IR 69-35   June 4, 1969

ACID AND BACTERIAL LEACHING OF PRODUCTS
FROM HEAVY-MEDIA SEPARATION TESTS PERFORMED
ON LOW-GRADE STANROCK ORE

by

V. F. Harrison

Extraction Metallurgy Division

Mines Branch Investigation Report IR 69-35

ACID AND BACTERIAL LEACHING OF PRODUCTS FROM HEAVY-MEDIA SEPARATION TESTS PERFORMED ON LOW-GRADE STANROCK ORE

by

V. F. Harrison*

SUMMARY

Acid and bacterial leaching of the sink and float products from heavy-media separation tests yielded an overall uranium recovery of 93.6% at a total cost in reagents of $0.90/T ore handled or $0.73/lb U_3O_8 produced. Acid leaching the whole ore extracted 93.6% of the uranium at a reagent cost of $0.86/T ore handled or $0.74/lb U_3O_8 produced.

*Research Scientist, Hydrometallurgy Section, Extraction Metallurgy Division, Mines Branch, Department of Energy, Mines and Resources, Ottawa, Canada.

INTRODUCTION

Mr. B.G. MacDermid, Mine Manager of Stanrock Uranium Mines Ltd., requested the Mines Branch in Ottawa to investigate the possibility of upgrading a sample of its ore. He stated that ore analysing <0.1% U_3O_8 is at present not economical to mine and treat by existing methods. It was suggested, therefore, that the situation might be resolved by dividing the ore into two fractions by heavy-media separation (HMS) from which the uranium could be removed by known hydrometallurgical processes. This would involve leaching a concentrate (sink product) with strong acid at elevated temperatures, and leaching a tailing (float product) with the help of bacteria at ambient temperatures. If this proposal was proven to be successful, then the economical recovery of the Company's low-grade ore might be realized.

The sink-float experiments were conducted in the Ferrous and Associated Minerals Section of the Mineral Processing Division on both a laboratory and pilot-plant scale[1]. Conventional acid leaching, and bacterial leaching of the products from the sink-float experiments were carried out in the Hydrometallurgical Section of the Extraction Metallurgy Division.

Previous work had shown that Stanrock ore as well as sink-float tails from another Elliot Lake ore responded well to conventional acid and bacterial-leaching methods[2,3]. Consequently, satisfactory recovery of values from the sink-float test products by these techniques was anticipated. To confirm this, conventional acid and bacterial-leach tests were done in which the feed materials were sink-float feed as well as sink-float products[1]. This report describes the procedure used in the leach work. The practical and economic significance of the results obtained are discussed.

PROCEDURE

Samples of the feed to and products from HMS tests conducted[1] on minus ¾-inch ore were submitted for the leaching studies. Uranium analyses of these materials are given in Table 1.

The samples of HMS feed and of the sink fraction were leached by conventional methods after wet-grinding them to about 65% minus 200 mesh. The conventional leach tests were done by mixing the solids with water and 75 lb sulphuric acid per ton of leach feed, and agitating the resulting slurry containing 70% solids

by weight for 48 hr at 75°C temperature. Sodium chlorate was added as required to the leach to maintain an oxidation potential in the solution of from 370 to 400 mV. Slurry samples were filtered, and the solids washed first with $\frac{1}{4}\%$ H_2SO_4 and then with water prior to chemical analysis. Uranium extractions were calculated from the U_3O_8 analyses of the feed and leached solids.

