in the settling of the pulp, owing to the comparatively large amount of talc in the ore.

## Ore Dressing and Metallurgical Investigation No. 707

GOLD ORE FROM THE SLADEN-MALARTIC MINES, LIMITED,
AMOS, QUEBEC

*Shipment.* One carload of gold ore, containing 201 bags and weighing 30 tons, was received on January 18, 1937, from St. B. Sladen, Managing Director, Sladen-Malartic Mines, Limited, 63 Sparks Street, Ottawa, Ontario.

*Location of Property.* The property of the Sladen-Malartic Mines, Limited, from which this shipment was received is situated in Fournière township, Abitibi county, Quebec.

*Sampling and Analysis.* After crushing, cutting, and grinding by standard methods a sample of the ore was obtained which assayed as follows:

| | |
|---|---|
| Gold | 0·385 oz./ton |
| Silver | 1·52 " |
| Lead | Trace |
| Copper | Trace |
| Zinc | Trace |
| Tellurium | 0·012 per cent |
| Iron | 2·30 " |
| Carbon dioxide | 1·73 " |
| Sulphur | 1·40 " |
| Silica | 76·45 " |

*Characteristics of the Ore.* Two samples, designated Sample No. 1—Quartz, and Sample No. 2—Porphyry, were submitted for microscopic examination. Six polished sections were prepared from each sample and were examined.

The gangue of *Sample No. 1* is grey translucent quartz, which has a slight bluish cast. The quartz is fine-grained with rather uniform sizing of the grains suggesting a replaced sediment, though there is no confirmatory evidence of this. The inter-grain boundaries of the quartz should be lines of weakness along which it will break most easily in grinding. An irregular network of fine fractures which contain very small quantities of carbonate is visible to the unaided eye.

The metallic minerals identified as present are as follows:

Pyrite
Sphalerite
Galena
Sylvanite
Petzite (?) or possibly hessite (??)
Native gold
Chalcopyrite

The metallic mineral content is very small and the minerals occur in a finely-divided state. Pyrite is disseminated in the quartz, in no apparent relation to the others, but the remaining minerals tend to associate together along the inter-grain boundaries of the quartz and along fine fractures. Sylvanite and petzite (?) are almost always found together, and a large part of the native gold is associated with these, usually enclosed within sylvanite. Most of the gold grains are irregular in shape, but a few show isometric crystal form. The grain size and modes of occurrence of the visible native gold are shown in the following table. From the fact that deposition of the pyrite appears to have definitely preceded that of the remaining metallic minerals, and because no visible gold was found associated with the pyrite, it is unlikely that this mineral contains submicroscopic gold.

*Grain Size of Visible Native Gold:*

| Mesh (Tyler) | Native Gold | | | Totals, per cent |
|---|---|---|---|---|
| | Free, per cent | Associated with sylvanite, per cent | Associated with petzite(?), per cent | |
| + 150 | ........... | 5·5 | ........... | 5·5 |
| − 150 + 200 | ........... | 8·1 | 3·2 | 11·3 |
| − 200 + 280 | ........... | 10·2 | 2·6 | 12·8 |
| − 280 + 400 | ........... | 10·2 | 1·7 | 11·9 |
| − 400 + 560 | 4·0 | 13·8 | 1·5 | 19·3 |
| − 560 + 800 | 3·4 | 12·3 | 0·9 | 16·6 |
| − 800 + 1100 | 3·0 | 10·2 | 0·5 | 13·7 |
| −1100 + 1600 | 1·7 | 4·2 | ........... | 5·9 |
| −1600 + 2300 | 0·3 | 1·8 | ........... | 2·1 |
| −2300 | 0·2 | 0·7 | ........... | 0·9 |
| | | 77·0 | 10·4 | |
| Totals | 12·6 | 87·4 | | 100·0 |

*Grain Size of the Gold-bearing Tellurides:*

| Mesh | Sylvanite, per cent | Petzite(?), per cent |
|---|---|---|
| + 100 | 14·9 | ........... |
| − 100 + 150 | 11·0 | 20·1 |
| − 150 + 200 | 17·2 | 25·5 |
| − 200 + 280 | 18·0 | 18·9 |
| − 280 + 400 | 16·1 | 15·8 |
| − 400 + 560 | 12·5 | 10·3 |
| − 560 + 800 | 7·1 | 7·0 |
| − 800 + 1100 | 2·4 | 1·8 |
| −1100 | 0·8 | 0·6 |
| | 100·0 | 100·0 |

Although the pyrite content of *Sample No. 2* is much higher than that of Sample No. 1, the gold mineralization appears to be similar. Native gold and tellurides are present in the same modes of occurrence but the quantities are much smaller. It is apparent that the pyrite mineralization was earlier than, and was not intimately connected with, the gold mineralization. Therefore, pyrite cannot be taken as an indicator of gold.

## EXPERIMENTAL TESTS

The test work included amalgamation, flotation, and cyanidation. Over 97 per cent of the gold and silver was extracted by straight cyanidation.

### AMALGAMATION

*Tests Nos. 1 and 2*

The ore at −14 mesh was ground in a ball mill to pass 54·1 per cent in Test No. 1 and 77·4 per cent in Test No. 2, through a 200-mesh screen. The pulps were amalgamated with mercury for 1 hour in a jar mill. The mercury was separated from the pulp, which was assayed for gold and silver.

A screen test showed the grinding as follows:

| Mesh | Weight, per cent | |
|---|---|---|
| | Test No. 1 | Test No. 2 |
| − 48 + 65 | 1·9 | ........... |
| − 65 + 100 | 8·6 | 0·7 |
| −100 + 150 | 16·9 | 6·1 |
| −150 + 200 | 18·4 | 15·8 |
| —200 | 54·1 | 77·4 |

*Amalgamation:*

| Test No. | Feed assay, oz./ton | | Tailing assay, oz./ton | | Recovery, per cent | |
|---|---|---|---|---|---|---|
| | Au | Ag | Au | Ag | Au | Ag |
| 1 | 0·385 | 1·52 | 0·255 | 1·51 | 33·8 | 0·7 |
| 2 | 0·385 | 1·52 | 0·21 | 1·40 | 45·5 | 7·9 |

The above tests were made to determine the total amounts of gold and silver set free by these particular degrees of comminution, and the results are not comparable to the amount of gold and silver that could be recovered by either plates or blankets.

### CYANIDATION

*Tests Nos. 3 to 6*

Portions of the ore at −14 mesh were ground to different sizes in ball mills and the pulps agitated in cyanide solutions having a strength of 1 pound of potassium cyanide per ton for 24 and 48 hours. Five pounds of lime was added per ton of ore to maintain protective alkalinity. The ratio of dilution was 2 of solution to 1 of ore. The cyanide tailings were assayed for gold and silver.

Screen tests showed the grinding as follows:

| Mesh | Weight, per cent | | | |
|---|---|---|---|---|
| | Test No. 3 | Test No. 4 | Test No. 5 | Test No. 6 |
| − 48 + 65 | 0·2 | ........ | ........ | ........ |
| − 65 + 100 | 1·8 | 0·9 | 0·2 | ........ |
| −100 + 150 | 8·5 | 5·8 | 3·1 | 1·3 |
| −150 + 200 | 19·0 | 13·8 | 11·9 | 7·9 |
| −200 | 70·4 | 79·3 | 84·5 | 90·8 |

*Cyanidation:*

Feed: gold, 0·385 oz./ton; silver, 1·52 oz./ton

| Test No. | Agitation, hours | Tailing, assay, oz./ton | | Extraction, per cent | | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|---|
| | | Au | Ag | Au | Ag | KCN | CaO |
| 3 | 24 | 0·01 | 0·10 | 97·4 | 93·4 | 0·3 | 4·1 |
| 3 | 48 | 0·0075 | 0·02 | 98·1 | 98·7 | 0·4 | 4·2 |
| 4 | 24 | 0·01 | 0·08 | 97·4 | 94·7 | 0·3 | 4·2 |
| 4 | 48 | 0·01 | 0·03 | 97·4 | 98·0 | 0·5 | 4·2 |
| 5 | 24 | 0·01 | 0·08 | 97·4 | 94·7 | 0·4 | 4·3 |
| 5 | 48 | 0·0075 | 0·01 | 98·1 | 99·3 | 0·5 | 4·3 |
| 6 | 24 | 0·01 | 0·06 | 97·4 | 96·0 | 0·5 | 4·3 |
| 6 | 48 | 0·01 | 0·01 | 97·4 | 99·3 | 0·6 | 4·4 |

*Tests Nos. 7 and 8*

The ore was ground in cyanide solution to pass 67·4 per cent through 200 mesh and agitated at dilution ratios of 1·5 : 1 in Test No. 7 and 2 : 1 in Test No. 8 for periods of 24 and 48 hours. Five pounds of lime was added per ton of ore.

A screen test showed the grinding as follows:

| Mesh | Weight, per cent |
|---|---|
| − 48 + 65 | 0·1 |
| − 65 + 100 | 3·0 |
| −100 + 150 | 11·4 |
| −150 + 200 | 17·9 |
| −200 | 67·4 |

*Cyanidation:*

Feed: gold, 0·385 oz./ton; silver, 1·52 oz./ton

| Test No. | Agitation, hours | Tailing assay, oz./ton | | Extraction, per cent | | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|---|
| | | Au | Ag | Au | Ag | KCN | CaO |
| 7 | 24 | 0·015 | 0·11 | 96·1 | 92·8 | 0·35 | 4·0 |
| 7 | 48 | 0·01 | 0·025 | 97·4 | 98·4 | 0·35 | 4·1 |
| 8 | 24 | 0·01 | 0·17 | 97·4 | 88·8 | 0·35 | 4·0 |
| 8 | 48 | 0·01 | 0·05 | 97·4 | 96·7 | 0·35 | 4·2 |

## FLOTATION

### *Test No. 9*

The ore was ground in a ball mill with 3 pounds of soda ash, 0·14 pound Barrett No. 4 oil, and 0·2 pound of amyl xanthate per ton to pass 84·5 per cent through 200 mesh. The pulp was transferred to a flotation cell and floated with 0·05 pound of pine oil per ton. A flotation concentrate was removed and assayed.

*Results:*

| Product | Weight, per cent | Assay, oz./ton | | Distribution, per cent | | Ratio of concentration |
|---|---|---|---|---|---|---|
| | | Au | Ag | Au | Ag | |
| Feed............... | 100·0 | 0·385 | 1·52 | 100·0 | 100·0 | |
| Flotation concentrate | 8·8 | 4·10 | 16·00 | 93·0 | 94·5 | 11·3 : 1 |
| Tailing............. | 91·2 | 0·03 | 0·09 | 7·0 | 5·5 | |

The assay of the flotation concentrate resulted as follows:

| | | | |
|---|---|---|---|
| Gold.................. | 4·10 oz./ton | Copper.................. | 0·06 per cent |
| Silver.................. | 16·00 " | Iron.................... | 14·75 " |
| Tellurium.............. | 0·075 per cent | Sulphur................. | 15·60 " |
| Lead.................. | 0·15 " | Silica.................. | 45·25 " |
| Zinc.................. | 0·03 " | | |

## SETTLING TESTS

### *Tests Nos. 10 and 11*

These tests were carried out in a tall glass tube having an inside diameter of 2 inches. The ore was ground in cyanide solution to pass 79·3 per cent through 200 mesh, five pounds of lime being added per ton of ore. The pulps were transferred to the glass tubes and the level of solids in decimals of feet read every 5 minutes. Readings were made for a one-hour period. At the end of the tests the solutions were titrated for alkalinity. The results of the tests are recorded in the following tables.

A screen test showed the grinding as follows:

| Mesh | Weight, per cent |
|---|---|
| − 65 + 100.......... | 0·9 |
| −100 + 150.......... | 5·8 |
| −150 + 200.......... | 13·8 |
| −200.......... | 79·3 |

### *Test No. 10*

| | |
|---|---|
| Ratio of liquid to solid.................. | 1·5 : 1 |
| Lime added per ton solid.................. | 5·0 pounds |
| Alkalinity of solution at end of test.................. | 0·50 lb./ton of solution |
| Overflow.................. | Clear |
| Rate of settling.................. | 0·62 ft./hour |

| Time | Settlement of solids, in feet | Cumulative settlement |
|---|---|---|
| 5 minutes.......... | 0·04 | 0·04 |
| 10 " .......... | 0·05 | 0·09 |
| 15 " .......... | 0·05 | 0·14 |
| 20 " .......... | 0·04 | 0·18 |
| 25 " .......... | 0·05 | 0·23 |
| 30 " .......... | 0·05 | 0·28 |
| 35 " .......... | 0·05 | 0·33 |
| 40 " .......... | 0·05 | 0·38 |
| 45 " .......... | 0·06 | 0·44 |
| 50 " .......... | 0·06 | 0·50 |
| 55 " .......... | 0·06 | 0·56 |
| 1 hour .......... | 0·06 | 0·62 |

*Test No. 11*

| | |
|---|---|
| Ratio of liquid to solid | 2 : 1 |
| Lime added per ton solid | 5·0 pounds |
| Alkalinity of solution at end of test | 0·50 lb./ton of solution |
| Overflow | Clear |
| Rate of settling | 1·31 ft./hour |

| Time | Settlement of solids, in feet | Cumulative settlement |
|---|---|---|
| 5 minutes | 0·08 | 0·08 |
| 10 " | 0·11 | 0·19 |
| 15 " | 0·10 | 0·29 |
| 20 " | 0·12 | 0·41 |
| 25 " | 0·11 | 0·52 |
| 30 " | 0·11 | 0·63 |
| 35 " | 0·11 | 0·74 |
| 40 " | 0·12 | 0·86 |
| 45 " | 0·12 | 0·98 |
| 50 " | 0·12 | 1·10 |
| 55 " | 0·11 | 1·21 |
| 1 hour | 0·10 | 1·31 |

*Milling Test (Grinding in Cyanide Solution)*

A mill run was carried out on the ore, using the following flow-sheet:

Automatic feeder (¼-inch feed):

↓
Ball mill
↓
Akins classifier
↓ Oversize ↓ (returned to ball mill) — ↓ Overflow
↓
Pump
↓
Thickener
↓
Pachuca agitator
↓
Denver super-agitator
↓
Oliver filter
↓ Solution to circuit — ↓ Tailing to waste

| | |
|---|---|
| Time of run | 59 hours |
| Rate of feed | 180 lb./hour |
| Density of mill discharge | 51·0 per cent solids |
| Density of classifier overflow | 17·1 " " |
| Density of Pachuca tank contents | 46·7 " " |
| Grinding: Classifier overflow | 65·0 per cent −200 mesh |

The ore was mixed with 5 pounds of lime per ton and fed to the ball mill at the rate of 180 pounds per hour, through a Hardinge constant weight feeder. Fresh cyanide solution of a strength of 1 pound per ton was fed to the ball mill from a feed tank. The mill discharged into a classifier, which overflowed at 17·1 per cent solids to a thickener, while the oversize was returned to the ball mill for regrinding. The thickener overflow was returned to the mill discharge launder to dilute the classifier overflow, and the thickener underflow went to the agitators. This product was agitated for 18 hours at 46·7 per cent solids and then passed to the Oliver filter where it was washed and the tailing sampled and discarded. The solution from the Oliver filter was returned to the circuit and fed to the ball mill. All solutions were kept at 1 pound of potassium cyanide per ton and 0·3 to 0·4 pound of lime per ton. Samples of the classifier overflow and the Oliver filter tailing were taken at one-hour intervals. The samples were filtered, washed, and assayed for gold and silver.

| | |
|---|---|
| Mill feed | 0·385 Au, oz./ton<br>1·52 Ag, oz./ton |
| Classifier overflow | 0·105 Au, oz./ton<br>1·05 Ag, oz./ton |
| Cyanide tailing | 0·015 Au, oz./ton<br>0·185 Ag, oz./ton |

These results were obtained after a 48-hour run, the classifier overflow building up from 0·04 ounce of gold and 0·57 ounce of silver per ton to 0·105 ounce of gold and 1·05 ounces of silver per ton. An attempt was made to precipitate the pregnant solution and return the resulting barren to the mill circuit. Owing to a break-down of the grinding unit it was impossible to complete this run, and the precipitation of the gold in the pregnant solution by zinc dust was not satisfactory.

### CYANIDATION (CYCLE TEST)

#### *Test No. 12*

This was to determine whether any noticeable fouling of cyanide solution occurs during the agitation periods. Any increase in the gold or silver content of the tailing would indicate a degree of fouling. The ore at −14 mesh was ground in cyanide solution of 1 pound of potassium cyanide per ton strength to pass 72·8 per cent through 200 mesh, 5 pounds of lime being added per ton of ore. The pulp was filtered and five portions agitated in the primary grinding solution for a 24-hour period. After agitation the tailings were filtered, washed, and assayed for gold and silver. A fresh portion of the ore was ground in the same cyanide solution and four portions of the pulp agitated for 24 hours. The process was repeated for four different cycles.

A screen test showed the grinding as follows:

| Mesh | Weight, per cent |
|---|---|
| − 65 + 100 | 1·0 |
| −100 + 150 | 8·7 |
| −150 + 200 | 17·4 |
| −200 | 72·8 |

*Results:* Feed: gold, 0·385 oz./ton; silver, 1·52 oz./ton

| Cycle No. | Average tailing assay, oz./ton | | Extraction, per cent | | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|
| | Au | Ag | Au | Ag | KCN | CaO |
| 1 | 0·010 | 0·11 | 97·4 | 92·5 | 0·38 | 4·38 |
| 2 | 0·012 | 0·22 | 96·7 | 83·0 | 0·44 | 4·00 |
| 3 | 0·015 | 0·30 | 96·1 | 80·3 | 0·51 | 4·10 |
| 4 | 0·015 | 0·29 | 96·1 | 80·9 | 0·51 | 4·10 |

An analysis of the cyanide solution from the fourth cycle was determined as follows:

| | |
|---|---|
| Reducing power: | 80 c.c. $\frac{N}{10}$ $KMnO_4$/litre |
| KCNS: | 0·04 grm./litre |

The results of the cycle test give an indication that a slight amount of fouling takes place in the solution. Although this does not appear to be serious, attention should be paid to this fact when starting operations. By bleeding solution from the circuit, the normal condition can probably be obtained.

## SUMMARY AND CONCLUSIONS

The ore cyanides readily with a normal consumption of cyanide and lime. Settling tests show a rate of settlement faster than normal.

A grind of 75 per cent through 200 mesh and a 24- to 30-hour period of agitation should extract 97 per cent of the gold and 95 per cent of the silver by straight cyanidation.

## Ore Dressing and Metallurgical Investigation No. 708

### GOLD ORE FROM HARD ROCK GOLD MINES, LIMITED, GERALDTON, ONTARIO

*Shipment.* A shipment of 20 sacks of ore, net weight 1,130 pounds, was received on February 9, 1937. The sample was submitted by J. C. Dumbrille, Manager, Hard Rock Gold Mines, Limited, Geraldton, Ontario.

*Location of Property.* This property is situated in the Little Long Lac area of the Thunder Bay district, near the town of Geraldton, Ontario.

*Characteristics of the Ore.* Six polished sections were prepared and examined microscopically for the purpose of determining the character of the ore and the mode of occurrence and grain size of the visible native gold.

The *gangue* in the sections is fine-grained, very dark silicates with some finely disseminated carbonate and some vein quartz.

Pyrite is the only abundant *metallic mineral* in the ore. It usually occurs as coarse-textured and somewhat fractured masses and medium to small crystals and grains. Rare scattered grains of pyrrhotite, chalcopyrite and hematite are present, and a small quantity of "limonite" has in some places replaced the pyrite grains.

Native gold is visible only in the pyrite. It occurs *(a)* along fractures, *(b)* in dense pyrite, and *(c)* in pyrite and associated with tiny grains of chalcopyrite and/or pyrrhotite.

The following table shows the grain analysis of the gold visible in the polished sections:

*Grain Size and Modes of Occurrence of Native Gold in Polished Sections:*

| Mesh | Along fractures in pyrite, per cent | In dense pyrite, per cent | Associated with chalcopyrite and/or pyrrhotite in pyrite, per cent | Totals, per cent |
|---|---|---|---|---|
| + 200 | | 16·8 | | 16·8 |
| − 200 + 280 | 10·5 | | | 10·5 |
| − 280 + 400 | 8·4 | | | 8·4 |
| − 400 + 560 | 12·5 | 6·3 | | 18·8 |
| − 560 + 800 | 8·1 | 4·2 | | 12·3 |
| − 800 + 1100 | 3·2 | | 3·1 | 6·3 |
| −1100 + 1600 | 2·1 | 3·9 | 8·0 | 14·0 |
| −1600 + 2300 | | 4·2 | 1·4 | 5·6 |
| −2300 | | 4·0 | 3·3 | 7·3 |
| | 44·8 | 39·4 | 15·8 | 100·0 |

*Sampling and Analysis.* The shipment of ore was crushed and a sample cut out and assayed, the results being as follows:

| | |
|---|---|
| Gold | 0·285 oz./ton |
| Silver | 0·12 " |
| Iron | 7·27 per cent |
| Sulphur | 11·08 " |
| Arsenic | 0·08 " |
| Pyrrhotite | 0·14 " |

## EXPERIMENTAL TESTS

The test work conducted on this ore included tests by cyanidation, amalgamation, and concentration, both alone and in combination. By straight cyanidation, 92·9 per cent of the gold can be extracted. By putting a table in the circuit, concentrating and regrinding the sulphides, extraction can be raised to 95 per cent or better. The ore can be floated down to a tailing assaying 0·005 ounce of gold per ton; and then by regrinding and cyaniding the concentrate, from 94 to 95 per cent of the total gold in the ore can be extracted. Although further extraction might result from roasting the concentrate, it could not be done economically on this grade of ore.

Work done on a previous shipment of ore from this property is covered in Investigation No. 652, Investigations in Ore Dressing and Metallurgy, Report No. 771. The sulphur content of the present shipment is much higher than that of the former.

The tests are described in detail as follows:

### CYANIDATION

*Tests Nos. 1 to 8*

Samples of the ore were ground in ball mills 69·0, 76·3, 83·2, and 87·3 per cent through 200 mesh and agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for periods of 24 and 48 hours. The pulps were not filtered between grinding and agitation.

*Summary of Tests Nos. 1 to 8:*

Feed: gold, 0·285 oz./ton

| Test No. | Grinding, per cent −200 mesh | Agitation, hours | Tailing assay, Au, oz./ton | Extraction, per cent | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| 1 | 69·0 | 24 | 0·03 | 89·5 | 0·58 | 3·25 |
| 2 | 76·3 | 24 | 0·025 | 91·2 | 0·58 | 3·28 |
| 3 | 83·2 | 24 | 0·025 | 91·2 | 0·64 | 3·34 |
| 4 | 87·3 | 24 | 0·02 | 92·9 | 0·64 | 3·37 |
| 5 | 69·0 | 48 | 0·025 | 91·2 | 0·67 | 3·40 |
| 6 | 76·3 | 48 | 0·02 | 92·9 | 0·69 | 3·43 |
| 7 | 83·2 | 48 | 0·02 | 92·9 | 0·73 | 3·53 |
| 8 | 87·3 | 48 | 0·02 | 92·9 | 0·76 | 3·54 |

## AMALGAMATION AND CYANIDATION

### *Tests Nos. 9 and 10*

Samples of the ore were ground 76·3 and 87·3 per cent through 200 mesh in ball mills and amalgamated with mercury for one hour in jar mills. The amalgamation tailings were sampled and assayed for gold and portions of each were agitated in cyanide solution, 1·0 pound of potassium cyanide per ton for 24 hours. The cyanide tailings were assayed for gold.

*Summary of Tests Nos. 9 and 10:*

Feed: gold, 0·285 oz./ton

| Test No. | Grinding, per cent −200 mesh | Amalgamation tailing assay, Au, oz./ton | Cyanide tailing assay, Au, oz./ton | Extraction, per cent | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| 9 | 76·3 | 0·11 | 0·025 | 91·2 | 0·32 | 2·84 |
| 10 | 87·3 | 0·09 | 0·025 | 91·2 | 0·28 | 2·84 |

## CYANIDATION WITH TABLE CONCENTRATION

### *Test No. 11*

A sample of the ore was ground 69·0 per cent through 200 mesh in a ball mill and agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 24 hours. The cyanide tailing was treated on a concentrating table, where a sulphide concentrate was taken off. The table tailing was assayed for gold and the table concentrate was reground nearly all through 325 mesh and reagitated in cyanide solution for 48 hours. The cyanide tailing from the reground concentrate was assayed for gold.

*Summary:*

Feed: gold, 0·285 oz./ton

| Product | Weight, per cent | Assay, Au, oz./ton | Extraction, per cent | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|
| | | | | KCN | CaO |
| Table tailing | 91·0 | 0·01 | ........ | ........ | ........ |
| Cyanide tailing from reground concentrate | 9·0 | 0·05 | ........ | ........ | ........ |
| Average tailing (cal.) | 100·0 | 0·014 | 95·1 | 0·60 | 4·5 |

## FLOTATION AND CYANIDATION

### *Test No. 12*

A sample of the ore was ground 69·0 per cent through 200 mesh in a ball mill and floated. The concentrate was reground nearly all through 325 mesh and agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 28 hours. The flotation tailing and the cyanide tailing from reground concentrate were assayed for gold.

*Charge to Ball Mill:*

| | |
|---|---|
| Ore | 2,000 grms. at −14 mesh |
| Water | 1,500 c.c. |
| Soda ash | 1·0 lb./ton |
| Potassium amyl xanthate | 0·10 " |

*Reagents to Cell:*

| | |
|---|---|
| Pine oil | 0·10 lb./ton |
| Copper sulphate | 1·0 " |

*Summary:*

Feed: gold, 0·285 oz./ton

| Product | Weight, per cent | Assay, Au, oz./ton | Extraction, per cent | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|
| | | | | KCN | CaO |
| Flotation tailing | 89·1 | 0·005 | | | |
| Cyanide tailing from reground concentrate | 10·9 | 0·105 | | | |
| Average tailing (cal.) | 100·0 | 0·016 | 94·4 | 0·44 | 2·10 |

## CONCLUSIONS

The work shows that the gold is nearly all associated with the sulphides and that most of it is quite fine. In order to expose this gold to the dissolving action of cyanide solution, extremely fine grinding of the sulphides will be necessary.

A good flow-sheet would be to grind the ore in cyanide solution about 70 per cent through 200 mesh in a ball mill-classifier circuit, with a trap or blankets between the mill and classifier to catch coarse gold. The classifier overflow would be thickened and the overflow sent to precipitation while the underflow would go to concentrating tables. The table tailing would go to cyanide agitators, and the concentrate to a regrind mill where it would be subjected to fine grinding and go on to the cyanide agitators. The trap and blanket concentrate would be reground and amalgamated and sent on with the reground table concentrate. In this way, about 95 per cent of the gold should be extracted.

As an alternative to the above, the ore could be floated and the concentrate reground and cyanided. This sample of ore floats particularly well, with a ratio of concentration of about 10 : 1, when it is ground 70 per cent through 200 mesh. This does not appear however to be the case with the former shipments of ore and the first process suggested should, therefore, be the one adopted if all kinds of the ore are to be milled together.

## Ore Dressing and Metallurgical Investigation No. 709

MILL PRODUCTS FROM LITTLE LONG LAC GOLD MINES, LIMITED, LITTLE LONG LAC, ONTARIO

*Samples.* Six samples of mill products from Little Long Lac Gold Mines, Limited, Little Long Lac, Ontario, were received on February 9, 1937. They were taken over a period of time and represent composite samples of the following:

No. 1010—Mill feed
No. 1011—Cyanide tailing
No. 1012—Flotation concentrate
No. 1013—Flotation tailing
No. 1014—Calcine
No. 1015—Roaster tailing

The numbers refer to the polished sections, of which six from each sample were prepared and examined microscopically.

*Purpose.* The examination was to determine, if possible, the mode of occurrence of the gold in each product.

### *No. 1010—Mill Feed*

The constituents are:

| | |
|---|---|
| Gangue | Major |
| Pyrite | Major |
| Arsenopyrite | Major |
| Pyrrhotite | Minor |
| Chalcopyrite | Minor |
| Magnetite | Minor |
| Native gold. | |

The sulphides are very largely free at this grinding and only a few small grains remain locked in the gangue. The gold is largely free and varies in grain size between 10 and 35 microns. One grain of gold, 12 microns in size, occurs within a 75-micron grain of pyrite.

### *No. 1011—Cyanide Tailing*

The appearance of this product is entirely similar to the mill feed except that no native gold was visible.

### *No. 1012—Flotation Concentrate*

The constituents are:

| | |
|---|---|
| Pyrite | Major |
| Arsenopyrite | Major |
| Gangue | Minor |
| Chalcopyrite | Minor |
| Magnetite | Trace |

In addition to the above, a single tiny grain of a soft grey mineral was seen in pyrite. The grain was lost before it could be subjected to enough tests to establish its identity, and further search of the sections failed to disclose any others.

The minerals are exceedingly well freed. The ratio of pyrite to arsenopyrite is roughly 7 : 3.

### *No. 1013—Flotation Tailing*

The constituents are:

| | |
|---|---|
| Gangue | Major |
| Pyrite<br>Arsenopyrite | Minor |
| Magnetite<br>Chalcopyrite | Trace |

A microscopic grain analysis of this product gives the following result concerning the condition of the pyrite and arsenopyrite:

(In percentages)

| Mesh | Pyrite | | Arsenopyrite | | Totals, per cent by volume |
|---|---|---|---|---|---|
| | Free | Combined with gangue | Free | Combined with gangue | |
| + 560 | 7·0 | ...... | ...... | ...... | 7·0 |
| − 560 + 800 | 9·0 | 2·5 | 2·5 | 2·0 | 16·0 |
| − 800 + 1100 | 11·0 | 4·5 | 3·0 | 2·5 | 21·0 |
| −1100 + 1600 | 5·0 | 1·0 | 7·0 | 2·0 | 15·0 |
| −1600 + 2300 | 3·6 | 2·2 | 5·8 | 1·4 | 13·0 |
| −2300 | 10·3 | 1·9 | 13·7 | 2·1 | 28·0 |
| Totals | 45·9 | 12·1 | 32·0 | 10·0 | 100·0 |
| | 58·0 | | 42·0 | | |

The table shows several features of this product:

(A) The ratio of pyrite to arsenopyrite is roughly 6 : 4, showing correspondingly more arsenopyrite relative to the pyrite in the tailing than in the concentrate and indicating that it has not floated so well as the pyrite.

(B) The sulphides are about 78 per cent free and only 22 per cent combined with gangue.

(C) The sulphides are extremely fine and the largest quantity in any one grain size is that in the −2300-mesh size, 28·0 per cent, of which 24 per cent is free.

### *No. 1014—Calcine*

The constituents are:

| | |
|---|---|
| Iron oxides—Hematite (artificial)<br>Magnetite (artificial) | Major |
| Pyrrhotite (artificial) (?) | Very minor |
| Pyrite | Trace |

Metallic gold.

The grains of iron oxides are angular in shape and very porous in character. Many are composed wholly of hematite, others are composed of hematite with a magnetite core, and still others show an outer zone of hematite, an intermediate zone of magnetite, and a core of pyrrhotite. Rarely, a core of unaltered sulphide is seen and in very rare cases a grain of pyrite has come through the roasting apparently unscathed.

The metallic gold occurs as free rounded particles, the largest about 15 microns in size. The average size of the visible gold, however, is about 2300 mesh or 6 microns, many grains being smaller than this. No grains less than one micron in size were measured. It seems odd that the metallic gold should not be enclosed within the iron oxides, but none was seen. The gold in the oxides is possibly very finely divided and submicroscopic in form, but the oxides may have been rendered porous enough to effect good dissolution by the cyanide solution.

*No. 1015—Roaster Tailing*

Entirely similar to the calcine, but no metallic gold is visible.

CONCLUSION

The evidence as seen under the microscope is conclusive in indicating that a very large proportion, if not all, of the gold in the flotation concentrates is present in the sulphides in submicroscopic form.

## Ore Dressing and Metallurgical Investigation No. 710

ORE AND CONCENTRATE FROM THE MINTO GOLD MINES, LIMITED, BRIDGE RIVER DISTRICT, BRITISH COLUMBIA

*Shipment.* A number of parcels of ore, concentrate, and tailing were received at different times, for tests, from the Minto Gold Mines, Limited, Bridge River district, British Columbia. The present investigation is a summary and discussion of the whole.

(1) A shipment of one sack of ore, net weight 100 pounds, was received on June 1, 1934. The sample was submitted by Warren A. Davidson, Superintendent, Minto Gold Mines, Limited.

(2) One bag of concentrate weighing 50 pounds was received on September 6, 1935, from the Minto Gold Mines, Ltd., per J. A. MacKenzie.

(3) Two bags of gold ore weighing 190 pounds were received on November 22, 1935, and a box containing 10 pounds of mill tailing arrived four days later. These were forwarded by J. A. MacKenzie, for the Minto Gold Mines, Ltd.

(4) One bag of concentrate weighing approximately 30 pounds was received on March 27, 1936, from the Minto Gold Mines, Ltd. The sample was submitted by W. Asselstine, Consulting Engineer for the company.

(5) One bag of concentrate weighing approximately 110 pounds was received on November 23, 1936.

One bag of ore weighing approximately 202 pounds was received on November 23, 1936.

(6) One bag of ore weighing 110 pounds was received on March 16, 1937. The sample was submitted by A. W. Holloway.

*Characteristics of Samples:*

*Shipment No. 1.* Samples showing the more heavily mineralized ore were selected and examined microscopically to determine the metallic minerals and their modes of occurrence, six polished sections being prepared and examined under the reflecting microscope. In addition the hand specimens were examined with the binocular microscope.

The gangue consists of fine-textured grey to white quartz and patches and veins of impure grey to white dolomite (or ankerite?), with inclusions of dense, fine-textured, dark grey country rock. A bright green transparent mineral occurs as fine stringers and spots in the dolomitic gangue, and this is probably mariposite.

The distribution of the metallic minerals is very erratic and spotty, and no two polished sections show quite the same mineralogical features. Much of the ore is barren of ore minerals, whereas some shows marked concentration of the sulphides, usually in the form of heavily mineralized stringers. In their order of abundance in the sections examined, the ore minerals are: pyrite, arsenopyrite, pyrrhotite, stibnite, sphalerite, unknown mineral "A", tetrahedrite, chalcopyrite, native bismuth, galena(?), and native gold.

Pyrite, arsenopyrite, and pyrrhotite are locally abundant. Pyrite and arsenopyrite are often much shattered and brecciated, and pyrrhotite commonly invades and replaces pyrite. All three minerals have been seen to contain fine veinlets of dolomitic gangue.

Stibnite is very abundant in one section. It occurs as thick elongated crystals and groups of crystals in a dolomitic gangue, and does not appear to be associated with other ore minerals.

Sphalerite, although present only in small amount, is rather widespread in association with the other sulphides. It occurs as small irregular grains, and invariably contains numerous tiny dots of chalcopyrite and in some places also of unknown "A" and galena(?).

Unknown "A" occurs as small irregular grains, often isolated in the dolomitic gangue but more rarely associated with sphalerite, tetrahedrite and arsenopyrite. The following results of tests on the mineral show that it is either jamesonite ($4PbS.FeS.3Sb_2S_3$) or a mineral closely allied in properties and composition.

| | |
|---|---|
| *Colour:* | Galena-white. |
| *Hardness:* | Soft; B. |
| *Crossed nicols:* | Strongly anisotropic. |
| *Etch tests:* | $HNO_3$—differentially iridescent to black. |
| | KOH—slowly tarnishes brown—rubs grey. |
| | HCl, KCN, $FeCl_3$, $HgCl_2$—negative. |
| *Microchemical analysis:* | S —positive—strong. |
| | Sb —positive—strong. |
| | As—trace. |
| | Bi —trace (?). |
| | Fe —positive to trace. |
| | Pb—doubtful—one test positive. |
| | Co, Ni, Cu, Se, Te—nil. |

Tetrahedrite occurs in very small amount as small grains associated with sphalerite, unknown "A" and galena(?). Microchemical tests failed to reveal the presence of silver in this mineral.

