

CA8205969



Energy, Mines and
Resources Canada

Énergie, Mines et
Ressources Canada

CANMET

Canada Centre
for Mineral
and Energy
Technology

Centre canadien
de la technologie
des minéraux
et de l'énergie

PRECONCENTRATION OF LOW-GRADE URANIUM ORES WITH ENVIRONMENTALLY
ACCEPTABLE TAILINGS; PART I: AGNEW LAKE ORE - 0.057% U

D. RAICEVIC, M. RAICEVIC AND D.R. MCCARTHY

MINERAL DRESSING SECTION
ORE PROCESSING LABORATORY

AUGUST 1979

Project MRP-4.3.6.0.04
Removal of Contaminants from Uranium Mill Effluents

MINERALS RESEARCH PROGRAM
MINERAL SCIENCES LABORATORIES
REPORT MRP/MSL 79-116 (IR)

PRECONCENTRATION OF LOW-GRADE URANIUM ORES
WITH ENVIRONMENTALLY ACCEPTABLE TAILINGS

by

D. Raicevic*, M. Raicevic**, D.R. McCarthy***

ABSTRACT

Ore Sample and Present Method of Uranium Extraction

The low-grade ore sample used for this investigation originated from Agnew Lake Mines Limited, Espanola, Ontario. It contained about 1% pyrite, 0.057% uranium, mainly as uranothorite, and a small amount of brannerite. Both of these minerals occur in the quartz-sericite matrix of a conglomerate.

Presently, the uranium is extracted from this ore by a bacterial leaching process which recovers 25-40% of the uranium in the ore.

Preconcentration Process and Results

A preconcentration process has been developed to give a high uranium recovery, reject pyrite, radium and thorium from the ore and produce environmentally acceptable tailing. By applying flotation, in combination with high intensity magnetic separation and gravity concentration, this concentration process:

- (1) removed about 96% of the pyrite from the ore in a pyrite concentrate comprising about 3.5% of the ore by weight;

*Research Scientist, **Head Technician, ***Technician, Ore Processing Laboratory, Mineral Sciences Laboratories, CANMET, Dept. of Energy, Mines and Resources, Ottawa, Ontario, Canada.

- (2) removed an additional 1-2% of the pyrite and recovered 95-97% of the uranium, 93-94% of the radium and 93-95% of the thorium in a radioactive concentrate grading between 2 and 3 lb of uranium per ton of concentrate depending on the preconcentration procedure applied. The pyrite and radioactive concentrates combined comprised between 32% and 50% of the original ore by weight;
- (3) produced a practically pyrite-free tailing which contained less than 0.1% sulphur as pyrite and comprised 50-68% of the ore by weight. This tailing would be suitable as mine backfill, or for surface storage and vegetation.

The uranium, radium and thorium would be extracted from the concentrates by leaching; the radium, thorium and pyrite could then be buried to avoid environmental concerns similar to those arising from tailings in the Elliot Lake area.

Because about 95% of the uranium is extractable from the pyrite and the radioactive concentrates by either the sulphuric or hydrochloric leaching process, the overall uranium recovery from this Agnew Lake ore sample would be 90-93% of the uranium in the ore depending on the preconcentration procedure applied.

CONTENTS

	<u>Page</u>
ABSTRACT	i
INTRODUCTION	1
Ore Sample Used for this Investigation	3
Present Uranium Extraction at Agnew Lake Mines	4
Analyses and Metallurgical Calculations	4
LABORATORY STUDY ON PRECONCENTRATION OF A LOW-GRADE AGNEW LAKE ORE (0.057% U)	5
Crushing and Grinding	5
Bulk Flotation	6
Tabling	9
High Intensity Wet Magnetic Separation	10
Tabling and Pyrite Flotation	11
Pyrite Flotation and High Intensity Wet Magnetic Separation	11
One-Stage High Intensity Magnetic Separation of Unsized Ore	12
Two-Stage High Intensity Magnetic Separation of Sized Ore	12
Rougher and Scavenger High Intensity Magnetic Separation with and Without Desliming	15
Pyrite Flotation, Tabling and High Intensity Magnetic Separation	16
Pyrite Flotation, Tabling with Desliming and High Intensity Magnetic Separation of 48-mesh Ground Ore	19
Pyrite Flotation, Uranium Flotation, High Intensity Magnetic Separation and Tabling	21
High Intensity Magnetic Treatment of Pyrite Concentrate	23
SUMMARY OF RESULTS AND DISCUSSION	24
CONCLUSIONS	28
ACKNOWLEDGEMENTS	29
REFERENCES	30

TABLES

<u>No.</u>		<u>Page</u>
1.	Size and Metal Distribution in the Ore Ground to Minus 65 Mesh	6
2.	Results from Bulk Flotation	8
3.	Results from Tabling	9
4.	Results from High Intensity Magnetic Separation	10
5.	Results from Tabling and Flotation	11
6.	Results from Pyrite Flotation and High Intensity Magnetic Separation	12
7.	Sala Conditions for Treatment of Coarse Fraction . . .	13
8.	Sala Conditions for Treatment of Fine Fraction	13
9.	Results from Pyrite Flotation and Two-Stage High Intensity Wet Magnetic Separation	14
10.	Results from Pyrite Flotation, High Intensity Magnetic Separation Without (AL-16) and With (AL-16A) Desliming	16
11.	Results from Pyrite Flotation, Tabling and High Intensity Magnetic Separation	18
12.	Results from Pyrite Flotation, Tabling with Desliming and High Intensity Magnetic Separation of 48-mesh Ground Ore	20
13.	Results from Pyrite and Uranium Flotation, High Intensity Magnetic Separation and Tabling	22
14.	Results of Removal of Uranium, Radium* and Thorium from Pyrite Concentrate	23