TABLE 1

Uranium Analyses of Heavy-Media Separation Products and Ore

Specific Gravity of Heavy Medium	Product		
	Sink (Conc)	Float (Tails)	HMS Feed
	U_3O_8,%	U_3O_8,%	U_3O_8,%
			0.062
2.68	0.081	0.021	
2.65	0.074	0.020	

The float products as submitted were already ground to minus 100 mesh. These were weighed into 400-g lots, and each lot was mixed with 25 g of minus 200-mesh pyrite concentrate from an Elliot Lake ore. This mixture was contacted with 600 ml $FeSO_4$-nutrient leaching solution (Table 1 of the Appendix). The leaching solution was adjusted to pH 2.0 with H_2SO_4 initially, and then was inoculated with bacteria culture. The leaching was done in percolation apparatus, and in a beaker where the slurry was stirred slowly. Extractions were calculated from the U_3O_8 content of the feed and leach liquor.

RESULTS

Results of acid-leaching the sink product from HMS separation are tabulated in Table 2. About 94% of the uranium could be extracted in 48 hours at 75°C from feed which contained 0.081% U_3O_8. A total of 7.0 lb/T $NaClO_3$ was required to maintain the oxidation potential of the leaching solution at greater than + 380 mV. Previous experience with the Elliot Lake leach ores has shown that an oxidation potential of at least 380 mV is needed for efficient uranium extraction.

TABLE 2

Results of Acid-Leaching Sink Products

Test No.	Analysis	Feed	Final Liquor	Final Residue	% Extr'n, U_3O_8 48 hr	$NaClO_3$, lb/T
1 Sink at sp gr 2.68	pH emf, mV U_3O_8, g/l, % Fe^{+2}, g/l Fe tot, g/l	0.081	0.90 + 400 1.74 8.5 11.8	0.005	93.8	7.0
2 Sink at sp gr 2.65	pH emf, mV U_3O_8, g/l Fe^{+2}, g/l Fe tot, g/l	0.074	0.40 + 370 1.64 9.2 11.4	0.005	93.3	6.0

The results of leaching the float products, shown in Table 1, with bacteria-bearing solutions are given in Table 3 and Figure 1. Both tests were carried on for 9 weeks at a temperature of 32°C. Pyrite amounting to 6% of the float-product weight was added to each test to be sure that there was sufficient sulphur and iron present to support efficient bacterial action. Test 3 was done with percolation leaching apparatus[3]. Test 4 was done by making a slurry of the ore and bacterially active solution. This slurry was stirred slowly and continuously.

Table 3 and Figure 1 show that the two HMS float fractions were leached to over 90% extraction in about 7 weeks. Sulphuric acid consumption was 8 lb H_2SO_4/ton ore in each of these tests. This amount of acid was what was added to adjust the initial pH of the leaching solution to 2.0. The extraction versus time curve shown in Figure 1 is similar to the curves obtained in an earlier study of the bacterial leaching of Elliot Lake ore[3].

Acid leaching of the HMS feed sample which contained 0.062 % U_3O_8 with 75 lb H_2SO_4/ton ore resulted in an extraction of 93.6% in 48 hr at 75°C with a reagent consumption of 5.0 lb/T $NaClO_3$ (Table 4). When the retention time was extended to 96 hr the extraction increased to 95.2% with no increase in reagent requirement. The extraction versus time curve for this test is shown in Figure 2.

TABLE 3

Results of Bacterial-Leaching Float Products

Test No.	Analysis	Feed	Final Liquor	Final Residue	% Extr'n, U_3O_8 4 wk	% Extr'n, U_3O_8 9 wk
3	pH emf, mV U_3O_8, g/1,% Fe^{+2}, g/1 Fe tot, g/1	0.021	1.30 +530 0.11 0.0 9.5	0.001	75.3	95.2
4	pH emf, mV U_3O_8, g/1,% Fe^{+2} g/1 Fe tot, g/1	0.020	1.35 +560 0.08 0.0 5.6	0.001	66.1	95.0

As Figure 2 shows, an extraction of about 94% was obtained in about 48 hr. These results are in agreement with the results obtained in the Stanrock mill when it was in operation. The Stanrock practice, resulted in 94.8% of the uranium being extracted from ore containing 0.096% U_3O_8 at 60-65°C in 56 hours of leaching with a reagent consumption of 43 lb/T H_2SO_4[2]. No free-acid titrations were made in this test to determine acid consumption of the ore, but there is no reason to assume that the acid consumption would be different from that experienced in the plant operation.

