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A PROCEDURE FOR PRODUCING A SELF-ROASTING FLOTATION CONCENTRATE FROM DEVELOPMENT ORE FOR CAMPBELL RED LAKE MINES LIMITED, BALMERTOWN, ONTARIO

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SUMMARY OF RESULTS

A modified procedure was developed which gave the necessary sulphur grade of a flotation concentrate required for the company's self-roasting operation.

A 92.5% gold extraction was obtained from this development ore by applying jigging, amalgamation, flotation, roasting and cyanidation.

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INTRODUCTION

Campbell Red Lake Mines Limited is a gold producer located in Balmer township, Red Lake area, in northwestern Ontario. Briefly, the operation consists of primary grinding the crushed ore followed by cycloning and then jigging of the cyclone underflow. This treatment recovers most of the coarse free gold, which is then extracted by amalgamation.

The jig tailing is re-ground to about 74% minus 200 mesh and floated to obtain a sulphide concentrate (mostly pyrite and arsenopyrite) and a tailing. The flotation tailing is cyanided in a conventional cyanidation circuit. The pyritic concentrate is roasted ("calcined") in a fluo-solids reactor without the addition of any fuel, i.e. a self-roasting operation. To maintain a satisfactory self-roasting product, it is essential to have the grade of the concentrate 18% sulphur or higher. The roasting of the concentrate exposes the gold, locked with the pyritic particles, making it amenable for dissolution by cyanide in a separate (calcine) cyanidation circuit. Gold is then recovered as precipitate from the both cyanidation circuits, and refined by standard procedure.

No difficulty is experienced in obtaining the required sulphur grade of the pyritic concentrate using the established plant flotation practice.

Purpose of Investigation

The company is presently developing a new underground area (1851 East Zone) and it appears that the ore from this zone has different flotation characteristics than has the normal ore. More specifically, the required sulphur grade of the pyritic concentrate (18% S) could not be obtained from this ore by the company's standard flotation procedure. As a result, Mr. J. Chisholm, former General Manager, requested an investigation on an ore sample from the 1851 East Zone to develop a simple modification, of their present flotation procedure, which would produce a pyritic concentrate with a minimum sulphur content of 18 percent.

Ore Sample

About 170 pounds of the ore from 1851 East Zone, crushed to about 3/4 inch, was received on September 4, 1969.

The sample was split into two equal portions. One portion was stored for future use and second portion was crushed to minus 10 mesh for investigation.

A head sample, cut from the shipment by standard procedure, was analysed as follows:

Gold (Au)	-	2.578 oz/ton
Silver (Ag)	-	0.298 oz/ton
Arsenic (As)	-	4.56 %
Antimony (Sb)	-	0.05 %
Sulphur (S)	-	3.02 %
Iron (Total Fe)	-	6.46 %

MINERALOGY

A mineralogical examination* was made of four hand specimens of the ore, submitted by the Campbell Red Lake Mines Limited, together with the 170-lb ore sample designated for testing.

The minerals identified in the samples are arsenopyrite, gold, pyrrhotite, chalcopyrite, pyrite and magnetite, all of which occur in a quartz-carbonate vein material.

Pyrrhotite, pyrite and arsenopyrite are the most abundant of the sulphide minerals. Pyrrhotite and pyrite generally occur as large masses several hundred microns in size. The arsenopyrite, on the other hand, occurs as clusters of individual crystals which seldom exceed 50 microns, and as individual crystals widely disseminated throughout the gangue. The individual crystals vary from 5 to 15 microns in size, (Figure 1).

Chalcopyrite usually occurs as inclusions in the pyrrhotite and pyrite. Gold occurs as individual grains ranging from 5 to 200 microns in diameter in the gangue (Figure 2). The larger gold grains either contain, or are in contact with the pyrrhotite and arsenopyrite. Very few gold grains are enclosed within the sulphide minerals.

Liberation of the pyrrhotite, pyrite, chalcopyrite and gold should offer no problem. However, the fine grain size of the arsenopyrite may present some difficulty in obtaining a low-arsenic sulphide concentrate.

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* From Internal report MS-70-26, by Dr. E.H. Nickel, Head, Mineralogy Section, Mineral Science Division.