FIGURES

<u>No.</u>		<u>Page</u>
1.	Flowsheet of Pyrite Flotation and High Intensity Magnetic Separation of Sized Ore	14
2.	Flowsheet of Pyrite Flotation, High Intensity Magnetic Separation Without or With Desliming	15

<u>No.</u>		<u>Page</u>
3.	Flowsheet of Pyrite Flotation, High Intensity Magnetic Separation and Tailing	14
4.	Flowsheet of Pyrite Flotation, Tabling with Desliming and High Intensity Magnetic Separation . .	20
5.	Flowsheet of Pyrite and Uranium Flotation, High Intensity Magnetic Separation and Tabling	22

INTRODUCTION

The conventional sulphuric acid leaching process for normal grade uranium ores (more than 0.1% uranium) recovers about 95% of the uranium in the ores. At a price of \$42-46 per pound of uranium, direct leaching of the ore produces economically attractive revenues. As expected, the economics of leaching uranium ores containing less than 0.1% uranium are less attractive than those of the normal ores due to higher operating costs for the same production of the uranium "yellow cake". For such low-grade ores the concept of preconcentrating the uranium-bearing minerals by ore dressing methods, and then leaching the concentrates instead of the whole ore, is becoming more acceptable^(1,2,3). Preconcentrates usually comprise only a fraction of the ore volume and require smaller leaching, thickening, filtration and other operating units in the plant thus improving the economics of the uranium extraction. Preconcentration would be particularly attractive if it could be carried out underground and if the preconcentration tailings could be used as mine backfill. This would avoid expensive hoisting of and return of the waste material underground. It would also reduce the number of mine pillars or ceiling supports containing economically treatable ore and consequently increase the overall uranium recovery.

The objective of this investigation was to develop a preconcentration process for the treatment of low-grade uranium ores that would:

- (1) produce a sulphide concentrate (if sulphides are present) and a concentrate of uranium-bearing minerals which would comprise less than 50% of the original ore weight and contain over 90% of the original uranium, radium and thorium;
- (2) produce an environmentally acceptable tailing, sulphide-free with a low uranium, radium and thorium content suitable for mine backfill or surface disposal;
- (3) have a sufficiently low cost to make the process economically attractive to industry.

The sulphide concentrate which would contain some uranium, radium and thorium, and the uranium-radium-thorium pre-concentrate would be leached separately either by the conventional sulphuric acid leaching process or by a hydrochloric acid leaching process. Sulphuric acid leaching of the concentrates would extract most of the uranium and thorium; however, over 95% of the radium would remain in the leach residues (tailings) and would continue to present environmental problems similar to those in the Elliot Lake area⁽⁴⁾.

Leaching of the sulphide concentrate and uranium-radium-thorium preconcentrate by the hydrochloric acid leaching^(5,6) and chlorination⁽⁷⁾ processes, both of which are under development at CANMET, would extract not only uranium but also most of the radium and thorium present in the concentrates and produce environmentally acceptable tailings. The leach solutions would then be subject to separate removal of uranium, radium and thorium by ion exchange, solvent extraction or other methods. The uranium "yellow cake" would then be produced from the uranium precipitate by conventional techniques while the radium precipitate would be safely buried and the thorium precipitate stored for possible future commercial use.

The pyrite could also be safely buried or used for sulphuric acid production if required. The preconcentration tailings produced would contain the coarse portion of the ground ore with low uranium, radium and thorium contents which would be suitable for mine backfill, or surface disposal and vegetation.

The HCl leach residue produced from the concentrate containing most of the fines from the ground ore could be mixed with lime, peat moss and fertilizer and used for vegetation of grass. A similar vegetation of the fine portion of the uranium tailings is presently accomplished by the Eldorado Nuclear Limited, Eldorado, Saskatchewan⁽⁸⁾.

Ore Sample Used for this Investigation

A low-grade uranium ore from Agnew Lake Mines Limited, Espanola, Ontario was used for this investigation. This ore contained:

0.057% uranium (U)

200 pCi/g radium-226 (Ra-226)

0.1% thorium (Th)

1.0% sulphur (S) - mainly as pyrite

The uranium ores from the Agnew Lake area have slightly different physical characteristics than other uranium-bearing ores mainly due to numerous fractures throughout the ore. A mineralogical investigation⁽⁹⁾ of this ore showed that uranothorite and monazite are the major radioactive minerals with minor amounts of brannerite also present. The radioactive minerals occur in the quartz-sericite matrix of a conglomerate, and are

randomly disseminated or concentrated in thin seams which also contain rutile, pyrite and occasionally pyrrhotite. The grains of uranothorite appear to be mainly between 48 and 65 mesh in size, and those of monazite, between 35 and 48 mesh. The rare-earth content is due primarily to the presence of monazite. Brannerite commonly occurs as an intergrowth in grains of rutile. Pyrite is the principal sulphide mineral but some pyrrhotite is also present⁽⁹⁾.