Chalcopyrite is very small in amount, and its only mode of occurrence is as tiny dots in sphalerite.

A very small amount of native bismuth is present as small irregular grains in pyrrhotite, and more rarely in unknown "A".

A few tiny irregular grains of a mineral closely resembling galena were seen in sphalerite and rarely in pyrite. Their identification is not positive.

Native gold was not seen in any of the polished sections. Examination of the hand specimens under the binocular microscope, however, showed a number of small flakes and grains of native gold occurring in a narrow discontinuous stringer of very dark quartz in light quartz. It is not known to what impurity the dark quartz owes its colour.

*Shipment No. 2.* This sample apparently represents a table concentrate made from a flotation concentrate, and contains 35 per cent of arsenopyrite, 50 per cent of pyrite, with small amounts of sphalerite, stibnite, etc., and less than 10 per cent of gangue material.

No microscopic examination was made.

*Shipment No. 3.* The constituents recognized in this sample of mill tailing are: gangue, pyrite, arsenopyrite, stibnite, sphalerite, chalcopyrite, and pyrrhotite.

A microscopic examination of polished sections of the gold ore showed that the shipment is similar in mineral-bearing character to the ore previously examined (Shipment No. 1). Two tiny grains of native gold are visible in the sections, both in dense pyrite and measuring 35 and 3 microns (400 and 2300 mesh).

*Shipment No. 4.* This represents a flotation concentrate, containing 24 per cent of arsenopyrite, 40 per cent of pyrite, with some sphalerite, stibnite, etc., and 25 to 30 per cent of gangue.

*Shipment No. 5.* This concentrate sample represents a low-grade flotation concentrate, containing 20 per cent of arsenopyrite, 40 per cent of pyrite, with some sphalerite, stibnite, etc.

The sample of ore sent with this shipment is apparently of somewhat lower grade than that employed for regular mill feed.

*Shipment No. 6.* This ore appears to be similar in general mineral characteristics to samples previously submitted.

*Analyses of Samples:*

*Shipment No. 1.* The analysis of this ore sample is as follows:

| | | | |
|---|---|---|---|
| Gold | 1·30 oz./ton | Silver | 2·43 oz./ton |
| Arsenic | 1·40 per cent | Copper | 0·06 per cent |
| Iron | 6·19 " | Lead | 0·17 " |
| Sulphur | 4·76 " | Antimony | 0·10 " |
| Zinc | 0·70 " | Insoluble | 44·14 " |

Considering the later results obtained in mine development, this sample is not at all representative of the average mill feed.

*Shipment No. 2.* This concentrate sample showed:

| | | | |
|---|---|---|---|
| Gold | 2·71 oz./ton | Silver | 5·11 oz./ton |
| Arsenic | 16·27 per cent | Copper | Not determined |
| Iron | 36·86 " | Lead | " |
| Sulphur | 33·34 " | Antimony | 0·58 per cent |
| Zinc | 1·29 " | Silica | 2·80 " |

*Shipment No. 3.* The mill tailing of this shipment showed:

| | |
|---|---|
| Gold | 0·0825 oz./ton |
| Silver | 0·57 " |

Metallic constituents not determined.

The gold ore sample showed on analysis:

| | | | |
|---|---|---|---|
| Gold | 0·305 oz./ton | Silver | 1·54 oz./ton |
| Arsenic | 2·56 per cent | Copper | Not determined |
| Iron | Not determined | Lead | 0·12 per cent |
| Sulphur | " | Antimony | 0·21 " |
| | | Zinc | 0·65 " |

*Shipment No. 4.* This concentrate showed:

| | | | |
|---|---|---|---|
| Gold | 3·04 oz./ton | Silver | 5·40 oz./ton |
| Arsenic | 10·95 per cent | Copper | 0·18 per cent |
| Iron | 27·60 " | Lead | 0·90 " |
| Sulphur | 26·20 " | Antimony | 0·68 " |
| Zinc | 3·40 " | Insoluble | 21·34 " |
| Lime (CaO) | 1·72 " | Magnesia | Present. |

*Shipment No. 5.* The sample of concentrate of this shipment assayed as follows:

| | | | |
|---|---|---|---|
| Gold | 1·24 oz./ton | Silver | 8·64 oz./ton |
| Arsenic | 9·32 per cent | Copper | 0·21 per cent |
| Iron | 28·13 " | Lead | 0·96 " |
| Sulphur | 27·29 " | Antimony | 0·89 " |
| Zinc | 3·15 " | Silica | 15·08 " |
| Lime | 1·10 " | Magnesia | 6·66 " |

The ore sample showed on analysis:

| | | | |
|---|---|---|---|
| Gold | 0·105 oz./ton | Silver | 0·88 oz./ton |
| Arsenic | 1·02 per cent | Copper | 0·01 per cent |
| Iron | 5·66 " | Lead | 0·13 " |
| Sulphur | 3·00 " | Antimony | 0·12 " |
| Zinc | 0·15 " | Insoluble | 46·34 " |
| Lime (CaO) | 9·27 " | Magnesia | 9·82 " |
| Alumina | 9·35 " | Bismuth | Nil |

*Shipment No. 6.* This ore showed:

| | |
|---|---|
| Gold | 0·415 oz./ton |
| Silver | 0·59 " |

## EXPERIMENTAL TESTS

### *Shipment No. 1

*Assay:*

| | |
|---|---|
| Gold | 1·30 oz./ton |
| Silver | 2·43 " |

### *Concentration Tests*

*Flotation.* A recovery of 92·9 per cent was made in a concentrate weighing 20·6 per cent of the feed. The concentrate assayed 5·0 ounces of gold per ton and the tailing, 0·10 ounce of gold per ton.

*Amalgamation.* By plate amalgamation the recovery was 26·2 per cent of the gold content. By barrel amalgamation the recovery was 56 per cent of the gold content.

*Cyanidation.* Grinding to —100 mesh, an extraction of around 87 per cent was obtained, with an average tailing of 0·175 ounce gold per ton.

*Flotation with Cyanidation of Tailing.* A flotation concentrate weighing 12·4 per cent of the feed and assaying 8·12 ounces of gold per ton recovered 83·3 per cent of the gold content. The flotation tailing at 0·23 ounce of gold was cyanided and a final tailing of 0·12 ounce of gold obtained. The cyanide extraction was 8 per cent and the total recovery 91·3 per cent. (Test No. 19.)

*Flotation and Blanketing.* The average of three tests showed a recovery by flotation and blanket concentration of 93·9 per cent of the gold. The flotation concentrate assayed from 4·48 to 5·27 ounces of gold per ton, with tailing of 0·06 ounce to 0·14 ounce. The combined concentrates comprised 22 to 29 per cent of the feed. (Tests Nos. 22, 26, 27.)

*Details of the test work on this shipment will be found in Invest. No. 589: Ore Dress. and Met., July-December 1934, Mines Branch, Dept. of Mines, Rept. 748, pp. 61–71 (1935).

### Shipment No. 2

*Cyanidation.* Straight cyanidation on −200-mesh concentrate gave 67·2 per cent extraction and a tailing of 0·89 ounce of gold per ton.

Cyanidation after pre-treatment with caustic soda and aluminium to break up antimony minerals gave an extraction of 55 per cent of the gold and a 1·22-ounce tailing.

*Roasting Concentrate and Cyaniding Calcine.* Roasting concentrate at low heat, grinding wet and filtering, and cyaniding ground calcine gave an extraction of 83·6 per cent of the gold and a tailing of 0·78 ounce.

### Shipment No. 3

*Mill Tailing*

*Assay:*

| | |
|---|---|
| Gold | 0·0825 oz./ton |
| Silver | 0·57 " |

*Flotation.* In Test No. 1, flotation on a sample ground 65 per cent −200 mesh recovered 21·3 per cent of the gold in 4·03 per cent of the feed as a concentrate assaying 0·45 ounce of gold per ton. The tailing was 0·07 ounce per ton. In Test No. 2, flotation on a sample ground 90·7 per cent −200 mesh recovered 25·1 per cent of the gold in 3·87 per cent of the feed as a concentrate assaying 0·54 ounce of gold per ton. The tailing was 0·065 ounce per ton.

A grain analysis of the sulphides showed:

| | |
|---|---|
| 75 per cent by volume | − 400 mesh |
| 48 " " | − 800 " |
| 32 " " | −1100 " |
| 17 " " | −1600 " |
| 8 " " | −2300 " |

89 per cent of the sulphides are free.
11 per cent of the sulphides are combined with gangue.

In the flotation concentrate no free gold was observed in Test No. 1, and two grains about 60 microns long and 6 to 8 microns in thickness were observed in the concentrate of Test No. 2.

*Ore Sample*

*Assay:*

| | |
|---|---|
| Gold | 0·305 oz./ton |
| Silver | 1·54 " |

*Cyanidation.* Several variations of the cyanide process were tried, with best results obtained on −100-mesh grind, the average extraction being 65·6 per cent with a tailing averaging 0·105 ounce of gold per ton.

*Flotation.* Grinding 91 per cent −200 mesh and floating, a concentrate weighing 20·3 per cent of the feed and assaying 1·39 ounces of gold per ton with a recovery of 93·4 per cent was obtained. The tailing assayed 0·025 ounce per ton.

*Roasting Concentrate and Cyaniding Calcine.* A quantity of concentrate was floated for roasting. The concentrate assayed 1·80 ounces of gold per ton and 9·36 ounces of silver per ton, copper 0·24 per cent, arsenic 6·07 per cent, lead 0·61 per cent, antimony 1·48 per cent, and zinc 2·55 per cent.

The concentrate was dead-roasted and water-washed. Analysis showed: gold 2·33 ounces per ton, silver 12·97 ounces per ton, copper 0·29 per cent, arsenic 0·60 per cent, lead 1·02 per cent, antimony 0·66 per cent, and zinc 2·80 per cent.

*Cyanidation.* Samples of the water-washed calcine were reground and cyanided. The gold extraction was 62·7 per cent and the silver 64·2 per cent.

**Shipment No. 4**

*Assay:*

| | |
|---|---|
| Gold | 3·04 oz./ton |
| Silver | 5·4 " |

*Cyanidation.* The concentrate was acid-washed with warm acid solution, water-washed, and cyanided. Cyaniding after a 45-minute grind gave a gold extraction of 68·12 per cent, and cyaniding after a 2-hour grind gave a gold extraction of 81·2 per cent. The tailing assays were 1·02 ounces and 0·56 ounce respectively.

*Roasting Concentrate and Cyaniding Calcine.* A series of tests, comprising 11 roasts of varying procedure with 28 cyanide tests of the calcines was made, and furnished some interesting results. Oxidizing roasting with 30-minute wet grinding and cyanidation with lime as protective alkali gave results of about 60 per cent extraction of gold. Neutral pulps gave extractions up to 83 per cent. One-hour grinding of the calcine increased the extraction to around 90 per cent.

Roasting with an addition of salt to the charge with one-hour grinding resulted in a further increase in extraction to 95 per cent. By passing the calcine over a blanket some 35 per cent of the gold is removed in a very high-grade concentrate.

The lowest tailing obtainable was 0·19 ounce per ton.

The highest silver extraction was 66 per cent and was obtained in a chloridizing roast test.

**Shipment No. 5**

*Concentrate*

*Assay:*

| | |
|---|---|
| Gold | 1·24 oz./ton |
| Silver | 8·64 " |

*Purpose of Test.* The purpose of the test work on this shipment was to determine if the middling product from floating the reground concentrate could be satisfactorily cyanided.

In refloating, a concentrate assaying 2·26 ounces of gold per ton was obtained with a middling assaying 0·67 ounce per ton, the ratio of concentration being 3·36 : 1.

Cyanidation of the middling gave a cyanide tailing of 0·45 ounce per ton. Cyanidation of the concentrate resulted in a cyanide tailing of 0·425 to 0·465 ounce per ton, showing an extraction of 65·7 to 62·5 per cent.

Roasting the concentrate and cyaniding the calcine after a short grind gave 80 per cent cyanide extraction.

Roasting with addition of salt improved extraction 5 per cent, and it is possible that longer grinding would result in a still higher cyanide extraction as in earlier test work.

*Ore Sample*

*Assay:*

| | |
|---|---|
| Gold | 0·105 oz./ton |
| Silver | 0·88 " |

This sample is much lower than the reported grade feed to mill.

Using caustic soda in place of sodium carbonate in flotation resulted in a cleaner higher grade concentrate and an increased recovery. A composite synthetic sample was prepared from this ore sample and the concentrate to give a 0·3 ounce of gold feed.

Comparative tests with soda ash and caustic soda showed a higher concentration and a higher recovery in favour of the latter.

**Shipment No. 6**

*Assay:*

| | |
|---|---|
| Gold | 0·415 oz./ton |
| Silver | 0·59 " |

*Purpose of Tests.* The purpose of the test work on this sample was to determine the maximum cyanide extraction obtainable. Recent tests on concentrate carried out by a neighbouring mill reported an extraction of slightly over 80 per cent of the gold.

*Cyanide Tests.* Cyanide tests on various sizes of the ore and under varying conditions were conducted. On 24-hour tests, grinding to −48 mesh, the gold extraction was 71 per cent; on −200-mesh material the extraction was 77 to 78 per cent. Cyaniding for 48 hours −48-mesh material showed 77 per cent extraction and −200 mesh 80·7 per cent extraction. The cyanide consumption on 24-hour −200-mesh material was 2·82 pounds of potassium cyanide per ton and on the 48-hour −200-mesh, 3·48 pounds of potassium cyanide per ton.

The fouling of the cyanide solution resulted from the formation of thiocyanate, ferrous salts, and copper salts.

*Amalgamation.* A barrel-amalgamation test showed a recovery of 53 per cent of the gold on a sample ground 63 per cent −200 mesh.

## DISCUSSION

From the microscopic examinations of the various samples submitted it is apparent that the metallic minerals and the modes of their occurrence are fairly consistent. The distribution of the metallic minerals is, however, somewhat erratic and spotty. The principal minerals in their order of abundance are: pyrite, arsenopyrite, pyrrhotite, sphalerite; with much lesser quantities of stibnite, tetrahedrite, chalcopyrite, native bismuth, an unknown mineral somewhat resembling jamesonite, and a mineral resembling galena. The gangue consists of quartz, siliceous rock, dolomite, and possibly other carbonates. Free native gold has been observed in an occasional hand specimen and also in two microscopic sections, and from the test work it is evident that the amount of free gold is variable.

The mode of occurrence of the silver has not been determined, nor has the silver mineral been isolated or identified.

The chemical analyses made of the various samples confirm the observations made above as to mineral occurrence and distribution.

The erratic and spotty nature of the mineral distribution and the varying and irregular occurrence of free native gold increase the difficulty of arriving at any definite conclusion as to the best mode of treatment to be employed, or as to the recoveries that can be maintained. The ore is not completely amenable to cyanidation, as a substantial proportion of the gold is locked up in the sulphides. Results of tests of various shipments have shown extractions ranging from 65 to 80 per cent of the gold by straight cyanidation.

The alternative, therefore, becomes one of concentration of the mineral constituents of the ore to produce a product having a value that will permit shipment to a smelter or treatment at site by means of roasting followed by cyanidation of the calcine. Laboratory tests have shown that 92 to 93 per cent of the precious metals can be obtained in a concentrate weighing about 20 per cent of the feed.

The grade (in gold) of this concentrate varies according to the grade of ore feed. For instance, on Shipment No. 1, with a feed of 1·30 ounces of gold per ton, a concentrate was obtained carrying 5·0 ounces of gold per ton, a recovery of 93 per cent; in Shipment No. 3, with an 0·30 ounce per ton feed, a concentrate carrying 1·39 ounces of gold per ton, a recovery of 93·4 per cent, was obtained; and in Shipment No. 5, with a feed of 0·105 ounce of gold per ton a concentrate carrying 0·54 ounce of gold, a recovery of 80 per cent, was obtained.

From the test work it may be deduced that with an ore feed around 0·5 ounce of gold per ton there would be no great difficulty in producing a primary concentrate carrying over 2 ounces of gold, with a reasonable recovery from the ore. With an ore feed between 0·2 ounce and 0·4 ounce of gold, a primary concentrate carrying less than 2 ounces of gold would be obtained, and with a feed below 0·2 ounce the concentrate would be under 1 ounce.

Recleaning the low-grade concentrate appears to be practical to obtain a shipping grade (over 2 ounces) of concentrate, but the greater the extent of recleaning the lower the recovery is likely to be. It is evident that the gold occurs as free native gold of varying sized particles and as gold locked in arsenopyrite and pyrite. The ore of higher grade probably carries proportionately more of the free gold than that of lower grade.

In the ores of Shipments Nos. 1 and 5 the ratio of pyrite to arsenopyrite is about 2 : 1. The ore sample of Shipment No. 3 was not fully analysed. In the sample of the concentrate of Shipment No. 5 the ratio of pyrite to arsenopyrite is similar; in Shipment No. 4 the ratio is 1·7 : 1, and in Shipment No. 2, which was a table concentrate, the ratio is not comparable.

Examination of the mill tailing sample in Shipment No. 3, by microscopic measurement of the sulphides, shows 48 per cent of the sulphides of a size −800 mesh, 32 per cent −1100 mesh, 17 per cent −1600 mesh, and 8 per cent −2300 mesh. It is estimated that 89 per cent of the sulphides occurring are free, with 11 per cent combined with gangue.

Flotation tests on this tailing, grinding to over 90 per cent −200 mesh, showed a recovery of only 25 per cent of the gold. The tailing dropped from 0·083 ounce to 0·065 ounce after re-treatment. Analysis of the final tailing was not made, but it was obvious that it contained sulphide minerals.

Treatment of the tailing with aqua regia, which dissolves the sulphides and gold contained therein, left a tailing assaying less than 0·01 ounce of gold. This indicates that a very minor proportion of the gold is contained in quartz or other gangue, and the non-recoverable gold is probably locked up in the very fine sulphides, which, being probably attached to gangue matter, may explain the difficulty in getting them to float. Even were they free, their fineness (−1100 mesh and under) would militate against their easy flotation. The conclusion is therefore that the tailing loss (in gold) will be governed by the amount of fine non-floatable sulphide in the ore feed. The maximum recovery should also be obtained at the finest practicable grinding. Talc in the ore increases the difficulty of obtaining a clean concentrate and also may have some effect (such as forming a coating of slime on fine sulphides) that decreases the flotability of the finer sulphides.

In recent test work on the ore sample of Shipment No. 5 the recovery by flotation was improved by some 5 per cent by using caustic soda instead of soda ash in the flotation. It is suggested that this procedure be tried in present plant operations.

Roasting the concentrate and cyaniding the calcine showed extraction up to 95 per cent of the gold. The latter figure was obtained by the special procedure of roasting with salt and grinding the calcine extremely fine. This procedure tends to improve the silver extraction also.

In following this procedure, however, it would be necessary to determine gold losses by volatilization, mechanical dusting, etc. Such losses will be governed by the type of roasting plant used and conditions pertaining thereto, and can not be determined with any accuracy under laboratory-scale conditions.

On Ore Sample (6) attention was directed entirely to the possibility of direct cyanidation, as requested by the company. The results indicated a maximum extraction of 80·7 per cent of the gold, but the cyanide consumption was high with excessive fouling of the solution. Amalgamation of this sample indicated, on a medium to fine grind, that 53 per cent of the gold was recoverable, so that at least 60 per cent of the gold content in this sample is in the nature of free gold.

## CONCLUSIONS

The difficulties associated with the treatment of this ore have been outlined above. The determining factor as to the most profitable mode of treatment depends upon the amount of free gold present in the ore.

A gold recovery of 92 per cent can be made by flotation, provided an ore feed averaging 0·4 ounce is procurable. With lower grade feed, necessitating recleaning of the concentrate, the recovery should be somewhat lower. The concentrate from flotation is not necessarily amenable to cyanidation, results varying according to the free gold content.

Roasting the concentrate and cyaniding the calcine gave a possible gold recovery of over 90 per cent on the calcine under special roasting and grinding conditions. With this procedure attention should be paid to possible losses through volatilization and dusting.

The possibility of direct cyanidation has received attention. Ore Sample (6), representing the latest feed ore, gave on direct cyanidation an extraction of 80 per cent of the gold. This is more promising and economical than floating a concentrate and shipping it to a smelter, or floating a concentrate, roasting, and cyaniding the calcine.

Direct cyanidation of the ore, however, involves high chemical consumption and entails the discarding of barren solution to obtain the maximum extraction of gold.

## Ore Dressing and Metallurgical Investigation No. 711

### GOLD ORE FROM THE KERR-ADDISON GOLD MINES, LIMITED, LARDER LAKE, ONTARIO

*Shipment.* A shipment of gold ore, consisting of 80 bags and weighing 10,652 pounds, was received on February 24, 1937, from the Kerr-Addison Gold Mines, Limited, Larder Lake, Ontario. The shipment was submitted by M. F. Fairlie, consulting engineer for the company, 12th Floor, Star Building, 80 King Street West, Toronto, Ontario.

*Characteristics of the Ore.* Six polished specimens were prepared and examined microscopically.

The *gangue* is mostly of a bright green colour and is complex in nature. The fine-textured and often schistose green material probably owes its colour to the presence of chrome mica (mariposite). This contains a considerable quantity of carbonate (probably dolomitic) and is cut by irregular stringers of white quartz.

Pyrite is the only *metallic mineral* present in noteworthy amount. It is sparsely disseminated as medium to small cubic crystals. A very small amount of chalcopyrite occurs as small irregular grains in the gangue and more rarely in the pyrite. Traces of two undetermined minerals occur as tiny grains in the gangue. A trace of bornite is associated with some of the chalcopyrite.

Tests of the two undetermined minerals failed to identify them, but it is possible that they may both be nickel minerals, for etch and optical tests suggest millerite (NiS) for one and gersdorffite (NiAsS) for the other.

Only one grain of native gold was seen in the sections examined. This is approximately 1100 mesh (13 microns) in size and occurs within the pyrite. Subsequent hydraulic classification and blanket concentration disclosed the presence of very fine visible grains of gold in the respective concentrates. Free gold, therefore, occurs, but its distribution in the ore is probably spotty.

*Sampling and Assaying.* The ore was crushed and sampled by standard methods. The analysis was as follows:

| | |
|---|---|
| Gold | 0·55 oz./ton |
| Silver | 0·07 " |
| Chromium | 0·15 " |

### EXPERIMENTAL TESTS

Standard cyanidation tests on different grinding sizes of dry-ground ore gave gold extractions of 97·27 per cent on −100-mesh feed (77 per cent −200-mesh) for 24-hour agitation at a 2: 1 pulp dilution. The consumption of cyanide was 0·76 pound per ton of ore. The tailings were gold, 0·015 ounce per ton.

On a cycle test, grinding in cyanide solution, the results were better. The tailing was lowered to 0·005 ounce gold per ton, showing an extraction of 99·09 per cent. There was no indication of serious fouling of the solution.

Settling tests indicated a satisfactory rate of settlement at a 2: 1 pulp dilution.

A large-scale semi-continuous cyanide test was run over a five-day period.

STANDARD CYANIDATION TESTS

*Tests Nos. 1 to 8*

Four samples of ore were ground dry to pass 48-, 100-, 150-, and 200-mesh screens respectively. Samples of each were cyanided in a solution of 2 to 1 dilution and a strength equivalent to 1 pound of potassium cyanide per ton for periods of 24 and 48 hours. Lime at the rate of 5 pounds per ton was added at the beginning of the test as protective alkalinity.

The results were as follows:

| Test No. | Agitation, hours | Mesh | Assay, Au, oz./ton | | Extraction of gold, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|---|
| | | | Feed | Tailing | | KCN | CaO |
| 1 | 24 | − 48 | 0·55 | 0·02 | 96·36 | 0·62 | 4·00 |
| 2 | 24 | −100 | 0·55 | 0·015 | 97·27 | 0·76 | 4·50 |
| 3 | 24 | −150 | 0·55 | 0·015 | 97·27 | 1·00 | 5·70 |
| 4 | 24 | −200 | 0·55 | 0·04 | 92·73 | 1·98 | 5·70 |
| 5 | 48 | − 48 | 0·55 | 0·04 | 92·73 | 0·72 | 4·10 |
| 6 | 48 | −100 | 0·55 | 0·015 | 97·27 | 0·96 | 5·40 |
| 7 | 48 | −150 | 0·55 | 0·015 | 97·27 | 1·20 | 7·50 |
| 8 | 48 | −200 | 0·55 | 0·03 | 94·54 | 2·30 | 7·60 |

*Screen Tests:*

| Mesh | Weight, per cent | | |
|---|---|---|---|
| | −48 mesh | −100 mesh | −150 mesh |
| + 65 | 11·5 | ........ | ........ |
| − 65+100 | 14·7 | ........ | ........ |
| −100+150 | 15·1 | 10·3 | ........ |
| −150+200 | 13·0 | 12·7 | 11·7 |
| −200 | 45·7 | 77·0 | 88·3 |
| Total | 100·0 | 100·0 | 100·0 |

The rise in the tailing on the −200-mesh ore is due probably to re-precipitation of gold caused by carbon or some other agent produced from the pulverizer at this fine stage of grinding.

## SETTLING TESTS

A sample of ore, weight 1,000 grammes, was ground in a cyanide-lime pulp (750 c.c. solution) to a fineness of approximately 77 per cent −200 mesh. The cyanide strength was 1 pound of potassium cyanide per ton. The pulp was made up to a dilution of 1·5 to 1 and transferred to a glass tube 2 inches inside diameter and the time of settling of the solids recorded for 1 hour.

| | |
|---|---|
| Overflow | Clear |
| Titration: KCN | 0·32 lb./ton |
| CaO | 0·84 " |
| Rate of settling | 0·39 ft./hour |

| Time | Settlement, feet | Cumulative settlement, feet |
|---|---|---|
| 5 minutes | 0·04 | 0·04 |
| 10 " | 0·04 | 0·08 |
| 15 " | 0·04 | 0·12 |
| 20 " | 0·03 | 0·15 |
| 25 " | 0·03 | 0·18 |
| 30 " | 0·03 | 0·21 |
| 35 " | 0·03 | 0·24 |
| 40 " | 0·03 | 0·27 |
| 45 " | 0·03 | 0·30 |
| 50 " | 0·03 | 0·33 |
| 55 " | 0·03 | 0·36 |
| 1 hour | 0·03 | 0·39 |

A second test of the same pulp was run, in which the dilution was made up to 2 to 1.

| | |
|---|---|
| Overflow | Clear |
| Titration: KCN | 0·24 lb./ton |
| CaO | 0·78 " |
| Rate of settling | 0·62 ft./hour |

| Time | Settlement, feet | Cumulative settlement, feet |
|---|---|---|
| 5 minutes | 0·06 | 0·06 |
| 10 " | 0·06 | 0·12 |
| 15 " | 0·05 | 0·17 |
| 20 " | 0·05 | 0·22 |
| 25 " | 0·05 | 0·27 |
| 30 " | 0·05 | 0·32 |
| 35 " | 0·05 | 0·37 |
| 40 " | 0·05 | 0·42 |
| 45 " | 0·05 | 0·47 |
| 50 " | 0·05 | 0·52 |
| 55 " | 0·05 | 0·57 |
| 1 hour | 0·05 | 0·62 |

The following three tests were run with the object of determining the amount and nature of the free gold in the ore.

### *Test No. 9*

This was a barrel-amalgamation test in which a sample of ore was ground to a fineness of about 77 per cent −200 mesh and amalgamated with 100 grammes of mercury for 1 hour.

| | |
|---|---|
| Gold in feed | 0·55 oz./ton |
| Gold in amalgamation tailing | 0·08 " |
| Recovery by amalgamation | 85·45 per cent |

### *Test No. 10*

A sample of ore was ground in a water pulp to about the same fineness as in Test No. 9 and passed over a hydraulic classifier. The oversize was panned to remove excess gangue. Fine grains of gold were visible in this product.

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| Feed | 100·00 | 0·49 (cal.) | 100·00 | |
| Oversize | 0·02 | 676·25 | 27·88 | 5,000 : 1 |
| Overflow | 99·98 | 0·35 | 72·12 | |

### *Test No. 11*

A sample of ore was ground as in Test No. 10 and the pulp passed over a corduroy blanket. The concentrate was panned to a small bulk and fine grains of gold were visible in this product.

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| Feed | 100·00 | 0·413 (cal.) | 100·00 | |
| Concentrate | 0·05 | 536·14 | 64·91 | 2,000 : 1 |
| Tailing | 99·95 | 0·145 | 35·09 | |

## CYANIDATION CYCLE TEST

A cycle test was carried out on the ore to determine if any minerals were present that would cause fouling of the solution. Five cycles were run and the method of procedure was as follows:

### *Cycle No. 1*

A charge of ore (1,100 grammes) was ground to a fineness of 78 per cent −200 mesh, with 5 pounds of lime per ton, in a cyanide solution of 1 pound of potassium cyanide per ton. The pulp dilution was 0·75 to 1. The ground pulp was filtered and five bottles made up each having a pulp dilution of 2 to 1. The solution used was composed of the pregnant solution and washings from the grinding mill.

Cyanide was added to each bottle to bring the cyanide strength to 1 pound of potassium cyanide per ton, and lime was added to provide protective alkalinity.

Agitation was carried out for 24 hours. The bottles were filtered and each cake washed with 45 c.c. water, which was added to the main filtrate (pregnant solution). This solution was treated with zinc dust and filtered (barren solution). The cakes were given further washings and assayed for gold. These washings were discarded.

*Cycle No. 2*

The barren solution, 750 c.c., from Cycle No. 1 was used in grinding with 1,000 grammes of fresh ore, cyanide being added to bring the strength up to 1 pound of potassium cyanide per ton. Grinding was carried out as in Cycle No. 1 and after filtering the pulp four bottles were made up for cyanidation. Washing of the grinding jar was done with barren solution from Cycle No. 1.

The bottles were made up to a dilution of 2 to 1 with the grinding solution and wash solution. The procedure was the same as in Cycle No. 1.

*Cycle No. 3*

One thousand grammes of fresh ore was ground with barren solution made up to 1 pound of potassium cyanide per ton and lime as in the previous cycle. Three bottles were made up and treated as above.

*Cycle No. 4*

One thousand grammes of fresh ore was ground as in the previous cycles and two bottles made up for 24 hours' agitation.

*Cycle No. 5*

One thousand grammes of fresh ore was ground as in previous cycles and one bottle made up for 24 hours' agitation.

The final solution from Cycle No. 5 was treated with zinc dust and analysed for cyanicides and fouling salts.

In all, 5,000 grammes of ore was treated, the same solution being used over again in each grinding and agitation cycle.

The assay results are as follows:

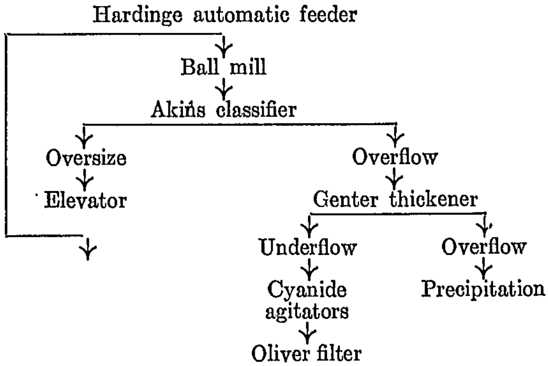
| Cycle No. | Average assays, Au, oz./ton | | Extraction of gold, per cent | Final titration, lb./ton solution | | Pulp dilution |
|---|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO | |
| 1 | 0·55 | 0·006 | 98·91 | 0·8 | 0·35 | 2 : 1 |
| 2 | 0·55 | 0·005 | 99·09 | 1·0 | 0·40 | 2 : 1 |
| 3 | 0·55 | 0·005 | 99·09 | 0·9 | 0·35 | 2 : 1 |
| 4 | 0·55 | 0·005 | 99·09 | 0·9 | 0·20 | 2 : 1 |
| 5 | 0·55 | 0·005 | 99·09 | 0·9 | 0·20 | 2 : 1 |

*Analysis of Final Solution:*

| | |
|---|---|
| Reducing power | 90 c.c. $\frac{N}{10}$ $KMnO_4$/litre |
| KCNS | 0·097 grm./litre |
| Chromium | Absent |
| Ferrous iron | Present |
| Copper | 0·01 grm./litre |

## *Mill Run*

A mill run was carried out on the ore using the following flow-sheet:




The ore was fed to the ball mill by a Hardinge automatic feeder at the rate of 150 pounds per hour. The ore feed was mixed with 5 pounds of lime per ton. Grinding was carried out in cyanide solution. The ball mill discharge was fed to an Akins classifier, the overflow of which was pumped to a Genter thickener. The average density of the classifier overflow was 17·3 per cent solids. The indicated average grinding, as shown by screen tests on classifier overflow, was as follows:

*Screen Test, Classifier Overflow:*

| Mesh | Weight, per cent |
|---|---|
| +100 | 2·3 |
| −100+150 | 13·2 |
| −150+200 | 15·6 |
| −200 | 68·9 |
| | 100·0 |

The underflow of the Genter thickener at a density of 52·5 per cent solids was fed to a Denver-Wallace agitator. The thickener overflow was pumped to the pregnant solution tank. For the first three days of the run the ball mill circuit was operated for 8 hours during each day. The agitator was run continuously and the tailing was filtered on an Oliver filter during the day's run only. On the last day a Pachuca tank was placed in series with the thickener and the agitator, and the entire unit was operated continuously for 24 hours. The total time of ball mill operation was 48 hours and of the agitator 144 hours.

The pregnant solution was fed over a weir to a mixing cone, to which zinc dust was added at a constant rate from a belt feeder. The solution was pumped through a small leaf filter press. The barren solution was pumped to a storage tank, from which a portion was used to wash the filter cake. The filtrate from the Oliver filter and the rest of the barren solution was returned to the ball mill. Strong cyanide solution was added from a solution feeder to maintain the cyanide strength in the grinding—classifier circuit at 1 pound of potassium cyanide per ton.

Lead acetate solution was fed to the mixing cone at the rate of 0·2 pound per ton of ore feed.

The amount of zinc dust added was as follows:

0·148 pound per ton of solution, which was equal to 0·117 pound per ton of ore feed.

The analysis of the solution on the final day of the run was as follows:

| | |
|---|---|
| Pregnant solution | 0·545 Au, oz./ton |
| Barren solution | 0·001 " |
| Efficiency of precipitation | 99·8 per cent |

Samples of the mill feed, classifier overflow, and filter cake (tailing) were taken every fifteen minutes during the runs. Assays were as follows:

| | |
|---|---|
| Average feed | 0·54 Au, oz./ton |
| Average classifier overflow | 0·144 " |
| Average tailing | 0·022 " |
| Tailing taken during emptying of agitator | 0·015 " |

In comparing the tailing results of the mill run with those of the small-scale tests it will be seen that the tailing from 24 hours' agitation on ore ground to a fineness of 45·7 per cent −200 mesh was 0·02 ounce gold per ton. At 77 per cent −200 mesh the tailing was 0·015 ounce gold per ton. In the cycle test the ore was ground in cyanide and the grinding was around 78 per cent −200 mesh, which gave a tailing of 0·005 ounce gold per ton. The method of running the mill test should be kept in mind when studying the results. The average tailing, 0·02 ounce gold per ton, given above is an arithmetical average of all the daily samples and includes the period during which the grinding circuit, thickener and filter were operated only during the day shift of eight hours. The process was therefore a semi-batch one for the first three days, and, although the theoretical displacement capacity of the single Wallace agitator was approximately 24 hours, there was undoubtedly short-circuiting of the pulp during the time the grinding circuit was in operation. To complete the run the whole operation was carried on continuously for a period of 24 hours. The daily sample from the last eight-hour period of this run was high, namely 0·04 ounce gold per ton; the maximum theoretical time in the agitators was 18 hours. However, during the period in which the agitators were slowly pumped empty, the filter cake samples assayed 0·015 ounce gold per ton. The period of emptying the agitators would approximate a total of 15 hours' additional agitation.

