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TWO STAGE FLOTATION TREATMENT OF URANIUM ORES FROM FARADAY URANIUM MINES LIMITED

by

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EXTRACTION METALLURGY DIVISION

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SUMMARY OF RESULTS

A series of laboratory flotation tests were done to study the ~~feasibility of treating~~ treatment of Faraday ore ~~using a~~ by two stage flotation ~~treatment~~ in which first the acid-consumers, and then the uranium minerals, were floated.

On ore ground in the Faraday mill grinding circuit, and using Faraday mill water in the flotation step, a recovery in the flotation concentrate of from 91 to 93% of the uranium was obtained in 45 to 50% of the weight, at a grade of 0.24% U_3O_8. The saving in acid in leaching the uranium-bearing flotation concentrate as compared to leaching the whole ore, ranged from 30 to 46%. The average overall extraction resulting from flotation and leaching of the flotation concentrate was 88.1%.

The best collector for the acid-consumers was sodium oleate. Acintol FA-1 emulsion was the best collector for the uranium minerals, and an acid pH in the slurry in the uranium flotation step was found to be necessary. The Reagents ~~required~~ were estimated to ~~cost~~ be from 22 to 26 cents per ton of ore.

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INTRODUCTION

The beneficiation by flotation of ore from Faraday Uranium Mines Limited, Bancroft, Ontario, has been the object of a continuing investigation[1]. This report deals with the results of recent flotation test work conducted with the object of producing a uranium-bearing concentrate which would contain only minor amounts of acid-consuming minerals.

Uranium ore is treated on a commercial scale by flotation in South Africa and Australia to produce a uranium-bearing concentrate suitable for use as feed to an acid leaching plant. In Canada, however, the uranium mining companies have not applied this technique to their ores. Although the uranium minerals can be concentrated from the Canadian ores by flotation, the uranium-bearing concentrate usually contains a concentration of the acid-consuming minerals as well. Since most of the acid-consumers report with the uranium minerals in the flotation concentrate, therefore it requires almost as much acid to leach this concentrate as it does to leach all of the ore. Consequently, there has been litttle incentive for the Canadian operator to pre-treat his ore by flotation with the object of acid leaching the concentrates.

It was with these points in mind that the This investigation, reported here, was begun in an attempt to devise a flotation technique which would make concentration by flotation more attractive to the Canadian operator. It was proposed to combine two earlier flotation processes[1][2],

one designed for the selective recovery of the acid-consuming minerals, and the other designed for the recovery of the uranium minerals. If the acid-consuming minerals were first removed and discarded, it was reasoned that a considerable reduction in acid consumption would be realized on leaching the uranium mineral concentrate produced subsequent to flotation of the acid-consumers. This report describes the flotation tests done to investigate the application of this technique to the Faradday ore, and the leaching tests done to evaluate the flotation results. Samples from the property of Faraday Uranium Mines, Limited at Bancroft, Ontario, are also discussed.

PROCEDURE

Two samples of Faraday ore were obtained for use as flotation feed in this investigation. One sample was 80 lb of minus one inch mill feed, said to be representative of plant feed; The other sample was 50 litres of pulp, containing 61% solids by weight, taken from the Faraday grinding circuit. The latter sample was representative of the grind produced at the Faraday plant to prepare the mill feed for leaching. In addition to the two samples of ore, two or three hundred litres of Faraday mill water were obtained for use as make-up water in the flotation testing.

The minus one inch sample was crushed at the Mines Branch to minus 10 mesh and split into 1150 g lots to be used as feed for the individual flotation tests. Each of these lots was ground at 67% solids

in water for 20 minutes in an Abbé porcelain ball mill charged with 20 lb of steel balls. This procedure resulted in a grind of 51.5% minus 200 mesh. After grinding the sample was filtered, and the filtrate was retained and recombined with the filter cake in the flotation cell, in the proportions required to produce the desired pulp density for flotation.