Present Uranium Extraction at Agnew Lake Mines

At present, uranium from Agnew Lake ores is extracted by bacterial leaching. About one third of the mined ore is leached on the surface and the remaining two thirds, in the mine stopes. The overall uranium recovery is low; between 25 and 40% of the uranium is recovered from the ore.

Analyses and Metallurgical Calculations

Uranium and thorium analyses for this report were made by optical and X-ray fluorescence methods, respectively. The radium-226 analyses in term of pCi/g were done by alpha spectrometry. Sulphur was determined by a combustion method.

Metallurgical calculations were done by a computer program developed for this purpose at CANMET⁽¹⁰⁾. The tables in this report are original computer printouts reduced in size.

The abbreviations for the computer terminology are listed below:

B	Bulk	MID	Middlings
CL	Cleaner	PY	Pyrite
CON	Concentrate	RO	Rougher
F	Flotation	TA	Table
GR	Gravity	T	Tailings
C	Combined	SC	Scavenger
HI	High Intensity	U	Uranium
J	Jones	SL	Slimes

LABORATORY STUDY ON PRECONCENTRATION OF A
LOW-GRADE AGNEW LAKE ORE (0.057% U)

There are four major methods for preconcentrating uranium ores:

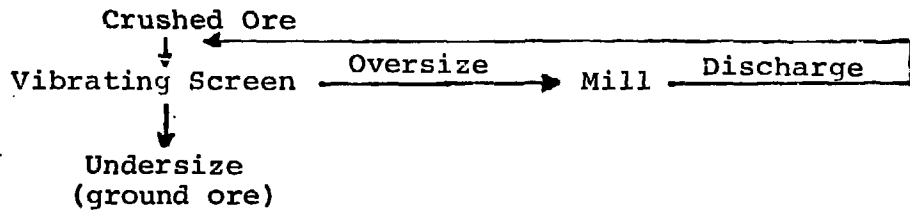
- (1) sorting (separation of lumps of ore on the basis of their uranium content as determined by their radioactivity)
- (2) gravity concentration (jigging, tabling, spirals, heavy-media separation⁽¹¹⁾, cycloning, etc.)
- (3) flotation
- (4) high intensity magnetic separation.

As a sorting machine was not available during this investigation, flotation, tabling, high intensity magnetic separation and their combinations were the major methods applied to preconcentrate this Agnew Lake ore sample.

Crushing and Grinding

The ore was crushed to pass a 10-mesh (1.68 mm) screen and then ground in stages using an Abbe porcelain mill with 3/4-inch steel balls in closed circuit with a 48-mesh (277 μ m)

or a 65-mesh (210 μm) vibrating screen as shown in the following diagram.



A typical screen analysis and the metal distribution of the various size fractions of a minus 65-mesh grind are given in Table 1.

TABLE 1

Size and Metal Distribution in the Ore Ground to Minus 65 Mesh

HEAD SAMPLE		ASSAY						DISTRIBUTION			
PRODUCTS * **	WEIGHT	WEIGHT %	U %	RA-226	TH %	S %	U %	RA-226	TH %	S %	
1_ +210 +65	0.000	0.000									
2_ +149 +100	19.000	11.411	.046	170.000	.062	.910	8.970	8.321	11.044	9.177	
3_ +105 +150	39.100	23.483	.052	175.000	.051	.990	20.868	17.628	18.696	20.546	
4_ +74 +200	17.600	10.571	.058	205.000	.066	1.290	10.477	9.295	10.691	12.051	
5_ +55 +270	16.600	9.970	.059	240.000	.065	1.410	16.032	10.264	10.116	12.424	
6_ +44 +325	10.900	6.547	.059	240.000	.058	1.480	6.600	6.739	5.927	8.583	
7_ +37 +400	10.500	6.306	.065	320.000	.064	1.560	7.005	8.456	6.300	8.674	
8_ +25 +500	9.100	5.465	.064	275.000	.069	1.780	5.977	6.447	5.007	8.500	
9_ -25 -500	43.700	26.246	.067	290.000	.076	.860	30.050	32.649	31.138	19.940	
TOTAL FEED	166.500	100.000	.059	233.129	.064	1.132	100.000	100.000	100.000	100.000	

* μm

** mesh

Bulk Flotation

The sulphide minerals, mainly pyrite, were floated at a natural pH of 7.8-8.2 using about 0.08 lb/ton of potassium amyl xanthate (K.A.X.) as a collector to assure a high pyrite rejection and thus a low pyrite content in the tailings. The sulphur content in the concentrate and tailings was determined by

analysing for the sulphide-sulphide content in these products rather than the iron content since iron could originate from other minerals present in the tailings.