The results obtained from the continuous run indicate that every care should be taken to prevent short-circuiting in the agitators. A capacity for 30 hours of agitation should be provided and a larger number of small units would be preferable to a small number of large units. This is recommended to meet and overcome any problem arising from coarse gold.

The results further indicate that it would be advisable to grind somewhat finer than the grinding done in this test. Comparing the results of the small-batch tests it would appear that a grind of about 75 per cent through 200 mesh would give satisfactory results.

## Ore Dressing and Metallurgical Investigation No. 712

CHROMITE ORE FROM THE CHROMIUM MINING AND SMELTING CORPORATION, LIMITED, OBONGA LAKE, ONTARIO

*Shipments.* One box, containing 200 pounds of chromite ore marked "Type A" and 200 pounds marked "Type B", was received on December 7, 1936. On December 29, a further shipment of 6 boxes, containing 1,800 pounds of ore, arrived. This was made up of both types of material and one box marked "Type B", Coarse. The two shipments contained 900 pounds of Type A ore and 1,447 pounds of Type B ore.

The samples were sent by H. H. Merritt, Mine Superintendent, Chromium Mining and Smelting Corporation, Limited, Collins, Ontario, from the property situated in the Obonga Lake area, Thunder Bay district, Ontario.

In the year 1931, a shipment of ore from this property was received from D. A. Mutch. Analysis of the ore showed it to be low in grade, containing only 4·44 per cent $Cr_2O_3$. Concentration tests showed that the large quantity of talc in the ore made it impossible to obtain an economic recovery of chromite or a concentrate containing from 48 to 50 per cent $Cr_2O_3$.

The present shipments were made to obtain data as a guide to improving or revising the flow-sheet at present in use at the property and also to determine the degree of grinding required to handle a mixture of the coarse- and fine-grained types of ore.

*Characteristics of the Ore.* Polished sections of the three types of ore were prepared and examined microscopically in the Mineragraphic Laboratory to determine the grain size and character of the chromite.

*Type A ore* consists of gangue, chromite, and a very small quantity of sparsely disseminated hematite. If magnetite is present, it was not possible to differentiate it from the chromite.

The chromite is coarse-grained. It has been somewhat coarsely shattered and now appears as large angular fragments in the gangue. The chromite itself contains varying quantities of tiny inclusions of gangue material, possibly ferromagnesian minerals. The sizes of these inclusions range from a maximum of about 25 microns down to the tiniest size visible under high powers of magnification, the average size being somewhere in the neighbourhood of 1600 mesh (9 microns). As it cannot be hoped to free these inclusions from the chromite, measurements were carried out to determine the size of the composite chrome grains, i.e. the chromite with inclusions.

| | Per cent, by volume |
|---|---|
| Composite chrome grains | 52·1 |
| Matrix gangue | 47·4 |
| Hematite | 0·5 |
| | 100·0 |

*Grain Size of Composite Chrome and Matrix Gangue in Type A:*

(Percentages by Volume)

| Mesh | Composite chrome, per cent | Matrix gangue, per cent | Totals, per cent | Cumulative totals, per cent |
|---|---|---|---|---|
| + 48 | 33·16 | 31·89 | 65·05 | 65·05 |
| − 48+ 65 | 7·67 | 4·60 | 12·27 | 77·32 |
| − 65+100 | 3·62 | 3·45 | 7·07 | 84·39 |
| −100+150 | 3·40 | 2·93 | 6·33 | 90·72 |
| −150+200 | 2·44 | 2·49 | 4·93 | 95·65 |
| −200+280 | 0·90 | 1·07 | 1·97 | 97·62 |
| −280+400 | 0·62 | 0·79 | 1·41 | 99·03 |
| −400+560 | 0·51 | 0·41 | 0·92 | 99·95 |
| −560 | 0·05 | ........... | 0·05 | 100·00 |
| Totals | 52·37 | 47·63 | 100·00 | |

Assuming the specific gravity of the chromite to be 4·45 and that of the gangue to be 3·00, and that the composition of the chromite is 50 per cent $Cr_2O_3$ and 28 per cent FeO, calculations based on the above data give the following:

*Character of the Composite Chrome:*

| Material | Percentage by volume | Percentage by weight |
|---|---|---|
| Chromite | 80·2 | 85·7 |
| Inclusions | 19·8 | 14·3 |
| | 100·0 | 100·0 |

*Relationship of Chromite to Total Gangue:*

| Material | Percentage by volume | Percentage by weight |
|---|---|---|
| Chromite | 41·3 | 51·9 |
| Total gangue (inclusions and matrix) | 58·7 | 48·1 |
| | 100·0 | 100·0 |

On the basis of these figures, the composite chromite might be expected to carry approximately 43 per cent $Cr_2O_3$. Therefore, if perfect separation were possible in practice, this would be the best grade of concentrate possible.

*Type B ore* is quite different in character from Type A. The constituents are: gangue, chromite (any magnetite which may be present is undifferentiated), and extremely rare pyrite.

The chrome grains are largely equidimensional and show rough crystal outlines, which have apparently been somewhat rounded and in places have been indented and slightly replaced by gangue. All of the larger grains exhibit a border zone, which is slightly lighter in shade than the central cores. This zone is consistently about 25 microns in width, and thus grains that are less than about 50 microns in diameter do not show the darker cores. This zone is probably due to slightly different composition of the chromite and is regarded as representing a type of reaction shell.

In some places gangue inclusions like those so prominent in Type A are common, but in general this chromite is notably free from them.

*Grain Size of the Chromite in Type B Ore:*

| Mesh | Chromite, per cent | Chromite, cumulative per cent |
|---|---|---|
| + 48 | 4·8 | 4·8 |
| − 48+ 65 | 7·5 | 12·3 |
| − 65+100 | 9·7 | 22·0 |
| −100+150 | 20·0 | 42·0 |
| −150+200 | 19·7 | 61·7 |
| −200+280 | 13·5 | 75·2 |
| −280+400 | 10·8 | 86·0 |
| −400+560 | 9·3 | 95·3 |
| −560+800 | 4·5 | 99·8 |
| −800 | 0·2 | 100·0 |
| | 100·0 | |

The ore of Type B, Coarse, is identical in character to Type A.

*Sampling and Analysis.* The samples marked "Type A" and "Type B" were sampled separately and analysed. They were found to contain:

| | Type A, per cent | Type B, per cent |
|---|---|---|
| Chromium sesquioxide ($Cr_2O_3$) | 16·40 | 14·48 |
| Iron (Fe) | 9·79 | 11·27 |
| Lime (CaO) | 3·35 | 1·55 |
| Magnesia (MgO) | 24·27 | 23·34 |
| Sulphur (S) | 0·16 | 0·06 |
| Silica ($SiO_2$) | 22·08 | 25·80 |

## Experimental Tests

A few flotation tests were made to discover if any separation might be expected by this process. To get a concentration of the chromite it was found necessary to float off most or all of the gangue. As the gangue is very talcy, the froth was voluminous and the grade of the finished product was only 20 to 25 per cent $Cr_2O_3$: the recovery was about 85 per cent.

Small-scale tests on classified products showed recoveries of from 65 to 72 per cent of the chromite in concentrates of about 42 per cent of $Cr_2O_3$.

Continuous mill runs indicated recoveries of 78 per cent and much the same grade of concentrate as was obtained in small-scale tests.

Most of the tests were made on Type B ore, which was finer grained than Type A. Some later tests and mill runs were made on a mixture of the two samples.

The tests in detail follow:

## FLOTATION

Six tests were made on samples of Type B ore ground in a ball mill to pass 80 per cent −200 mesh. Various reagent combinations were tried. In all cases it was found that over 60 per cent of the weight of feed floated. This material contained from 8 to 12 per cent of $Cr_2O_3$. The residue not floated contained 23 per cent of $Cr_2O_3$.

## TABLE CONCENTRATION OF CLASSIFIED FEED, TYPE B

### *Test No. 1*

Samples of the ore were ground to pass 8, 14, 48, 65, and 100 meshes, and were then run through a pulsating classifier that produced three products, i.e. Spigot No. 1, Spigot No. 2, and Classifier Overflow. Each of these products was passed over a Wilfley table and a concentrate and tailing obtained. No separation was obtained on the −8-mesh feed.

*Results: —14-Mesh Feed:*

| Product | Weight, per cent | Assay, per cent | | $Cr_2O_3$ distribution, per cent |
|---|---|---|---|---|
| | | $Cr_2O_3$ | Fe | |
| Feed (calculated) | 100·0 | 14·70 | 12·04 | 100·0 |
| Table concentrate, Spigot No. 1 | 16·4 | 20·28 | 14·64 | 22·6 |
| Table tailing, Spigot No. 1 | 13·9 | 12·87 | 10·75 | 12·2 |
| Table concentrate, Spigot No. 2 | 5·4 | 31·21 | 22·39 | 11·5 |
| Table tailing, Spigot No. 2 | 20·1 | 10·89 | 10·35 | 14·9 |
| Table concentrate, classifier overflow | 8·9 | 36·12 | 28·66 | 21·8 |
| Table tailing, classifier overflow | 35·3 | 7·08 | 6·53 | 17·0 |
| Total concentrates | 30·7 | 26·80 | 20·10 | 55·9 |

*Screen Tests, Weight, Per Cent:*

| Mesh | Spigot No. 1 | | Spigot No. 2 | | Classifier overflow | |
|---|---|---|---|---|---|---|
| | Table concentrate | Table tailing | Table concentrate | Table tailing | Table concentrate | Table tailing |
| − 14+ 20 | 33·3 | 73·2 | ........... | 4·5 | 2·8 | 1·1 |
| − 20+ 28 | 27·4 | 17·3 | 2·1 | 23·5 | 2·7 | 3·2 |
| − 28+ 35 | 18·5 | 6·5 | 10·9 | 35·8 | 2·9 | 5·9 |
| − 35+ 48 | 11·8 | 2·1 | 31·1 | 23·9 | 3·5 | 6·4 |
| − 48+ 65 | 4·3 | 0·5 | 20·5 | 6·2 | 4·8 | 5·5 |
| − 65+100 | 2·6 | 0·2 | 19·7 | 2·3 | 19·8 | 11·9 |
| −100+150 | 1·5 | 0·1 | 12·0 | 1·7 | 39·3 | 12·6 |
| −150+200 | 0·5 | ........... | 3·1 | 1·0 | 18·9 | 9·9 |
| −200 | 0·1 | ........... | 0·6 | 1·1 | 5·3 | 43·5 |
| | 100·0 | 100·0 | 100·0 | 100·0 | 100·0 | 100·0 |

An examination of the products from the screen tests showed that the chromite grains in the concentrates were not freed from the gangue under 65-mesh size. The −65-mesh sizes of the table tailings contained small crystals of chromite.

The recovery of chromite was poor and the concentrates produced in this test were low grade.

*Results: −48-Mesh Feed:*

| Product | Weight, per cent | Assay, per cent | | $Cr_2O_3$ distribution, per cent |
|---|---|---|---|---|
| | | $Cr_2O_3$ | Fe | |
| Feed (calculated) | 100·0 | 14·69 | 12·56 | 100·0 |
| Table concentrate, Spigot No. 1 | 5·0 | 40·10 | 32·60 | 13·7 |
| Table tailing, Spigot No. 1 | 7·1 | 15·44 | 12·80 | 7·5 |
| Table concentrate, Spigot No. 2 | 4·5 | 41·80 | 35·60 | 12·8 |
| Table tailing, Spigot No. 2 | 11·1 | 13·50 | 12·80 | 10·2 |
| Table concentrate, classifier overflow | 8·5 | 41·00 | 32·00 | 23·7 |
| Table tailing, classifier overflow | 63·8 | 7·40 | 6·70 | 32·1 |
| Total concentrates | 18·0 | 40·95 | 33·00 | 50·2 |

*Screen Tests, Weight, Per Cent:*

| Mesh | Spigot No. 1 | | Spigot No. 2 | | Classifier overflow | |
|---|---|---|---|---|---|---|
| | Table concentrate | Table tailing | Table concentrate | Table tailing | Table concentrate | Table tailing |
| − 48+ 65 | 7·4 | 27·6 | 1·2 | 13·3 | 2·0 | 3·8 |
| − 65+100 | 29·3 | 50·8 | 15·4 | 45·5 | 11·9 | 17·3 |
| −100+150 | 49·9 | 14·5 | 49·6 | 24·4 | 33·7 | 22·4 |
| −150+200 | 11·4 | 2·8 | 26·7 | 6·9 | 36·3 | 12·3 |
| −200 | 2·0 | 4·3 | 7·1 | 9·9 | 16·1 | 44·2 |

The grade of concentrates obtained from ore ground −48 mesh is considerably higher than those from ore ground −14 mesh. The recovery is still low.

*Results: −65-Mesh Feed:*

| Product | Weight, per cent | Assay, per cent | | $Cr_2O_3$ distribution, per cent |
|---|---|---|---|---|
| | | $Cr_2O_3$ | Fe | |
| Feed (calculated) | 100·0 | 14·09 | 11·72 | 100·0 |
| Table concentrate, Spigot No. 1 | 4·3 | 42·25 | 30·88 | 12·9 |
| Table tailing, Spigot No. 1 | 3·8 | 17·86 | 13·54 | 4·8 |
| Table concentrate, Spigot No. 2 | 4·3 | 41·66 | 30·12 | 12·7 |
| Table tailing, Spigot No. 2 | 5·5 | 10·52 | 9·38 | 4·1 |
| Table concentrate, classifier overflow | 13·5 | 41·40 | 30·88 | 39·7 |
| Table tailing, classifier overflow | 68·6 | 5·31 | 5·68 | 25·8 |
| Total concentrates | 22·1 | 41·62 | 30·73 | 65·3 |

*Screen Tests, Weight, Per Cent:*

| Mesh | Spigot No. 1 | | Spigot No. 2 | | Classifier overflow | |
|---|---|---|---|---|---|---|
| | Table concentrate | Table tailing | Table concentrate | Table tailing | Table concentrate | Table tailing |
| − 65+100 | 14·8 | 26·2 | 9·9 | 10·5 | 4·4 | 8·4 |
| −100+150 | 52·9 | 38·4 | 48·2 | 34·4 | 40·5 | 13·5 |
| −150+200 | 24·0 | 15·9 | 29·7 | 21·0 | 38·6 | 14·8 |
| −200 | 8·3 | 19·5 | 12·2 | 34·1 | 16·5 | 63·3 |

This test shows that treating −65-mesh material yields a somewhat higher grade concentrate and a considerable increase in recovery over previous tests.

*Results: −100-Mesh Feed:*

| Product | Weight, per cent | Assay, per cent | | $Cr_2O_3$ distribution, per cent |
|---|---|---|---|---|
| | | $Cr_2O_3$ | Fe | |
| Feed (calculated) | 100·0 | 14·37 | 12·16 | 100·0 |
| Table concentrate, Spigot No. 1 | 4·8 | 42·36 | 31·42 | 14·1 |
| Table tailing, Spigot No. 1 | 2·4 | 18·22 | 12·68 | 3·0 |
| Table concentrate, Spigot No. 2 | 4·2 | 42·14 | 31·64 | 12·3 |
| Table tailing, Spigot No. 2 | 2·9 | 10·34 | 9·94 | 2·2 |
| Table concentrate, classifier overflow | 14·9 | 40·94 | 30·72 | 42·4 |
| Table tailing, classifier overflow | 70·8 | 5·27 | 5·88 | 26·0 |
| Total concentrates | 23·9 | 41·44 | 31·02 | 68·8 |

*Screen Tests, Weight, Per Cent:*

| Mesh | Spigot No. 1 | | Spigot No. 2 | | Classifier overflow | |
|---|---|---|---|---|---|---|
| | Table concentrate | Table tailing | Table concentrate | Table tailing | Table concentrate | Table tailing |
| −100+150 | 48·9 | 22·8 | 30·0 | 12·6 | 26·4 | 2·9 |
| −150+200 | 38·7 | 23·8 | 49·2 | 22·8 | 36·7 | 9·6 |
| −200 | 12·4 | 53·4 | 20·8 | 64·6 | 36·9 | 87·5 |

These tests show that as the fineness of grinding increases the recovery also increases. The grade of concentrate remains much the same, i.e. 40 to 41·5 per cent of $Cr_2O_3$.

## Table Concentration, Types A and B Mixed

### *Test No. 2*

The two types of ore were mixed in equal proportions and a series of tests similar to the preceding series was made on this mixture.

*Results: —48-Mesh Feed:*

| Product | Weight, per cent | Assay, per cent | | $Cr_2O_3$ distribution, per cent |
|---|---|---|---|---|
| | | $Cr_2O_3$ | Fe | |
| Feed (calculated) | 100·0 | 13·01 | 10·01 | 100·0 |
| Table concentrate, Spigot No. 1 | 6·0 | 38·56 | 23·94 | 17·8 |
| Table tailing, Spigot No. 1 | 7·0 | 8·18 | 8·82 | 4·4 |
| Table concentrate, Spigot No. 2 | 4·3 | 40·20 | 25·56 | 13·3 |
| Table tailing, Spigot No. 2 | 6·6 | 8·54 | 8·82 | 4·3 |
| Table concentrate, classifier overflow | 9·0 | 40·16 | 25·94 | 27·8 |
| Table tailing, classifier overflow | 67·1 | 6·28 | 5·88 | 32·4 |
| Total concentrates | 19·3 | 39·67 | 25·23 | 58·9 |

*Screen Tests, Weight, Per Cent:*

| Mesh | Spigot No. 1 | | Spigot No. 2 | | Classifier overflow | |
|---|---|---|---|---|---|---|
| | Table concentrate | Table tailing | Table concentrate | Table tailing | Table concentrate | Table tailing |
| — 48+ 65 | 6·1 | 27·3 | 4·8 | 11·9 | 1·9 | 0·9 |
| — 65+100 | 44·1 | 42·2 | 27·2 | 31·0 | 17·7 | 9·1 |
| —100+150 | 34·2 | 17·3 | 41·6 | 24·8 | 35·6 | 13·3 |
| —150+200 | 13·0 | 5·7 | 19·4 | 12·8 | 28·5 | 7·8 |
| —200 | 2·6 | 7·5 | 7·0 | 19·5 | 16·3 | 68·9 |

Grinding at —48 mesh is not fine enough to obtain a high-grade concentrate or a high recovery.

*Results: —65-Mesh Feed:*

| Product | Weight, per cent | Assay, per cent | | $Cr_2O_3$ distribution, per cent |
|---|---|---|---|---|
| | | $Cr_2O_3$ | Fe | |
| Feed (calculated) | 100·0 | 16·95 | 12·14 | 100·0 |
| Table concentrate, Spigot No. 1 | 8·3 | 41·18 | 23·04 | 20·2 |
| Table tailing, Spigot No. 1 | 1·7 | 12·98 | 10·79 | 1·3 |
| Table concentrate, Spigot No. 2 | 3·2 | 42·48 | 26·22 | 8·0 |
| Table tailing, Spigot No. 2 | 2·7 | 9·58 | 6·62 | 1·5 |
| Table concentrate, classifier overflow | 17·9 | 41·90 | 26·22 | 44·2 |
| Table tailing, classifier overflow | 66·2 | 6·34 | 6·54 | 24·8 |
| Total concentrates | 29·4 | 41·75 | 25·32 | 72·4 |

*Screen Tests, Weight, Per Cent:*

| Mesh | Spigot No. 1 | | Spigot No. 2 | | Classifier overflow | |
|---|---|---|---|---|---|---|
| | Table concentrate | Table tailing | Table concentrate | Table tailing | Table concentrate | Table tailing |
| — 65+100 | 57·2 | 36·9 | 6·0 | 6·0 | 3·5 | 2·0 |
| —100+150 | 31·8 | 15·7 | 46·0 | 27·2 | 39·1 | 9·9 |
| —150+200 | 8·4 | 11·9 | 32·4 | 20·7 | 30·8 | 8·6 |
| —200 | 2·6 | 35·5 | 15·6 | 46·1 | 26·6 | 79·5 |

When the ore is ground to pass 65 mesh, the grade of concentrate rises, and the recovery also is increased.

*Results: —100-Mesh Feed:*

| Product | Weight, per cent | Assay, per cent | | $Cr_2O_3$ distribution, per cent |
|---|---|---|---|---|
| | | $Cr_2O_3$ | Fe | |
| Feed (calculated) | 100·0 | 14·84 | 11·17 | 100·0 |
| Table concentrate, Spigot No. 1 | 5·1 | 42·04 | 27·12 | 14·4 |
| Table tailing, Spigot No. 1 | 3·6 | 8·82 | 8·42 | 2·1 |
| Table concentrate, Spigot No. 2 | 4·3 | 40·88 | 26·54 | 11·8 |
| Table tailing, Spigot No. 2 | 2·6 | 8·02 | 8·22 | 1·4 |
| Table concentrate, classifier overflow | 13·9 | 41·14 | 27·32 | 38·6 |
| Table tailing, classifier overflow | 70·5 | 6·66 | 6·14 | 31·7 |
| Total concentrates | 23·3 | 41·29 | 27·13 | 64·8 |

*Screen Tests, Weight, Per Cent:*

| Mesh | Spigot No. 1 | | Spigot No. 2 | | Classifier overflow | |
|---|---|---|---|---|---|---|
| | Table concentrate | Table tailing | Table concentrate | Table tailing | Table concentrate | Table tailing |
| —100+150 | 53·0 | 19·0 | 58·6 | 22·4 | 23·8 | 8·8 |
| —150+200 | 33·6 | 23·3 | 28·2 | 15·2 | 39·2 | 7·2 |
| —200 | 13·4 | 57·7 | 13·2 | 62·4 | 37·0 | 84·0 |

It is apparent that finer grinding than —65-mesh does not increase the grade of concentrate. The recovery also is decreased.

*Results: —150-Mesh Feed:*

| Product | Weight, per cent | Assay, per cent | | $Cr_2O_3$ distribution, per cent |
|---|---|---|---|---|
| | | $Cr_2O_3$ | Fe | |
| Feed (calculated) | 100·0 | 14·76 | 12·68 | 100·0 |
| Table concentrate, Spigot No. 1 | 0·8 | 42·58 | 28·22 | 2·4 |
| Table tailing, Spigot No. 1 | 1·7 | 12·76 | 10·50 | 1·5 |
| Table concentrate, Spigot No. 2 | 1·3 | 42·54 | 28·12 | 3·7 |
| Table tailing, Spigot No. 2 | 3·2 | 13·08 | 10·05 | 2·8 |
| Table concentrate, classifier overflow | 11·3 | 41·32 | 27·52 | 31·6 |
| Table tailing, classifier overflow | 81·7 | 10·48 | 10·39 | 58·0 |
| Total concentrates | 13·4 | 41·51 | 27·62 | 37·7 |

*Screen Tests: Weight, Per Cent:*

| Mesh | Spigot No. 1 | | Spigot No. 2 | | Classifier overflow | |
|---|---|---|---|---|---|---|
| | Table concentrate | Table tailing | Table concentrate | Table tailing | Table concentrate | Table tailing |
| —150+200 | 29·7 | 1·4 | 30·5 | 1·2 | 25·2 | 1·0 |
| —200 | 70·3 | 98·6 | 69·5 | 98·8 | 74·8 | 99·0 |

The recovery from ore ground −150 mesh is much lower than that from −100-mesh material.

The degree of grinding is important, as indicated by the results of the above tests. Coarser than 65 mesh does not yield good recovery, owing to unliberated grains of chromite, whereas finer grinding than 100 mesh causes higher losses in the slimes.

## TABLE CONCENTRATION OF SIZED FEED

### *Test No. 3*

To note any difference between tabling a classified feed and a sized feed, a sample was ground dry to pass 65 mesh and then passed through 100-, 150-, and 200-mesh screens. The portion remaining on each screen was then tabled.

This screening operation split the feed in the following proportions:

| Mesh | Weight, per cent |
|---|---|
| − 65+100 | 27·9 |
| −100+150 | 20·5 |
| −150+200 | 13·7 |
| −200 | 37·9 |
| | 100·0 |

*Table Concentration:*

| Product | Weight, per cent | $Cr_2O_3$ Assay, per cent | $Cr_2O_3$ Distribution, per cent |
|---|---|---|---|
| *−65+100 Mesh:* | | | |
| Feed (calculated) | 100·0 | 21·23 | 100·0 |
| Table concentrate | 40·4 | 42·80 | 81·4 |
| Table tailing | 59·6 | 6·61 | 18·6 |
| *−100+150 Mesh:* | | | |
| Feed (calculated) | 100·0 | 22·11 | 100·0 |
| Table concentrate | 42·7 | 42·70 | 82·4 |
| Table middling | 7·7 | 18·46 | 6·4 |
| Table tailing | 49·6 | 4·96 | 11·2 |
| *−150+200 Mesh:* | | | |
| Feed (calculated) | 100·0 | 21·17 | 100·0 |
| Table concentrate | 33·7 | 42·92 | 68·3 |
| Table middling | 10·8 | 36·64 | 18·7 |
| Table tailing | 55·5 | 4·96 | 13·0 |
| *−200 Mesh:* | | | |
| Feed (calculated) | 100·0 | 9·04 | 100·0 |
| Table concentrate | 7·3 | 41·16 | 31·9 |
| Table tailing | 92·7 | 6·64 | 68·1 |

*Compilation of Results:*

| Product | Weight, per cent | Assay, $Cr_2O_3$, per cent | Distribution, $Cr_2O_3$, per cent |
|---|---|---|---|
| Feed (calculated) | 100·0 | 16·86 | 100·0 |
| — 65-mesh concentrate | 11·3 | 42·80 | 28·7 |
| —100-mesh " | 8·7 | 42·70 | 22·0 |
| —150-mesh " | 4·6 | 42·92 | 11·7 |
| —200-mesh " | 2·9 | 41·16 | 7·1 |
| Total concentrate | 27·5 | 42·62 | 69·5 |
| Total tailing and middling | 72·5 | 7·09 | 30·5 |

The results of this test, when compared with those obtained on —65-mesh classified, show about 3 per cent lower recovery when sized products are concentrated.

## MILL RUNS

The remainder of the ore, 2,000 pounds in weight, was used in continuous runs, the flow-sheets of which were arranged to give much the same treatment as in the small-scale tests on classified feeds.

The ore was fed to a small ball mill which discharged into a hydraulic jig. The jig tailing was passed over a Hummer screen with a 28-mesh opening and the screen oversize was returned to the ball mill. The —28-mesh product flowed to a two-spigot hydraulic classifier, where two sand products were made. The slime overflow passed to a cone for thickening. The three products were tabled, the first sand on a ¼-size Butchart table, the second on a ¼-size Plato table, and the cone underflow on a full-sized Wilfley table. A middling product was cut off the two sand-tables and pumped to an Akins classifier, where the sand was returned to the ball mill, and the slime overflow was passed back to the slime thickening cone. The overflow from this cone was run to waste.

### *Mill Run No. 1*

The ore was fed at the rate of 200 pounds per hour. The ball mill contained a light load of balls, to avoid overgrinding of the chromite grains. It was found that the jig was not balanced with the rate of feed, and although coarse particles of clean chromite accumulated, it was impossible to maintain a uniform discharge of high-grade material. The jig, therefore, was taken out of the circuit and the ball mill discharge was passed directly over the 28-mesh screen.

The overflow from the cone that thickened the slime overflow from the hydraulic classifier was practically clear, containing a small quantity of very fine chromite.

*Assays:*

| — | $Cr_2O_3$, per cent | Fe, per cent |
|---|---|---|
| Feed | 14·89 | 11·10 |
| Ball mill discharge | 14·31 | 10·59 |
| Classifier feed, −28 mesh | 12·66 | 10·39 |
| Spigot No. 1—Butchart table feed | 17·71 | 12·88 |
| " " concentrate | 42·18 | 26·46 |
| " " tailing | 4·68 | 5·73 |
| Spigot No. 2—Plato table feed | 16·16 | 12·72 |
| " " concentrate | 43·22 | 30·22 |
| " " tailing | 4·50 | 5·63 |
| Cone underflow—Wilfley table feed | 9·48 | 9·08 |
| " " concentrate | 37·94 | 26·76 |
| " " tailing | 4·20 | 5·37 |
| Cone overflow | 14·11 | 13·54 |

*Screen Tests, Weight, Per Cent:*

| Mesh | Ball mill discharge | Classifier feed, −28 mesh | Spigot No. 1, Butchart feed | Spigot No. 2, Plato feed | Cone underflow, Wilfley feed |
|---|---|---|---|---|---|
| + 6 | 3·0 | ........ | ........ | ........ | ........ |
| − 6+ 8 | 6·1 | ........ | ........ | ........ | ........ |
| − 8+ 10 | 11·4 | ........ | ........ | ........ | ........ |
| − 10+ 14 | 10·8 | ........ | ........ | ........ | ........ |
| − 14+ 20 | 10·2 | ........ | ........ | ........ | ........ |
| − 20+ 28 | 7·5 | ........ | ........ | ........ | ........ |
| − 28+ 35 | 8·3 | 7·8 | 17·0 | 3·4 | ........ |
| − 35+ 48 | 7·2 | 9·5 | 20·8 | 8·0 | 0·1 |
| − 48+ 65 | 6·0 | 14·5 | 18·2 | 12·9 | 2·3 |
| − 65+100 | 7·9 | 14·8 | 15·4 | 18·5 | 9·5 |
| −100+150 | 7·9 | 15·0 | 13·3 | 20·5 | 20·8 |
| −150+200 | 2·9 | 8·2 | 4·0 | 10·5 | 15·2 |
| −200 | 10·7 | 30·2 | 11·3 | 26·2 | 52·1 |

*Mill Run No. 2*

In the previous run the harder portions of the ore were accumulating in the grinding mill. The weight of balls was therefore increased and the rate of feed stepped up to 250 pounds per hour, when the sand concentrating tables performed better with the heavier feed rate. In this run, the underflow from the cone that thickened the classifier slime overflow was pulled fast enough to prevent an overflow from the cone.

*Assays:*

| — | $Cr_2O_3$, per cent | Fe, per cent |
|---|---|---|
| Feed | 15·82 | 12·38 |
| Ball mill discharge | 15·76 | 12·26 |
| Classifier feed, −28 mesh | 15·92 | 12·38 |
| Spigot No. 1—Butchart table feed | 23·66 | 12·98 |
| " " concentrate | 43·30 | 25·56 |
| " " tailing | 5·14 | 6·58 |
| Spigot No. 2—Plato table feed | 20·28 | 15·22 |
| " " concentrate | 43·98 | 29·72 |
| " " tailing | 4·48 | 6·48 |
| Cone underflow—Wilfley table feed | 8·78 | 8·52 |
| " " concentrate | 42·00 | 29·92 |
| " " tailing | 4·44 | 4·96 |

*Screen Tests, Weight, Per Cent:*

| Mesh | Ball mill discharge | Classifier feed, −28 mesh | Spigot No. 1, Butchart | Spigot No. 2, Plato feed | Cone underflow, Wilfley table feed |
|---|---|---|---|---|---|
| + 6 | 1·6 | ........ | ........ | ........ | ........ |
| − 6+ 8 | 4·9 | ........ | ........ | ........ | ........ |
| − 8+ 10 | 8·9 | ........ | ........ | ........ | ........ |
| − 10+ 14 | 11·7 | ........ | ........ | ........ | ........ |
| − 14+ 20 | 14·0 | ........ | ........ | ........ | ........ |
| − 20+ 28 | 8·0 | ........ | ........ | ........ | ........ |
| − 28+ 35 | 10·2 | 7·7 | 36·6 | 5·9 | ........ |
| − 35+ 48 | 8·4 | 12·9 | 36·2 | 12·8 | 0·4 |
| − 48+ 65 | 5·5 | 13·0 | 18·0 | 19·6 | 1·6 |
| − 65+100 | 5·4 | 11·7 | 7·1 | 22·7 | 6·7 |
| −100+150 | 4·8 | 12·0 | 1·6 | 20·2 | 9·8 |
| −150+200 | 3·6 | 8·8 | 0·2 | 8·4 | 8·9 |
| −200 | 13·0 | 33·9 | 0·1 | 10·4 | 72·6 |

After the conclusion of the run, the machines were cleaned out and all material run over the −28-mesh screen and over the tables. The concentrates from the two runs were combined, weighed, and assayed, also any middling and oversize left from the clean-up. Owing to spills and losses at various points, the actual recovery of concentrate was low compared to that shown by assays.

*Results:*

| — | Weight, pounds | Assay, per cent | |
|---|---|---|---|
| | | $Cr_2O_3$ | Fe |
| Butchart table concentrates | 85·0 | 41·36 | 28·54 |
| Plato table concentrates | 202·3 | 42·92 | 30·22 |
| Wilfley table concentrates | 103·0 | 41·22 | 29·64 |
| Middling and oversize | 48·0 | 17·98 | 13·42 |
| Total concentrate | 390·3 | 42·13 | 39·49 |

Average tailing from—

| | |
|---|---|
| Mill Runs Nos. 1 and 2 | 4·57 $Cr_2O_3$, per cent |
| Recovery based on assays | 78·5 per cent |
| Ratio of concentration | 3·5 : 1 |

## SUMMARY AND CONCLUSIONS

Small-scale tests made on Type B ore show that the ore requires to be ground −65 mesh before the larger chromite grains are freed from gangue. A slightly higher recovery is indicated when the grinding is −100 mesh.

Tests on a mixture of coarse-grained Type A ore and fine-grained Type B ore show again that −65 mesh is the critical point of grinding. When the fineness of grinding is increased, recovery falls off.

Mill Run No. 2 shows that the coarser sand product from Spigot No. 1 containing 90·8 per cent of −28+65-mesh material gives a tailing containing 5·14 per cent of $Cr_2O_3$, whereas the product from Spigot No. 2, containing 38·3 per cent −28+65 material, shows a tailing of 4·48 per cent of $Cr_2O_3$. The concentrates in both cases are over 43 per cent of $Cr_2O_3$. The coarse grains of chromite from Type A ore doubtless are the reason for the grade of concentrate obtained from the coarser grind.

A mixture of ores of these types lends itself to a flow-sheet in which the ore is coarsely ground and the larger particles of clean chromite removed by jigs placed at the discharge of the grinding mill. As the ore contains chromite grains ranging from coarse to microscopic size, a system of tables designed to treat these sizes is imperative. These tables therefore must be fed from a system of efficient classification to obtain maximum recovery.

The grade of concentrate to be expected will be about 42 per cent $Cr_2O_3$. This point is clearly shown by the microscopic examination. The chromite grains have inclusions of gangue that hold the $Cr_2O_3$ content down to 42 to 43 per cent.

The results of this investigation indicate that this grade of concentrate can be obtained with about 78 per cent recovery, but only if the flow-sheet used in the mill test, and briefly outlined previously in this report, be followed closely in principle.

The following are the essential points: (1) Although the small-scale tests show that the critical point of grinding is 65 mesh, not all the ore should be ground to that fineness before concentration. (2) As the chromite mineral tends to overgrind and to slime, it should be concentrated out as soon as it is freed.

In order to meet the above conditions a rod mill should be used (as it produces less slime than a ball mill) to grind the ore from either $\frac{3}{4}$-inch or $\frac{1}{2}$-inch feed to a final tailing product containing all material passing a 65-mesh screen. A jig or jigs should be placed to concentrate the ore as it comes from the rod mill. The jig tailing should be screened on about a 20-mesh screen and the oversize returned to the mill. The throughs from the screen should be classified in a hydraulic-type classifier to obtain a classified feed for concentration on tables. At least three spigot sizes are recommended. A clean concentrate should be taken from each table and a middling cut should be returned for regrinding. On the coarser-sized classified products, it will be necessary to return both the tailing and middling for regrinding. These products must be dewatered before regrinding, and for this purpose an Akins or a Dorr type of classifier is recommended.