TABLE 4

Results of Acid-Leaching HMS Feed

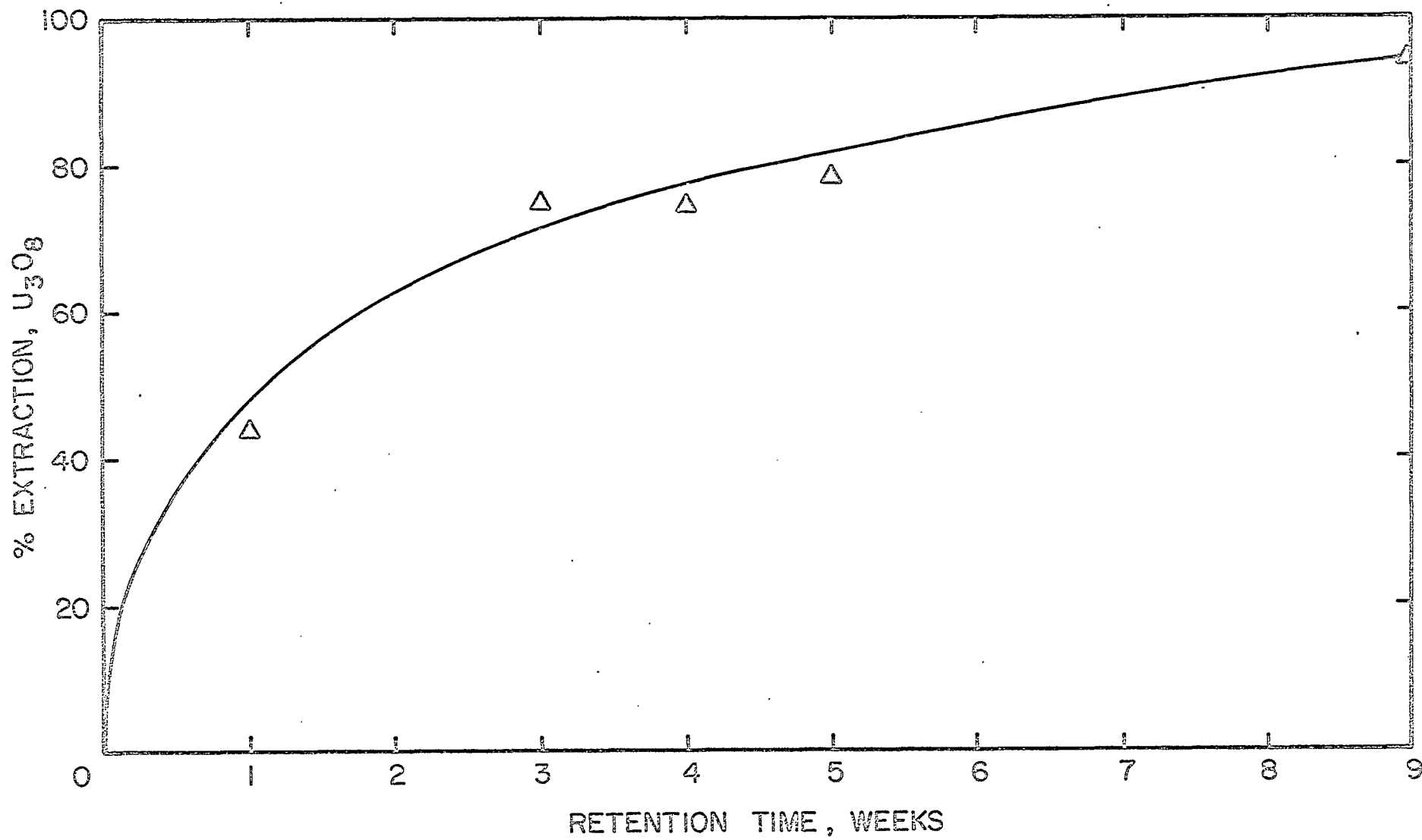| Analysis	Feed	Final Liquor	Residue 48 hr	Residue 96 hr	% Extr'n, U_3O_8 48hr	% Extr'n, U_3O_8 96hr	$NaClO_3$, lb/T
pH emf, mV U_3O_8, g/1,% Fe^{+2}, g/1 Fe tot, g/1	0.062	1.01 +400 1.30 4.7 10.8	0.004	0.003	93.6	95.2	5.0