Research on flotation of oxide minerals has shown that the collecting power of fatty acids increases with the increase in the number of unsaturated bonds in the fatty acids hydrocarbon chains. Thus, saturated fatty acids, such as palmitic and stearic acids, are poorer collectors than oleic acid containing one double bond in its hydrocarbon chain. The highly unsaturated fatty acids with larger numbers of unsaturated bonds, such as linoleic and linolenic, are good oxide collectors. For this reason, a flotation reagent called "Otanol", produced from northern pines of Finland by Traversaari Pulp and Paper Mills of Finland, containing mostly linoleic and linolenic unsaturated fatty acids, was tested for the concentration of radioactive minerals.

A similar Canadian product manufactured under the commercial name of "Single Distilled Oleic Acids", manufactured by Canada Packers Ltd., was also used as a collector for the same purpose. The Canadian product of single distilled oleic acids contained:

<u>% Unsaturated fatty acids</u>	<u>% Saturated fatty acids</u>
73% Oleic	5% Palmitic
8% Linoleic	2% Myristic
7% Palmitoleic	1% Stearic
2% Myristoleic	<u>1% Margarin</u>
<u>1% Linolenic</u>	
91% Unsaturated	9% Saturated

Two tests were conducted in which ore was ground to pass 100 mesh (150 μ m). Pyrite and the radioactive minerals were floated together as a bulk concentrate using 0.03 lb of K.A.X. and 1.5 lb of Otanol per ton of ore at a natural pH of 7.8 (Test AL-1) and at a pH of 6.5 using H_2SO_4 as an acidifier. Both reagents were added in stages. The results of these two tests are recorded in Table 2.

TABLE 2
Results from Bulk Flotation

TEST AL-1			ASSAY		DISTRIBUTION	
PRODUCTS	WEIGHT	WEIGHT %	U %	RA-226	U %	RA-226
1-B KO CON1	208.000	10.379	.143	470.000	29.454	25.188
2-B KO CON2	219.000	10.928	.102	440.000	22.120	24.827
3-B KO CON3	281.000	14.022	.070	280.000	19.478	20.272
4-B KO CON4	246.000	12.275	.042	170.000	10.231	10.775
C11234000000	954.000	47.605	.086	329.790	81.284	81.063
5-B KO T	1050.000	52.395	.018	70.000	18.716	18.937
C1CAL.FEED	2004.000	100.000	.050	193.673	100.000	100.000
TEST AL-2			ASSAY		DISTRIBUTION	
PRODUCTS	WEIGHT	WEIGHT %	U %	RA-226	U %	RA-226
1-B KU CON1	212.400	10.597	.135	445.000	23.779	25.096
2-B KU CON2	465.900	23.244	.124	350.000	47.909	43.297
3-B KU CON3	356.900	17.806	.057	165.000	16.870	15.636
4-B KU CON4	199.700	9.963	.019	70.000	3.147	3.712
C11234000000	1234.900	61.609	.090	267.593	91.704	87.741
5-B KO T	769.500	38.391	.013	40.000	8.296	12.259
C1CAL.FEED	2004.400	100.000	.060	187.897	100.000	100.000

Tabling

The ore sample was ground to pass 100 mesh (150 μ m) and tabled on a Deister sand deck to produce a table rougher concentrate and a table rougher tailing. The rougher concentrate was cleaned on the sand deck and a table cleaner concentrate and table cleaner tailing produced.

The table cleaner tailing and rougher tailings were combined and sized on a 325-mesh (44 μ m) screen. The plus 325-mesh fraction was tabled on the Deister sand deck and the minus 325-mesh fraction was tabled on the slime deck. A plus 325-mesh scavenger concentrate (#1) and a minus 325-mesh scavenger concentrate (#2) with two corresponding scavenger tailings were produced. The results of this method (Test AL-3A) are recorded in Table 3.

TABLE 3

Results from Tabling

TEST AL-3A			ASSAY			DISTRIBUTION		
PRODUCTS	WEIGHT	WEIGHT %	U %	RA-224	TH %	U %	RA-224	TH %
1-TA CL CON	130.050	7.875	.300	820.000	.088	46.728	36.874	9.954
2-TA SC CON1	171.300	10.374	.024	100.000	.081	4.926	5.925	12.074
3-TA SC CON2	44.200	2.677	.140	370.000	.220	7.414	5.657	8.461
C11230000000	345.500	20.924	.143	405.453	.101	59.068	48.457	30.489
4-TA SC T1	602.200	36.470	.010	43.000	.017	7.215	8.957	8.908
5-TA SC T2	703.500	42.605	.040	175.000	.099	33.716	42.586	40.403
C14500000000	1305.700	79.076	.026	114.120	.061	40.932	51.543	49.511
CICAL FEED	1651.200	100.000	.051	175.079	.070	100.000	100.000	100.000

High Intensity Wet Magnetic Separation

Two tests were done under slightly different conditions, using a Sala high intensity wet magnetic separator, Laboratory Model No. 10-15-20.

In one test (AL-8), the ore was ground to pass 48 mesh and fed unsized to the Sala separator at 2200 gauss. This produced magnetics (concentrate) and a rougher non-magnetics (tailing).

In the other test (AL-9), the minus 48-mesh ground ore was passed through the Sala separator as in test AL-8 but the rougher non-magnetics were retreated to produce a scavenger magnetic fraction (concentrate) and a scavenger non-magnetic fraction (tailing). No cleaning of the rougher and scavenger concentrates was carried out. The results of these two tests are given in Table 4.