The regrinding can be done either in a separate rod mill or in the original mill. If the products are returned to the original mill the grinding capacity of that mill will be greatly reduced. The slimes from all classifiers should be settled either in cones or in rake thickeners and the thickened product fed to slime tables.

At some later date it may be found advisable to thicken the slime tables tailing again and to concentrate it by flotation.

## Ore Dressing and Metallurgical Investigation No. 713

GOLD-COPPER ORE FROM THE SLAVE LAKE GOLD MINES, LIMITED, GREAT SLAVE LAKE, N.W.T.

*Shipment.* A shipment of 15 sacks of ore, total weight 1,063 pounds, was received on February 24, 1937, from Alphonse A. Paré, Consulting Engineer of the N. A. Timmins Corporation, 1010 Canada Cement Building, Montreal, Quebec.

*Location of the Property.* The property of the Slave Lake Gold Mines, Limited, from which this shipment was received, is situated on Outpost island, Great Slave lake, N.W.T.

*Sampling and Analysis.* After cutting, crushing, and grinding by standard methods, a sample was obtained which assayed as follows:—

| | |
|---|---|
| Gold | 3·15 oz./ton |
| Silver | 0·29 " |
| Copper | 1·67 per cent |
| $WO_3$ | 1·20 " |
| Tin | 0·20 " |
| Iron | 5·25 " |
| Sulphur | 3·19 " |
| Arsenic | Nil |
| Zinc | Nil |

*Characteristics of the Ore.* Six polished sections were prepared and examined microscopically for determining the character of the ore.

The *gangue* is largely fine-textured grey quartz. It commonly exhibits traces of schistosity and occasional small flakes of chloritic material are arranged in parallel streaks. The sulphides also show a tendency to concentration along parallel directions. Some of the ore shows intense staining by brown iron oxide.

The *metallic minerals* identified in the polished sections are as follows:—

*Pyrite.*—Moderately abundant; coarse grains associated with chalcopyrite and marcasite.

*Chalcopyrite.*—Common; masses and irregular grains, usually with associated pyrite and marcasite.

*Marcasite.*—Common, but small quantity; small grains and crystals associated with, and often within, pyrite. This is not the fibrous secondary variety of marcasite, but the well crystallized, twinned, primary type.

*Wolframite* ( $(Fe,Mn)WO_4$).—Small to moderate quantity, as elongated crystals and irregular grains; shows replacement by a transparent mineral. It is inseparable from either of the end members, ferberite ($FeWO_4$) and hübnerite ($MnWO_4$), except by chemical tests.

*Cassiterite* ($SnO_2$).—Small quantity, as irregular rounded grains in gangue. The variety appears to be quite light in colour, as evidenced by the light straw-coloured to white internal reflections instead of the usual deep red internal reflections commonly noted in most cassiterites.

*Bornite.*—Very small quantity associated with chalcopyrite.

*Covellite.*—Occasional replacements of copper minerals by covellite, which has evidently resulted from surface alteration.

*Molybdenite* ($MoS_2$).—Very small quantity as small blunt plates with crystal terminations.

*Native Gold.*—With the exception of one small grain, which occurs within chalcopyrite, all the native gold seen is present as small flakes and rounded grains in quartz.

NOTE:—The minerals wolframite, cassiterite, molybdenite, and marcasite were subjected to investigation by microchemical and spectrographic methods, and although their identification may not be exact, their compositions are essentially those of the minerals named and the significant elements, tungsten and tin, are present.

The *grain size* of the gold in the sections, as determined microscopically, is shown in the following table:—

| Mesh | Gold in quartz, per cent | Gold in chalcopyrite, per cent | Totals, per cent |
|---|---|---|---|
| + 200 | 14·2 | ........... | 14·2 |
| − 200+ 280 | 9·5 | ........... | 9·5 |
| − 280+ 400 | 7·9 | ........... | 7·9 |
| − 400+ 560 | 15·0 | ........... | 15·0 |
| − 560+ 800 | 13·5 | ........... | 13·5 |
| − 800+1100 | 14·2 | 2·4 | 16·6 |
| −1100+1600 | 12·7 | ........... | 12·7 |
| −1600+2300 | 4·6 | ........... | 4·6 |
| −2300 | 6·0 | ........... | 6·0 |
| | 97·6 | 2·4 | 100·0 |

## Experimental Tests

The test work consisted of gravity concentration, flotation, amalgamation, and cyanidation. Over 80 per cent of the gold and 90 per cent of the copper was recovered by amalgamation followed by flotation.

### BARREL AMALGAMATION

#### *Tests Nos. 1 to 3*

In these tests, the ore at −14 mesh was ground in a ball mill to pass 47·5 per cent through a 200-mesh screen in Test No. 1, 62·1 per cent in Test No. 2, and 74·7 per cent in Test No. 3. The pulps were amalgamated with mercury for 1 hour in a jar mill. The amalgamation tailings were assayed for gold.

*Results:*

Feed: gold, 3·15 oz./ton.

| Test No. | Tailing assay, Au, oz./ton | Recovery, per cent |
|---|---|---|
| 1 | 0·87 | 72·4 |
| 2 | 0·59 | 81·3 |
| 3 | 0·485 | 84·6 |

The above tests show the total amount of gold set free by these particular degrees of comminution.

GRAVITY CONCENTRATION AND FLOTATION

*Test No. 4*

In this test the ore at −14 mesh was ground in a ball mill to pass 61·6 per cent through 200 mesh. The pulp was passed through a hydraulic classifier, or trap, and the trap tailing passed over a corduroy blanket. The blanket tailing was conditioned with 2 pounds of soda ash per ton and floated with 0·1 pound of Barrett No. 4 oil, 0·05 pound of pine oil and 0·10 pound of amyl xanthate per ton. A flotation concentrate was removed. The three concentrates were combined and then barrel-amalgamated with mercury, and the amalgamation tailing was assayed for gold.

A screen test showed the grinding as follows:—

| Mesh | Weight, per cent |
|---|---|
| − 48+ 65 | 0·2 |
| − 65+100 | 2·8 |
| −100+150 | 14·1 |
| −150+200 | 21·2 |
| −200 | 61·7 |
| | 100·0 |

*Results:*

| Product | Weight, per cent | Assay | | Distribution, per cent | | Ratio of concentration |
|---|---|---|---|---|---|---|
| | | Au, oz./ton | Cu, per cent | Au | Cu | |
| *Trap Concentration:* | | | | | | |
| Feed | 100·00 | 3·15 | 1·67 | 100·0 | 100·0 | |
| Trap concentrate | 0·75 | 167·24 | .......... | 39·8 | .......... | 133 : 1 |
| Trap tailing | 99·25 | 1·91 | .......... | 60·2 | .......... | |
| *Blanket Concentration:* | | | | | | |
| Feed | 100·00 | 1·91 | .......... | 100·0 | .......... | |
| Blanket concentrate | 2·39 | 21·51 | .......... | 26·9 | .......... | 42 : 1 |
| Blanket tailing | 97·61 | 1·43 | .......... | 73·1 | .......... | |
| *Flotation:* | | | | | | |
| Feed | 100·0 | 1·43 | 1·77* | 100·0 | 100·0 | |
| Flotation concentrate | 8·8 | 14·38 | 19·60 | 88·5 | 97·4 | 11·4 : 1 |
| Final tailing | 91·2 | 0·18 | 0·05 | 11·5 | 2·6 | |

*Calculated.

*Amalgamation of Combined Concentrates:*

| Assay, Au, oz./ton | | Recovery, per cent |
|---|---|---|
| Feed | Tailing | |
| 26·88 | 4·48 | 83·3 |

*Summary:*

| | Per cent |
|---|---|
| Gold recovered in trap concentrate | 39·8 |
| Gold recovered in blanket concentrate | 16·2 |
| Gold recovered in flotation concentrate | 38·9 |
| Gold recovered by amalgamation | 79·1 |
| Copper recovered in flotation concentrate | 97·4 |

GRAVITY CONCENTRATION AND FLOTATION

*Test No. 5*

In this test the ore at −14 mesh was ground to pass 81·4 per cent through 200 mesh and the pulp treated similarly to Test No. 4. The combined trap and blanket concentrates were amalgamated separately from the copper concentrate. A screen test showed the grinding as follows:

| Mesh | Weight, per cent |
|---|---|
| − 65+100 | 0·4 |
| −100+150 | 3·6 |
| −150+200 | 14·4 |
| −200 | 81·6 |
| | 100·0 |

*Trap and Blanket Concentration:*

| Product | Weight, per cent | Assay | | Distribution, per cent | | Ratio of concentration |
|---|---|---|---|---|---|---|
| | | Au, oz./ton | Cu, per cent | Au | Cu | |
| Feed | 100·00 | 3·15 | 1·67 | 100·0 | 100·0 | |
| Concentrates | 8·16 | 30·84 | 6·29 | 79·9 | 30·7 | 12·3 : 1 |
| Blanket tailing | 91·84 | 0·69 | 1·26 | 20·1 | 69·3 | |

*Flotation:*

| Product | Weight, per cent | Assay | | Distribution, per cent | | Ratio of concentration |
|---|---|---|---|---|---|---|
| | | Au, oz./ton | Cu, per cent | Au | Cu | |
| Feed | 100·00 | 0·69 | 1·26 | 100·0 | 100·0 | |
| Flotation concentrate | 7·10 | 6·84 | 16·95 | 70·4 | 95·5 | 13·7 : 1 |
| Final tailing | 92·90 | 0·22 | 0·06 | 29·6 | 4·5 | |

*Barrel Amalgamation:*

| Product | Assay, Au, oz./ton | | Recovery, per cent |
|---|---|---|---|
| | Feed | Tailing | |
| Trap and blanket concentrates | 30·84 | 2·84 | 90·8 |
| Flotation concentrate | 6·84 | 3·22 | 52·9 |

*Summary:*

| | Per cent |
|---|---|
| Gold recovered in trap and blanket concentrates | 79·9 |
| Gold recovered in flotation concentrate | 14·1 |
| Gold recovered by amalgamation | 79·9 |
| Copper recovered in flotation concentrate | 66·2 |

## GRAVITY CONCENTRATION, FLOTATION AND CYANIDATION

### *Test No. 6*

In this test the ore at −14 mesh was ground in a ball mill to pass 84·6 per cent through 200 mesh. The pulp was passed through a hydraulic trap and the trap tailing passed over a corduroy blanket. The combined trap and blanket concentrates were amalgamated and the amalgam residue added to the blanket tailing. This product was conditioned with 2 pounds of lime per ton and floated with 0·2 pound of cresylic acid and 0·1 pound of butyl xanthate per ton. The flotation concentrate was cleaned and amalgamated with mercury, as was the flotation middling. Portions of the flotation tailing were agitated in cyanide solution of a strength of 1 pound per ton for 24- and 48-hour periods. A screen test showed the grinding as follows:—

| Mesh | Weight, per cent |
|---|---|
| − 65+100 | 0·5 |
| −100+150 | 3·4 |
| −150+200 | 11·5 |
| −200 | 84·6 |
| | 100·0 |

*Trap and Blanket Concentration:*

| Product | Weight, per cent | Assay | | Distribution, per cent | | Ratio of concentration |
|---|---|---|---|---|---|---|
| | | Au, oz./ton | Cu, per cent | Au | Cu | |
| Feed | 100·00 | 3·15 | 1·67 | 100·0 | 100·0 | |
| Concentrates | 5·26 | 47·81 | 3·11 | 79·8 | 9·8 | 19 : 1 |
| Blanket tailing | 94·74 | 0·67 | 1·59 | 20·2 | 90·2 | |

*Flotation:*

| | | | | | | |
|---|---|---|---|---|---|---|
| Feed | 100·00 | 0·80 | 1·73* | 100·0 | 100·0 | |
| Flotation concentrate | 6·24 | 7·00 | 24·90 | 54·5 | 89·5 | 16 : 1 |
| Flotation middling | 3·14 | 1·79 | 2·90 | 7·0 | 5·2 | |
| Flotation tailing | 90·62 | 0·34 | 0·10 | 38·5 | 5·3 | |

*Calculated.

7060—6½

*Barrel Amalgamation:*

| Product | Assay, Au, oz./ton | | Recovery, per cent |
|---|---|---|---|
| | Feed | Tailing | |
| Flotation concentrate | 7·00 | 2·30 | 66·0 |
| Flotation middling | 1·79 | 1·30 | 27·4 |

*Cyanidation of Flotation Tailing:*

Feed: gold, 0·34 oz./ton.

| Agitation, hours | Tailing assay, Au, oz./ton | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|
| | | | KCN | CaO |
| 24 | 0·015 | 95·6 | 3·15 | 3·2 |
| 48 | 0·01 | 97·0 | 3·60 | 4·1 |

*Summary:*

| | Per cent |
|---|---|
| Gold recovered in trap and blanket concentrates | 79·8 |
| Gold recovered by amalgamation of trap and blanket concentrates | 74·6 |
| Gold recovered in flotation concentrate | 13·8 |
| Gold recovered by amalgamation of flotation concentrate | 9·1 |
| Gold remaining in flotation concentrate | 4·7 |
| Gold extracted by cyanidation of flotation tailing | 9·5 |
| Total gold recovered by amalgamation | 83·7 |
| Copper recovered in flotation concentrate | 89·5 |

### *Test No. 7*

This was similar to Test No. 6. The fineness of grinding was identical and the metallurgical treatment of the ore followed the same procedure.

*Trap and Blanket Concentration:*

| Product | Weight, per cent | Assay | | Distribution, per cent | | Ratio of concentration |
|---|---|---|---|---|---|---|
| | | Au, oz./ton | Cu, per cent | Au | Cu | |
| Feed | 100·00 | 3·15 | 1·67 | 100·0 | 100·0 | |
| Concentrate | 4·19 | 59·86 | .......... | 79·6 | .......... | 23·9 : 1 |
| Blanket tailing | 95·81 | 0·67 | .......... | 20·4 | .......... | |

*Flotation:*

| Product | Weight, per cent | Assay | | Distribution, per cent | | Ratio of concentration |
|---|---|---|---|---|---|---|
| | | Au, oz./ton | Cu, per cent | Au | Cu | |
| Feed | 100·00 | 0·79 | 1·89* | 100·0 | 100·0 | |
| Concentrate | 6·81 | 8·73 | 25·58 | 75·2 | 92·2 | 14·7 : 1 |
| Middling | 3·79 | 0·92 | 1·07 | 4·4 | 2·1 | |
| Tailing | 89·40 | 0·18 | 0·12 | 20·4 | 5·7 | |

*Calculated.

*Barrel Amalgamation:*

| Product | Assay, Au, oz./ton | | Recovery, per cent |
|---|---|---|---|
| | Feed | Tailing | |
| Flotation concentrate | 8·73 | 2·60 | 70·2 |
| Flotation middling | 0·92 | 0·34 | 63·0 |

*Cyanidation of Flotation Tailing:*

Feed: gold, 0·18 oz./ton.

| Agitation, hours | Tailing assay, Au, oz./ton | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|
| | | | KCN | CaO |
| 24 | 0·025 | 86·1 | 3·30 | 3·2 |
| 48 | 0·015 | 91·7 | 3·80 | 4·1 |

*Summary:*

| | Per cent |
|---|---|
| Gold recovered in trap and blanket concentrates | 79·6 |
| Gold recovered by amalgamation of trap and blanket concentrates | 74·9 |
| Gold recovered in flotation concentrate | 18·8 |
| Gold recovered by amalgamation of flotation concentrate | 13·2 |
| Gold remaining in flotation concentrate | 5·6 |
| Gold extracted by cyanidation of flotation tailing | 4·7 |
| Total gold recovered by amalgamation | 88·1 |
| Copper recovered in flotation concentrate | 92·2 |

The following tests show the amounts of tungsten and tin that can be recovered from the ore, the tungsten being present as wolframite and assaying 1·20 per cent of $WO_3$, and the tin as cassiterite and assaying 0·20 per cent of Sn.

*Test No. 8*

In this test a portion of the flotation tailing from Test No. 6 was passed over a Wilfley table. The different products were assayed for $WO_3$ and Sn.

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | | Ratio of concentration |
|---|---|---|---|---|---|---|
| | | $WO_3$ | Sn | $WO_3$ | Sn | |
| Feed | 100·00 | 1·18* | 0·18* | 100·0 | 100·0 | |
| Concentrate | 0·77 | 26·34 | 9·09 | 17·2 | 38·9 | 130·0 : 1 |
| Middling | 5·65 | 4·08 | 0·84 | 19·5 | 26·4 | |
| Tailing | 93·58 | 0·80 | 0·09 | 63·3 | 34·7 | |

*Calculated.

*Test No. 9*

In this test the flotation tailing from Test No. 7 was passed over a Wilfley table and the table tailing concentrated on a Haultain panner.

*Table Concentration:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | | Ratio of concentration |
|---|---|---|---|---|---|---|
| | | $WO_3$ | Sn | $WO_3$ | Sn | |
| Feed | 100·00 | 1·16* | 0·20* | 100·0 | 100·0 | |
| Concentrate | 1·94 | 15·66 | 5·35 | 26·2 | 51·9 | |
| Middling | 2·94 | 0·80 | 0·20 | 2·0 | 29·4 | 51·5 : 1 |
| Tailing | 95·12 | 0·87 | 0·04 | 71·8 | 18·7 | |

*Calculated.

*Panning Concentration of Table Tailing:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | | Ratio of concentration |
|---|---|---|---|---|---|---|
| | | $WO_3$ | Sn | $WO_3$ | Sn | |
| Feed | 100·00 | 0·87 | 0·04 | 100·0 | 100·0 | |
| Concentrate | 0·82 | 27·66 | 4·70 | 26·1 | 98·5 | 122 : 1 |
| Tailing | 99·18 | 0·65 | ......... | 73·9 | 1·5 | |

*Test No. 10*

Owing to the large amount of tungsten being lost in the slimes in the two previous tests, it was decided to try a coarser grind prior to passing the ore over the Wilfley table.

A screen test showed the grinding as follows:—

| Mesh | Weight, per cent |
|---|---|
| − 35+ 48 | 0·2 |
| − 48+ 65 | 2·2 |
| − 65+100 | 9·8 |
| −100+150 | 17·5 |
| −150+200 | 17·6 |
| −200 | 52·7 |
| | 100·0 |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | | Ratio of concentration |
|---|---|---|---|---|---|---|
| | | $WO_3$ | Sn | $WO_3$ | Sn | |
| Feed | 100·00 | 1·07* | 0·25* | 100·0 | 100·0 | |
| Concentrate | 3·67 | 14·92 | 3·26 | 50·9 | 47·7 | 27·2 : 1 |
| Middling | 26·75 | 0·75 | 0·23 | 18·7 | 24·5 | |
| Tailing | 69·58 | 0·47 | 0·10 | 30·4 | 27·8 | |

*Calculated.

In an endeavour to determine the amounts of wolframite and cassiterite that could be recovered by a table concentration in mill practice, the remainder of the ore, weighing 900 pounds, was ground in a ball mill to pass 31·4 per cent through 200 mesh. The ball mill was in closed circuit with a Hummer screen of 28-mesh size, the oversize returning to the ball mill for further grinding and the undersize being fed to a Plato table. The rate of feed was 180 pounds per hour. A screen test showed the grinding as follows:—

| Mesh | Weight, per cent |
|---|---|
| + 28 | 0·1 |
| − 28+ 35 | 1·1 |
| − 35+ 48 | 4·8 |
| − 48+ 65 | 9·7 |
| − 65+100 | 17·8 |
| −100+150 | 19·2 |
| −150+200 | 15·8 |
| −200 | 31·5 |
| | 100·0 |

*Table Concentration:*

| Product | Weight, per cent | Assay | | | | Distribution, per cent | | | | Ratio of concentration |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Au, oz./ton | Cu, per cent | $WO_3$, per cent | Sn, per cent | Au | Cu | $WO_3$ | Sn | |
| Feed | 100·00 | 3·15 | 1·59* | 1·01* | 0·18* | 100·0 | 100·0 | 100·0 | 100·0 | |
| Concentrate | 2·50 | 78·00* | 10·09 | 15·65 | 3·36 | 61·9 | 15·8 | 38·8 | 46·9 | 40 : 1 |
| 1st middling | 5·66 | 8·60 | 11·60 | 5·31 | 0·79 | 15·5 | 41·2 | 29·8 | 25·0 | |
| 2nd middling | 1·00 | 6·86 | 3·15 | 1·61 | 0·49 | 2·2 | 2·0 | 1·6 | 0·2 | |
| Tailing | 90·84 | 0·71 | 0·72 | 0·33 | 0·05 | 20·4 | 41·0 | 29·8 | 27·9 | |

*Calculated.

The table concentrate, assaying 78·0 ounces of gold per ton, was barrel-amalgamated to produce an amalgam residue assaying 18·60 ounces of gold per ton. A portion of this product was reground with 2 pounds of lime and 0·2 pound of Barrett No. 4 oil per ton, and floated with 0·1 pound of amyl xanthate and 0·05 pound of pine oil per ton. The first middling product was treated similarly without the preliminary amalgamation.

*Results:*

| Product | Weight, per cent | Assays | | | | Distribution, per cent | | | | Ratio of concentration |
|---|---|---|---|---|---|---|---|---|---|---|
| | | Au, oz./ton | Cu, per cent | $WO_3$, per cent | Sn, per cent | Au | Cu | $WO_3$ | Sn | |
| *Flotation of Table Concentrate:* | | | | | | | | | | |
| Feed | 100·00 | 18·60 | 10·09 | 15·65 | 3·36 | 100·0 | 100·0 | 100·0 | 100·0 | |
| Flotation concentrate | 56·17 | 24·28 | 17·62 | 2·68 | 0·82 | 70·2 | 98·5 | 9·5 | 13·2 | 1·8 : 1 |
| Tailing | 43·83 | 13·20 | 0·35 | 32·82 | 6·92 | 29·8 | 1·5 | 90·5 | 86·8 | |
| *Flotation of First Middling:* | | | | | | | | | | |
| Feed | 100·00 | 8·60 | 11·60 | 5·31 | 0·79 | 100·0 | 100·0 | 100·0 | 100·0 | |
| Flotation concentrate | 47·10 | 16·48 | 24·04 | 0·70 | 0·12 | 93·8 | 99·6 | 6·7 | 5·8 | 1·9 : 1 |
| Tailing | 52·90 | 0·98 | 0·11 | 8·67 | 1·73 | 6·2 | 0·4 | 93·3 | 94·2 | |

Combining the recovery of wolframite and cassiterite obtained in the flotation tailings from the table concentrate and first middling product, the results are as follows: a recovery of 62·9 per cent of the tungsten, assaying 22·1 per cent $WO_3$; and a recovery of 64·2 per cent of tin, assaying 5·0 per cent Sn.

## SUMMARY AND CONCLUSIONS

The test work shows that 85 per cent of the gold can be recovered by amalgamation of trap, blanket, and flotation concentrates. Of the remaining 15 per cent, the copper concentrates would retain approximately 5 per cent and the flotation tailing 10 per cent. Cyanidation of the flotation tailing was successful in extracting most of the gold remaining in this product.

Over 90 per cent of the copper was recovered in a flotation concentrate assaying 25 per cent of copper.

The recoveries of tungsten and tin were low, the tendency of the wolframite to slime on the table accounting for most of the losses.

Considering the location of the property and the result of the test work, the flow-sheet to use would comprise grinding in water, followed by traps and blankets to remove the coarse gold. A gold jig might replace the traps if the mill run of ore contains as much coarse gold as the test sample. The concentrates recovered from these procedures would be amalgamated and the amalgam residues added to the blanket tailing. This product would be passed to a flotation machine and the copper recovered. The copper concentrate would be amalgamated and the amalgam residue roasted in an electric furnace, to form a copper-gold matte as a shipping product. If a recovery of the tungsten and tin is deemed economic the flotation tailing could be tabled and a concentrate of these products recovered.

## Ore Dressing and Metallurgical Investigation No. 714

### GOLD ORE FROM THE MONETA PORCUPINE MINES, LIMITED, TIMMINS, ONTARIO

*Shipment.* Two shipments of ore were received on March 1 and March 17, 1937. The first contained 18 pounds of drill core rejects and the second contained two separate lots taken from a test pit, one being 247 pounds of normal sulphide ore and the other 26 pounds of graphitic ore. The samples were submitted by W. E. Segsworth, President, Moneta Porcupine Mines, Limited, 67 Yonge Street, Toronto, Ontario.

*Location of Property.* This property is in Tisdale township, Timiskaming county, Ontario, and adjoins the west side of the property of Hollinger Consolidated Gold Mines, Limited.

*Character of the Ore.* Twenty-one polished sections were prepared from the samples, allocated as follows:—

No. 1030—Six sections: From Hole 11
No. 1031—Six sections: From Hole 27
No. 1040—Six sections: From "Normal Sulphide Ore"
No. 1041—Three sections: From "Graphitic Ore"

The sections were examined microscopically to determine the general character of the samples and the mode of occurrence of the visible gold.

*Samples from Drill Cores.* The samples from drill holes Nos. 11 and 27 exhibited no difference in character and are, therefore, not treated individually.

The *gangue* is medium- to fine-grained and of even texture. The groundmass is siliceous in character and contains abundant finely disseminated carbonate. Occasionally narrow stringers of light grey calcite are to be seen.

Pyrite is the only *metallic mineral* in appreciable quantity. It occurs as medium to small cubic crystals and irregular grains disseminated abundantly in the gangue. In polished sections the mineral appears to be dense in character, with only slight fracturing. When it is etched with nitric acid, however, numerous tiny incipient fractures and inter-grain boundaries are brought out; this indicates that many of the grains are potentially composed of a mosaic of grains which appear only on etching.

Chalcopyrite is in very small quantity as small, irregular grains in gangue and rarely in pyrite. In some sections replacement of the chalcopyrite by covellite is apparent. Rare tiny grains of pyrrhotite occur in the pyrite.

Native gold occurs only as tiny irregular grains in apparently dense pyrite. When the pyrite was etched with nitric acid most of the gold was seen to occur along the incipient fractures and grain boundaries brought out by the etching.

The *Normal Sulphide Ore* was similar, microscopically, to the drill core samples. A fewer number of grains of gold were visible, but all occur within apparently dense pyrite and exhibit about the same degree of fineness of grain.

The *Graphitic Ore* containing graphite is quite different in character from those previously described. In the hand specimen it is fine-textured and exhibits a number of black streaks and bands; slickensided surfaces are abundant and these show smears of shiny graphitic material. In the polished sections the gangue is extremely fine-textured and shows a distinct schistosity; it does not polish well and the particles of graphite are too small or amorphous in character and are not visible under the microscope.

As in the other samples, pyrite is abundant. It occurs as small granular masses, large crystals, and medium to very small cubes and irregular grains. Chalcopyrite, covellite, and pyrrhotite occur as in the drill core samples, but no native gold was visible.

Pyrite does not appear to have been involved in the movement responsible for the shearing along the graphitic zones, because fracturing is not marked and no pyrite is smeared on the slickensided surfaces. It seems rather that the distinctly crystallized, disseminated pyrite has been locally concentrated along certain bands in the already sheared gangue. There is a possibility that this well crystallized and often comparatively coarse pyrite is not of the same age as the disseminated pyrite of the non-graphitic gangue, but no evidence in support of this hypothesis was detected. Perhaps the most significant fact is that no gold is visible in the coarsely crystallized or massive type of pyrite.

*Mode of Occurrence of the Gold.* Native gold is visible only in pyrite, and from the large number of grains seen it seems probable that this is the only important mode of occurrence. The pyrite in which it occurs is the evenly disseminated form of medium to fine grain and usually not well crystallized. The coarsely crystallized pyrite and that associated with the graphitic shears does not appear to carry visible gold, although the evidence of this is only suggestive.

The grain size of the gold is extremely small. The particles appear to occur within dense pyrite, but when the sample is etched with nitric acid the incipient fractures and inter-grain boundaries already referred to are seen to be in contact with the gold particles in almost all cases. The effect of such lines of weakness in the pyrite will be to lessen the difficulty of grinding to expose the gold particles.

The grain size of the gold in the drill core samples is shown in the following table:—

*Grain Size of the Visible Gold:*

| Mesh | Gold in pyrite, per cent |
|---|---|
| + 560 | 4·0 |
| − 560+ 800 | 6·8 |
| − 800+1100 | 17·0 |
| −1100+1600 | 26·0 |
| −1600+2300 | 21·7 |
| −2300 | 24·5 |
| | 100·0 |

*Sampling and Assaying.* The drill core rejects were sampled by standard methods and a complete analysis was made. A composite sample was prepared from the second shipment by mixing the graphitic ore with the normal sulphide ore in the proportion of 6 to 94 respectively. The two samples assayed as follows:—

*Drill Cores:*

| | |
|---|---|
| Insoluble | 51·55 per cent |
| Silica | 31·00 " |
| Alumina | 10·50 " |
| Lime | 1·56 " |
| Magnesia | 0·22 " |
| Titanium oxide | 0·61 " |
| Iron | Nil |
| Soluble:— | |
| Iron | 5·29 per cent |
| Alumina | 11·14 " |
| Lime | 10·84 " |
| Magnesia | 2·24 " |
| Arsenic | 0·13 " |
| Antimony | 0·05 " |
| Bismuth | Trace |
| Copper | 0·02 per cent |
| Lead | Nil |
| Zinc | Nil |
| Sulphur | 6·89 per cent |
| Nickel | Nil |
| Cobalt | Trace |
| Carbon dioxide | 11·26 per cent |
| Molybdenum | 0·02 " |
| Manganese | 0·23 " |
| Graphite | 0·17 " |
| Gold | 0·705 oz./ton |
| Silver | 0·165 " |

*Normal Sulphide Ore:*

| | |
|---|---|
| Gold | 0·625 oz./ton |
| Silver | 0·10 " |
| Graphite | 0·08 per cent |

*Graphitic Ore:*

| | |
|---|---|
| Gold | 0·03 oz./ton |
| Silver | 2·47 " |
| Graphite | 1·60 per cent |

*Composite Mixture:*

(6 per cent Graphitic ore and 94 per cent Normal Sulphide ore)

| | |
|---|---|
| Gold | 0·60 oz./ton |
| Silver | 0·12 " |
| Graphite | 0·19 per cent |
| Insoluble | 55·13 " |
| Silica | 37·22 " |
| Iron | 10·33 " |
| Sulphur | 7·00 " |

## Experimental Tests

The work conducted on this ore consisted of tests by straight cyanidation of the ore and concentration with regrinding and cyanidation of the concentrate. Maximum extraction obtained by either method was in the neighbourhood of 85 per cent, and this can be obtained with six hours' agitation. There did not appear to be any evidence of precipitation of gold by graphite, but the low extraction is due to the presence of very fine gold which cannot be exposed to the action of cyanide solution by any amount of grinding within economic limits.

The tests are described in detail as follows:—

### DRILL CORES

### *Test No. 1*

A sample of the ore was ground 85 per cent through 200 mesh in a ball mill with 0·80 pound of kerosene oil added per ton of charge. A graphite concentrate was then floated off using pine oil as frother, and a pyrite concentrate was floated with the following reagents:—

| | |
|---|---|
| Soda ash | 1·0 lb./ton |
| Potassium amyl xanthate | 0·10 " |
| Barrett No. 4 oil | 0·09 " |
| Pine oil | 0·10 " |

The flotation tailing was treated on a blanket set at a slope of 2·5 inches per foot. The pyrite concentrate and the blanket concentrate were reground 99·8 per cent through 325 mesh in cyanide solution and agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 48 hours. The products were assayed for gold and graphite.

*Summary:*

| Product | Weight, per cent | Assay | | Distribution, per cent | | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|---|
| | | Au, oz./ton | Graphite, per cent | Gold | Graphite | KCN | CaO |
| Graphite concentrate | 7·5 | 3·86 | 1·07 | 41·06 | 54·8 | | |
| Cyanide tailing from pyrite concentrate | 15·1 | 0·24 | 0·13 | 5·14 | 13·4 | 1·50 | 3·45 |
| Blanket tailing | 77·4 | 0·025 | 0·06 | 2·74 | 31·8 | | |
| Feed | 100·0 | 0·705 | *0·146 | 48·94 | 100·0 | | |

*Calculated.

Extraction by cyanidation of concentrates..............51·06 per cent total gold

### *Test No. 2*

A sample of the ore was ground 85 per cent through 200 mesh in a ball mill with 0·80 pound of kerosene oil added per ton of charge. A graphite concentrate was then floated using pine oil as a frother and a pyrite concentrate was floated with the following reagents:—

| | |
|---|---|
| Soda ash | 1·0 lb./ton |
| Potassium amyl xanthate | 0·10 " |
| Reagent No. 208 | 0·08 " |
| Reagent No. 301 | 0·08 " |
| Pine oil | 0·10 " |

The graphite concentrate was recleaned and the cleaner tailing added to the pyrite concentrate. The flotation tailing was treated on a blanket set at a slope of 2·5 inches per foot. The pyrite and blanket concentrates were reground 99·2 per cent through 325 mesh and agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 48 hours. The products were assayed for gold and graphite.

*Summary:*

| Product | Weight, per cent | Assay | | Distribution, per cent | | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|---|
| | | Au, oz./ton | Graphite, per cent | Gold | Graphite | KCN | CaO |
| Graphite concentrate | 5·1 | 6·36 | 1·03 | 46·01 | 38·5 | | |
| Cyanide tailing from pyrite concentrate | 28·0 | 0·14 | 0·18 | 5·56 | 37·0 | 1·84 | 2·95 |
| Blanket tailing | 66·9 | 0·02 | 0·05 | 1·90 | 24·5 | | |
| Feed | 100·0 | 0·705 | *0·136 | 53·47 | 100·0 | | |

*Calculated.

Extraction by cyanidation of pyrite concentrate......46·53 per cent total gold

### *Tests Nos. 3 and 4*

Samples of the ore were ground 88 and 93 per cent through 200 mesh in ball mills without kerosene oil. Graphite concentrates were floated using cresylic acid as a frother, and pyrite concentrates were floated with the following reagents:—

| | |
|---|---|
| Soda ash | 1·0 lb./ton |
| Reagent No. 208 | 0·08 " |
| Reagent No. 301 | 0·08 " |
| Pine oil | 0·10 " |

The pyrite concentrates were reground nearly all through 325 mesh and agitated in cyanide solution, 1·0 pound of potassium cyanide per ton, for 48 hours. All the products were assayed for gold. A sample of the flotation tailing from Test No. 2 was treated on a Haultain super-panner to see if any free gold could be found in it. No gold was found in it but a small amount of free sulphide was panned out. The pan tailing, however, assayed the same as the flotation tailing.

*Summary of Test No. 3:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | | | Gold | KCN | CaO |
| Graphite concentrate | 1·9 | 11·86 | 32·0 | | |
| Cyanide tailing from pyrite concentrate | 18·4 | 0·23 | 6·0 | 1·67 | 3·23 |
| Flotation tailing | 79·7 | 0·02 | 2·2 | | |
| Feed | 100·0 | 0·705 | | | |

Extraction by cyanidation of concentrates...............59·8 per cent total gold

*Summary of Test No. 4:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | | | Gold | KCN | CaO |
| Graphite concentrate | 2·6 | 12·28 | 45·3 | 1·70 | 3·45 |
| Cyanide tailing from pyrite concentrate | 19·8 | 0·23 | 6·5 | | |
| Flotation tailing | 77·6 | 0·015 | 1·6 | | |
| Feed | 100·0 | 0·705 | | | |

Extraction by cyanidation of concentrates................46·6 per cent total gold

All further work was done on the composite sample prepared from the two samples of ore taken from the test pit.

## CYANIDATION

### *Tests Nos. 1 to 7*

Samples of the ore were ground in ball mills for varying periods of time and agitated in cyanide solution, 1·0 pound per ton, for periods of time ranging from 6 to 48 hours. In each test the dilution ratio was 1·5 : 1.