Figure 1. Photomicrograph showing the grain size and distribution of arsenopyrite in the quartz-carbonate matrix.

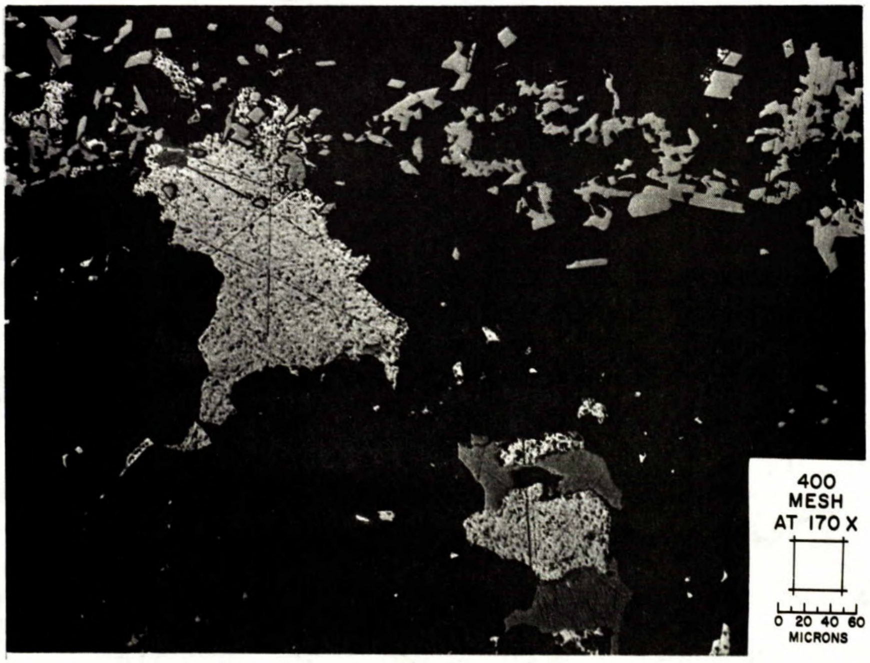

Figure 2. Photomicrograph of native gold (pitted) with pyrrhotite (dark grey) and arsenopyrite (lighter grey).

DETAILS OF INVESTIGATION

Company's Standard Laboratory Procedure

The laboratory procedure, used by the company to test ore which simulates their plant practice, consists of grinding the ore to about 74% minus 200 mesh, passing it over a corduroy blanket to recover free gold in a blanket (gravity) concentrate followed by amalgamation of the blanket concentrate. To recover the gold associated with the sulphides in the ore, the blanket tailing and amalgam tailing are combined and floated under the following flotation conditions.

Ball Mill Grind
2000 gm minus 10-mesh ore
1200 ml water

Flotation (2000-gm machine)
#1 conditioning - 3 minutes
flotation - 5 minutes
#2 conditioning - 3 minutes
flotation - 5 minutes

Reagent Additions

Ball Mill		
Z - 9	0.3	lb per ton
#1 conditioning		
Dowfroth 250	0.035	" " "
A.F. 25	0.020	" " "
#2 conditioning		
Copper Sulphate	0.25	" " "

The concentrate is roasted on a cast iron tray in the Hoskins Furnace at 710°C for 2 hrs & 45 min. The calcine is ground in an Abbé pebble mill to 85% minus 325 mesh and then cyanided 96 hours keeping the NaCN and CaO concentrations at 1.85 and 2.0 lb/ton of calcine respectively.

The flotation (scavenger) tailing is cyanided for 24 hours in a separate circuit using the same concentration of NaCN and CaO as is used in the cyanidation of the calcine.

Preliminary Tests

(a) Rougher Flotation

Ore, ground to minus 28 mesh, was jigged; the jig concentrate was ground to about 75% minus 200 mesh and amalgamated. The amalgamation tailing and the jig tailing, ground to about 75% minus 200 mesh, were combined and used for flotation feed, applying the flotation conditions of the company's standard laboratory procedure, i.e. by adding collectors and forthers, in two stages, to the rougher conditioner and to the scavenger conditioner.