TABLE 4

Results from High Intensity Magnetic Separation

TEST AL-8			ASSAY				DISTRIBUTION			
PRODUCTS	WEIGHT	WEIGHT %	U %	RA-226	TH %	S %	U %	RA-226	TH %	S %
1. HI RD CON	488.000	24.763	.190	625.000	.370	2.180	83.900	74.611	83.537	51.342
2. HI RD T	1482.700	75.237	.012	70.000	.024	.680	16.100	25.389	16.463	48.658
CICAL.FEED	1970.700	100.000	.056	207.433	.110	1.051	100.000	100.000	100.000	100.000
TEST AL-9			ASSAY				DISTRIBUTION			
PRODUCTS	WEIGHT	WEIGHT %	U %	RA-226	TH %	S %	U %	RA-226	TH %	S %
1. HI RD CON	476.500	23.873	.170	580.000	.320	1.900	71.206	63.868	72.357	49.103
2. HI SC CON	472.800	23.607	.056	220.000	.090	1.210	23.274	24.038	20.192	31.028
C:1200000000	949.300	47.560	.113	400.702	.205	1.556	94.480	87.906	92.550	80.131
3. HI SC T	1046.700	52.440	.006	50.000	.015	.350	5.520	12.094	7.450	19.869
CICAL.FEED	1996.000	100.000	.057	216.794	.106	.924	100.000	100.000	100.000	100.000

Tabling and Pyrite Flotation

Ore ground to 100-mesh was sized into plus 270-mesh and minus 270-mesh fractions. Rougher and cleaner sand and slime deck tabling was then carried out on each fraction. This produced two table cleaner concentrates, two rougher and two cleaner tailings. The rougher and cleaner tailings were then combined and the pyrite and uranium rougher concentrates were floated using the same conditions as in the bulk flotation tests. The results obtained from this procedure (Test AL-5) are recorded in Table 5.

TABLE 5

Results from Tabling and Flotation

TEST AL-5 PRODUCTS	ASSAY			DISTRIBUTION		
	WEIGHT	WEIGHT %	U %	RA-226	U %	RA-226
1. TA RD CON1	246.000	8.354	.260	720.000	41.181	23.831
2. TA RD CON2	83.600	2.839	.260	740.000	13.995	8.324
3. PY RD CON	62.000	2.106	.103	470.000	4.112	3.921
4. U RD CON	533.000	18.101	.058	285.000	19.904	20.439
C1123400000	924.600	31.400	.133	454.282	79.191	56.514
5. F RD T	2020.000	68.600	.016	140.000	20.809	43.486
CICAL.FEED	2944.600	100.000	.053	252.404	100.000	100.000

Pyrite Flotation and High Intensity Wet Magnetic Separation

Two tests were carried out using this procedure. In both cases, ore was ground to pass 65 mesh and a pyrite rougher flotation was performed in the usual manner (as described in "Bulk Flotation") to produce a pyrite rougher concentrate.

One-Stage High Intensity Magnetic Separation of Unsized Ore

In the first test (AL-7), the unsized pyrite flotation tailing was treated once by the Sala high intensity wet magnetic separator to produce a rougher magnetic concentrate and a rougher tailing. The results are recorded in Table 6.

TABLE 6

Results from Pyrite Flotation and High Intensity Magnetic Separation

TEST AL-7	ASSAY						DISTRIBUTION			
	PRODUCTS	WEIGHT	WEIGHT %	U %	RA-226	TH %	S %	U %	RA-226	TH %
1_PY RO CON	69.000	3.463	.240	750.000	.280	23.240	13.767	12.159	8.626	84.165
2_HI RU CON	497.000	24.944	.180	580.000	.340	.320	74.373	67.730	75.450	8.347
C11200000000	566.000	28.407	.187	600.724	.333	3.114	88.141	79.890	84.077	92.513
3_HI RO T	1426.500	71.593	.010	60.000	.025	.100	11.859	20.110	15.923	7.487
C:CAL.FEED	1992.500	100.000	.060	213.601	.112	.956	100.000	100.000	100.000	100.000

Two-Stage High Intensity Magnetic Separation of Sized Ore

In the second test (AL-11), the tailing of the pyrite rougher flotation was separated into two fractions by screening. The minus 270-mesh fraction was subjected to a rougher, scavenger and cleaner treatment using the Sala separator to produce a minus 270-mesh scavenger tailing, cleaner tailing and a radioactive cleaner concentrate. The plus 270 mesh was also subjected to a rougher and cleaner treatment by the Sala separator. This produced a plus 270-mesh rougher tailing and a radioactive cleaner concentrate.

The operating conditions of the Sala separator for the treatment of the coarse ore fraction (plus 270 or plus 200 mesh) and for the treatment of the fine fraction (minus 270 or minus 200 mesh) are recorded in Table 7 and Table 8.

TABLE 7

Sala Conditions for Treatment of Coarse Fraction

Sample		% Solids	Magn. Field, Gauss	Matrix	Flow Restrictor	Rinse, Min
Name	Grams					
Coarse fraction	100	10	14,000	57	0.073	2

TABLE 8

Sala Conditions for Treatment of Fine Fraction

Sample		% Solids	Magn. Field, Gauss	Matrix	Flow Restrictor	Rinse, Min
Name	Grams					
Fine fraction	100	10	14,000	2	108	1

Details of the procedure are given in Figure 1 and the results (Test AL-11), in Table 9.