*Summary of Tests Nos. 1 to 7:*

Feed: gold, 0·60 oz./ton.

| Test No. | Grinding, per cent −200 mesh | Agitation, hours | Tailing assay, Au, oz./ton | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| 1 | 88·8 | 48 | 0·12 | 80·00 | 2·0 | 4·7 |
| 2 | 92·9 | 48 | 0·11 | 81·67 | 2·1 | 4·7 |
| 3 | 96·1 | 48 | 0·10 | 83·33 | 2·1 | 4·9 |
| 4 | 98·0 | 48 | 0·10 | 83·33 | 2·6 | 5·0 |
| 5 | 96·1 | 6 | 0·11 | 81·67 | 1·5 | 3·90 |
| 6 | 96·1 | 16 | 0·12 | 80·00 | 2·2 | 4·20 |
| 7 | 96·1 | 24 | 0·115 | 80·83 | 2·2 | 4·30 |

A settling test was made on the pulp from Test No. 2 as it came from the agitator. The pulp was put in a glass tube 2 inches in diameter and allowed to settle for one hour, readings being taken at 5-minute intervals.

The conditions under which the test was conducted are as follows:—

Dilution ratio....................1·5 : 1
Potassium cyanide....................1·0 lb./ton solution
Lime....................0·25 "
Height of pulp column at beginning of test....................2·48 feet

The results of the test are as follows:—

| Time | Height of pulp column, in feet |
|---|---|
| Start | 2·48 |
| 5 minutes | 2·44 |
| 10 " | 2·41 |
| 15 " | 2·37 |
| 20 " | 2·34 |
| 25 " | 2·31 |
| 30 " | 2·28 |
| 35 " | 2·26 |
| 40 " | 2·23 |
| 45 " | 2·20 |
| 50 " | 2·18 |
| 55 " | 2·16 |
| One hour | 2·13 |
| Drop in pulp level in one hour | 0·35 feet |
| Overflow | Clear |

In view of the low extraction obtained by cyanidation, as well as the slow settling rate, it was decided to try flotation with regrinding and cyanidation of the concentrate. The intimate association of the gold with the pyrite and the extremely fine grains in which it occurs, as revealed by the microscopic examination of the ore, hold promise of this being a satisfactory method, as the gold-bearing pyrite can be more efficiently ground after concentration.

After a number of preliminary tests a satisfactory reagent combination was found and flotation tailings were reduced to 0·01 ounce of gold per ton. As the ore contains quite an appreciable quantity of sulphides the ratio of concentration will be necessarily low and the test work has shown that the quantity of flotation reagents needed will be comparatively high.

## FLOTATION AND BLANKET CONCENTRATION

### *Test No. 8*

A sample of the ore was ground 85 per cent through 200 mesh in a ball mill, and a bulk concentrate was floated from it. The flotation tailing was sampled and assayed and the remainder of it was treated on a corduroy blanket set at a slope of 2·5 inches per foot. All products were assayed for gold.

*Charge to Ball Mill:*

| | |
|---|---|
| Ore | 1,000 grms. −14 mesh |
| Water | 750 c.c. |
| Soda ash | 0·50 lb./ton |
| Barrett No. 4 oil | 0·18 " |
| Potassium amyl xanthate | 0·20 " |

*Reagents to Cell:*

| | |
|---|---|
| Pine oil | 0·10 lb./ton |

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, per cent |
|---|---|---|---|
| Flotation concentrate | 19·86 | 2·94 | 96·6 |
| Blanket concentrate | 2·06 | 0·24 | 0·8 |
| Blanket tailing | 78·08 | 0·02 | 2·6 |
| Feed (cal.) | 100·00 | 0·604 | 100·0 |
| Flotation tailing | 80·14 | 0·025 | 3·4 |

### *Test No. 9*

A sample of the ore was ground 85 per cent through 200 mesh in a ball mill and a bulk concentrate floated from it in natural pulp. The flotation tailing was sampled and assayed and the remainder of it treated on a blanket. All products were assayed for gold.

*Charge to Ball Mill:*

| | |
|---|---|
| Ore | 1,000 grms. −14 mesh |
| Water | 750 c.c. |
| Barrett No. 4 oil | 0·18 lb./ton |
| Potassium amyl xanthate | 0·20 " |

*Reagents to Cell:*

| | |
|---|---|
| Pine oil | 0·20 lb./ton |
| Copper sulphate | 1·0 " |

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, per cent |
|---|---|---|---|
| Flotation concentrate | 15·07 | 3·58 | 92·05 |
| Blanket concentrate | 4·54 | 0·23 | 1·78 |
| Blanket tailing | 80·39 | 0·045 | 6·17 |
| Feed (cal.) | 100·00 | 0·586 | 100·00 |
| Flotation tailing | 84·93 | 0·06 | |

### *Test No. 10*

A sample of the ore was ground 85 per cent through 200 mesh in a ball mill and a bulk concentrate floated from it in caustic soda pulp. The flotation tailing was treated on a blanket as before.

*Charge to Ball Mill:*

| | |
|---|---|
| Ore | 1,000 grms. −14 mesh |
| Water | 750 c.c. |
| Caustic soda | 0·50 lb./ton |
| Barrett No. 4 oil | 0·18 " |
| Potassium amyl xanthate | 0·20 " |

*Reagents to Cell:*

Pine oil .......... 0·20 lb./ton
Copper sulphate .......... 1·0 "
Potassium amyl xanthate .......... 0·20 "

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, per cent |
|---|---|---|---|
| Flotation concentrate | 19·9 | 2·94 | 97·8 |
| Blanket concentrate | 5·7 | 0·10 | 1·0 |
| Blanket tailing | 74·4 | 0·01 | 1·2 |
| Feed (cal.) | 100·0 | 0·598 | 100·0 |
| Flotation tailing | 80·1 | 0·015 | |

## *Test No. 11*

This test was a duplicate of Test No. 10, except that flotation was carried out in natural pulp. In comparing this test with No. 9 the lower tailing assays obtained here are believed to be due to the extra quantity of xanthate added to the flotation cell.

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold, per cent |
|---|---|---|---|
| Flotation concentrate | 17·8 | 3·31 | 96·9 |
| Blanket concentrate | 5·0 | 0·07 | 0·6 |
| Blanket tailing | 77·2 | 0·02 | 2·5 |
| Feed (cal.) | 100·0 | 0·608 | 100·0 |
| Flotation tailing | 82·2 | 0·025 | |

## FLOTATION

## *Test No. 12*

A sample of the ore was ground 85 per cent through 200 mesh in a ball mill and a bulk concentrate floated from it. The flotation tailing was not treated on a blanket and the concentrate was set aside for cyanidation tests.

*Charge to Ball Mill:*

Ore .......... 2,000 grms. −14 mesh
Water .......... 1,500 c.c.
Aerofloat No. 31 .......... 0·14 lb./ton
Potassium amyl xanthate .......... 0·40 "

*Reagents to Cell:*

Pine oil .......... 0·10 lb./ton
Copper sulphate .......... 1·0 "
Potassium amyl xanthate .......... 0·10 "

The flotation tailing assayed 0·01 ounce of gold per ton with a ratio of concentration of 3·85 : 1. This means that 98·76 per cent of the gold has been recovered in the concentrate. A number of similar tests were conducted with the same result, and the concentrates were all mixed together for cyanidation tests, which are described in the following paragraph. Some of the foregoing tests were made in soda ash pulp and some in caustic soda pulp but recoveries and ratios of concentration were the same in each case.

### CYANIDATION OF BULK CONCENTRATES

*Tests Nos. 13 to 16*

The concentrates produced in the series of tests described in Test No. 12 were mixed together and ground 99·4 per cent through 325 mesh. Samples of the reground concentrate were agitated in cyanide solution, 3·0 pounds of potassium cyanide per ton, for periods of 6, 10, 14, and 20 hours at 2·5 : 1 dilution. The cyanide tailings were assayed for gold.

*Summary of Tests Nos. 13 to 16:*

Feed: gold, 2·3 oz./ton.

| Test No. | Agitation, hours | Tailing assay, Au, oz./ton | Extraction, per cent | Extraction, per cent total gold | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| 13 | 6 | 0·35 | 84·8 | 83·74 | 2·18 | 2·60 |
| 14 | 10 | 0·34 | 85·2 | 84·14 | 2·44 | 2·63 |
| 15 | 14 | 0·33 | 85·7 | 84·63 | 2·49 | 3·00 |
| 16 | 20 | 0·34 | 85·2 | 84·14 | 2·62 | 3·12 |

### BULK FLOTATION AND CYANIDATION OF CONCENTRATES

*Test No. 17*

A batch of bulk concentrate was made similar to that produced in Test No. 12. The concentrate contained 98·76 per cent of the gold and the tailing assayed 0·01 ounce per ton. The ratio of concentration was 4·27 : 1.

The concentrate was reground nearly all through 325 mesh in cyanide solution and agitated for 6 hours at 2·5 : 1 dilution. The solution was kept up to the equivalent of 3·0 pounds of potassium cyanide per ton. At the end of 6 hours the cyanide tailing was filtered and washed and a sample cut out for assay. The remainder was repulped in fresh cyanide solution of the same strength and given another 24-hour agitation. The second cyanide tailing was sampled and assayed and the remainder roasted at 550° to 600° C. The calcine was washed and agitated in cyanide solution, 3·0 pounds of potassium cyanide per ton, for 24 hours. The tailing was assayed for gold. Loss in weight during roasting was 16·5 per cent.

*Summary:*

| Product | Assay, Au, oz./ton | Extraction, per cent | Extraction, per cent total gold | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|
| | | | | KCN | CaO |
| Feed sample | 2·60 | | | | |
| 6-hour cyanide tailing | 0·35 | 86·54 | 85·46 | 2·06 | 2·46 |
| 30-hour cyanide tailing | 0·34 | 86·92 | 85·84 | 0·33 | 0·29 |
| Calcine | 0·40 | | | | |
| Cyanide tailing from calcine | 0·26 | 35·0 | 4·52 | 0·40 | 2·55 |

FLOTATION AND CYANIDATION

*Test No. 18*

A sample of the ore was ground 85 per cent through 200 mesh and a graphite concentrate taken off using cresylic acid as a frother. A pyrite concentrate was then taken off using the following reagents:—

| | |
|---|---|
| Caustic soda | 0·50 lb./ton |
| Aerofloat No. 31 | 0·14 " |
| Potassium amyl xanthate | 0·50 " |
| Pine oil | 0·10 " |
| Copper sulphate | 1·0 " |

The graphite concentrate was cleaned and the cleaner tailing added to the pyrite concentrate.

The pyrite concentrate was then reground in cyanide solution all through 325 mesh and agitated for 6 hours in solution containing the equivalent of 3·0 pounds of potassium cyanide per ton. The cyanide tailing was sampled and assayed and the remainder of it repulped in fresh cyanide solution of equal strength and agitated for another 24 hours. The products were assayed for gold.

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, gold, per cent | Extraction, per cent total gold | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| Graphite concentrate | 0·19 | 5·46 | 1·9 | | | |
| Pyrite concentrate | 32·98 | 1·60 | 96·3 | | | |
| Flotation tailing | 66·83 | 0·015 | 1·8 | | | |
| Feed (cal.) | 100·00 | 0·548 | 100·0 | | | |
| 1st cyanide tailing | | 0·34 | | 75·84 | 1·75 | 2·45 |
| 2nd cyanide tailing | | 0·24 | | 81·86 | 0·20 | 0·43 |

*Test No. 19*

This test is a duplicate of Test No. 18 except that the Aerofloat No. 31 added to the cell was reduced to 0·07 pound per ton and a better ratio of concentration and better grade of concentrate were obtained.

The concentrate was not assayed but was all reground and agitated for 6 hours and re-agitated for 24 hours as before. Extractions were calculated by difference from the feed sample.

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent total gold | Extraction, per cent total gold | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| Flotation tailing | 75·76 | 0·015 | 1·89 | | | |
| Graphite concentrate | 0·24 | 5·27 | 2·11 | | | |
| 1st cyanide tailing from concentrate | 24·00 | 0·40 | 16·00 | 80·0 | 1·67 | 2·58 |
| 2nd cyanide tailing from concentrate | 24·00 | 0·30 | 12·00 | 84·0 | 0·44 | 0·80 |
| Feed | 100·00 | 0·60 | | | | |

## BULK FLOTATION AND CYANIDATION OF CONCENTRATES AFTER EXTREMELY FINE REGRINDING

### *Test No. 20*

A sample of the ore was ground 85 per cent through 200 mesh in a ball mill and a bulk concentrate floated off. The concentrate was reground 99·95 per cent through 325 mesh, sampled for assay, and repulped in the water in which it was ground in the ball mill. Cyanide and lime were added and the pulp agitated for 24 hours at approximately 2·5 : 1 dilution. The cyanide solution was kept at the equivalent of 3·0 pounds of potassium cyanide per ton. The cyanide tailing was filtered and sampled for assay and the remainder of it repulped in the same solution and re-agitated for another 24 hours. All the products were assayed for gold.

This concentrate was reground in a ball mill for two hours. In any of the foregoing tests the maximum grinding time was one hour and therefore this concentrate will have a higher percentage of fines in the sizes smaller than 325 mesh than did the former ones. In fact, this concentrate is probably finer than any classifier would deliver in plant practice but the test was done merely to show that to obtain any additional extraction the sulphides will have to be ground finer in order to expose the fine gold, some of which may be sub-microscopic, to the dissolving action of the cyanide solution.

*Charge to Ball Mill:*

| | |
|---|---|
| Ore | 2,000 grms. —14 mesh |
| Water | 1,500 c.c. |
| Soda ash | 0·50 lb./ton |
| Potassium amyl xanthate | 0·20 " |
| Aerofloat No. 31 | 0·14 " |

*Reagents to Cell:*

| | |
|---|---|
| Pine oil | 0·10 lb./ton |
| Potassium amyl xanthate | 0·30 " |
| Copper sulphate | 1·0 " |

*Summary:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent total gold | Extraction, per cent total gold | Reagents consumed, lb./ton ore | |
|---|---|---|---|---|---|---|
| | | | | | KCN | CaO |
| Flotation concentrate | 27·21 | 2·11 | 98·75 | | | |
| Flotation tailing | 72·79 | 0·01 | 1·25 | | | |
| Feed (cal.) | 100·00 | 0·581 | 100·00 | | | |
| 1st cyanide tailing from concentrate | 27·21 | 0·26 | | 86·58 | 3·46 | 3·31 |
| 2nd cyanide tailing from concentrate | 27·21 | 0·25 | | 87·05 | 0·55 | 0·74 |

## CYANIDATION FOLLOWED BY FLOTATION

### *Test No. 21*

A sample of the ore was ground 93 per cent through 200 mesh in cyanide solution, 1·0 pound of potassium cyanide per ton, and agitated for 18 hours. The cyanide tailing was filtered, washed three times, and sampled for assay, and the remainder of it was floated with the following reagents:—

| | |
|---|---|
| Soda ash | 0·50 lb./ton |
| Caustic soda | 0·50 " |
| Copper sulphate | 1·0 " |
| Potassium amyl xanthate | 0·10 " |
| Cresylic acid | 0·20 " |
| Pine oil | 0·03 " |

The flotation concentrate and tailing were assayed for gold.

*Summary:*

| | |
|---|---|
| Feed sample | 0·60 Au, oz./ton |
| Cyanide tailing | 0·125 " |
| Extraction by cyanidation | 79·17 per cent total gold |

*Flotation of Cyanide Tailing:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent total gold |
|---|---|---|---|
| Flotation concentrate | 16·1 | 0·62 | 17·83 |
| Flotation tailing | 83·9 | 0·02 | 3·00 |
| Cyanide tailing (cal.) | 100·0 | 0·117 | 20·83 |

### *Test No. 22*

This test was the same as Test No. 21 except for the reagent combination used in flotation of the cyanide tailing, which was as follows:—

| | |
|---|---|
| Caustic soda | 1·0 lb./ton |
| Copper sulphate | 0·50 " |
| Barrett No. 4 oil | 0·09 " |
| Potassium amyl xanthate | 0·20 " |
| Cresylic acid | 0·20 " |
| Pine oil | 0·05 " |

*Summary:*

| | |
|---|---|
| Feed sample | 0·60 Au, oz./ton |
| Cyanide tailing | 0·125 " |
| Extraction by cyanidation | 79·17 per cent total gold |

*Flotation of Cyanide Tailing:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent total gold |
|---|---|---|---|
| Flotation concentrate | 16·04 | 0·59 | 16·45 |
| Flotation tailing | 83·96 | 0·03 | 4·38 |
| Cyanide tailing (cal.) | 100·00 | 0·12 | 20·83 |

CYANIDATION FOLLOWED BY FLOTATION

*Test No. 23*

This test also differed from Tests Nos. 21 and 22 only in the matter of the reagent combination used to float the cyanide tailing.

The reagents used were as follows:—

| | |
|---|---|
| Soda ash | 2·50 lb./ton |
| Copper sulphate | 0·50 " |
| Potassium amyl xanthate | 0·20 " |
| Reagent No. 208 | 0·10 " |
| Pine oil | 0·15 " |

*Summary:*

| | |
|---|---|
| Feed sample | 0·60 Au, oz./ton |
| Cyanide tailing | 0·12 " |
| Extraction by cyanidation | 80·0 per cent total gold |

*Flotation of Cyanide Tailing:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent total gold |
|---|---|---|---|
| Flotation concentrate | 19·3 | 0·56 | 18·00 |
| Flotation tailing | 80·7 | 0·015 | 2·00 |
| Cyanide tailing (cal.) | 100·0 | 0·12 | 20·00 |

CONCLUSIONS

The investigation has brought out the following points about this sample of ore.

(1) The gold is practically all associated with the sulphides and can be concentrated, but the ratio of concentration will be low.

(2) The gold is very fine, some of it probably sub-microscopic, and no amount of grinding within economic limits will expose more than about 85 per cent to the dissolving action of cyanide solution.

(3) The graphite is all amorphous and does not appear to cause precipitation of the gold from cyanide solution.

(4) Any gold exposed by grinding is readily soluble in cyanide solution.

(5) The amount of cyanide consumed is on the high side either for direct treatment of the ore or for treatment of a reground concentrate.

There are two methods of treatment to choose between, the one being straight cyanidation of the ore and the other flotation with regrinding and cyanidation of the concentrate. By the latter method about 85 per cent of the total gold can be extracted and by the first method about 83 per cent will be extracted if the ore is ground 96 per cent through 200 mesh.

To concentrate by flotation the ore should be ground 85 per cent through 200 mesh and to get 85 per cent of the gold from the concentrate it should be ground nearly all through 325 mesh. By roasting a cyanide tailing from a reground concentrate, extraction was raised to 90 per cent of the total gold.

There is, however, about 25 per cent of the weight of the original ore to be reground and cyanided and possibly roasted after the ore has been floated. Whether the additional extraction, as compared with straight cyanidation, will more than pay for the additional cost of carrying out these steps will have to be decided on the basis of those costs.

There is also the possibility of cyaniding the ore and floating the cyanide tailing to produce a concentrate for sale to a smelter. This, however, will be a marginal proposition, as the concentrate will be low in grade, from 0·50 to 0·60 ounce of gold per ton, and will have to bear the cost of several operations as well as freight and smelter charges.

## Ore Dressing and Metallurgical Investigation No. 715

GOLD ORE FROM THE NAYBOB GOLD MINES, LIMITED, TIMMINS, ONTARIO

*Shipment.* A shipment of three samples of gold ore was received on February 11, 1937. These samples were designated "CA", "CB", and "CC", and contained 1,956, 2,209, and 1,305 pounds, respectively.

The shipment was made by the Naybob Gold Mines, Limited (which was formerly the Hayden Gold Mines, Limited), Porcupine district, Ontario, and was submitted by R. V. Neily, Mine Manager.

*Purpose of the Experimental Test.* The shipment was made to determine the value of the three samples separately and to determine a method of treatment to recover the contained gold.

*Characteristics of the Ore.* Eighteen polished sections of ore and two of classifier concentrate from ore ground 75 per cent −200 mesh and concentrated by a hydraulic trap were prepared and examined microscopically.

The *gangue* is dark grey, fine-textured and somewhat schistose siliceous rock with abundant finely disseminated carbonate and a considerable quantity of light grey translucent quartz.

The *metallic minerals* present are, in their order of abundance: pyrite, arsenopyrite, chalcopyrite, covellite, galena, pyrrhotite, and native gold.

Pyrite occurs as abundantly disseminated coarse to fine crystals and grains, and contains numerous tiny inclusions of gangue. A lesser quantity of arsenopyrite is present as disseminated crystals, often associated with pyrite. A very small quantity of chalcopyrite occurs as small grains in gangue and rarely in pyrite; it has been replaced by covellite in places. Rare tiny grains of galena and pyrrhotite occur in pyrite.

Native gold was seen only in dense pyrite. The tiny grains vary in size from 15 microns to less than a micron. Only nine grains of gold were seen.

This product consists almost entirely of pyrite and arsenopyrite, all of which appear to be free from gangue. Native gold occurs free, in pyrite, and adhering to a small piece of gangue. Most of it is free, however, the largest grain being about 100 mesh in size. The gold in this product indicates what the ore samples do not, namely, that much of the gold occurs in larger sizes and in the gangue and not entirely as very finely divided particles in pyrite as was indicated by the ore sections. Obviously, the 75 per cent −200-mesh grind has not freed the finely divided gold present in the pyrite.

*Sampling and Analysis.* Each of the three samples was crushed, sampled separately by standard methods, and assayed. The samples were then mixed and sampled and used in the following tests

*Results:*

| Sample | Au, oz./ton | Ag, oz./ton |
|---|---|---|
| CA | 0·09 | 0·03 |
| CB | 0·20 | 0·05 |
| CC | 0·315 | 0·05 |
| (CA, CB, CC) | 0·195 | 0·05 |

| | |
|---|---|
| (CA, CB, CC)—Arsenic | 0·61 per cent |
| Copper | Nil |
| Pyrrhotite | 0·04 per cent |

## Experimental Tests

The tests on the ore include straight cyanidation and combinations of concentration by different means. The concentrates were treated to remove the valuable metals by amalgamation, cyanidation, and roasting.

### CONCENTRATION BY HYDRAULIC TRAP AND WILFLEY TABLE

*Test No. 1*

This test was to determine the amount of free gold in the ore by means of a hydraulic trap and a Wilfley table.

A sample of −14-mesh ore was ground in a ball mill, at a dilution of 4 parts of ore to 3 parts of water (4 : 3), to give a product 75 per cent −200 mesh.

The ground ore was concentrated in a hydraulic trap and the trap tailing was concentrated on a Wilfley table.

The products of the test were assayed for gold and the trap concentrate was examined under a microscope. It showed numerous particles of free gold, some of which were visible only under the microscope.

*Results:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent |
|---|---|---|---|
| *Trap Concentration:* | | | |
| Feed | 100·00 | 0·195 | 100·00 |
| Trap concentrate | 1·02 | 9·80 | 52·88 |
| Trap tailing | 98·98 | 0·09 | 47·12 |
| *Wilfley Table Concentration:* | | | |
| Table feed | 100·00 | 0·09 | 100·00 |
| Table concentrate | 4·30 | 0·99 | 47·08 |
| Table tailing | 95·70 | 0·05 | 52·92 |

| | Per cent |
|---|---|
| Distribution of gold in the trap concentrate | 52·88 |
| Distribution of gold in the table concentrate, 47·08 × 47·12 | 22·18 |
| Distribution of gold in both concentrates | 75·06 |
| Loss of gold in the final tailing, 52·92 × 47·12 | 24·94 |
| | 100·00 |

STRAIGHT CYANIDATION

*Test No. 2*

The purpose of this test was the extraction of gold by straight cyanidation.

Four samples of ore were ground, for different periods of time, in a solution of sodium cyanide equivalent in strength to 1·0 pound of potassium cyanide per ton of solution. Lime was added to the grinding mill to supply protective alkalinity.

After grinding, each charge was agitated for 24 hours at a dilution of 1·5 parts of cyanide solution to 1 part of ore. The strength of the solution was made up to 1·0 pound of potassium cyanide per ton. The solutions were kept up to strength by addition of reagents as required.

After agitation each charge was filtered, washed, and assayed. A screen test was made to determine the degree of grinding.

*Results of Cyanidation:*

| Sample No. | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| 1 | 0·195 | 0·04 | 79·50 | 0·50 | 3·75 |
| 2 | 0·195 | 0·04 | 79·50 | 0·73 | 4·15 |
| 3 | 0·195 | 0·035 | 82·05 | 0·81 | 4·25 |
| 4 | 0·195 | 0·035 | 82·05 | 0·82 | 4·50 |

*Screen Test:*

| Mesh | Weight, per cent | | | |
|---|---|---|---|---|
| | Sample No. 1 | Sample No. 2 | Sample No. 3 | Sample No. 4 |
| + 48 | 0·2 | ........ | ........ | ........ |
| − 48+ 65 | 2·5 | ........ | ........ | ........ |
| − 65+100 | 6·9 | 0·9 | 0·3 | ........ |
| −100+150 | 13·4 | 5·4 | 2·6 | 0·7 |
| −150+200 | 13·2 | 10·8 | 8·1 | 4·5 |
| −200 | 63·8 | 82·9 | 89·0 | 94·8 |
| | 100·0 | 100·0 | 100·0 | 100·0 |
| −350 | 89·0 | 90·0 | 93·0 | 97·0 |

The test shows that a maximum extraction of 82 per cent of the gold was obtained at a grind of 89 per cent −200 mesh.

*Test No. 2A*

This test was to determine the effect of very fine grinding and prolonged agitation with cyanide solution.

Two samples were ground in water, dilution 4 : 3, lime at the rate of 4·0 pounds per ton of ore being added to the charge. The ground ore was filtered, washed, and repulped in cyanide solution, 1·0 pound of potassium cyanide per ton, at a dilution of 2 : 1. The agitation was discontinued at the end of 24 hours and the pulp was filtered, washed, and sampled. The remainder was then repulped in fresh solution, 1·0 pound of potassium cyanide per ton, dilution 2 : 1. The agitation was concluded after 48 hours. The alkalinity of the solutions was maintained by adding lime.

A screen test on the tailings shows the following:—

Sample No. 5........................................ 98·6 per cent −200 mesh
Sample No. 6........................................ 99·8 per cent −200 mesh

*Results:*

*Cyanidation (24 hours):*

| Sample No. | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| 5........................................ | 0·195 | 0·035 | 82·05 | 0·78 | 3·20 |
| 6........................................ | 0·195 | 0·04 | 79·50 | 0·94 | 3·75 |
| 5........................................ | 0·035 | 0·035 | Nil | 0·06 | 3·20 |
| 6........................................ | 0·04 | 0·04 | Nil | 0·17 | 3·10 |

The results of the test show that the extraction was completed in 24 hours and that fine grinding did not increase the extraction.

### *Test No. 3*

The purpose of this test was to note the effect of grinding in cyanide solution, concentrating the sulphides, regrinding them in cyanide solution, and then cyaniding them with the main portion of the ore.

A sample of ore was ground in cyanide solution, 1·0 pound of potassium cyanide per ton, to give a product 80 per cent −200 mesh. The pulp was filtered and the solution saved.

The ground ore was concentrated in a hydraulic trap and on a Wilfley table. The concentrate was reground in cyanide solution, filtered, and mixed with the table tailing.

Two charges of the mixed pulp were cyanided for 24 hours at a dilution of 1·5 : 1, using the solutions from the grind, which were made up to 1·0 pound of potassium cyanide per ton. Lime was used to give protective alkalinity in grinding and agitation.

A sample of the mixed pulp was assayed before agitation, and the extraction before agitation was calculated.

*Results:*

*Grinding in Cyanide Solution and Concentration:*

| Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|
| Feed | Cyanidation feed | | KCN | CaO |
| 0·195 | 0·075 | 61·54 | 1·77 | 3·75 |

*Agitation in Cyanide Solution (24 hours):*

| Test No. | Assay, Au oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | | Total reagents, lb./ton | |
|---|---|---|---|---|---|---|---|
| | Cyanidation feed | Cyanidation tailing | | KCN | CaO | KCN | CaO |
| 1.................... | 0·075 | 0·03 | 60·0 | 0·50 | 3·45 | 2·27 | 7·20 |
| 2.................... | 0·075 | 0·03 | 60·0 | 0·44 | 3·45 | 2·21 | 7·20 |

Gold extracted in grinding.................................... 61·54 per cent
Gold extracted in cyanidation, 60·0 × (100·0—61·54)............ 23·08 per cent

Overall extraction of gold.................................... 84·62 per cent

The results of the test show that 61·54 per cent of the gold was extracted during the grinding of the ore and regrinding of the concentrates and that 23·08 per cent was extracted during the agitation period, making a total extraction of gold of 84·62 per cent.

### *Tests Nos. 4 and 5*

These tests were repetitions of Test No. 3. The details of the tests were similar. The final cyanide tailing for both tests was 0·035 ounce of gold per ton, giving an overall extraction of 82·05 per cent of the gold, with a similar consumption of reagents.

### *Test No. 6*

This test was made in order to note the effect of grinding in cyanide solution, concentrating on a Wilfley table, regrinding the concentrate in cyanide solution, and cyaniding both products separately.

A sample of ore was ground in cyanide solution, 1·0 pound of potassium cyanide per ton, to give a product 85 per cent —200 mesh. The pulp was filtered and the solution was saved.

The ground ore was concentrated on a Wilfley table. The table concentrate was reground in cyanide solution and agitated in cyanide solution, 2·0 pounds of potassium cyanide per ton, for 48 hours. The table tailing was agitated in two lots in cyanide solution, 1·0 pound of potassium cyanide per ton, one lot for 24 hours and the other for 48 hours.

*Results:*

*Grinding the Ore in Cyanide Solution:*

| Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|
| Original feed | Table feed | | KCN | CaO |
| 0·195 | 0·185 | 5·13 | 0·30 | 3·80 |

*Table Concentration:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold in products, per cent | Ratio of concentration |
|---|---|---|---|---|
| Table feed | 100·00 | 0·185 | 100·00 | 14·3 : 1 |
| Table concentrate | 7·01 | 2·11 | 79·89 | |
| Table tailing | 92·99 | 0·04 | 20·11 | |

*Cyanidation of Table Concentrate (48 hours):*

| Assay, Au, oz./ton | | Extraction of gold, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|
| Feed | Tailing | | KCN | CaO |
| 2·11 | 0·24 | 88·63 | 9·26 | 77·0 |

*Cyanidation of Table Tailing (24 and 48 hours):*

| Agitation, hours | Assay, Au, oz./ton | | Extraction, of gold, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| 24 | 0·04 | 0·02 | 50·00 | 0·60 | 7·00 |
| 48 | 0·04 | 0·015 | 62·50 | 0·75 | 7·25 |

*Summary of Test No. 6. Cyanidation of Table Tailing (24 hours):*

| Product | Weight, per cent | Assay, Au, oz./ton | Units, gold | Distribution, per cent |
|---|---|---|---|---|
| Feed | 100·00 | 0·195 | | 100·00 |
| Extracted during grinding | | | 1·00 | 5·13 |
| Table concentrate cyanided | 7·01 | 0·24 | 1·68 | 8·62 |
| Extracted by cyanidation of table concentrate | | | 13·10 | 67·19 |
| Table tailing cyanided | 92·99 | 0·02 | 1·86 | 9·53 |
| Extracted by cyanidation of table tailing | | | 1·86 | 9·53 |
| Total extraction | | | | 81·85 |
| Combined tailings from cyanidation | | 0·035 | | |

*Cyanidation of Table Tailing (48 hours):*

| Product | Weight, per cent | Assay, Au, oz./ton | Units, gold | Distribution, per cent |
|---|---|---|---|---|
| Feed | 100·00 | 0·195 | | 100·00 |
| Extracted during grinding | | | 1·00 | 5·13 |
| Table concentrate cyanided | 7·01 | 0·24 | 1·68 | 8·62 |
| Extracted by cyanidation of table concentrate | | | 13·10 | 67·19 |
| Table tailing cyanided | 92·99 | 0·015 | 1·39 | 7·13 |
| Extracted by cyanidation of table tailing | | | 2·33 | 11·93 |
| Total extraction | | | | 84·25 |
| Combined tailings from cyanidation | | 0·031 | | |

AMALGAMATION AND CYANIDATION

*Test No. 7*

The purpose of this test was to note the effect of grinding in water, concentrating by hydraulic trap, amalgamating the concentrate, and cyaniding the residue and tailing for 24 hours.

Two samples of ore were ground in water, one to give 75 per cent −200 mesh and the other 92 per cent −200 mesh.

Each sample after grinding was concentrated in a hydraulic trap. The concentrates were amalgamated with mercury by grinding in an iron mortar. After separating the amalgam, the residues were added to the tailings and filtered.

The filter cake was then repulped in cyanide solution, 1·0 pound of potassium cyanide per ton, at a dilution of 1·5 : 1 and agitated for 24 hours, using lime to give the solution protective alkalinity. The tailings were assayed for gold.

*Results:*

| Test No. | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| 7A (75 per cent −200 mesh) | 0·195 | 0·04 | 79·5 | 0·60 | 3·15 |
| 7B (92 per cent −200 mesh) | 0·195 | 0·04 | 79·5 | 0·70 | 3·70 |

The results obtained by amalgamation of the free gold in the concentrates prior to cyanidation did not show increased extraction.

*Test No. 8*

This test was made to note the effect of grinding in cyanide solution, concentrating by hydraulic trap, amalgamating the concentrate, and cyaniding the residue and tailing.

Two samples of ore were ground in cyanide solution, 1·0 pound of potassium cyanide per ton, one to give 75 per cent −200 mesh and the other 92 per cent −200 mesh. The pulps were filtered and the cyanide solution from each was saved and used for the agitation of tailings.

Each sample, after grinding, was concentrated in a hydraulic trap and the concentrates were amalgamated as in the previous test.

The residues were returned to their respective tailings, filtered, and repulped in cyanide solution at a dilution of 1·5 : 1. The strength of the solution was made up to 1·0 pound of potassium cyanide per ton. Agitation was discontinued after 24 hours. The tailings were assayed for gold.

*Results:*

| Test No. | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| 8A (75 per cent −200 mesh) | 0·195 | 0·035 | 82·05 | 1·04 | 7·30 |
| 8B (92 per cent −200 mesh) | 0·195 | 0·04 | 79·50 | 1·05 | 7·50 |

The results obtained by amalgamating the free gold in the concentrates prior to cyanidation did not indicate increased extraction.

*Tests Nos. 9, 10, and 11*

The purpose of these tests was to note the effect of grinding in cyanide solution, concentrating by hydraulic trap, amalgamating the trap concentrate, concentrating the trap tailing on a Wilfley table, regrinding the table concentrate and amalgamation residue in cyanide solution, mixing with the table tailing, and agitating in cyanide solution.

An attempt was made in the following three tests to lower the residual gold in the final tailing.

The following procedure was used in each test:—

Samples of ore were ground in cyanide solution, 1·0 pound of potassium cyanide per ton, to approximately 87 per cent −200 mesh. The pulp was filtered and the cyanide solution saved for use in the agitation. The ore was then concentrated by a hydraulic trap. The concentrate was amalgamated, and after separating the amalgam the residue was mixed with the table concentrate obtained from concentrating the trap tailing on a Wilfley table.

The concentrate and residue were reground in cyanide solution, 2·0 pounds of potassium cyanide per ton, filtered, and mixed with the table tailing. The solution from the grind was added to the first solution from the ore.

Two charges were then repulped in the cyanide solution, 1·0 pound of potassium cyanide per ton, at a dilution of 1·5 : 1 and were agitated for 24 and 48 hours.