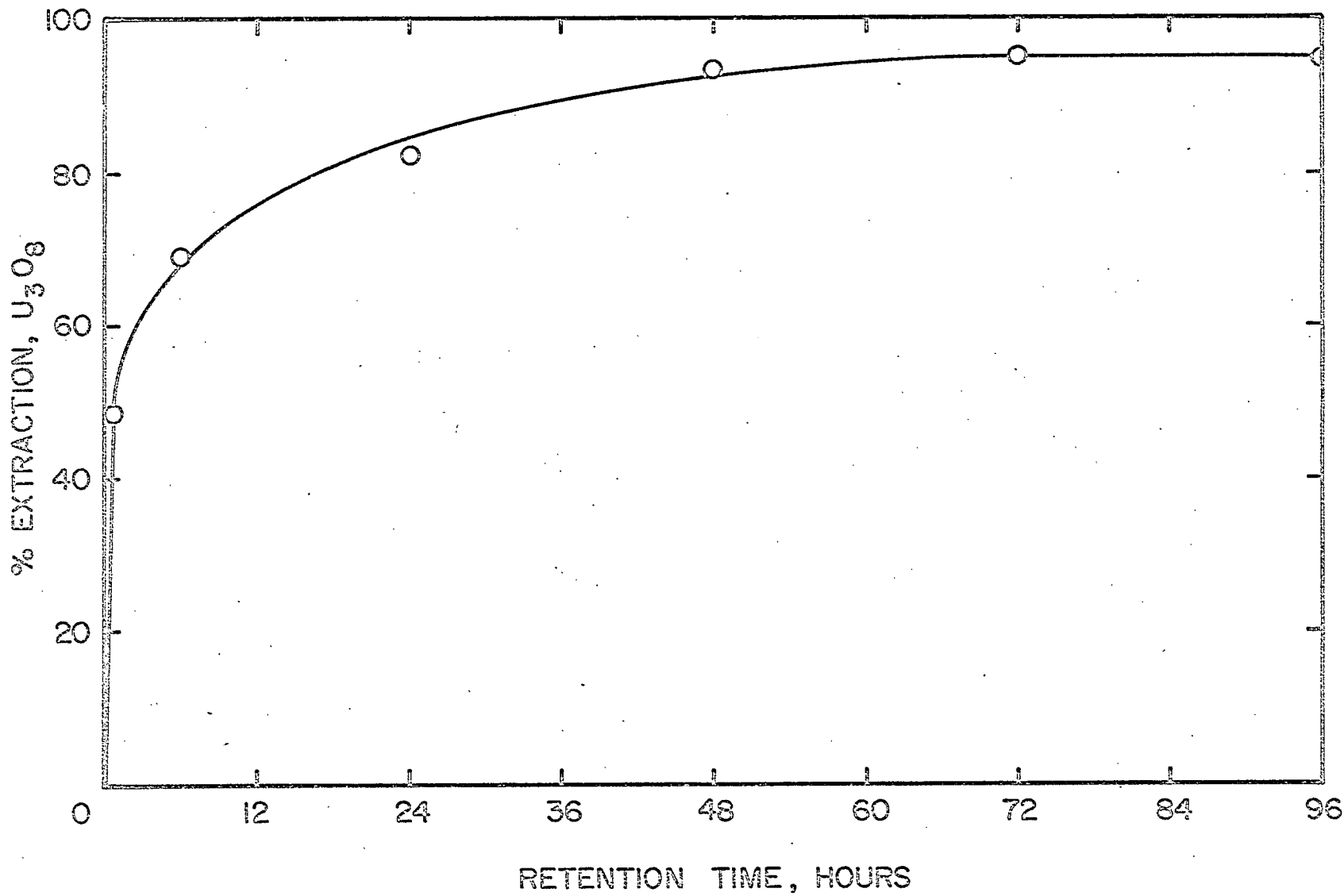Figure 1.   Results of bacterial-leaching float products.