The ore in the pulp sample taken from the grinding circuit in the Faraday plant was ground to 54.9% minus 200 mesh, a grind suitable for flotation. Consequently, to prepare a sample of this material for flotation, it was only necessary to dip a volume of slurry containing about 1000 g from the well-mixed main sample. The 1000 g sample was transferred to the flotation cell and diluted with water to a pulp density suitable for flotation.

In all the tests, flotation of the samples was carried out at a pulp density of 30% solids in a 500 g Fagergren laboratory flotation cell. The acid-consuming minerals were removed in the first flotation concentrate. On completion of this first step, the flotation tailings were deslimed and the uranium minerals floated from the deslimed portion as a uranium concentrate. Finally, the slimes and the uranium concentrate were combined for the subsequent acid leaching tests.

In the acid-consuming minerals float, the only reagents used were sodium oleate as the collector, sodium silicate as the depressant

for silica, and sodium carbonate to adjust the pH. This reagent combination had been shown in a previous investigation[1] to be satisfactory for the recovery of the acid-consuming minerals in Faraday ore.

In the uranium minerals float, Acintol FA-1* was ~~the main~~ the reagent used as ~~a~~ collector for the uranium minerals. The choice of this reagent for this purpose was based on the results of a previous investigation[2]. Other reagents used in this step were sodium silicate and sodium silicofluoride as gangue depressants, Dowfroth** and cresylic acid as frothers, and sodium carbonate and sulphuric acid as pH regulators.

The overall detailed procedures used in all the tests were similar except for variations in the type and amount of reagents used. The detailed procedure used in Test MF-11 is given as an example, as follows:

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*Acintol FA-1 supplied by Charles Albert Smith, Ltd., (address?)

**Dowfroth supplied by Dow Chemical of Canada Ltd. (")

In this work the Acintol FA-1 was used as an emulsion consisting of 8% Acintol and 0.1% sodium hydroxide in water.

Reagents Added	lb/ton
Acid-consuming Minerals Float	
Sodium silicate	0.5
Sodium carbonate	0.4
Sodium oleate	0.4
Conditioned--3 min (pH=8.5, temp 31°C)	
Rougher float--3 min	
Sodium oleate	0.2
Conditioned--3 min	
1st Scavenger float--3 min	

Combined floats were cleaned with 0.04 lb Dowfroth/ton. The cleaner tailing plus the rougher tailing were combined for desliming.

Desliming Step	
Sodium silicate	0.5
Settled 5 min--the slimes were removed by syphoning off the unsettled fraction. This whole operation was repeated once but no additional sodium silicate was used in the second step.	
Uranium Float	
Sulphuric acid	1.0
Acintol FA-1 (emulsion)	0.5
Conditioned--5 min (pH=6.4, temp 32°C)	
Rougher float--3 min	
Sulphuric acid	0.5
Acintol FA-1 (emulsion)	0.5
Conditioned--3 min (pH=5.6)	
1st Scavenger float--3 min	
Sulphuric acid	0.25
Acintol FA-1 (emulsion)	0.5
Cresylic acid	0.04
Conditioned--3 min (pH=6.3)	
2nd Scavenger float--4 min	

The uranium concentrates (one rougher and two scavenger) were combined with the slimes fraction as feed for the leaching test work.

Leaching tests were carried out on the combined flotation concentrate and slimes from those flotation tests that had given satisfactory uranium recovery. Leaching tests were also done on the two original ore samples. These data were used to evaluate the saving in acid effected by flotation. The leach charges were prepared by mixing the slimes and concentrate fractions together in the relative proportions obtained in the flotation work. The samples were pulped in water to a pulp density of 60% solids in glass beakers and stirred for 24 hours at 25°C. During leaching the pH was maintained at 1.5 by the addition of sulphuric acid. Sodium chlorate was used as an oxidant in all of the tests and was added after the pH had reached a steady value of 1.5.