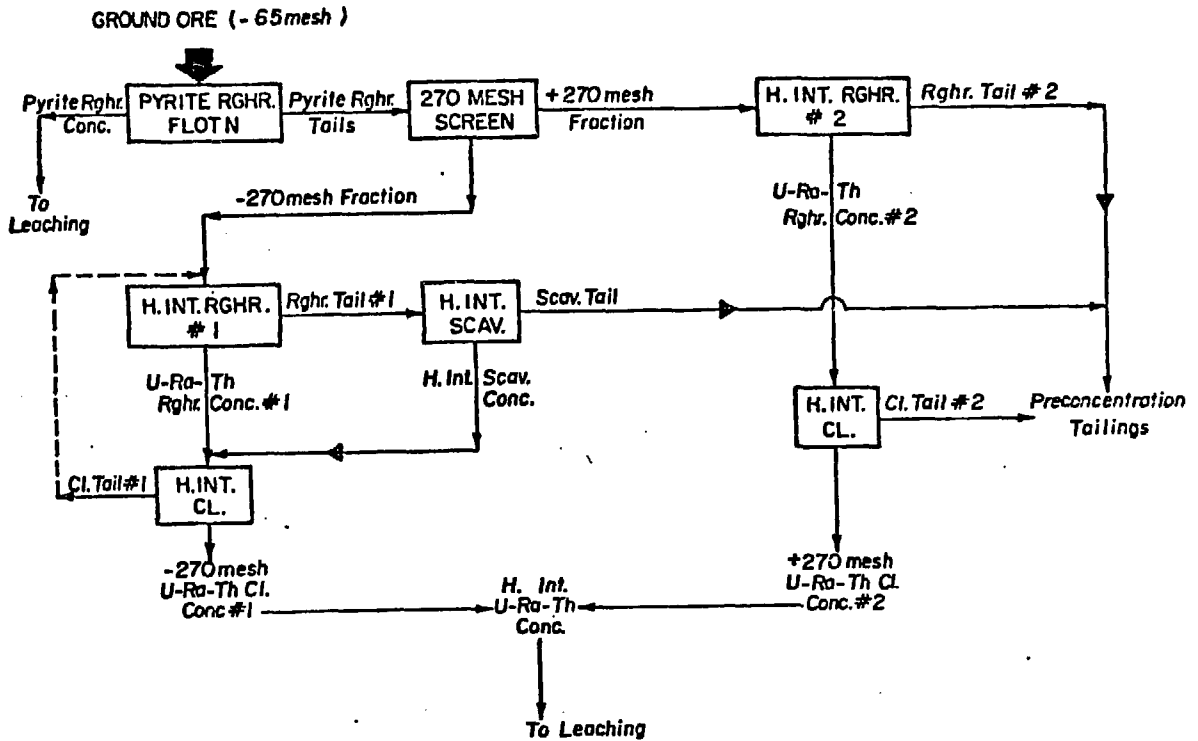


Fig. 1 - Flowsheet of Pyrite Flotation and High Intensity Magnetic Separation of Sized Ore

TABLE 9

Results from Pyrite Flotation and Two-Stage High Intensity Wet Magnetic Separation

TEST AL-11											
PRODUCTS	WEIGHT	WEIGHT %	ASSAY				DISTRIBUTION				
			% U	RA-226	% TH	% S I	% U	RA-226	% TH	% S I	
1. PY RD CON	290.000	2.900	.220	3650.000	.340	27.220	10.531	35.591	7.017	87.971	
2. HI CL CON2	1730.000	17.300	.180	595.000	.450	.170	51.399	34.611	55.402	3.278	
3. HI CL CON1	1180.000	11.800	.170	585.000	.340	.400	33.111	23.211	28.552	5.260	
C1123000000	3200.000	32.000	.180	868.172	.39%	2.706	95.040	93.413	90.971	96.508	
14. HI CL T1	400.000	4.000	.018	105.000	.042	.170	1.188	1.412	1.196	.758	
15. HI SC T	1930.000	19.300	.005	45.000	.016	.050	1.593	2.920	2.198	1.075	
16. HI CL T2	210.000	2.100	.002	15.000	.012	.100	.069	.106	.179	.234	
17. HI RD T2	4260.000	42.600	.003	15.000	.018	.030	2.109	2.149	5.457	1.424	
C1567000000	6400.000	64.000	.004	24.047	.017	.038	3.772	5.175	7.834	2.734	
C1CAL.FEED	10000.000	100.000	.061	297.405	.141	.897	100.000	100.000	100.000	100.000	

Rougher and Scavenger High Intensity Magnetic Separation
With and Without Desliming

Two tests (AL-16 and AL-16A) were carried out using the procedure illustrated in Figure 2. The ore samples were ground to minus 65 mesh, the pyrite was floated in the usual manner and the ore was sized into plus and minus 200-mesh fractions. The coarse fraction was treated by the Sala separator and a rougher concentrate and tailing were produced. The fine fraction was subjected to rougher and scavenger Sala separations. In test AL-16, the Sala scavenger tailing was analysed as produced but in test AL-16A, this scavenger tailing was deslimed (dotted lines on the flowsheet), the slimes produced analysed and added to the combined (final) high intensity concentrate. The deslimed tailing was then discarded as part of the combined preconcentration tailings. The results of this procedure are given in Table 10.