*Results:*

| Test No. | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| *Extraction by Grinding and Amalgamation:* | | | | | |
| 9 | 0·195 | 0·09 | 53·85 | 1·93 | 6·40 |
| 10 | 0·195 | 0·085 | 56·41 | 1·94 | 4·27 |
| 11 | 0·195 | 0·075 | 61·54 | 2·12 | 7·45 |
| *Extraction by Cyanidation:* | | | | | |
| 9—24-hour | 0·09 | 0·035 | 61·11 | 0·45 | 3·40 |
| 48-hour | 0·09 | 0·04 | 55·55 | 0·70 | 3·60 |
| 10—24-hour | 0·085 | 0·035 | 58·82 | 0·44 | 3·55 |
| 48-hour | 0·085 | 0·035 | 58·82 | 0·53 | 3·85 |
| 11—24-hour | 0·075 | 0·04 | 46·67 | 0·59 | 2·05 |
| 48-hour | 0·075 | 0·04 | 46·67 | 0·82 | 2·25 |

*Summary of Tests Nos. 9, 10, and 11:*

| Test No. | Agitation, hours | Extraction by grinding, etc., per cent | Total extraction, per cent | Total reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | | | | KCN | CaO |
| 9 | 24 | 53·85 | 82·05 | 2·38 | 9·80 |
| | 48 | | 79·50 | 2·63 | 10·00 |
| 10 | 24 | 56·41 | 82·05 | 2·38 | 7·82 |
| | 48 | | 82·05 | 2·47 | 8·12 |
| 11 | 24 | 61·54 | 79·50 | 2·71 | 9·50 |
| | 48 | | 79·50 | 2·94 | 9·70 |

The tests show no appreciable increase in extraction by amalgamating and selective grinding of the sulphide portion of the ore.

## BLANKET CONCENTRATION AND FLOTATION

### *Test No. 12*

This test was made to note the effect of grinding in water, concentrating on blanket, regrinding blanket tailing, and concentrating by flotation.

It was desired to find out if the gold-bearing minerals could be removed by blanketing and flotation.

A sample of ore was ground in water to give a product 82 per cent −200 mesh. The ground ore was concentrated on a corduroy blanket sloping 2·5 inches in 12 inches. The blanket concentrate was panned to remove gangue and the sulphides were dried, weighed, and the gold content calculated.

The panning was returned to the blanket tailing, filtered, and reground in the ball mill at the same dilution with the following reagents:—

| | |
|---|---|
| Soda ash | 1·0 lb./ton |
| Aerofloat No. 31 | 0·035 " |

The following reagents were added to the flotation cell:—

| | |
|---|---|
| Potassium amyl xanthate | 0·1 lb./ton |
| Pine oil | 0·05 " |

The pH of the pulp during flotation was 8·8 to 8·9.

*Results:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| *Blanket Concentration:* | | | | |
| Feed | 100·00 | 0·195 | 100·00 | |
| Blanket concentrate | 0·67 | 10·82 | 37·18 | 149·3 : 1 |
| Blanket tailing | 99·33 | 0·1225 | 62·82 | |
| *Flotation:* | | | | |
| Feed (blanket tailing) | 100·00 | 0·1225 | 100·00 | |
| Flotation concentrate | 11·00 | 0·79 | 70·94 | 9·1 : 1 |
| Flotation tailing | 89·00 | 0·04 | 29·06 | |

*Summary of Results:*

| | Per cent |
|---|---|
| Distribution of gold in blanket concentrate | 37·18 |
| Distribution of gold in flotation concentrate, 70·94 × 62·82 | 44·56 |
| Distribution of gold in both concentrates | 81·74 |
| Loss of gold in the tailing, 29·06 × 62·82 | 18·26 |
| | 100·00 |

To discover if free gold or sulphides remained in the flotation tailing, a portion was concentrated on a Haultain super-panner.

The panner concentrate was found to contain considerable arsenopyrite and sulphides, which had not been recovered by flotation.

*Results of Concentration by Super-panner:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| Feed | 100·00 | 0·04 | 100·00 | |
| Panner concentrate | 0·31 | 9·69* | 75·10 | 322·6 : 1 |
| Panner tailing | 99·69 | 0·01 | 24·90 | |

*Calculated.

Part of the arsenopyrite had not been recovered in the flotation concentrate.

*Test No. 13*

This test was to find out the effect of adding copper sulphate in the flotation of the sulphides.

A sample of ore was ground in water to 89 per cent −200 mesh and treated as in Test No. 12.

The following reagents were added to the mill when regrinding the blanket tailing:—

| | |
|---|---|
| Soda ash | 1·0 lb./ton |
| Aerofloat No. 31 | 0·035 " |

The following reagents were added to the flotation cell:—

| | |
|---|---|
| Copper sulphate | 0·5 lb./ton |
| Potassium amyl xanthate | 0·1 " |
| Pine oil | 0·05 " |

*Results:*

| Products | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| *Blanket Concentration:* | | | | |
| Feed | 100·00 | 0·195 | 100·00 | |
| Blanket concentrate | 0·43 | 13·88 | 30·62 | 232·6 : 1 |
| Blanket tailing | 99·57 | 0·135 | 69·38 | |
| *Flotation:* | | | | |
| Feed (blanket tailing) | 100·00 | 0·135 | 100·00 | |
| Flotation concentrate | 13·77 | 0·92 | 93·63 | 7·3 : 1 |
| Flotation tailing | 86·23 | 0·01 | 6·37 | |

*Summary of Results:*

| | Per cent |
|---|---|
| Distribution of gold in blanket concentrate | 30·62 |
| Distribution of gold in flotation concentrate, 93·63 × 69·38 | 64·96 |
| Distribution of gold in both concentrates | 95·58 |
| Loss of gold in tailing, 6·37 × 69·38 | 4·42 |
| | 100·00 |

This test shows that the addition of copper sulphate reduced the tailing from 0·04 to 0·01 ounce of gold per ton.

### CONCENTRATION BY FLOTATION AND HYDRAULIC TRAP

*Test No. 14*

The purpose of this test was to note the effect of concentrating by flotation and then removing the free gold left in the flotation tailing by means of a hydraulic trap.

A sample of ore was ground in water to 85 per cent −200 mesh with the following reagents:—

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

The reagents added to the flotation cell were:—

| | |
|---|---|
| Copper sulphate | 0·5 lb./ton |
| Potassium amyl xanthate | 0·2 " |
| Pine oil | 0·1 " |

After flotation, the tailing was passed through a hydraulic trap. Free gold was found in the trap concentrate. The trap concentrate was added to the flotation concentrate.

*Results:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| Feed | 100·00 | 0·195 | 100·00 | |
| Flotation and classifier concentrate | 15·01 | 1·24 | 95·64 | 6·7 : 1 |
| Flotation tailing | 84·99 | 0·01 | 4·36 | |

The examination of the trap tailing by the Haultain super-panner showed only a very minute quantity of sulphides remaining in the tailing.

This panning operation reduced the flotation tailing from 0·01 to 0·005 ounce of gold per ton. This indicates that total removal of the sulphides should give a tailing of 0·005 ounce of gold per ton.

*Test No. 15*

The test was made to find out the extraction obtained by cyaniding a flotation concentrate.

A representative portion of −14-mesh ore was ground in ball mills, dilution 4 : 3 in water, to give a product 90 per cent −200 mesh.

The same reagents were used in flotation as were used in Test No. 14, i.e.:—

*To the Ball Mill:*

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

*To the Flotation Cell:*

| | |
|---|---|
| Copper sulphate | 0·5 lb./ton |
| Potassium amyl xanthate | 0·2 " |
| Pine oil | 0·1 " |

The flotation tailing was concentrated by a hydraulic trap and the trap concentrate was added to the flotation concentrate.

This mixture was reground in a ball mill in cyanide solution, 2·0 pounds of potassium cyanide per ton, for 1 hour. The pulp was agitated in cyanide solution, 2·0 pounds of potassium cyanide per ton, at a dilution of 2·5 : 1. After 48 hours a sample was cut out for assay and the agitation was continued for 72 hours, when another sample was cut. The agitation was stopped after 92 hours, and the remaining pulp was assayed.

*Results:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| *Flotation:* | | | | |
| Feed | 100·00 | 0·195 | 100·00 | |
| Flotation and classifier concentrate | 15·99 | 1·17 | 95·69 | 6·3 : 1 |
| Tailing | 84·01 | 0·01 | 4·31 | |

*Cyanidation of Flotation Concentrate:*

| Agitation, hours | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton of concentrate | | | | | |
|---|---|---|---|---|---|---|---|---|---|
| | | | | During agitation | | In ball mill | | Total | |
| | Feed | Tailing | | KCN | CaO | KCN | CaO | KCN | CaO |
| 48 | 1·17 | 0·22 | 81·20 | 4·76 | 8·78 | 2·60 | 57·0 | 7·36 | 65·78 |
| 72 | 1·17 | 0·20 | 82·91 | 5·58 | 9·25 | 2·60 | 57·0 | 8·18 | 66·25 |
| 92 | 1·17 | 0·18 | 84·62 | 8·26 | 11·70 | 2·60 | 57·0 | 10·86 | 68·70 |

*Summary (Extraction in Terms of Feed):*

| Agitation, hours | Extraction by cyanidation | Recovery by flotation | Total extraction, per cent |
|---|---|---|---|
| 48 | 81·20 | 95·69 | 77·70 |
| 72 | 82·91 | 95·69 | 79·34 |
| 92 | 84·62 | 95·69 | 80·97 |

The results of the test show that prolonged agitation in cyanide solution is required to dissolve the gold in the concentrate.

CYANIDATION AND FLOTATION OF CYANIDE TAILING

*Test No. 16*

This was to note the effect of grinding in cyanide solution, agitating for 24 hours, filtering, washing, and concentrating the cyanide tailing by flotation and regrinding and agitating the flotation concentrate in cyanide solution.

A sample of ore was ground in cyanide solution, 1·0 pound of potassium cyanide per ton, to 72 per cent −200 mesh. The pulp was diluted to 1·5 : 1 and the solution was made up to 1·0 pound of potassium cyanide per ton. The agitation was concluded after 24 hours and the pulp was filtered, washed, and sampled. The cyanide solution was saved.

The cyanide tailing was transferred to a flotation cell and conditioned for 10 minutes with 2·0 pounds of soda ash per ton and for 5 minutes with 0·5 pound of copper sulphate per ton. Then 0·2 pound of potassium amyl xanthate and 0·10 pound of pine oil per ton were added and a concentrate was removed.

The flotation tailing was passed through a hydraulic trap and the trap concentrate was added to the flotation concentrate. The mixed concentrates were then reground for 1 hour in a ball mill, using the cyanide solution from the primary grind, made up to 2·0 pounds of potassium cyanide per ton. The reground concentrate was then diluted to 2·5 : 1 in cyanide solution, made up to 2·0 pounds of potassium cyanide per ton, and agitated for periods of 30 and 78·5 hours. Lime was used to supply protective alkalinity.

*Grinding and Agitation in Cyanide Solution:*

| Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|
| Feed | Tailing | | KCN | CaO |
| 0·195 | 0·045 | 76·92 | 0·94 | 3·42 |

*Flotation of Cyanide Tailing:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution of gold in products, per cent | Ratio of concentration |
|---|---|---|---|---|
| Feed | 100·00 | 0·045 | 100·00 | |
| Flotation concentrate | 13·52 | 0·301 | 90·44 | 7·4 : 1 |
| Flotation tailing | 86·48 | 0·005 | 9·56 | |

*Regrinding and Agitation of Flotation Concentrate in Cyanide Solution:*

| Agitation, hours | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| 30·0 | 0·301 | 0·24 | 20·27 | 5·73 | 71·1 |
| 78·5 | 0·301 | 0·23 | 23·59 | 7·67 | 75·7 |

*Summary of Results:*

| | 30 hr. | 78·5 hr. |
|---|---|---|
| Extraction by cyanidation of the ore | 76·92 | 76·92 |
| Gold remaining in the ore, 100—76·92 .......23·08 per cent | | |
| Concentration by flotation, 23·1 × 90·44 ....... 20·89 per cent | | |
| Extracted by cyanidation of concentrate— | | |
| 20·89 × 20·27 | 4·23 | |
| 20·89 × 23·59 | | 4·92 |
| Total extraction of gold | 81·15 | 81·84 |

The results of this test again show the difficulty of extracting gold from the sulphides.

*Test No. 17*

The details of this test were similar to those of Test No. 16 except that the ore was ground to 87 per cent −200 mesh instead of 72 per cent −200 mesh.

The extraction by cyanidation of the ore was the same, 76·92 per cent of the gold.

*Grinding and Cyanidation of the Ore:*

| Agitation, hours | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| 24 | 0·195 | 0·045 | 76·92 | 0·91 | 3·50 |

*Flotation of Cyanide Tailing:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| Feed | 100·00 | 0·045 | 100·00 | |
| Flotation concentrate | 13·50 | 0·301 | 90·44 | 7·41 : 1 |
| Flotation tailing | 86·50 | 0·005 | 9·56 | |

*Grinding and Cyanidation of Flotation Concentrate:*

| Agitation, hours | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| 43 | 0·301 | 0·28 | 6·98 | 5·1 | 57·8 |

*Summary of Results:*

| | Per cent |
|---|---|
| Extraction by cyanidation of the ore | 76·92 |
| Gold remaining in ore, 100·0—76·92 .......23·08 per cent | |
| Concentration by flotation, 23·1 × 90·44 .......20·89 per cent | |
| Extracted by cyanidation of concentrate, 20·89 × 6·98 | 1·45 |
| Total extraction of gold | 78·37 |

The flotation concentrate when reground and cyanided for 43 hours had a tailing assay of 0·28 ounce of gold per ton from a calculated feed of 0·30 ounce of gold per ton. This extraction when referred to the ore as in Test No. 16 was only 1·45 per cent, making a total extraction of 78·37 per cent of the gold.

### *Test No. 18*

This was to note the effect of shorter periods of agitation in the treatment of reground flotation concentrate prepared similarly to that in Tests Nos. 16 and 17.

A screen test showed 68 per cent −200 mesh.

*Grinding and Cyanidation of the Ore:*

| Agitation, hours | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| 24 | 0·195 | 0·04 | 79·50 | 0·67 | 4·10 |

*Flotation of Cyanide Tailing:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| Feed | 100·00 | 0·04 | 100·00 | |
| Flotation concentrate | 14·29 | 0·25 | 89·25 | 7 : 1 |
| Flotation tailing | 85·71 | 0·005 | 10·75 | |

*Grinding and Cyanidation of Flotation Concentrate:*

| Agitation, hours | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|---|
| | Feed | Tailing | | KCN | CaO |
| 7 | 0·25 | 0·22 | 12·00 | 3·71 | 29·78 |
| 18 | 0·25 | 0·22 | 12·00 | 3·76 | 36·94 |

*Summary of Results:*

| | Per cent |
|---|---|
| Gold extracted by cyanidation of ore | 79·50 |
| Gold left in tailing, 100−79·50 . . . 20·50 per cent | |
| Gold concentrated in flotation concentrate, 20·50 × 89·25 . . . 18·31 per cent | |
| Gold extracted by cyanidation of concentrate, 18·31 × 12·0 | 2·20 |
| Total extraction of gold | 81·70 |

Although 18·31 per cent of the gold was reported in the flotation concentrate, only 2·2 per cent was extracted by cyanidation.

CYANIDATION OF ROASTED CONCENTRATE

*Test No. 19*

This was to note the effect of cyaniding a roasted flotation concentrate prepared similarly to that of Test No. 15.

Four samples of ore were ground in water, using the following reagents, soda ash, 2·0 pounds per ton, and Barrett No. 4 oil, 0·1 pound per ton, for different lengths of time in order to note the effect of coarse and fine grinding on obtaining a clean flotation tailing.

The reagents added to the cell were:—

| | |
|---|---|
| Copper sulphate | 1·0 lb./ton |
| Potassium amyl xanthate | 0·2 " |
| Pine oil | 0·1 " |

The flotation tailing from each portion was sampled separately and a screen test was made on each.

The four concentrates were mixed, dried, and after sampling for assay were roasted, first at a low temperature at about 400° C. in order to drive off arsenic. The temperature then was raised to about 600° C. to drive off all the sulphur to give a dead roast. When cool, the calcine was weighed and the ignition loss was found to be about 61 per cent.

After sampling for assay, the calcine was reground in a ball mill with lime. Two lots were agitated in cyanide solution, 3·0 pounds of potassium cyanide per ton, at a dilution of 3 : 1, for periods of 24 and 48 hours. A screen test shows the grind to be 99 per cent −325 mesh.

*Results of Flotation Concentration:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| Feed | 100·00 | 0·183* | 100·00 | |
| Flotation concentrate | 13·79 | 1·29 | 97·04 | 7·25 : 1 |
| Flotation tailing | 86·21 | 0·0063* | 2·96 | |

*Calculated values from assays of the products of the test.

*Tailings from Four Flotation Tests:*

| Grind, per cent −200 mesh | Assay of tailings, Au, oz./ton |
|---|---|
| 63·4 | 0·01 |
| 74·6 | 0·005 |
| 83·5 | 0·005 |
| 89·1 | 0·005 |
| Combined tailing | 0·0063 |

*Cyanidation of Roasted Concentrates (Calcine):*

| Agitation, hours | Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton calcine | |
|---|---|---|---|---|---|
| | (Calcine) Feed | Tailing | | KCN | CaO |
| 24 | 2·12 | 0·45 | 78·77 | 3·18 | 15·10 |
| 48 | 2·12 | 0·75 | 64·62 | 4·68 | 16·25 |

Extraction from 24-hour cyanidation, 78·77 × 97·04... 76·44 per cent of the gold
Extraction from 48-hour cyanidation, 64·62 × 97·04... 62·71 " "

The extraction obtained from the roasted flotation concentrate was less than that obtained from the unroasted flotation concentrate of Test 15; either the gold did not dissolve in the time allowed or it was reprecipitated from the solution during the 48-hour test.

It is apparently unnecessary to grind finer than 75 per cent −200 mesh to obtain a maximum recovery by flotation.

### CYANIDATION AND FLOTATION OF THE CYANIDE TAILING

*Test No. 20*

The purpose was to make a high-grade flotation concentrate from the cyanide tailing.

A sample of ore was ground to 76 per cent −200 mesh in a cyanide solution, 1·0 pound of potassium cyanide per ton. The pulp was diluted to 2 : 1 and the strength of the solution was brought up to 1·0 pound of potassium cyanide per ton. The agitation was stopped after 24 hours.

The cyanide tailing was filtered, washed, and sampled. The filter cake was transferred to a flotation cell and conditioned for 10 minutes with 2·0 pounds of soda ash per ton and for 5 minutes with 1·0 pound of copper sulphate per ton. The following reagents were then added and a concentrate was removed:—

| | |
|---|---|
| Potassium amyl xanthate | 0·2 lb./ton |
| Pine oil | 0·1 " |

The flotation concentrate was recleaned in a cleaner cell without using any reagents. The cleaner cell tailing is shown as flotation middling in the results.

*Cyanidation:*

| Assay, Au, oz./ton | | Extraction, per cent | Reagents consumed, lb./ton | |
|---|---|---|---|---|
| Feed | Tailing | | KCN | CaO |
| 0·195 | 0·04 | 79·50 | 0·72 | 3·95 |

*Flotation:*

| Product | Weight, per cent | Assay, Au, oz./ton | Distribution, per cent | Ratio of concentration |
|---|---|---|---|---|
| Feed | 100·00 | 0·04 | 100·00 | |
| Flotation concentrate | 6·70 | 0·46 | 73·35 | 15 : 1 |
| Flotation middling | 2·87 | 0·075 | 5·12 | |
| Flotation tailing | 90·43 | 0·01 | 21·53 | |

*Summary of Test:*

| | | Per cent |
|---|---|---|
| Extraction of gold by cyanidation | | 79·50 |
| Gold left in cyanide tailing, 100−79·5 | 20·50 per cent | |
| Concentration by flotation, 73·35 × 20·50 | 15·04 per cent | |
| Gold in middling, 5·12 × 20·50 | 1·05 " | |
| Gold in tailing, 21·53 × 20·50 | 4·41 " | |
| | 20·50 per cent | |
| | | 20·50 |
| | | 100·00 |

In practice, part of the gold in the middling would be divided between the concentrate and the tailing.

It should not be difficult to make a flotation concentrate of 0·5 ounce of gold per ton, which could be profitably treated at a smelter.

## SUMMARY OF THE TEST WORK

The free gold in the ore is liberated at a grind of about 75 per cent −200 mesh and may be recovered by traps, jigs, or blankets (Test No. 1). A concentrate of this nature was examined and described under "Characteristics of the Ore".

Straight cyanidation gave a maximum extraction of 82 per cent of the gold at 89 per cent −200 mesh, in 24 hours. Grinding to 99 per cent −200 mesh and agitating for 72 hours did not increase the extraction, which was finished in 24 hours. Coarser grinding dropped the extraction to 79·5 per cent of the gold. (Test No. 2.)

Regrinding the sulphide portion of the ore and agitating this in cyanide solution with the main portion resulted in substantially the same extraction, i.e. 84·62 per cent of the gold and a tailing of 0·03 ounce of gold per ton. (Test No. 3.) In two similar tests (Tests Nos. 4 and 5), the extraction dropped to 82 per cent of the gold and the tailings slightly increased to 0·035 ounce of gold per ton.

Amalgamating the trap concentrate to remove free gold and cyaniding the residue with the main portion of the ore gave the same result as straight cyanidation. (Tests Nos. 7, 8, 9, 10, and 11.)

With blanket concentration followed by flotation, the results of flotation show quite clearly that copper sulphate is necessary. (Test No. 12.) As indicated in Tests Nos. 13, 14, and 19, it lowers the tailing from 0·04 to 0·005 ounce of gold per ton.

Cyanidation of trap and flotation concentrates gave a maximum extraction of 81 per cent of the gold. (Test No. 15.)

Cyanidation of the ore followed by flotation of the cyanide tailing and cyanidation of the reground concentrate from the cyanide tailing gave an overall extraction of 81 per cent. (Test No. 16.) Twelve per cent of the gold in the reground concentrate was dissolved in 7 hours and only 23·6 per cent of the gold was taken into solution in 78·5 hours of agitation. Referred to the feed this amounts to 2 per cent for the 7-hour agitation and 4 per cent for the 78-hour agitation. (Tests Nos. 18 and 16.)

Cyanidation of a roasted concentrate reground to 99 per cent −325 mesh and cyanided for 24 hours gave an extraction of 76·4 per cent of the gold. Increased time of agitation did not improve the results but gave a lower extraction. (Test No. 19.)

A flotation concentrate containing 0·46 ounce of gold per ton can be made from a cyanide tailing of 0·04 ounce of gold per ton and this could be shipped to a smelter for treatment. (Test No. 20.)

## CONCLUSIONS

The test work shows that a maximum extraction of 82 per cent of the gold can be obtained by straight cyanidation in 24 hours at a grind of 89 per cent −200 mesh. Grinding to 99 per cent −200 mesh and agitating up to 72 hours effects no improvement in extraction. An extraction of 79·5 per cent of the gold is obtained in 24 hours at a grinding of 76 per cent −200 mesh.

Recyanidation of the sulphide part of the tailing did not extract an appreciable amount of gold. This indicates that the gold is enclosed in the sulphides and is not attacked by the solution and therefore cannot be extracted at the degree of grinding that is economical.

## RECOMMENDATIONS FOR TREATMENT

It is recommended that the ore be treated by straight cyanidation at a grind of from 85 to 90 per cent −200 mesh. If a higher recovery is desired, the cyanide tailing may be concentrated by flotation to make a shipping grade of concentrate for treatment at a smelter.

## Ore Dressing and Metallurgical Investigation No. 716

### BULK FLOTATION OF ALDERMAC CHALCOPYRITE-PYRITE ORE

*Shipment.* A carload of copper-iron pyrite ore, weighing 56 tons, was received March 18, 1937, from the Aldermac Copper Corporation, Limited, Arntfield, Quebec.

The shipment was taken from ore being fed to the mill and was sent for tests to determine whether bulk flotation of the copper and iron sulphides with subsequent separation of the copper sulphides from the pyrite would be more advantageous than the selective flotation now being used at the property.

The company is endeavouring to produce a high-grade copper concentrate for shipment to a smelter, and to obtain an iron-pyrite concentrate containing over 48 per cent sulphur and with a minimum zinc content.

The ore was of the same character as shipments previously examined and reported on in Mines Branch Report No. 724—Investigations in Ore Dressing and Metallurgy, 1930. The present samples came from sections of the ore-body that contained a low percentage of zinc. Pyrrhotite constituted a considerable percentage of the sulphides.

### Experimental Tests

#### Section I

##### BULK FLOTATION WITH POTASSIUM AMYL XANTHATE AS THE COLLECTING REAGENT

In these runs, each of which was continuous for a period of eight hours, the intent was to produce a bulk concentrate and to condition this concentrate with lime to depress iron pyrite. The copper sulphides were then refloated, leaving a tailing which constituted the iron-pyrite concentrate.

The ore was fed at the rate of 500 pounds per hour to a ball mill in closed circuit with a classifier. The classifier overflow passed to a conditioning tank in which contact was maintained for about 20 minutes. Air was admitted at the bottom of this tank for air conditioning. The pulp from the conditioning tank flowed to Cell No. 1 of a bank of ten mechanically agitated flotation cells each of about one cubic foot capacity. The first five produced the bulk concentrate, while the concentrate taken off the last five cells was returned to Cell No. 2.

The bulk concentrate was pumped to a second conditioning tank where it was conditioned with lime for 15 to 20 minutes and then passed to Cell No. 2 of a second unit of ten cells. The concentrate from Cells

Nos. 2 and 3 was recleaned in Cell No. 1, producing a finished copper concentrate. The concentrate from Cells Nos. 4 to 10 was returned to Cell No. 2 with the feed. The tailing from Cell No. 10 constituted the pyrite concentrate.

The bulk flotation tailing was passed over a Wilfley table, which served as a guide to flotation. Operation was considered satisfactory when no iron pyrite was visible on the table. Large quantities of pyrrhotite were concentrated.

*Mill Run No. 1*

In this run, copper sulphate was added to the bulk flotation cells, and cyanide in addition to lime was used in the conditioning of the bulk concentrate to depress pyrite.

A screen test made on the classifier overflow shows:

| Mesh | Weight, per cent |
|---|---|
| + 48 | 0·9 |
| − 48+ 65 | 2·3 |
| − 65+100 | 7·3 |
| −100+150 | 10·2 |
| −150+200 | 15·6 |
| −200 | 63·7 |
| | 100·0 |

*Reagents to Ball Mill:*

| | Lb./ton |
|---|---|
| Soda ash | 2·0 |

*Reagents, Bulk Flotation:*

| | | |
|---|---|---|
| To Conditioning Tank: | Amyl xanthate | 0·216 |
| Cell No. 1: | Pine oil | 0·066 |
| Cell No. 5: | Amyl xanthate | 0·081 |
| Cell No. 6: | Amyl xanthate | 0·081 |
| | Copper sulphate | 0·26 |
| Cell No. 9: | Amyl xanthate | 0·054 |
| | Total xanthate | 0·431 |

*Reagents, Copper Flotation:*

| | |
|---|---|
| Lime | 6·9 |
| Sodium cyanide | 0·11 |
| Feed to flotation | 35 per cent solids |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent |
|---|---|---|---|
| Feed | 1·41 | 0·50 | 35·08 |
| Classifier overflow | 1·56 | 0·45 | 32·86 |
| Bulk concentrate | 2·14 | 0·70 | 40·30 |
| Copper concentrate | 23·80 | 5·18 | 29·13 |
| Pyrite concentrate | 0·10 | 0·20 | 38·80 |
| Mill tailing | 0·05 | 0·05 | 9·83 |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·56 | 0·45 | 100·0 | 100·0 |
| Copper concentrate | 6·24 | 23·80 | 5·18 | 94·9 | 68·8 |
| Pyrite concentrate | 66·22 | 0·10 | 0·20 | 4·2 | 28·3 |
| Mill tailing | 27·54 | 0·05 | 0·05 | 0·9 | 2·9 |

This combination of reagents floated practically all the sulphides. The pyrite concentrate was low grade, below the limit of 48 per cent sulphur, as indicated by the iron content. The zinc in the ore floated into the copper concentrate.

## EFFECT OF CYANIDE

### *Mill Run No. 2*

In view of the large quantity of material floated in the preceding day's run, the reagents were slightly reduced, the xanthate to 0·405 pound per ton and the pine oil to 0·04 pound per ton. All other conditions were the same as in Run No. 1.

Half way through the run, the cyanide added to the bulk concentrate to depress iron pyrite was omitted, and a second set of samples of the copper flotation circuit was taken.

Classifier overflow................ 35 per cent solids, 60·4 per cent −200 mesh

### *Mill Run No. 2A—Cyanide Used*

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·40 | 0·25 | 36·2 | .......... |
| Classifier overflow | 1·32 | 0·25 | 34·3 | .......... |
| Bulk concentrate | 2·36 | 0·35 | 43·3 | .......... |
| Copper concentrate | 23·78 | 4·52 | 29·2 | .......... |
| Pyrite concentrate | 0·19 | 0·05 | 43·9 | 45·3 |
| Mill tailing | 0·11 | Nil | 21·2 | .......... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·32 | 0·25 | 100·0 | 100·0 |
| Copper concentrate | 4·95 | 23·76 | 4·52 | 89·1 | 90·2 |
| Pyrite concentrate | 48·81 | 0·19 | 0·05 | 7·0 | 9·8 |
| Mill tailing | 46·24 | 0·11 | Nil | 3·9 | ......... |

*Mill Run No. 2B—No Cyanide Used*

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Copper concentrate | 22·33 | 4·05 | 29·9 | ........... |
| Pyrite concentrate | 0·19 | 0·08 | 44·7 | 48·18 |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·32 | 0·25 | 100·0 | 100·0 |
| Copper concentrate | 5·27 | 22·33 | 4·05 | 89·2 | 84·6 |
| Pyrite concentrate | 48·49 | 0·19 | 0·08 | 6·9 | 15·4 |
| Mill tailing | 46·24 | 0·11 | Nil | 3·9 | ......... |

The use of cyanide seems slightly to accelerate the flotability of the zinc minerals. When it is omitted, more zinc is found in the pyrite concentrate. There is no apparent advantage in its use, as the lime used to depress the pyrite seems quite sufficient. Therefore, cyanide was used in the later tests only when it is so stated.

*Mill Run No. 3—No Copper Sulphate*

In the previous runs, the zinc sulphides floated readily and reported in the copper concentrate. In this run, in order to decrease the flotability of the zinc minerals, no copper sulphate was added to the bulk float.

Classifier overflow.... 35 per cent solids, 60·4 per cent −200 mesh, 5·0 per cent +65 mesh

*Reagents:*

| | Lb./ton |
|---|---|
| To Ball Mill: | |
| Soda ash | 2·0 |
| To Bulk Float: | |
| Amyl xanthate | 0·451 |
| Pine oil | 0·04 |
| To Copper Float: | |
| Lime | 7·0 |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·42 | 0·25 | 36·5 | ......... |
| Classifier overflow | 1·51 | 0·25 | 34·8 | ........... |
| Bulk concentrate | 2·72 | 0·13 | 43·5 | ........... |
| Copper concentrate | 25·20 | 1·46 | 30·0 | ........... |
| Pyrite | 0·17 | 0·08 | 44·7 | 48·96 |
| Mill tailing | 0·14 | 0·38 | 25·0 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed | 100·00 | 1·51 | 0·25 | 100·0 | 100·0 |
| Copper concentrate | 5·42 | 25·20 | 1·46 | 90·3 | 26·8 |
| Pyrite concentrate | 47·78 | 0·17 | 0·08 | 5·4 | 12·9 |
| Mill tailing | 46·80 | 0·14 | 0·38 | 4·3 | 60·3 |

The effect of copper sulphate is very noticeable. In Mill Run No. 2B, which is the same as the present one in all respects apart from the use of copper sulphate, 84·6 per cent of the zinc was floated into the copper concentrate and none left in the mill tailing. In this test, 60·3 per cent of the zinc passed out of the circuit with the mill tailing, and 26·8 per cent is found with the copper concentrate. The same percentage of zinc is found in the iron pyrite in both cases.

### *Mill Run No. 4—Copper Sulphate Added to Copper Flotation*

To determine if the zinc content of the iron-pyrite concentrate could be reduced, copper sulphate at the rate of 0·25 pound per ton of feed was added to the bulk concentrate before this was conditioned with lime. All other conditioning was as in Mill Run No. 3.

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·53 | 0·28 | 35·7 | ............ |
| Classifier overflow | 1·53 | 0·28 | 34·7 | ............ |
| Bulk concentrate | 2·92 | 0·38 | 43·4 | ............ |
| Copper concentrate | 26·08 | 1·36 | 30·3 | ............ |
| Pyrite concentrate | 0·17 | 0·08 | 44·6 | 48·67 |
| Mill tailing | 0·17 | 0·20 | 27·1 | ............ |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·53 | 0·28 | 100·0 | 100·0 |
| Copper concentrate | 5·26 | 26·08 | 1·36 | 89·5 | 34·4 |
| Pyrite concentrate | 44·32 | 0·17 | 0·08 | 4·9 | 17·1 |
| Mill tailing | 50·42 | 0·17 | 0·20 | 5·6 | 48·5 |

Copper sulphate added to the copper flotation circuit has no apparent effect on reducing the zinc content of the iron-pyrite concentrate.

### *Mill Run No. 5*

Seven pounds of lime added to the bulk concentrate depressed the iron pyrite, making it possible to obtain a high-grade copper concentrate. In this run the quantity of lime was reduced to 2·9 pounds. Other reagents were the same as in the preceding run.

The grind was slightly finer, the classifier overflow having the following screen analysis:

| Mesh | Weight, per cent |
|---|---|
| + 48 | 0·9 |
| − 48+ 65 | 2·5 |
| − 65+100 | 5·0 |
| −100+150 | 10·5 |
| −150+200 | 14·3 |
| −200 | 66·8 |
| | 100·0 |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·46 | 0·30 | 36·1 | ........... |
| Classifier overflow | 1·51 | 0·30 | 34·5 | ........... |
| Bulk concentrate | 2·86 | 0·30 | 43·8 | ........... |
| Copper concentrate | 25·08 | 1·41 | 31·9 | ........... |
| Pyrite concentrate | 0·20 | Nil | 44·6 | 49·10 |
| Mill tailing | 0·12 | 0·30 | 27·7 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·51 | 0·30 | 100·0 | 100·0 |
| Copper concentrate | 5·40 | 25·08 | 1·41 | 90·1 | 33·9 |
| Pyrite concentrate | 45·10 | 0·20 | Nil | 6·0 | .......... |
| Mill tailing | 49·50 | 0·12 | 0·30 | 3·9 | 66·1 |

A microscopic examination of polished sections prepared from the mill tailing showed the constituents to be:

| | |
|---|---|
| Gangue | Major quantity |
| Pyrite | Minor quantity |
| Pyrrhotite | Minor quantity |
| Sphalerite | Small quantity |
| Chalcopyrite | Very small quantity |

The chalcopyrite is largely combined with coarse particles of gangue, although a small proportion is combined with sphalerite and very rarely with pyrite. A small proportion of the chalcopyrite occurs as very fine free grains (slime).

From 2·9 to 3 pounds of lime and from 15 to 20 minutes' conditioning are sufficient to depress the iron pyrite and to produce a high-grade copper concentrate. The slightly finer grind apparently liberates the zinc from the pyrite, as no zinc was found in the pyrite concentrate.

### *Mill Run No. 6*

The quantity of soda ash added to the mill was reduced in this test from two pounds to one pound per ton and the copper sulphate omitted from the copper flotation circuit. The number of cells in the copper circuit was reduced. Four cells were taken off the scavengers. The lime-conditioned bulk concentrate entered the second cell. Cells Nos. 2 and 3 produced a copper concentrate, which was cleaned in Cell No. 1. The scavenger concentrate from Cells Nos. 4, 5, and 6 was returned to Cell No. 2.