RESULTS

The two samples of Faraday ore obtained for this investigation were analysed chemically with the results shown in Table 1. Also, a mineralogical study[3] was made of pieces of ore selected from the minus one inch sample. The results of the mineralogical study, given in detail in a separate report[3], showed that uraninite and uranothorite were the main uranium-bearing minerals identified. Acid-consuming

In Tests A-1 and A-2 In the first two tests (Tests A-1, and A-2) in this series, the uranium loss in the final flotation tailing was about 45% indicating poor selectivity. In the light of these results, the use of fuel oil and sulphonated whale oil in the uranium float was discontinued, and in Test A-3 additional Acintol FA-1 was used to compensate for their elimination. Also, in Test A-3, sodium silicate was added as a depressant for silica gangue. The results of this test showed a considerable improvement over those obtained in A-1 and A-2, since the uranium loss in the tailings was reduced to about 10%.

The conditions used in Tests A-4 and A-11 were similar to those of Test A-3 except for an increase in the amount of sodium silicate used from 0.5 to 1.2 and 0.9 lb/ton respectively. This had the effect of further improving the uranium flotation, ~~and~~ the uranium loss as in the tailings ~~was~~ being reduced to 3.8% and 7.8% respectively.

The conditions used in Tests A-5, A-7 and A-10 were substantially the same as those used in Test A-4 except for the substitution of sodium silicofluoride for some of the sodium silicate. In these tests the uranium losses in the flotation tailings were 2.1, 4.9 and 7.7% respectively. Consequently, on the basis of Tests A-4, A-5, A-7, A-10 and A-11, there was no conclusive indication that the use of sodium silicofluoride improves the recovery of the uranium minerals in the flotation concentrate.

In Test A-8, the ore was not deslimed prior to the uranium flotation step. The high uranium loss (18.6%) in the tailing obtained in this test, as compared with those obtained in tests where the slimes were removed (A-5, A-7, A-10), clearly demonstrated the need for the desliming operation.

In Test A-9, tap water was substituted for distilled water throughout the test procedure. The very poor flotation results in this test, compared with those obtained in Tests A-7 or A-10, where the flotation conditions were similar except for the type of water used, indicated the importance of the effect of impurities in the flotation water on the flotation results.

Test Series MF -- Pulp Sample from Grinding Circuit

Because of the apparent importance of the water used in the flotation operation, the next series of tests was done using the pulp sample from the Faraday grinding circuit with Faraday mill water for make-up in the flotation tests. The results of these tests are given in Table 3.

Tests MF-2 and MF-3 were done using the same general procedure that was found to be successful in the previous series. The results of these two tests were quite unsatisfactory since the uranium losses in the final flotation tailings were 20.6 and 36.7% respectively. In Test MF-4, two extra scavenger floats were employed in the uranium flotation step in an attempt to recover more uranium in the flotation

dispersion obtained using sodium silicate. Since only four tests were done using ultrasonic desliming techniques, the results cannot be considered as conclusive. However, the indications are that the desliming of Faraday pulp is not improved by the use of ultrasonic slime dispersion.

CONCLUSIONS

The sulphuric acid consumed in leaching Faraday uranium ore was reduced by 30 to 46% by concentrating the ore prior to leaching. This involved:

(1) flotation of acid-consuming minerals and discarding the concentrate;

(2) desliming the flotation tailing;

(3) flotation of the uranium minerals from the deslimed flotation tailing;

and (4) combining the slimes and the uranium concentrates.

About 2% of the uranium in the ore was lost to the acid-consuming mineral concentrate, and from 5 to 7% was lost to the uranium flotation tailing; the overall recovery in the combined slimes and uranium concentrate was 91 to 93%. The combined slimes and uranium concentrate amounted to 45 to 50% of the weight of the original ore and contained about 0.24% U_3O_8.

The flotation reagents required were 0.6 to 0.7 lb sodium oleate per ton of ore for the acid-consuming minerals float, and 1.5

to 1.7 lb Acintol FA-1 per ton of ore for the uranium float. The pH for the uranium flotation step had to be less than 7.0 for best recovery to be obtained. The cost of the flotation reagent was estimated to be from 22 to 26 cents per ton of ore.

In addition to the reduction in acid consumption, this procedure, by reducing the amount of material to be leached by about 50%, would approximately double the capacity of a given leaching plant. On the other hand, application of this procedure would result in the loss of 7 to 9% of the uranium in the ore during the flotation step.

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