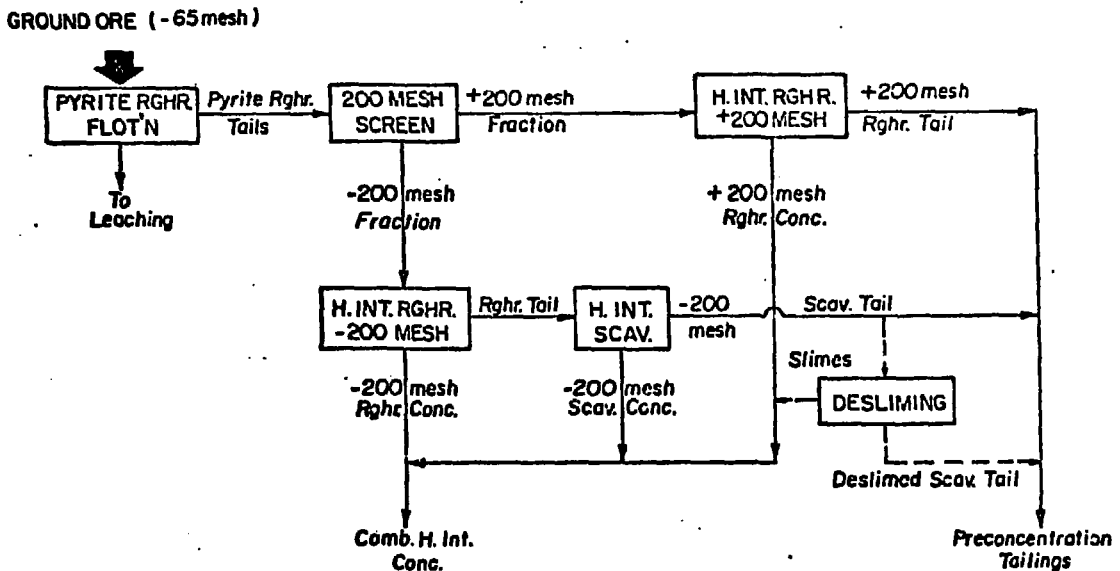


Fig. 2 - Flowsheet of Pyrite Flotation, High Intensity Magnetic Separation Without or With Desliming

TABLE 10

Results from Pyrite Flotation, High Intensity Magnetic Separation
Without (AL-16) and With (AL-16A) Desliming

TEST AL-16											
PRODUCTS	WEIGHT	WEIGHT %	ASSAY				DISTRIBUTION				
			U %	RA-226	TH %	S %	U %	RA-226	TH %	S %	
1. FY RO CON	114.500	5.796	.180	700.000	.330	13.760	17.059	19.845	13.995	91.877	
2. HI RO CON1	328.000	16.603	.110	380.000	.270	.100	29.864	30.861	32.801	1.913	
3. HI RO CON2	413.000	20.906	.140	360.000	.320	.130	47.859	36.814	40.949	3.151	
C:1230000000	855.500	43.305	.134	413.174	.302	1.943	94.782	07.521	95.745	96.921	
4. HI RO T1	640.000	32.397	.002	15.000	.007	.030	.848	2.377	1.508	1.120	
5. HI RO T2	480.000	24.290	.011	85.000	.015	.070	4.370	10.102	2.667	1.959	
C:4560000000	1120.000	56.695	.006	45.000	.010	.047	5.218	12.479	4.255	3.079	
C:CAL.FLED	1975.500	100.000	.061	204.439	.137	.868	100.000	100.000	100.000	100.000	
TEST AL-16A											
PRODUCTS	WEIGHT	WEIGHT %	ASSAY				DISTRIBUTION				
			U %	RA-226	TH %	S %	U %	RA-226	TH %	S %	
1. FY RO CON	114.500	5.820	.180	700.000	.330	13.760	19.466	10.794	15.636	91.120	
2. SC SI	217.000	11.029	.042	250.000	.120	.180	8.608	12.721	10.776	2.259	
3. HI RO CON1	328.000	16.671	.110	300.000	.270	.100	34.078	29.226	36.648	1.897	
4. HI RO CON2	263.000	13.367	.140	500.000	.300	.160	34.777	30.835	32.651	2.434	
C:1234000000	922.500	46.887	.111	423.350	.251	1.831	96.929	91.576	95.712	97.718	
5. HI RO T1	640.000	32.529	.002	15.000	.007	.030	.967	2.251	1.774	1.111	
6. HI RO T2	405.000	20.584	.006	65.000	.015	.050	2.104	6.173	2.514	1.171	
C:5640000000	1045.000	53.113	.003	34.378	.010	.038	3.071	0.424	4.280	2.282	
C:CAL.FLED	1947.500	100.000	.054	216.755	.123	.879	100.000	100.000	100.000	100.000	

Pyrite Flotation, Tabling and High Intensity Magnetic Separation

Four tests with slightly different conditions were conducted by applying the following.