Figure 2, Results of acid-leaching HMS feed.

DISCUSSION

It is apparent from the results of this work, which was done on the feed and products from HMS laboratory tests, that the leaching of the HMS products by conventional and bacterial-leaching techniques will result in about the same overall extraction as would be obtained by leaching the ore (HMS feed) by conventional techniques (Calculation in Appendix). Consequently, if the leaching of HMS products is to have an advantage over the direct leaching of whole ore, the operating costs of the former approach must be significantly lower than those of the latter. This test work allows a comparison of the reagent costs incurred by the two approaches, but these are the larger fraction of the total operating costs incurred in leaching.

The reagent requirements for the whole ore (HMS feed) were 75 lb H_2SO_4 and 5 lb $NaClO_3$ per ton of leach feed. These figures compare with a reported reagent requirement, when the plant was in operation, of 87 lb H_2SO_4 and 3 lb $NaClO_3$ per ton of leach feed[2]. The difference between the amount of acid used in the work reported here and that used in past plant practice is not considered to be significant. Probably the sample used in this work had a slightly lower concentration of acid-consuming minerals than that experienced on the average in past practice. The difference in the sodium chlorate requirements are significant and may result from the fact that, as has been observed in the past, oxidation by aeration in laboratory tests is much less effective than it is in large plant-scale agitators. It would appear from these results that the sodium chlorate requirement in a plant operating on ore similar to the sample used in this work would be 2 lb/ton of ore less than that needed in the laboratory tests.

The conventional acid leaching of the HMS sink fraction required 75 lb H_2SO_4 and 7.0 lb $NaClO_3$ per ton of leach feed. Again, if it is assumed that the laboratory-scale testing needs about 2.0 lb $NaClO_3$ per ton of ore more than will be required in plant practice, then conventional acid leaching of HMS sink product in the plant would require 5.0 lb $NaClO_3$ per ton of leach feed.

Although the bacterial-leach tests were conducted on minus 100-mesh size feed, in actual practice it would be done on minus ¾-inch material. This difference in size could account for at least a tenfold increase in the retention time to reach better than 90% extraction of the uranium. In other words, approximately a year and a half will be required to leach the values from the sink-float tails instead of about 8 weeks, as shown in Figure 1.

If heap leaching were to be done underground, the reduction in temperature from the 32°C used in the laboratory to the 15°C that has been found to be the ambient mine temperature would result in a further, substantial lengthening of the leach time.

In a full-scale operation, the necessity of introducing pyrite to the tails to ensure the presence of sufficient sulphur for efficient bacterial action would be a serious problem. In the laboratory, mixing the small amount of pyrite used to enrich the tails was no problem since the particle size of the pyrite was close to that of the float product which had been ground to minus 100 mesh. At the mine, on the other hand, the only means of obtaining sulphides would be by recovering them by flotation from the leached sink-product.

Combining the results of the leaching investigation reported here along with the HMS results reported by Raicevic[1], it can be shown that the overall extraction obtained by HMS separation followed by conventional leaching of the sink fraction, and bacterial leaching of the float fraction would be about 94% (Appendix). This is the same extraction that would be obtained by leaching the whole ore. Furthermore, as shown in the Appendix, the cost of reagents for the two processes are similar at about $0.74/lb U_3O_8 produced. The reagent cost does not include the cost of recovering the sulphides needed for bacterial leaching from the leached sink-fraction by flotation. The HMS route would also incur an operating cost for the HMS step.