The grind also was somewhat coarser than in preceding runs, as shown by the following screen test on the classifier overflow:

| Mesh | Weight, per cent |
|---|---|
| + 48 | 1·1 |
| − 48+ 65 | 2·9 |
| − 65+100 | 6·8 |
| −100+150 | 12·0 |
| −150+200 | 20·7 |
| −200 | 56·5 |
| | 100·0 |

*Reagents:*

| | Lb./ton |
|---|---|
| To Ball Mill: | |
| Soda ash | 1·0 |
| To Bulk Float: | |
| Amyl xanthate | 0·451 |
| Pine oil | 0·04 |
| To Copper Float: | |
| Lime | 2·9 |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·53 | 0·30 | 35·4 | ............ |
| Classifier overflow | 1·48 | 0·30 | 35·4 | ............ |
| Bulk concentrate | 3·14 | 0·20 | 43·4 | ............ |
| Copper concentrate | 19·40 | 0·40 | 34·3 | ............ |
| Pyrite concentrate | 0·25 | 0·20 | 44·4 | 48·4 |
| Mill tailing | 0·14 | 0·40 | 26·9 | ............ |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·48 | 0·30 | 100·0 | 100·0 |
| Copper concentrate | 6·74 | 19·40 | 0·40 | 88·4 | 8·3 |
| Pyrite concentrate | 37·90 | 0·25 | 0·20 | 6·4 | 23·4 |
| Mill tailing | 55·36 | 0·14 | 0·40 | 5·2 | 68·3 |

A microscopic examination of polished sections of the iron-pyrite concentrate shows the constituents to be:

| | |
|---|---|
| Pyrite | Major quantity |
| Gangue | Minor quantity |
| Sphalerite | Small quantity |
| Chalcopyrite | Very small quantity |

The chalcopyrite occurs as small grains combined with gangue, with sphalerite, and free. Exceedingly rare small grains are combined with pyrite. In general, the chalcopyrite has been ground free of the pyrite.

An examination of the mill tailing showed it to be similar to that of Mill Run No. 5.

This set of conditions gave unsatisfactory flotation and poorer results. There was a lower recovery of iron pyrite, a lower grade copper concentrate, and more zinc in the iron-pyrite concentrate.

### *Mill Run No. 7*

In previous tests, the ball mill discharge was about 75 per cent solids. For this and all succeeding runs the mill discharge was held at 90 per cent solids. To make a maximum recovery of pyrite, as indicated by the performance of the table concentrating the mill tailing, the bulk flotation cells were operated at maximum capacity. In this run, the quantity of amyl xanthate added was increased by 0·027 pound per ton of ore feed and the soda ash fixed at 1·75 pound. The original flow-sheet was used, that employing ten cells on the copper flotation.

Classifier overflow....35 per cent solids.

*Screen Test:*

| Mesh | Weight, per cent |
|---|---|
| + 48 | 0·3 |
| − 48+ 65 | 2·5 |
| − 65+100 | 10·2 |
| −100+150 | 9·1 |
| −150+200 | 11·2 |
| −200 | 66·7 |
| | 100·0 |

*Reagents:*

| | Lb./ton |
|---|---|
| To Ball Mill: | |
| Soda ash | 1·75 |
| To Bulk Flotation: | |
| Amyl xanthate | 0·47 |
| Pine oil | 0·04 |
| To Copper Flotation: | |
| Lime | 2·9 |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent | Au, oz./ton |
|---|---|---|---|---|---|
| Feed | 1·50 | 0·25 | 35·6 | .......... | .......... |
| Classifier overflow | 1·60 | 0·25 | 33·5 | .......... | 0·01 |
| Bulk concentrate | 2·80 | 0·15 | 43·6 | .......... | .......... |
| Copper concentrate | 28·66 | 0·41 | 30·5 | .......... | 0·07 |
| Pyrite concentrate | 0·14 | 0·10 | 44·6 | 48·82 | 0·01 |
| Mill tailing | 0·15 | 0·43 | 25·3 | 12·80 | 0·005 |

Determinations of pyrrhotite showed that the copper concentrate contained 4·94 per cent and the pyrite concentrate 1·80 per cent.

*Results:*

| Product | Weight, per cent | Assay Per cent Cu | Assay Per cent Zn | Assay Oz./ton Au | Distribution, per cent Cu | Distribution, per cent Zn | Distribution, per cent Au |
|---|---|---|---|---|---|---|---|
| Feed—classifier overflow | 100·00 | 1·60 | 0·25 | 0·01 | 100·0 | 100·0 | 100·0 |
| Copper concentrate | 5·10 | 28·66 | 0·41 | 0·07 | 91·4 | 7·9 | 33·1 |
| Pyrite concentrate | 49·55 | 0·14 | 0·10 | 0·01 | 4·3 | 18·7 | 45·9 |
| Mill tailing | 45·35 | 0·15 | 0·43 | 0·005 | 4·3 | 73·4 | 21·0 |

### *Mill Run No. 8*

This run is the same as Mill Run No. 7 with 0·013 pound of amyl xanthate per ton added to the copper flotation in an endeavour to lower the copper content of the pyrite concentrate.

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·54 | 0·23 | 35·4 | ........... |
| Classifier overflow | 1·62 | 0·25 | 35·4 | ........... |
| Bulk concentrate | 3·00 | 0·13 | 43·6 | ........... |
| Copper concentrate | 28·36 | 0·40 | 31·1 | ........... |
| Pyrite concentrate | 0·17 | 0·07 | 45·0 | 49·36 |
| Mill tailing | 0·17 | 0·40 | 25·3 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent Cu | Assay, per cent Zn | Distribution, per cent Cu | Distribution, per cent Zn |
|---|---|---|---|---|---|
| Feed—classifier overflow | 100·00 | 1·62 | 0·25 | 100·0 | 100·0 |
| Copper concentrate | 5·15 | 28·36 | 0·40 | 90·1 | 8·3 |
| Pyrite concentrate | 46·13 | 0·17 | 0·07 | 4·8 | 13·0 |
| Mill tailing | 48·72 | 0·17 | 0·40 | 5·1 | 78·7 |

Additional xanthate added for the flotation of copper from the bulk concentrate is of no apparent benefit.

### *Mill Run No. 9*

This was a demonstration run made in the presence of J. Legg, Superintendent of the Aldermac Concentrator. It was a duplicate of Mill Run No. 8.

Classifier overflow........................................ 35 per cent solids

*Grind: Flotation Feed*

| Mesh | Weight, per cent |
|---|---|
| + 48 | 1·4 |
| − 48+ 65 | 2·4 |
| − 65+100 | 5·4 |
| −100+150 | 9·7 |
| −150+200 | 16·5 |
| −200 | 64·6 |
| | 100·0 |

*Reagents:*

| | Lb./ton |
|---|---|
| To Ball Mill: | |
| Soda ash | 1·75 |
| To Bulk Flotation: | |
| Amyl xanthate | 0·48 |
| Pine oil | 0·04 |
| To Copper Flotation: | |
| Lime | 2·9 |

*Assays:*

| | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent | Au, oz./ton |
|---|---|---|---|---|---|
| Feed | 1·50 | 0·30 | 34·9 | .......... | .......... |
| Classifier overflow | 1·54 | 0·30 | 35·4 | .......... | 0·01 |
| Bulk concentrate | 2·60 | 0·13 | 43·6 | .......... | .......... |
| Copper concentrate | 26·54 | 0·35 | 31·1 | .......... | 0·09 |
| Pyrite concentrate | 0·15 | 0·07 | 45·0 | 49·0 | 0·005 |
| Mill tailing | 0·19 | 0·40 | 25·3 | .......... | 0·0025 |

*Results:*

| Product | Weight, per cent | Assay | | | Distribution, per cent | | |
|---|---|---|---|---|---|---|---|
| | | Per cent | | Oz./ton | | | |
| | | Cu | Zn | Au | Cu | Zn | Au |
| Feed—classifier overflow | 100·00 | 1·54 | 0·30 | 0·01 | 100·0 | 100·0 | 100·0 |
| Copper concentrate | 5·22 | 26·54 | 0·35 | 0·09 | 89·7 | 8·0 | 56·3 |
| Pyrite concentrate | 50·96 | 0·15 | 0·07 | 0·005 | 4·9 | 15·6 | 30·6 |
| Mill tailing | 43·82 | 0·19 | 0·40 | 0·0025 | 5·4 | 76·4 | 13·1 |

These results show the uniformity of results and ease of operation. Consistent results can easily be obtained.

*Mill Run No. 10*

To note if the quantity of potassium amyl xanthate could be reduced by adding it at a different point in the circuit, the 0·216 pound per ton of ore which in previous runs was added to the conditioner ahead of the bulk flotation unit was now added direct to the grinding mill. The remaining xanthate added to the bulk flotation cells was reduced by one-third of the quantity previously used.

*Grind—Flotation Feed:*

| Mesh | Weight, per cent |
|---|---|
| + 48 | 1·5 |
| − 48+ 65 | 2·7 |
| − 65+100 | 5·8 |
| −100+150 | 9·7 |
| −150+200 | 14·9 |
| −200 | 65·4 |
| | 100·0 |

*Reagents:*

| | Lb./ton |
|---|---|
| To Ball Mill: | |
| Soda ash | 1·75 |
| Amyl xanthate | 0·216 |
| To Cell No. 1: | |
| Pine oil | 0·04 |
| To Cell No. 5: | |
| Amyl xanthate | 0·054 |
| To Cell No. 6: | |
| Amyl xanthate | 0·054 |
| To Copper Float: | |
| Lime | 2·9 |
| Amyl xanthate | 0·01 |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·52 | 0·30 | 35·3 | ........... |
| Classifier overflow | 1·56 | 0·30 | 35·3 | ........... |
| Bulk concentrate | 6·62 | 0·28 | 41·4 | ........... |
| Copper concentrate | 27·70 | 0·70 | 31·0 | ........... |
| Pyrite concentrate | 0·62 | 0·13 | 41·4 | 44·9 |
| Mill tailing | 0·17 | 0·33 | 35·1 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·56 | 0·30 | 100·0 | 100·0 |
| Copper concentrate | 4·78 | 27·70 | 0·70 | 84·8 | 10·7 |
| Pyrite concentrate | 16·77 | 0·62 | 0·13 | 6·7 | 6·9 |
| Mill tailing | 78·45 | 0·17 | 0·33 | 8·5 | 82·4 |

The froth on the bulk flotation cells was much thinner and less heavily mineralized in Cell No. 1 than in previous tests in which the same quantity of xanthate was added to the conditioning tank ahead of Cell No. 1. Apparently when the xanthate is added to the grinding mill it loses its effectiveness. The results obtained were poor, a very low recovery of iron pyrite being obtained and that of the copper being decreased.

*Mill Run No. 11*

In this run the quantity of amyl xanthate ordinarily added to the conditioning tank of the bulk float circuit was reduced from 0·216 to 0·16 pound per ton and added directly to Cell No. 1. This changed the character of the froth in the first cell, and the pine oil had to be increased from 0·04 to 0·05 pound per ton.

*Grind—Classifier Overflow:*

| Mesh | Weight, per cent |
|---|---|
| + 48 | 1·0 |
| − 48+ 65 | 2·1 |
| − 65+100 | 6·3 |
| −100+150 | 9·5 |
| −150+200 | 13·9 |
| −200 | 67·2 |
| | 100·0 |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·32 | 0·33 | 35·30 | ........... |
| Classifier overflow | 1·47 | 0·30 | 34·50 | ........... |
| Bulk concentrate | 2·88 | 0·23 | 43·70 | ........... |
| Copper concentrate | 25·64 | 0·70 | 31·60 | ........... |
| Pyrite concentrate | 0·13 | 0·13 | 45·30 | 49·76 |
| Mill tailing | 0·17 | 0·43 | 25·90 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·47 | 0·30 | 100·0 | 100·0 |
| Copper concentrate | 5·16 | 25·64 | 0·70 | 90·2 | 11·4 |
| Pyrite concentrate | 42·69 | 0·13 | 0·13 | 3·8 | 17·6 |
| Mill tailing | 52·15 | 0·17 | 0·43 | 6·0 | 71·0 |

It is apparent that amyl xanthate is most effective when added directly to the flotation cells.

*Mill Run No. 12*

This was a demonstration day's run made in the presence of Mr. McLeod, representing the Aldermac Corporation. The reagents used were those considered as standard:

| | Lb./ton |
|---|---|
| Soda ash | 1·75 |
| Amyl xanthate | 0·47 |
| Pine oil | 0·04 |
| Lime | 2·9 |

To note if the copper content of the iron-pyrite concentrate could be reduced 0·05 pound of diphenylquandine per ton of feed was mixed with the lime.

The grind, as shown by a screen test on the classifier overflow, was 64·5 per cent −200 mesh with 2·5 per cent +65 mesh.

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·34 | 0·30 | 35·3 | ........... |
| Classifier overflow | 1·53 | 0·30 | 34·7 | ........... |
| Bulk concentrate | 2·92 | 0·25 | 42·8 | ........... |
| Copper concentrate | 20·08 | 0·80 | 23·7 | ........... |
| Pyrite concentrate | 0·15 | 0·18 | 44·5 | 48·38 |
| Mill tailing | 0·13 | 0·43 | 23·9 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·53 | 0·30 | 100·0 | 100·0 |
| Copper concentrate | 6·98 | 20·08 | 0·80 | 91·5 | 16·1 |
| Pyrite concentrate | 43·27 | 0·15 | 0·18 | 4·2 | 22·4 |
| Mill tailing | 49·75 | 0·13 | 0·43 | 4·3 | 61·5 |

These results check very closely with the results of previous mill runs. Diphenylquandine had a beneficial effect on the character of the froth in the copper cleaner cell. When lime alone was used the froth was inclined to be dry. In this day's run the froth was brighter and flowed freely.

### *Mill Run No. 13*

Attempting to lower the copper content of the mill tailing, four additional cells were included in the bulk flotation circuit, making 14 cells in all. The first five produced a finished concentrate, while that from the last nine cells was returned to the feed end of the unit. The same reagents as used in Mill Run No. 12 were applied. The grind, as shown by the screen test on the classifier overflow, was 61·1 per cent −200 mesh with 4·2 per cent +65 mesh.

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·44 | 0·33 | 35·30 | ........... |
| Classifier overflow | 1·55 | 0·28 | 34·90 | ........... |
| Bulk concentrate | 2·82 | 0·20 | 42·60 | ........... |
| Copper concentrate | 18·60 | 0·60 | 34·90 | ........... |
| Pyrite concentrate | 0·17 | 0·13 | 43·50 | 47·04 |
| Mill tailing | 0·14 | 0·43 | 25·90 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·55 | 0·28 | 100·0 | 100·0 |
| Copper concentrate | 7·57 | 18·60 | 0·60 | 90·8 | 14·8 |
| Pyrite concentrate | 45·06 | 0·17 | 0·13 | 4·9 | 19·0 |
| Mill tailing | 47·37 | 0·14 | 0·43 | 4·3 | 66·2 |

Increasing the number of cells for the bulk flotation does not reduce the copper content of the mill tailing. The noticeable effect was to lower the grade of both the copper and the pyrite concentrates.

### *Mill Run No. 14*

This test has the same flow-sheet as that of Mill Run No. 13. The quantity of amyl xanthate used in previous runs was reduced from 0·48 to 0·29 pound per ton of feed. The effect of this reduction was very apparent. It was not possible to float all the iron pyrite, a large proportion of which was observed on the pilot Wilfley table.

| | |
|---|---|
| Grind | 65·7 per cent −200 mesh |
| Classifier overflow | 35 per cent solids |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·46 | 0·30 | 34·9 | ............ |
| Classifier overflow | 1·53 | 0·30 | 34·7 | ............ |
| Bulk concentrate | 3·68 | 0·25 | 42·8 | ............ |
| Copper concentrate | 26·04 | 0·76 | 31·0 | ............ |
| Pyrite concentrate | 0·16 | 0·10 | 45·3 | 49·10 |
| Mill tailing | 0·18 | 0·38 | 30·6 | ............ |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·53 | 0·30 | 100·0 | 100·0 |
| Copper concentrate | 5·25 | 26·04 | 0·76 | 89·3 | 13·0 |
| Pyrite concentrate | 33·36 | 0·16 | 0·10 | 3·5 | 10·9 |
| Mill tailing | 61·39 | 0·18 | 0·38 | 7·2 | 76·1 |

Reduction of the quantity of xanthate results in a low yield of iron-pyrite concentrate. In other tests, from 45 to 50 per cent of the weight of feed was recovered as a pyrite concentrate. In this run, 33 per cent was recovered.

## Section II

### FLOTATION OF ZINC SULPHIDES

The Aldermac ore-body contains varying percentages of zinc. The present sample contains only 0·3 per cent, but ore from other parts of the mine contains much more.

A few mill runs were made to determine the response of the zinc minerals to flotation.

The mill runs made at the beginning of this investigation showed that copper sulphate added to the bulk float resulted in the zinc minerals floating with the copper and iron sulphides. When no copper sulphate was used, a high percentage of the zinc was discarded in the mill tailing.

### *Mill Run No. 1*

The flow-sheet and reagents for the bulk and copper flotations were the same as in previous runs; 0·25 pound of copper sulphate per ton of feed was added to the bulk flotation tailing, which was pumped to a conditioning tank. The overflow from this tank passed to the first of three cells. The concentrate from these was then cleaned in one cell.

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·39 | 0·32 | 34·48 | ........... |
| Classifier overflow | 1·51 | 0·35 | 34·48 | ........... |
| Bulk concentrate | 3·04 | 0·22 | 43·66 | ........... |
| Copper concentrate | 26·82 | 0·25 | 31·22 | ........... |
| Pyrite concentrate | 0·15 | 0·15 | 44·68 | 49·32 |
| Zinc concentrate | 0·37 | 3·67 | 45·90 | ........... |
| Mill tailing | 0·17 | 0·35 | 26·92 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·51 | 0·35 | 100·0 | 100·0 |
| Copper concentrate | 5·06 | 26·82 | 0·25 | 90·1 | 7·2 |
| Pyrite concentrate | 41·67 | 0·15 | 0·15 | 4·2 | 35·8 |
| Zinc concentrate | 2·71 | 0·37 | 3·67 | 0·7 | 57·0 |
| Mill tailing | 50·56 | 0·15 | Nil | 5·0 | .......... |

Copper sulphate reactivates the zinc sulphides and allows them to be concentrated.

### *Mill Run No. 2*

This is a duplicate of the previous day's run, with the exception that 0·05 pound of diphenylquandine per ton of feed was added with the lime to the bulk copper-pyrite concentrate going to the conditioning tank. A zinc float was made on the bulk flotation tailing.

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·36 | 0·30 | 34·9 | ........... |
| Classifier overflow | 1·49 | 0·35 | 33·7 | ........... |
| Bulk concentrate | 2·56 | 0·25 | 43·0 | ........... |
| Copper concentrate | 19·40 | 0·25 | 32·6 | ........... |
| Pyrite concentrate | 0·10 | 0·20 | 45·5 | 49·32 |
| Zinc concentrate | 0·30 | 3·12 | 50·0 | ........... |
| Bulk flotation tailing | 0·15 | 0·35 | 24·5 | ........... |
| Mill tailing | 0·14 | Nil | 22·6 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·49 | 0·35 | 100·0 | 100·0 |
| Copper concentrate | 7·08 | 19·40 | 0·25 | 92·2 | 8·4 |
| Pyrite concentrate | 48·48 | 0·10 | 0·20 | 3·3 | 46·0 |
| Zinc concentrate | 3·08 | 0·30 | 3·12 | 0·6 | 45·6 |
| Mill tailing | 41·36 | 0·14 | Nil. | 3·9 | ......... |

Diphenylquandine increases the effectiveness of the copper float. The copper content of the pyrite concentrate is lower in this test than in any of the preceding ones. The grade of the copper concentrate is somewhat lower.

Flotation of zinc is again quite easily obtained.

### *Mill Run No. 3*

In the two previous tests, the grade of zinc concentrate was low. In this test, the bulk tailing constituting the feed to zinc flotation was conditioned for 15 minutes with 3 pounds of lime and 0·10 pound of cyanide per ton of original feed; 0·25 pound of copper sulphate was added and the zinc floated. Diphenylquandine was again added to the copper float.

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·34 | 0·30 | 34·5 | ........... |
| Classifier overflow | 1·46 | 0·25 | 32·6 | ........... |
| Bulk concentrate | 3·02 | 0·20 | 43·5 | ........... |
| Copper concentrate | 22·60 | 0·45 | 32·2 | ........... |
| Pyrite concentrate | 0·07 | 0·17 | 45·1 | 49·02 |
| Zinc concentrate | 0·31 | 9·22 | 48·6 | ........... |
| Mill tailing | 0·15 | 0·05 | 23·9 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·46 | 0·25 | 100·0 | 100·0 |
| Copper concentrate | 6·00 | 22·60 | 0·45 | 92·4 | 11·7 |
| Pyrite concentrate | 39·87 | 0·07 | 0·17 | 1·9 | 29·5 |
| Zinc concentrate | 1·18 | 0·31 | 9·22 | 0·2 | 47·3 |
| Mill tailing | 52·95 | 0·15 | 0·05 | 5·5 | 11·5 |

Conditioning the zinc flotation feed with lime and cyanide depresses the iron pyrite and pyrrhotite and allows the flotation of the zinc minerals. The production of a zinc concentrate of commercial grade from a feed containing a fair percentage of zinc should present no difficulty. Diphenylquandine is again effective in increasing the recovery of copper by reducing the copper content of the pyrite concentrate.

## Section III

### SPECIAL REAGENTS

Attention has already been drawn to the use of diphenylquandine as an auxiliary reagent, added to the copper flotation circuit. This water-soluble reagent improves the separation of copper sulphides from the iron pyrite. It, however, produces a slightly lower grade copper concentrate.

### *Mill Run No. 1*

In this run, the 0·05 pound of diphenylquandine that previously had been added to the bulk concentrate passing to the copper flotation circuit was added to the grinding mill. All other reagents were maintained as formerly. No zinc flotation was made.

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·44 | 0·33 | 34·9 | ........... |
| Classifier overflow | 1·51 | 0·33 | 34·3 | ........... |
| Bulk concentrate | 2·66 | 0·25 | 42·8 | ........... |
| Copper concentrate | 28·24 | 0·50 | 31·4 | ........... |
| Pyrite concentrate | 0·33 | 0·20 | 44·9 | 49·14 |
| Mill tailing | 0·16 | 0·48 | 25·3 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·51 | 0·33 | 100·0 | 100·0 |
| Copper concentrate | 4·69 | 28·24 | 0·50 | 87·6 | 6·8 |
| Pyrite concentrate | 49·36 | 0·23 | 0·20 | 7·5 | 28·8 |
| Mill tailing | 45·95 | 0·16 | 0·48 | 4·9 | 64·4 |

Diphenylquandine added to the grinding mill has no effect on lowering the grade of the mill tailing. The recovery of copper as concentrate is lower than usual owing to the copper content of the pyrite concentrate. This probably is due to the making of a too high-grade copper concentrate.

*Mill Run No. 2.—Dupont "Floto"*

Dupont "Floto" was stated to be a good collector reagent for copper and iron pyrite in solutions of low pH value, and a poor collector of zinc sulphides.

This reagent was substituted in equal quantities for amyl xanthate. Grinding was done in a natural circuit, no alkaline reagent being added. The classifier overflow showed a pH value of 8·4, and the feed to flotation after aeration in the conditioning tank showed pH value of 8·0.

*Reagents:*

| | Lb./ton |
|---|---|
| To Conditioning Tank: | |
| "Floto" | 0·216 |
| To Cell No. 1: | |
| Pine oil | 0·04 |
| To Cell No. 5: | |
| "Floto" | 0·08 |
| To Cell No. 6: | |
| "Floto" | 0·08 |
| To Cell No. 9: | |
| "Floto" | 0·05 |
| To Copper Flotation: | |
| Lime | 2·9 |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·46 | 0·25 | 35·3 | ........... |
| Classifier overflow | 1·38 | 0·25 | 35·3 | ........... |
| Bulk concentrate | 2·56 | 0·13 | 44·5 | ........... |
| Copper concentrate | 29·16 | 0·25 | 30·4 | ........... |
| Pyrite concentrate | 0·31 | 0·05 | 45·5 | 50·72 |
| Mill tailing | 0·46 | 0·40 | 29·2 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·38 | 0·25 | 100·0 | 100·0 |
| Copper concentrate | 3·42 | 29·16 | 0·25 | 72·1 | 3·4 |
| Pyrite concentrate | 40·44 | 0·31 | 0·05 | 9·1 | 7·9 |
| Mill tailing | 56·14 | 0·46 | 0·40 | 18·8 | 88·7 |

The air in the air-conditioning tank apparently had a detrimental effect on "Floto", as a very poor froth resulted. The recovery of copper in the bulk flotation circuit was very low. A screen analysis of the tailing from this circuit showed 15·5 per cent +100 mesh, which assayed 0·78 per cent copper. The −100-mesh portion contained 0·38 per cent copper.

*Mill Run No. 3*

This run is similar to the preceding, with the exception that the "Floto" reagent was added directly to Cell No. 1 of the bulk flotation circuit. To improve flotation of copper, 0·5 pound of soda ash per ton was added to the grinding mill, which gave the feed to flotation a pH value of 8·0.

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·36 | 0·30 | 35·3 | ........... |
| Classifier overflow | 1·48 | 0·30 | 35·3 | ........... |
| Bulk concentrate | 2·48 | 0·18 | 43·7 | ........... |
| Copper concentrate | 28·56 | 0·50 | 31·2 | ........... |
| Pyrite concentrate | 0·29 | 0·10 | 45·3 | 50·08 |
| Mill tailing | 0·30 | 0·55 | 25·5 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·48 | 0·30 | 100·0 | 100·0 |
| Copper concentrate | 4·19 | 28·56 | 0·50 | 80·9 | 6·5 |
| Pyrite concentrate | 49·86 | 0·29 | 0·10 | 9·8 | 15·4 |
| Mill tailing | 45·95 | 0·30 | 0·55 | 9·3 | 78·1 |

Under these conditions, the recovery of iron pyrite is excellent, about 50 per cent of the weight of feed being recovered as a concentrate assaying 50 per cent sulphur. However, the recovery of copper is again low.

*Mill Run No. 4—Sodium Silicate*

In this run, 1·5 pounds of sodium silicate was added to the grinding mill, in place of soda ash. Potassium amyl xanthate, 0·47 pound per ton, was used as collector reagent. In all other details this run is the same as the standard bulk flotation runs.

*Grind—Classifier Overflow:*

| Mesh | Weight, per cent |
|---|---|
| + 48 | 0·8 |
| − 48+ 65 | 2·1 |
| − 65+100 | 5·1 |
| −100+150 | 9·9 |
| −150+200 | 13·9 |
| −200 | 68·2 |
| | 100·0 |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·56 | 0·28 | 35·55 | ........... |
| Classifier overflow | 1·54 | 0·30 | 35·13 | ........... |
| Bulk concentrate | 2·34 | 0·20 | 41·01 | ........... |
| Copper concentrate | 16·18 | 0·40 | 35·00 | ........... |
| Pyrite concentrate | 0·29 | 0·17 | 42·42 | 45·16 |
| Mill tailing | 0·09 | 0·38 | 23·83 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·54 | 0·30 | 100·0 | 100·0 |
| Copper concentrate | 8·32 | 16·18 | 0·40 | 87·4 | 12·6 |
| Pyrite concentrate | 56·20 | 0·29 | 0·17 | 10·6 | 36·2 |
| Mill tailing | 35·48 | 0·09 | 0·38 | 2·0 | 51·2 |

The pyrrhotite in the ore floated very freely, resulting in low-grade concentrates. The recovery of copper in the bulk float was high, 98 per cent, but the grade of final copper concentrate was considerably lower than in tests in which soda ash was used. This is due to the extreme flotability of the pyrrhotite.

### *Mill Run No. 5*

In this run, 0·10 pound of Barrett No. 4 per ton was added to the grinding mill. In all other respects the conditions were those of the standard bulk flotation run.

*Screen Test—Classifier Overflow:*

| Mesh | Weight, per cent |
|---|---|
| + 48 | 0·8 |
| − 48+ 65 | 2·4 |
| − 65+100 | 5·5 |
| −100+150 | 11·8 |
| −150+200 | 15·6 |
| −200 | 64·0 |

*Reagents:*

| | Lb./ton |
|---|---|
| To Ball Mill: | |
| Soda ash | 1·75 |
| Barrett No. 4 | 0·10 |
| To Bulk Flotation: | |
| Potassium amyl xanthate | 0·47 |
| Pine oil | 0·04 |
| To Copper Flotation: | |
| Lime | 2·9 |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·48 | 0·28 | 34·9 | ........... |
| Classifier overflow | 1·51 | 0·25 | 34·3 | ........... |
| Bulk concentrate | 2·92 | 0·23 | 42·4 | ........... |
| Copper concentrate | 24·26 | 0·60 | 31·1 | ........... |
| Pyrite concentrate | 0·22 | 0·15 | 43·0 | 47·26 |
| Mill tailing | 0·12 | 0·30 | 29·5 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·51 | 0·25 | 100·0 | 100·0 |
| Copper concentrate | 5·59 | 24·26 | 0·60 | 89·6 | 13·4 |
| Pyrite concentrate | 44·16 | 0·22 | 0·15 | 6·4 | 26·4 |
| Mill tailing | 50·25 | 0·12 | 0·30 | 4·0 | 60·2 |

Barrett No. 4 used as a supplementary collector reagent results in the production of a low-grade pyrite concentrate. This probably is due to gangue being floated in the bulk concentrate.

### *Mill Run No. 6*

To note the effect of a reagent cheaper than potassium amyl xanthate, sodium ethyl xanthate in equal quantities was substituted. Very poor results were obtained. The quantity of xanthate was raised to 0·96 pound per ton. Flotation was poor, a thin watery froth resulted, and a high loss of pyrite was apparent on the pilot table concentrating the bulk flotation tailing. No samples were taken.

## Section IV

### SELECTIVE FLOTATION

The object of the investigation was to determine the superiority, if any, of bulk flotation over selective flotation as practised at the mill.

### *Mill Run No. 1*

The flow-sheet in this run was one producing a copper concentrate and an iron-pyrite concentrate from the copper flotation tailing.

The ore at the rate of 500 pounds per hour was ground and aerated as in previous runs. The aeration tank discharged into the second cell of a 10-cell flotation unit. The concentrate from Cells Nos. 2 and 3 was cleaned in Cell No. 1, yielding a copper concentrate. The cleaner tailing from Cell No. 1 passed to Cell No. 2. Cells Nos. 4 to 10 gave off a scavenger concentrate, which was returned to Cell No. 2 with the feed. The tailing from the copper circuit was pumped to the second flotation unit and an iron-pyrite concentrate was taken off in the first two cells. The concentrate from Cells Nos. 3 to 10 was returned to the feed end.

| | |
|---|---|
| Classifier overflow | 35 per cent solids |

*Screen Test—Classifier Overflow:*

| Mesh | Weight, per cent |
|---|---|
| + 65 | 0·5 |
| − 65+100 | 3·4 |
| −100+150 | 9·7 |
| −150+200 | 18·9 |
| −200 | 67·5 |
| | 100·0 |

*Reagents:*

| | Lb./ton |
|---|---|
| To Ball Mill: | |
| Soda ash | 1·75 |
| Reagent No. 208 | 0·04 |
| Cyanide | 0·07 |
| Aerofloat No. 25 | 0·12 |
| To Cell No. 1: | |
| Cresylic acid | 0·10 |
| To Iron-pyrite Float: | |
| Copper sulphate | 1·0 |
| Amyl xanthate | 0·24 |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent | Au, oz./ton |
|---|---|---|---|---|---|
| Feed | 1·53 | 0·25 | 36·5 | .......... | .......... |
| Classifier overflow | 1·63 | 0·30 | 34·9 | .......... | 0·01 |
| Copper concentrate | 17·50 | 0·40 | 32·6 | .......... | 0·08 |
| Copper tailing | 0·13 | 0·25 | 35·3 | .......... | 0·0025 |
| Pyrite concentrate | 0·16 | 0·50 | 45·5 | 47·66 | 0·01 |
| Mill tailing | 0·13 | Nil | 20·8 | .......... | 0·0025 |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·63 | 0·30 | 100·0 | 100·0 |
| Copper concentrate | 8·64 | 17·50 | 0·40 | 91·8 | 11·5 |
| Pyrite concentrate | 53·43 | 0·16 | 0·50 | 5·2 | 88·5 |
| Mill tailing | 37·93 | 0·13 | Nil | 3·0 | .......... |

The recovery of copper is equal to that obtained in the bulk flotation runs, although the grade of concentrate is lower. The grade of iron pyrite is below 48 per cent sulphur, whereas the zinc content is much higher than that in previous runs. To reduce this zinc content it would be necessary to scalp off this zinc in cells prior to the flotation of the pyrite.

*Mill Run No. 2*

From time to time the application of selective flotation of this ore in a lime circuit has been advocated.

To demonstrate the unsuitability of such a circuit, this run was carried out as follows:

The same flow-sheet as in the preceding run was maintained. The classifier overflow passed to the aerating tank and then to the copper circuit. The tailing from this was conditioned with copper sulphate and xanthate, and a pyrite concentrate was recovered.

*Screen Test—Classifier Overflow:*

| Mesh | Weight, per cent |
|---|---|
| − 48+ 65 | 0·1 |
| − 65+100 | 4·3 |
| −100+150 | 9·7 |
| −150+200 | 18·4 |
| −200 | 67·5 |
| | 100·0 |

*Reagents:*

| | Lb./ton |
|---|---|
| To Ball Mill: | |
| Lime | 3·1 |
| Potassium amyl xanthate | 0·108 |
| To Copper Cells: | |
| Amyl xanthate | 0·027 |
| Pine oil | 0·04 |
| To Iron-pyrite Flotation: | |
| Copper sulphate | 1·0 |
| Amyl xanthate | 0·38 |
| Pine oil | 0·01 |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·52 | 0·28 | 35·9 | ........ |
| Classifier overflow | 1·59 | 0·28 | 35·8 | ........ |
| Copper concentrate | 28·04 | 0·20 | 29·2 | ........ |
| Copper tailing | 0·45 | 0·28 | 35·5 | ........ |
| Pyrite concentrate | 0·82 | 0·66 | 44·9 | 47·68 |
| Mill tailing | 0·13 | Nil | 26·9 | ........ |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·59 | 0·28 | 100·0 | 100·0 |
| Copper concentrate | 4·13 | 28·04 | 0·20 | 72·9 | 2·7 |
| Pyrite concentrate | 44·38 | 0·82 | 0·66 | 22·9 | 97·3 |
| Mill tailing | 51·49 | 0·13 | Nil | 4·2 | ........ |

The results of floating in a lime circuit are readily seen. The recovery of copper is low, and large amounts of reagents are necessary to float the iron pyrite. This concentrate is under 48 per cent sulphur.

### *Mill Run No. 3*

This run is a continuation of the preceding one. The xanthate to the copper circuit was increased from 0·135 pound to 0·162, and in the pyrite circuit from 0·38 to 0·49 pound per ton of feed.

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent |
|---|---|---|---|---|
| Feed | 1·54 | 0·30 | 35·9 | ........... |
| Classifier overflow | 1·65 | 0·30 | 35·5 | ........... |
| Copper concentrate | 27·74 | 0·20 | 29·2 | ........... |
| Copper tailing | 0·46 | 0·30 | 35·9 | ........... |
| Pyrite concentrate | 0·62 | 0·50 | 44·9 | 47·22 |
| Mill tailing | 0·14 | Nil | 22·9 | ........... |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·65 | 0·30 | 100·0 | 100·0 |
| Copper concentrate | 4·36 | 27·74 | 0·20 | 73·3 | 2·5 |
| Pyrite concentrate | 63·76 | 0·62 | 0·50 | 24·0 | 97·5 |
| Mill tailing | 31·88 | 0·14 | Nil | 2·7 | .......... |

Increasing the xanthate has no material effect on the copper recovery. The only effect apparent was to increase the weight of iron-pyrite concentrate recovered. This product contains 47·2 per cent sulphur and represents 63·76 per cent of the weight of ore milled.