To observe the relationship between per cent weight, sulphur grade and sulphur recovery in the pyritic rougher concentrate, several tests were carried out in which the rougher concentrates were obtained in fractions after 2 and 5 minutes of flotation time and each fraction was assayed separately. The best results obtained are recorded in Table 1.

TABLE 1

Pyrite Rougher and Scavenger Flotation

Two-stage Addition of Collectors

Test No.	Grind, -200 mesh	Flotation Products	% Weight	% S Assay	% S Distn
C-4	75.6	Py rghr conc 1	8.24	17.84	46.7
		" " " 2	5.10	10.80	17.5
		Py rghr conc 1+2	13.34	15.10	64.2
		Py scav conc	2.36	10.28	7.6
		Py scav tail	84.30	1.06	28.2
		Head (calcd)	100.00	3.15	100.0

These results showed that the company's laboratory procedure did not produce a pyrite rougher concentrate of the 18% sulphur grade even when the concentrate was floated in two stages.

(b) Cleaning of the Flotation Rougher Concentrate

When a simple cleaning step of the rougher concentrate was applied without any additional flotation reagents, the required sulphur grade of the pyritic concentrate was obtained. These results are recorded in Table 2.

TABLE 2

Cleaning of Pyrite Rougher Concentrate

Test No.	Grind, -200 mesh	Flotation Products	% Weight	% S Assay	% S Distribution
C-3	82.5	Py cleaner conc	9.4	18.10	56.5
		" " tail	3.6	9.35	11.5
		Py rghr and scav conc (calcd)	13.0	15.65	68.0
		Py scav tail	87.0	1.11	32.0
		Head (calcd)	100.0	3.00	100.0

Modification of Flotation Rougher Procedure

The cleaning operation on the rougher concentrate would require a few additional flotation cells in the company's flotation rougher circuit, i.e. an additional capital cost. Space for the new circuit, and maintenance would be required.

To avoid the cleaning operation and thus eliminate the additional cost, an effort was made to modify the present flotation rougher procedure so that the pyrite rougher concentrate of the required sulphur grade would be produced. The modified procedure was as follows:

a) 0.15 lb/ton of Z-9 (instead of 0.3 lb/ton) was added to the grind followed by addition of 0.012 lb/ton of A-25 (instead of 0.20 lb/ton) and 0.02 lb/ton of Dowfroth 250 (instead of 0.035 lb/ton) to the first rougher conditioner, conditioned for 3 minutes, and then a first pyrite rougher concentrate was floated (1 1/2-minute flotation time).

b) 0.15 lb/ton of Z-9 and 0.012 lb/ton of A-25 were added to the second rougher conditioner, conditioned for 3 minutes, and a second pyrite rougher concentrate was floated (2-minute flotation time).

c) After the second rougher concentrate was obtained, 0.2 lb/ton of $CuSO_4$, 0.05 lb/ton of Z-9 and 0.01 lb/ton of Dowfroth 250 were added to the scavenger conditioner, conditioned for 5 minutes, and then a scavenger concentrate was floated (3-minute flotation time).

This flotation procedure, which would require an additional conditioner to the present circuit, gave a more selective froth than the standard procedure and produced a satisfactory grade of the pyrite rougher concentrate. The flotation results of this test (C-6) are recorded in Table 3.

TABLE 3

Pyrite Rougher and Scavenger Flotation

Three-stage addition of collectors

Test No.	Grind, -200 mesh	Flotation Products	% Weight	% S Assay	% S Distn
C-6	75.6	Py rghr conc #1	2.5	22.48	18.8
		" " " #2	4.4	19.05	28.2
		Py rghr conc 1+2	6.9	20.60	47.0
		Py scav conc	7.9	9.95	26.8
		Py scav tail	85.2	0.92	26.2
		Head (calcd)	100.0	2.98	100.0

Locked-Cycle Test

To indicate the sulphur grade and sulphur recovery as well as the overall gold recovery that might be expected from a plant operation, a six-stage locked-cycle test was carried out. A description of the procedure follows:

Ore was ground (primary grind) to minus 28 mesh and jigged. The jig concentrate, containing coarse and the most of the free gold in the ore, was ground to about 65% minus 200 mesh and amalgamated. The amalgamation tailing and jig tailing were combined and then split into six equal portions. Each portion was then ground to about 75% minus 200 mesh and used as flotation feed for each stage of the cycle test applying the modified flotation procedure as described earlier. The scavenger concentrate (middling) of each flotation stage was added (recycled) to the succeeding rougher conditioner. After the second cycle, the amounts of collector (Z-9) and frother (Dowfroth 250) were reduced to compensate for the amounts of these reagents contained in the recycled scavenger concentrate. This eliminated the possibility of overcollection and overfrothing of the succeeding cycles.