CONCLUSION

An analysis of some of the operating costs incurred by the HMS route taken in conjunction with the metallurgy obtained as compared to a similar analysis relative to conventional acid leaching of the whole ore shows that there would be little chance of reducing the cost of producing a pound of uranium by utilizing the HMS route.

ACKNOWLEDGEMENTS

Thanks are expressed to Mr. H.H. McCreedy for his professional advice, and to Mr. P. Prud'homme for his technical assistance.

REFERENCES

1. D. Raicevic, "Beneficiation studies of A Low-Grade Uranium Ore from The Elliot Lake District, Ontario," Min. Proc. Div. Internal Report MPI (O) 69-25, (May, 1969).

2. Private communication.

3. V.F. Harrison, W.A. Gow and M.R. Hughson, "Factors Influencing the Application of Bacterial Leaching to a Canadian Uranium Ore", J. Met., 18, 1189-1194 (1966).

APPENDIX

TABLE 1

<u>Composition of Unoxidized Iron-Nutrient (9K Medium)</u>

Constituent	g/l
$FeSO_4 . 7H_2O$	44.2
$(NH_4)_2SO_4$	3.0
K_2HPO_4	0.5
KCl	0.1
$MgSO_4 . 7H_2O$	0.5
$Ca(NO_3)_2 . 4H_2O$	0.02
H_2SO_4	to pH 2.0

<u>Calculations</u>

(a) <u>Uranium Recovery</u>

U_3O_8, g

For the sink, 0.728 x 0.081 x 0.938   = 0.0553

For the float, 0.272 x 0.025 x 0.920   = <u>0.0063</u>

    Total   = 0.0616

    Calc'd head,

                 0.728 x 0.081   = 0.0590

                 0.272 x 0.025   = <u>0.0068</u>

    Total   = 0.0658

Overall uranium recovery,

$$\frac{0.0616}{0.0658} \times 100 \qquad = 93.6 \ \%$$

For the ore, $\frac{(0.062 - 0.004)}{0.062} \times 100 \qquad = 93.6 \ \%$

(b) Reagent Costs

For the sink, $\dfrac{2 \times 728}{2000} \times 43 = 31.3$ lb H_2SO_4 @ \$0.02/lb = \$0.63/T

$\dfrac{2 \times 728}{2000} \times 2 = 1.5$ lb $NaClO_3$ @ \$0.15/lb = \$0.23/T

For the float, $\dfrac{2 \times 272}{2000} \times 8 = 2.2$ lb H_2SO_4 @ \$0.02/lb = \$0.04/T

Total = \$0.90/T

which is equivalent to $\dfrac{0.90}{0.0616 \times 20}$ = \$0.73/lb U_3O_8

For the ore, 43 lb H_2SO_4 @ \$0.02/lb = \$ 0.86/T

which is equivalent to $\dfrac{0.86}{0.062 \times 20 \times 0.936}$ = \$0.74/lb U_3O_8

TABLE 2

Summary of Results of Leaching the
Whole Ore and Products From Upgrading
(MPI (O) 69-25)

Products Leached	% Wt in Original Ore	U_3O_8, %		Leaching Process	% Extr'n, U_3O_8	
		Analysis*	Distn		Sample	Original Ore
Whole Ore	100.0	0.062	100.0	H_2SO_4		93.6
Concentrate	72.8	0.081	90.5	H_2SO_4	93.8	84.9
Tails	27.2	0.024	9.5	Bacterial	92.0	8.7
Pre-Concentrate and Waste Combined: % Extr'n, U_3O_8						93.6

* HMS pilot-plant samples.