It is quite apparent that the flotation of both sulphides in a lime circuit should not be attempted.

### *Mill Run No. 4*

In the selective flotation of sulphide ores containing copper, gold, and iron pyrite, the customary practice is to grind in a soda ash circuit and use cyanide to depress the pyrite. In some cases the cyanide dissolves part of the gold, entailing losses that are usually overlooked.

The effect of adding ammonium sulphate to the grinding mill to depress iron and zinc was studied in this run. No iron-pyrite concentrate was made. The flow-sheet for the grinding and the copper flotation was that used in the other selective flotation tests.

*Screen Test, Classifier Overflow:*

| Mesh | Weight, per cent |
|---|---|
| + 65 | 0·1 |
| − 65+100 | 2·6 |
| −100+150 | 8·1 |
| −150+200 | 14·6 |
| −200 | 74·6 |
| | 100·0 |

*Reagents:*

| | Lb./ton |
|---|---|
| To Ball Mill: | |
| Soda ash | 1·75 |
| Ammonium sulphate | 1·5 |
| To Copper Flotation: | |
| Potassium amyl xanthate | 0·16 |
| Pine oil | 0·04 |

*Results:*

| Product | Weight, per cent | Assay, per cent | | Distribution, per cent | |
|---|---|---|---|---|---|
| | | Cu | Zn | Cu | Zn |
| Feed—classifier overflow | 100·00 | 1·60 | 0·30 | 100·0 | 100·0 |
| Copper concentrate | 5·24 | 24·56 | 0·25 | 80·5 | 4·4 |
| Copper tailing | 94·76 | 0·33 | 0·30 | 19·5 | 95·6 |

Ammonium sulphate depresses both the iron pyrite and the zinc sulphides, but the recovery of copper is poor.

### *Mill Run No. 5*

Ammonium sulphate was again used in this run. Both a copper concentrate and an iron-pyrite concentrate were made. The flow-sheet was similar to previous selective flotation runs.

*Reagents:*

| | Lb./ton |
|---|---|
| To Ball Mill: | |
| Soda ash | 1·75 |
| Ammonium sulphate | 1·5 |
| Reagent No. 208 | 0·135 |
| To Copper Cells: | |
| Reagent No. 208 | 0·08 |
| Pine oil | 0·05 |
| To Pyrite Float: | |
| Copper sulphate | 1·0 |
| Amyl xanthate | 0·21 |
| Pine oil | 0·03 |

*Assays:*

| — | Cu, per cent | Zn, per cent | Fe, per cent | S, per cent | Au, oz./ton |
|---|---|---|---|---|---|
| Feed | 1·54 | 0·30 | 35·8 | .......... | .......... |
| Classifier overflow | 1·61 | 0·30 | 35·3 | .......... | 0·01 |
| Copper concentrate | 23·32 | 0·20 | 30·2 | .......... | 0·08 |
| Copper tailing | 0·44 | 0·33 | 34·7 | .......... | 0·005 |
| Pyrite concentrate | 0·59 | 0·58 | 45·5 | 51·48 | 0·01 |
| Mill tailing | 0·27 | Nil | 24·5 | .......... | 0·0025 |

*Results:*

| Product | Weight, per cent | Assay | | | Distribution, per cent | | |
|---|---|---|---|---|---|---|---|
| | | Per cent | | Oz./ton | | | |
| | | Cu | Zn | Au | Cu | Zn | Au |
| Feed—classifier overflow | 100·00 | 1·61 | 0·30 | 0·01 | 100·0 | 100·0 | 100·0 |
| Copper concentrate | 5·11 | 23·32 | 0·20 | 0·08 | 74·0 | 3·4 | 39·9 |
| Pyrite concentrate | 50·47 | 0·59 | 0·58 | 0·01 | 18·5 | 96·6 | 49·3 |
| Mill tailing | 44·42 | 0·27 | Nil | 0·0025 | 7·5 | ........ | 10·8 |

The copper recovery again was low. A very good recovery of pyrite was obtained, 50·47 per cent of the weight of feed with a sulphur content of 51·48 per cent.

## SUMMARY AND CONCLUSIONS

Flotation of the copper and iron sulphides in one concentrate with subsequent separation of the two minerals is superior to their selective flotation.

This is quite clear from a comparison of the two methods, as exemplified by Mill Run No. 7, Section I, and Mill Run No. 1, Section IV:

| — | Bulk flotation | Selective flotation |
|---|---|---|
| *Copper Concentrate:* | | |
| Grade, copper per cent | 28·66 | 17·50 |
| Recovery, per cent | 91·4 | 91·8 |
| Tons recovered per 100 tons feed | 5·10 | 8·64 |
| *Iron-pyrite Concentrate:* | | |
| Grade, sulphur per cent | 48·82 | 47·66 |
| Tons recovered per 100 tons feed | 49·55 | 53·43 |
| *Reagents:* (pounds per ton of feed) | | |
| Soda ash | 1·75 | 1·75 |
| Cyanide | None | 0·07 |
| Reagent No. 208 | None | 0·04 |
| Aerofloat No. 25 | None | 0·12 |
| Cresylic acid | None | 0·10 |
| Pine oil | 0·04 | None |
| Amyl xanthate | 0·47 | 0·24 |
| Copper sulphate | None | 1·0 |
| Lime | 2·9 | None |

The cost of the reagents for the selective flotation of the copper is practically the same as that for the bulk flotation including that of the lime to depress the iron pyrite. Taking into consideration the extra reagents necessary to recover the pyrite contained in the copper tailing, the cost of bulk flotation is considerably less than selective flotation.

Apart from the reagent cost, the grade of copper concentrate obtained by bulk flotation is much higher.

The simplicity of operation is a factor in favour of bulk flotation. The circuits are not sensitive to slight changes, and the whole operation requires little attention from the operators.

The investigation discloses the necessity of adding enough xanthate to the flotation cells to float the large weight of iron pyrite. The pyrrhotite in the ore also can be floated by an increase in reagents. The quantity of this taken will govern the sulphur content of the pyrite concentrate.

The point at which the xanthate is added is quite important. When added to the grinding mill, recovery fell off rapidly. Apparently the amyl xanthate is most effective when added directly to the cells.

Reagent diphenylquandine has an effect on improving the character of the froth in the copper circuit and also slightly raises the copper recovery by assisting in the flotation of coarse chalcopyrite.

Bulk flotation also has the advantage that most of the zinc in the feed can be discarded with the mill tailing. If desired, this zinc may be recovered in an extra flotation circuit. In selective flotation the zinc will float with the iron pyrite and contaminate this product.

Aeration of the classifier overflow is necessary. If omitted, the flotation of copper is difficult. Although not so noticeable in the small unit in the laboratories, operations at the mine are conclusive. Without air conditioning the results are very unsatisfactory.

Sodium ethyl xanthate is useless on this ore. Large quantities have to be added, and the character of the froth and the recoveries are very poor.

Dupont "Floto" in solutions of low pH is good for the flotation of iron pyrite but does not give a high recovery of copper.

Selective flotation of copper in a lime circuit also results in a low copper recovery. The iron pyrite then is difficult to float.

Bulk flotation followed by separation of the different minerals offers much to the operators of mills now using selective flotation. This has special application to ores containing gold in which the cyanide employed to depress part of the sulphides also attacks the precious metals.

# III

## INVESTIGATIONS THE RESULTS OF WHICH ARE SYNOPSIZED

*Gold-Antimony Ore from the Congress Gold Mines, Ltd., Bridge River, B.C.* On July 24, 1936, 42 pounds of ore was received, consisting of white to grey quartz with much carbonate. The metallic minerals are stibnite, arsenopyrite, pyrite, and small quantities of sphalerite and galena. It assayed 0·135 ounce of gold per ton and 0·9 per cent of antimony.

A recovery of 70 to 75 per cent of the antimony was made in a concentrate assaying 11 to 12 per cent of antimony, and over 70 per cent of the gold in a concentrate assaying 0·6 ounce of gold per ton. It is doubtful if an economic milling procedure could be evolved from the grade of ore submitted—the object of the test.

*Flotation Concentrate from the Algoma-Summit Gold Mines, Ltd., at Goudreau, Ontario.* On December 7, 1936, 50 pounds of concentrate was received, which assayed 2·98 ounces of gold and 0·52 ounce of silver per ton.

Amalgamation recovered 70 per cent of the gold. Test work showed that the concentrate should be washed after regrinding, or reground with an excess of lime, to prevent serious flouring of mercury.

*Gold-Silver-Copper-Lead Ore from Greenbridge Gold Mines, Ltd., Greenwood, B.C.* On November 4, 1936, 205 pounds of ore was received containing free gold, chalcopyrite, and galena. The sample assayed 0·655 ounce of gold and 3·01 ounces of silver per ton, 0·27 per cent of copper, 0·93 of lead, and no zinc.

A recovery of 0·52 per cent of the gold was made as trap concentrate assaying 32·26 ounces of gold and 52·36 ounces of silver per ton; 2·95 per cent of the feed was recovered as a flotation concentrate assaying 12·86 ounces of gold, 66·943 ounces of silver, 7·96 per cent of copper, and 26·7 per cent of lead; and 1·42 per cent of the feed was recovered as a flotation middling assaying 2·8 ounces of gold and 14·32 ounces of silver; 1·45 per cent of copper and 5·17 of lead. The total recoveries were: gold, 92·1 per cent, silver, 90·8, copper, 96·5, lead, 98. Cyanidation reduced the flotation tailing from 0·055 ounce of gold to 0·01 ounce, an additional recovery of 6·5 per cent.

*Mill Products from Bralorne Mines, Ltd., Bralorne, Bridge River District, B.C.* On November 13, 1936, 85 pounds of flotation and 73 pounds of classifier overflow were received, assaying 0·1 ounce and 0·14 ounce of gold per ton respectively.

Cyanidation gave an extraction of 82 to 85 per cent after further grinding. Almost 50 per cent of the remaining gold was associated with sulphides, mostly minus 200-mesh, according to tests by the Haultain super-panner.

*Graphite Ore from the Black Donald Mine, Calabogie, Ont.* On December 5, 1936, 15 tons analysing 40·68 per cent of carbon was received for testing by flotation to get as much coarse flake as possible and to remove gangue impurities.

A recovery of 200 pounds of +65-mesh concentrate analysing 85·4 per cent of carbon was made from each ton, representing 20·6 per cent of the carbon. At least 20 to 25 per cent of the carbon as plus 65-mesh flake and 70 to 78 per cent as minus 65-mesh, or a total of 95 per cent of the graphite, should be possible.

*Gold Ore from Amca Mines, Ltd., at Matheson, Ont.* On December 9, 1936, 1,000 pounds each of fresh and of oxidized ore were received, the fresh ore assaying 0·62 ounce of gold and 0·08 ounce of silver per ton, the oxidized ore 0·685 ounce of gold.

The ore was too slow in settling for straight cyanidation. Barrel amalgamation of a flotation concentrate from a mixture of 90 per cent of the fresh ore and 10 per cent of the oxidized ore extracted 93 per cent of the gold.

*Silver-Lead Ore from Mr. Stanley Brooks, c/o E. T. Kenney, Ltd., Terrace, B.C.* On October 26, 1936, 100 pounds of ore was received consisting of approximately 50 per cent white quartz, the remainder being pyrite, chalcopyrite, galena, and sphalerite, with small amounts of limonite, tetrahedrite (or tennantite), and covellite. This ore assayed 0·11 ounce of gold and 19·8 ounces of silver per ton; 21·3 per cent of lead, 4·6 of zinc, 0·9 of copper, 4·0 of iron, and 8·18 of sulphur.

Bulk flotation followed by table concentration, with a ratio of concentration of 2·6 : 1, recovered 91·1 per cent of the gold, 91·7 of the silver, 91·6 of the lead, 90·6 of the zinc, and 98·1 of the copper.

Differential flotation was not feasible, owing to the oxidized condition of the ore.

*Flotation concentrate from Minto Gold Mines, Ltd., Minto, Bridge River District, B.C.* On November 23, 1936, 110 pounds was received analysing, 1·24 ounces of gold and 8·64 ounces of silver per ton; 9·32 per cent of arsenic, 0·89 of antimony, 28·13 of iron, 0·21 of copper, 0·96 of lead, 3·51 of zinc, 15·08 of silica, 1·10 of lime, 6·66 of magnesia, 27·29 of sulphur, and no bismuth.

Cyanidation of the flotation middling from the reground concentrate was attempted, 87 per cent being −200 mesh. The tailing was about the same as that from cyanidation of the original concentrate so that the refractory gold could not be separated by regrinding and making a cleaner concentrate.

*Gold Ore from the Paymaster Consolidated Mines, Ltd., South Porcupine, Ont.* On November 24, 1936, a sample of gold ore, weighing 110 pounds, and assaying 0·205 ounce of gold per ton, was received.

A check run according to the mill practice of making a high-grade concentrate, regrinding and recyaniding it together with the main portion of the ore, gave an extraction of 90·24 per cent. By concentrating the sulphides, regrinding and recyaniding them and their tailing separately an extraction of 93·8 per cent was obtained. Straight cyanidation at 98 per cent −200 mesh gave a maximum extraction of 92·68 per cent in 48 hours.

*Gold Ore from MacLeod-Cockshutt Gold Mines, Ltd., Geraldton, Ont.* On December 28, 1936, 520 pounds was received consisting of an altered schist, with veins of grey quartz and some white carbonate. The metallic minerals present were pyrite, arsenopyrite, chalcopyrite, and small irregular grains of native gold in gangue and dense pyrite. The sample assayed 0·4 ounce of gold and 0·04 ounce of silver per ton; 0·36 per cent of arsenic, and 3·73 of sulphur.

Straight cyanidation recovered 93 per cent of the gold from a grind of 90 per cent −200 mesh. Roasting the flotation concentrate from the cyanide tailing and cyaniding the calcine raised the recovery to 97 per cent.

*Arsenical Gold Ore from the Monte Carlo Exploration Gold Mines, Ltd., Timagami, Northern Ontario.* On December 19, 1936, 280 pounds was received of about equal quantities by volume of grey quartz and of arsenopyrite and pyrrhotite, with a small quantity of chalcopyrite. A few microscopic grains of gold, 2300 mesh, were seen in the dense arsenopyrite. It assayed 0·6 ounce of gold and 0·19 ounce of silver per ton; 18·80 per cent of arsenic, 7·96 of sulphur, 15·8 of iron, and a trace of copper.

Only 60 per cent of the gold was extractable by straight cyanidation. Bulk flotation recovered 98 per cent in a concentrate assaying 1·5 ounces of gold per ton, the ratio of concentration being 2·3 : 1. Roasting raised its content to 2 ounces per ton.

*Gold Ore from McMillan Gold Mines, Ltd., Footbanks, Ont.* On January 15, 1937, 50 pounds was received assaying 0·36 ounce of gold and 0·04 ounce of silver per ton. The ore was submitted as it contained more sulphides than the ordinary mill feed and it would illustrate better the difficulties in treatment.

Concentration of the sulphides, regrinding, and amalgamation was not successful. To get the maximum extraction of the gold, 88·9 per cent, it was necessary to aerate with lime, filter, wash to remove the cyanicides, and cyanide with two solutions with intermediate filtering. The cyanide solution requires aeration to restore its oxygen before using again.

*Silver-Pitchblende Ore and Mill Tailing from Eldorado Gold Mines, Ltd, Echo Bay, Great Bear Lake, N.W.T.* On January 5, 1937, 196 pounds of ore and 184 pounds of tailing were received, assaying as follows:

| — | Ore | Mill tailing |
|---|---|---|
| | oz./ton | oz./ton |
| Silver | 82·39 | 40·16 |
| | per cent | per cent |
| Uranium oxide ($U_3O_8$) | 1·03 | 0·25 |
| Copper | 1·8 | 1·48 |
| Manganese | 7·95 | 7·58 |
| Iron oxide (FeO) | 7·52 | 7·2 |
| Lime (CaO) | 6·01 | 6·09 |
| Carbon dioxide | 14·2 | 14·1 |
| Sulphur | 1·83 | 1·59 |
| Insoluble | 43·27 | 46·82 |

Table tests were made on the ore screened to different sizes. The tailing from material coarser than 48 mesh always contained less than 0·1 per cent of uranium oxide, but was higher from the −48-mesh material, and the slime accounted for over 13 per cent of the uranium oxide. Hydraulic classification of the table feed was recommended for reducing the sliming during grinding.

By using caustic soda instead of soda ash in flotation of the mill tailing, a tailing of about 7 ounces of silver per ton and 0·17 per cent of uranium oxide could be made—a recovery of about 80 per cent of the silver and 40 per cent of the uranium oxide.

*Copper-Zinc Ore from the Abana Mine, Desmeloizes Township, Abitibi County, Que.* On September 8, 1936, a sample of ore from the third level of the Abana mine was received. It assayed 0·04 ounce of gold and 7·06 ounces of silver per ton; 3·68 per cent of copper and 11·75 of zinc.

Owing to the unfavourable copper-zinc ratio, it was not possible to make a bulk float of the copper and zinc and to extract the gold in the iron-pyrite tailing as in former shipments. Fine grinding is essential to obtain a copper concentrate containing 20 per cent of copper and 10 per cent of zinc, a recovery of 80 to 90 per cent of the copper. From 78 to 80 per cent of the zinc can be recovered in a 46 per cent zinc concentrate; and about 50 per cent of the gold and 60 per cent of the silver is in the copper concentrate.

*Gold Ore from the Minto Gold Mines, Ltd., Minto, Bridge River District, B.C.* On November 23, 1936, a shipment of 202 pounds was received assaying 0·105 ounce of gold and 0·88 ounce of silver per ton; 1·02 per cent of arsenic, 0·12 of antimony, 9·27 of lime, and 9·82 of magnesia.

Flotation tests were made to show that caustic soda is better than sodium carbonate as a modifying agent, the best results being obtained with 3 to 6 pounds per ton of ore feed. Using 3 pounds per ton the concentrate carried 1·06 ounces of gold and 8·64 ounces of silver per ton, and 8·89 per cent of arsenic, a recovery of 94·8 per cent of the gold and a ratio of concentration of 3·26 to 1. The tailing assayed 0·025 ounce of gold per ton. Using 6 pounds of caustic soda, the concentrate carried 1·14 ounces of gold per ton, a recovery of 93·1 per cent, the tailing assayed 0·03 ounce per ton, and the ratio of concentration was 3·82 to 1. Three pounds of soda ash per ton gave a tailing having 0·05 ounce per ton, a recovery of 89·8 per cent.

*Silver Ore from the Lily of the Valley Mine, Thunder Bay District, Ont.* On March 4, 1937, 50 pounds was received, consisting of a coarse-textured, white calcite with much light grey translucent quartz. The metallic minerals were pyrite, sphalerite, galena, and native silver. It assayed 286·7 ounces of silver per ton and no gold; 0·79 per cent of lead and 0·36 per cent of zinc.

Tabling a −14-mesh unclassified feed gave a concentrate assaying 1,820·2 ounces of silver per ton, or 67·13 per cent of the silver. The tailing was reground to 74·5 per cent −200-mesh using 1·5 pounds of caustic soda per ton and 0·192 pound of coal-tar creosote, and was floated using 1 pound of sodium sulphide and 0·10 pound of butyl xanthate per ton. The flotation concentrate assayed 1,562 ounces of silver per ton, accounting for 31·52 per cent of the silver. The overall recovery was 98·65 per cent. The combined concentrates showed a ratio of concentration of 6·67 : 1.

*Gold Ore from the B.C. Goldfield Group, Spud Creek, Vancouver Island, B.C.* On March 16, 1937, a sample, 150 pounds, was received consisting of silicate minerals with some carbonate. The metallic minerals were pyrite, arsenopyrite, magnetite, "limonite", sphalerite, galena, chalcopyrite, and native gold. The assay was: 1·54 ounces of gold and 0·52 ounce of silver per ton; 0·5 per cent of arsenic, 0·02 of copper, 3·73 of iron, 0·1 of lead, 0·13 of zinc, and 2·55 of sulphur.

Amalgamating the concentrates from traps, blankets, and flotation gave a recovery of 87 per cent of the gold as bullion. The combined concentrates assayed 14 and 5 ounces per ton of gold and silver respectively and represented 10 per cent of the weight of the feed. A recovery of 97 per cent was made by traps and blankets and cyanidation of the tailing. The cyanide solutions tended to foul and fine grinding was necessary to free the gold from the dense pyrite.

*Gold Ore from Minto Gold Mines, Ltd., Minto Mines, B.C.* On March 16, 1937, 110 pounds was received, assaying 0·415 ounce of gold and 0·59 ounce of silver per ton.

Barrel amalgamation of ore ground to 63·2 per cent −200 mesh recovered 53 per cent of the gold; 77 to 80 per cent was recovered by cyanidation of 53·2 and 100 per cent −200-mesh ore respectively, and agitating for 48 hours. Fine grinding increased cyanide consumption from 0·82 to over 3 pounds per ton of ore. The solution tended to foul, according to an analysis of that from the 48-hour treatment of the −200-mesh ore.

*Gold-Silver Ore from the Lakeview Mining Syndicate, Slocan Mining District, B.C.* On March 8, 1937, 140 pounds of ore was received assaying 0·16 ounce of gold and 3·53 ounces of silver per ton, and consisting of milky-white quartz, the metallic minerals being pyrite, galena, sphalerite, chalcopyrite, native silver, and native gold.

Either flotation or straight cyanidation gave high recovery of both gold and silver. The choice would depend on the cost of freight and the treatment of the flotation concentrate.

*Gold Concentrate from the Murray-Algoma Mining Co., Ltd., Township 28, Range 24, District of Algoma, Northern Ontario.* A sample of concentrate, 7·5 pounds, was received on May 10, 1937, assaying 1·27 ounces of gold and 1·15 ounces of silver per ton. A screen test showed 29·7 per cent −200 mesh.

A recovery of 70 per cent of the gold was made by barrel amalgamation after regrinding to 65 per cent −200 mesh.

*Arsenopyrite-Pyrite Gold Ore from the Athelstan-Jackpot Mine, Grand Forks Mining Division, B.C.* On January 21, 1937, a shipment of gold ore, 300 pounds, was received, which assayed 0·75 ounce of gold and 0·32 ounce of silver per ton; 6·99 per cent of arsenic, 26·26 of iron, 24·99 of sulphur, 3·56 of lime, and 3·25 of magnesia.

The highest extraction obtained by cyanide was 78·7 per cent. Roasting gave a calcine having 1·04 ounces of gold per ton. The calcium and magnesium made the removal of arsenic and sulphur difficult.

*Gold Ore from the Wisik Gold Mines, Ltd., Sullivan, Que.* On March 5, 1937, a shipment weighing 437 pounds was received. The predominant sulphide was pyrrhotite and the ore assayed 0·4 ounce of gold and 0·09 ounce of silver per ton; 1·84 per cent of sulphur.

Satisfactory cyanidation was not possible due to fouling of the solution by the pyrrhotite. The addition of lead nitrate (1·5 pounds per ton) overcame this which gave an extraction of 97·5 per cent. Barrel amalgamation and grinding to 78·5 per cent −200 mesh recovered 68·75 per cent of the gold.

*Gold Ore from Lake Caswell Mines, Ltd., Shining Tree, Ont.* On May 25, 1937, a shipment of 2,000 pounds, was received consisting of quartz in which pyrite was rather sparingly disseminated and native gold, relatively finely divided, occurred chiefly in the gangue. The assay was 0·27 ounce of gold and 0·05 ounce of silver per ton.

Direct cyanidation gave an extraction of 96·29 per cent at a grinding of 91 per cent −200 mesh.

*Silver-Lead Ore from the Silver Standard Mine, Hazelton District, B.C.* On May 25, 1937, a shipment, 58 pounds, of much oxidized ore was received, consisting of fine-textured milky-white quartz, the metallic minerals being galena, pyrite, sphalerite, arsenopyrite, limonite, and tetrahedrite. The sample assayed 0·05 ounce of gold, and 93·46 ounces of silver per ton; 51·5 per cent of lead, 3·03 of zinc, and 0·36 of copper, but it probably did not represent the average ore.

Bulk flotation recovered over 95 per cent of the lead and 97 per cent of the silver.

*Gold Ore from the Venus-Juno Group, in the Nelson Mining Division, near Nelson, B.C.* On January 14, 1937, 213 pounds of Venus ore, 14 pounds from Venus waste dump, and 12 pounds of Juno ore were received. The Venus ore alone was investigated. It consisted of pyrite and limonite in quartz and assayed 1·935 ounces of gold and 2·63 ounces of silver per ton; with 0·56 per cent of zinc.

Cyanidation extracted 98 per cent of the gold. A very high-grade flotation concentrate recovered 88 per cent.

*Mill Tailing from the Orelia Mines, Ltd., Rainy River District, Ont.* On June 7, 1937, a shipment of mill tailing, 167 pounds, was received assaying 0·455 ounce of gold and 0·44 ounce of silver per ton.

Amalgamation at a grind of 56 per cent −200 mesh recovered 62·6 per cent of the gold. Cyanidation at a grind of 76 per cent −200 mesh extracted 97 per cent in 24 hours. Consumption of cyanide was excessive and was not remedied by water-washing. Flotation gave a recovery of 88·6 per cent at a ratio of concentration of 25 : 1.

*Molybdenite Ore from Martel Gold Mines, Ltd., near Martel, B.C.* On May 10, 1937, a sample of molybdenite ore, 25 pounds, was received consisting of quartz with some calcite containing molybdenite, pyrite, pyrrhotite, chalcopyrite, sphalerite, and arsenopyrite. It assayed 0·015 ounce of gold and 0·04 ounce of silver per ton; 1·48 per cent of molybdenum sulphide, and 0·11 of copper.

Flotation at a grind of 70 to 80 per cent −200 mesh gave a concentrate assaying 71·5 per cent of molybdenum sulphide and 0·71 per cent of copper.

*Five Samples Submitted from the Hope, Yale Mining Division, B.C.* These five small samples were examined microscopically for identification of the metallic minerals, and were analysed in the Chemical Laboratory and assayed for gold and silver. The metallic minerals visible in the sections include pyrite, pyrrhotite, magnetite, hematite, "limonite", marcasite, sphalerite, galena, chalcopyrite, tetrahedrite, and jamesonite.

Some of the samples contain very little gold and silver. Spectrographic analyses of the silver buttons proved the absence of platinum-group metals in all samples.

*Two Mill Products from Bidgood Kirkland Gold Mines Limited, Kirkland Lake, Ont.* The mode of occurrence of the gold in samples from the tube mill discharge and from the tailing were studied to see whether the metallic or non-metallic fractions might be given preference in further grinding.

Both samples were panned in the Haultain super-panner, and the products were assayed, and studied microscopically in polished sections. The gold visible in the tube mill discharge occurs largely as extremely fine particles in the gangue, and that in the tailing is largely with the non-metallic fraction. The non-metallic fraction will, therefore, require extremely fine grinding to liberate the gold, but as little is known of the amount of gold in the sulphides, it is impossible to say whether their removal would do any good.

*Metallic Minerals in Special Sample from Macassa Mines, Limited, Kirkland Lake, Ont.* This was examined microscopically to determine what tellurides were present. In addition to pyrite, chalcopyrite, and native gold, two tellurides were recognized. The common telluride is altaite, PbTe; and a rare, creamy coloured mineral possessing the characteristics of calaverite $(Au,Ag)\,Te_2$) occurs as rounded grains in pyrite. Other tellurides, such as coloradoite and petzite, which have been reported elsewhere in the mines of the Kirkland Lake camp, were not noted.

*An Examination of the Steel from Two Austenitic Manganese Steel Castings.* Two samples of austenitic manganese steel, one from a crusher jaw plate that had failed in service after about half of its expected life, and one from a gyratory mantle that also had failed prematurely, were sent in by a Canadian steel company.

The composition of both steels was within the limits usually specified for this type of material; both were quite clean. The jaw crusher steel was found to be not entirely austenitic, for it contained large amounts of carbide at the grain boundaries and some acicular martensite within the grains. The grain size of the gyratory mantle steel was found to be excessively large.

The presence of acicular martensite and grain boundary carbide in the jaw crusher steel indicated that it had been inefficiently quenched, and the large grain size of the gyratory mantle proved that it had been poured from too high a temperature.

The failure of the jaw crusher plate was attributed directly to the embrittlement of the casting by the carbide and martensite constituents. Large-grained austenitic manganese steel has considerably lower tensile strength than fine-grained material of a similar analysis. The premature failure of the gyratory mantle was considered, then, to have been at least partially caused by its coarseness of grain.

*An Examination of the Steels from Three Ball Mill Liners.* Samples taken from three ball mill liners made respectively in Canada, England, and the United States were sent in.

The Canadian and United States materials were found to have compositions within the limits usually specified for austenitic manganese steels. The English steel proved to be a high-carbon chrome steel and probably was intended for use as an end liner.

The Canadian and United States steels were fairly clean, the English dirty. The Canadian steel was finely grained and the United States steel was coarse-grained, a proof that the pouring temperature had been satisfactory for the first material and high for the second, for heat treatment does not refine the grain in austenitic manganese steels. The English steel was not austenitic, consequently its fine grain size indicated only that it had received a proper heat treatment. The absence of carbide and martensite in the two austenitic materials showed that their heat treatment had also been satisfactory.

It is difficult to compare materials of such different analyses. It was concluded, however, that the English steel would give the best service under conditions where abrasion is not accompanied by cold work. Where abrasion is associated with impact, however, the other two steels would have better wear resistance, and of these two it was considered that the finer-grained Canadian steel would give the better service.

*A Metallographic Examination of the Steel of a Stainless Fork End Bolt.* A lug of a stainless steel fork end bolt which had failed in service was sent in by the Department of National Defence.

The steel proved to be a clean, fine-grained, 12 per cent chromium material with a Rockwell hardness of C. 43. There was no metallographic evidence to account for the failure of the bolt.

The impact strength of the steel might have been lowered by a draw at too high a temperature. There was not enough material available, however, to test the steel in impact.

The form of the fracture, however, indicated that the failure was due to fatigue, probably caused by an overloading of the member.

*An Examination of the Steel in Three Castings.* Risers from a mild steel casting, and pieces of two austenitic manganese steel gyratory crusher mantles, were received. The mild steel casting had been very brittle, breaking with a very coarse flat fracture. One of the manganese steel castings (No. 1) had failed after about two-thirds of its normal life; and the other riser had also failed prematurely.

The materials had the following analyses:

| — | Carbon, per cent | Manganese, per cent | Silicon, per cent | Sulphur, per cent | Phosphorus per cent |
|---|---|---|---|---|---|
| Mild steel riser.................. | 0·11 | 0·81 | 0·28 | 0·050 | 0·126 |
| Casting No. 1.................. | 1·15 | 11·20 | 0·61 | 0·012 | 0·103 |
| Casting No. 2.................. | 1·11 | 12·90 | 0·56 | 0·013 | 0·063 |

The phosphorus content of the mild steel was unduly high, even allowing for the effect of segregation. The composition of the austenitic manganese steels, with the exception of the phosphorus in Casting No. 1, was within the bounds usually specified for this material.

The mild steel was dirty and exhibited a Widmanstatten structure, probably as a result of pouring from too high a temperature. The poor properties of this steel were attributed to its structure and its high phosphorus content.

Both austenitic manganese steels were coarse-grained. Casting No. 1 was fairly dirty.

The large grain structure and fairly high phosphorus content of Casting No. 1 were believed to have been responsible for its premature failure. Casting No. 2 was even more coarse-grained than No. 1 and it was believed that this coarseness of grain caused its early failure.

*An Examination of an Austenitic Manganese Steel Crusher Jaw Plate.* A manganese steel crusher jaw plate that had worn very rapidly in service was received for investigation.

The steel contained only 0·90 per cent of carbon, but with this exception its composition was within the limits usually specified for this class of material.

The Brinell hardness of the casting was 175; heat-treated austenitic manganese steel is usually somewhat harder. The steel was found to be quite dirty but fairly fine-grained and free from carbides, an indication that it had been poured from a proper temperature and properly heat-treated. The poor wear resistance of the jaw plate was probably due to its low carbon content.

*An Examination of Three Austenitic Manganese Steels.* Pieces of two ball mill liners, one of Canadian and one of English make, and a sample of a gyratory bowl liner that had failed prematurely were received for testing.

The Canadian ball mill liner was found to be slightly low in carbon, but with this exception the composition of the three materials was within limits usually specified for austenitic manganese steels. Both ball mill liner steels were clean, fine-grained, and free from carbides, indicating that they had been manufactured and heat-treated properly. The bowl liner steel, although clean and free from carbides, was coarse-grained, a proof that it had been poured from too high a temperature. The premature failure of the ball mill liner was at least partly due to the coarse-grained structure of the casting.

*Corrosion Tests on Aluminium Bronze.* A bar of aluminium bronze was sent in for the determination of its resistance to the attack of various acid solutions.

Disks were cut and ground on an abrasive wheel so that all surfaces might be uniform. They were immersed in the corrosive medium, the loss in weight per surface area being taken as a measure of the attack.

Tests were made at room temperature with quiet and aerated solutions of sulphuric, hydrochloric, acetic, and phosphoric acids. In the case of the sulphuric and hydrochloric acids the respective strengths were 10, 20, and 80 per cent. The acetic and phosphoric acids were each used in strengths of 20 and 80 per cent. Quiet beaker tests were also conducted at 180°F., using all the solutions except the 20 per cent sulphuric and hydrochloric acids.

The attack of the sulphuric and phosphoric acids was negligible under all conditions. Hot acetic acid and aerated hydrochloric acid, however, attacked the alloy appreciably. Corrosion by hot hydrochloric acid was very rapid.

The bronze, therefore, had superior resistance to both sulphuric and phosphoric acids and had at least average resistance to hydrochloric and acetic acids.

*An Examination of Interlocking Piling Steel.* Corrosion-resisting steel of the following analyses was specified for piling:

| Carbon | Silicon | Manganese | Sulphur | Phosphorus | Copper | Chromium |
|---|---|---|---|---|---|---|
| 0·25–0·30 | 0·20 max. | 0·70–1·00 | 0·05 max. | 0·05 max. | 0·30—0·60 | 0·30–0·60 |

This steel was specified to have an ultimate strength of between 85,000 and 105,000 pounds per square inch and a minimum stress yield point of 54,000 pounds per square inch.

Some piling that had bent in rolling cracked when submitted to a cold-rolling straightening operation. The broken material had been analysed, and the tensile strength near the point of fracture determined, and it was found that the ultimate strengths exceeded or approached the upper limit specified. The heat from which the piling had been rolled was reported as being within specifications. In spite of this, some of the broken steels had carbon contents slightly higher than that specified. The other elements, however, were found to be within the specified limits.

Six broken tensile test pieces, machined from the defective piling, and five broken tensile test pieces machined from steel that had not been cold rolled, were brought to the Ore Dressing and Metallurgical Laboratories.

Drillings taken from the specimens were analysed and in every case but one the carbon was found to be higher than that specified, one of the steels containing ten points more than the specified maximum.

A Meyer hardness analysis, a method for determining the capacity of a material for work-hardening, was made of the various specimens. Heavily cold-worked material that will crack if further work-hardened has an "n" value of 2·00. The "n" value of the material that had not received the cold straightening treatment was found to be 2·25. One of the steels that had cracked on rolling had an "n" value of 2·05. The other steels' "n" values were all 2·15. The material near the fracture, then, had been appreciably work-hardened.

A microscopic examination of the steels was made. All materials were found to be clean and in good condition. The steel from the fractured piling, however, was somewhat dirtier than the unworked material. The high pearlite-ferrite ratio indicated that the carbon content exceeded the specification limit.

It was recommended that a closer check be kept on the carbon content of the piling and that changes be made in the rolling practice to reduce the amount of local cold work involved in the straightening operation.

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