All pyrite rougher concentrates were combined and assayed as one concentrate. The sixth (final) scavenger concentrate was assayed separately and then added to the combined rougher concentrates. These combined concentrates were calcined and cyanided using the company's standard procedure for cyaniding the flotation tailing.

The results of jigging, amalgamation and flotation are recorded in Table 4.

TABLE 4

Cycle Test

Results of Jigging, Amalgamation and Flotation

Treatment Product	% Weight	Assay		Metal Content		Dist'n	
		Au Oz/ton	S %	Au	S	Au %	S %
Jigging and Amalgamation	-		-	0.826*	-	32.0	-
Pyrite rghr conc	7.00	19.56	20.44	1.370	1.434	53.2	48.3
" scav. conc	1.25	5.56	16.44	0.070	0.202	2.7	6.9
Pyrite conc (comb)	8.25	17.45	19.80	1.440	1.636	55.9	55.2
Scav. tail	91.75	0.34	1.45	0.312	1.330	12.1	44.8
Head (assayed)	100.00	2.578	2.996	2.578	2.966	100.0	100.0

* Calculated by difference.

These results showed that jigging and amalgamation recovered 32.0% of the gold in the ore. The flotation produced 19.8% sulphur grade in the pyrite concentrate and contained 55.9% of the gold and 55.2% in the sulphur in the ore. The remaining 12.1% of the gold and 44.8% of the sulphur were left in the scavenger tailing.

The results of cyanidation of the scavenger tailing and calcine are recorded in Table 5.

TABLE 5

Results of Cyanidation of Calcine and Scavenger Tailing

Cyanidation Products	% Weight	Assay oz Au/ton	Content, Oz Au		Au Recovery %		Au Loss %
			Lost	Recovered	In Product	over-all	over-all
Cyanidation residue from calcine	6.05	0.47	0.028	1.412	98.0	54.8	1.1
L.O.I.	2.20	-	-	-	-	-	-
Cyanidation residue from scavenger tail	91.75	0.18	0.165	0.146	47.0	5.7	6.4
Total	100.00					60.5	7.5

* Based on the assayed head - 2.578 oz Au/ton of ore.

These results showed that cyanidation of the calcine recovered 98% of the gold in the pyrite (calcine) concentrate which represents 54.8% of the gold in the ore, i.e. losing only 1.1% of the gold in the ore.

The highest gold loss (6.4%) in this cycle test was in the cyanidation residue from the scavenger tailing.

A summary of results from Table 4 and Table 5, showing the gold recovered in the locked-cycle test, is given in Table 6.

TABLE 6

Summary of Results of Gold Recovery in Cycle Test

Operation	Au Dist'n %	
	Lost	Recovered
Jigging and amalgamation	-	32.0
Cyanidation of calcine	1.1	54.8
Cyanidation of scavenger tail	6.4	5.7
Total	7.5	92.5

The overall gold extraction in this cycle test was 92.5% of the gold in the ore.

CONCLUSIONS

A modified flotation procedure consisting of adding the flotation reagents in two stages (rather than one) and floating the pyritic concentrate in two short stages, within a three minute conditioning period, was successful in producing a concengrate containing over 18 per cent sulphur.

The total reagent additions were the same as used in plant practice but the total flotation time was 3 1/2 minutes instead of 5 minutes. Scavenger flotation time was reduced to 3 minutes with some collector added after the addition of copper sulphate; frother was added as required.

The procedure was confirmed in a six-stage locked-cycle test in which a high-grade sulphur concentrate was obtained (Table 4). An overall gold recovery in the range of 93 per cent can be expected in processing this ore by this method.

ACKNOWLEDGEMENTS

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