In the first two tests, AL-6 and AL-10, the ore samples were ground to minus 48 and minus 65 mesh, respectively. Pyrite was floated in the usual manner and the pyrite rougher tailing was separated into plus and minus 270-mesh fractions by screening. The plus 270-mesh fractions of each test were tabled on Deister tables which produced plus 270-mesh table rougher and cleaner (when applied) concentrates and corresponding table tailings. The tables were operated in such a way to produce

slightly different concentrate grades in order to observe the variation in the uranium, radium, thorium and sulphur contents in the table tailings. The minus 270-mesh fractions were treated by high intensity magnetic separation; in test AL-16, by the Jones separator and in test AL-10, by the Sala separator.

In the third (AL-12) and fourth (AL-13) tests, the pyrite flotation was done in the same manner but the pyrite rougher tailings were screened into plus and minus 200-mesh fractions; the plus 200-mesh fractions were tabled on Deister sand rougher and cleaner decks and the minus 200-mesh fractions were treated by Sala separator at 15,000 gauss (AL-12) and 23,000 gauss (AL-13).

The flowsheet of this procedure is given in Figure 3 and the results in Table 11.

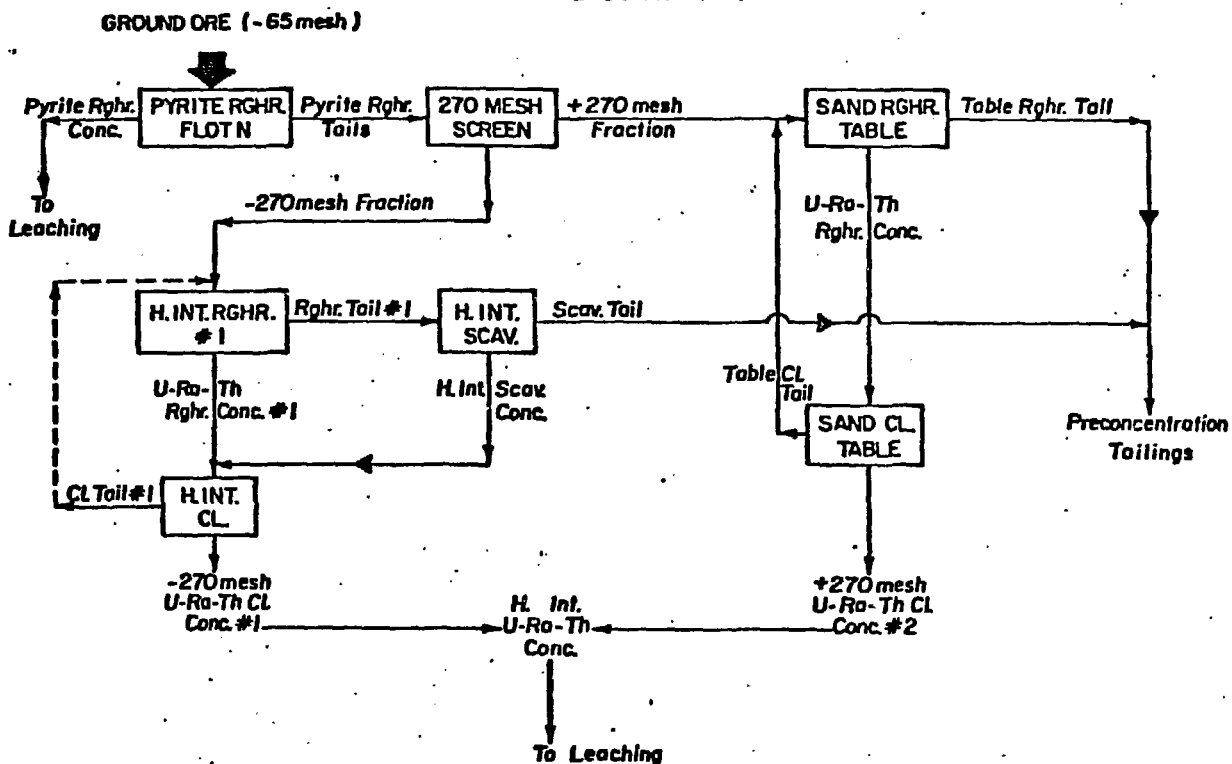


Fig. 3 - Flowsheet of Pyrite Flotation, High Intensity Magnetic Separation and Tailing

TABLE II

Results from Pyrite Flotation, Tabling
and High Intensity Magnetic Separation

TEST AL-6			ASSAY					DISTRIBUTION			
PRODUCTS	WEIGHT	WEIGHT %	U %	RA-226	TH %	S %	U %	RA-226	TH %	S %	
1_PY RD CON	77.900	3.799	.260	3400.000	.390	22.800	17.739	47.899	13.935	87.729	
2_GR RD CON	272.100	13.270	.170	495.000	.390	.230	40.512	24.358	48.676	3.091	
3_J RD CON	143.000	6.974	.200	595.000	.380	.230	25.048	15.387	24.925	1.625	
C:123000000	493.000	24.043	.193	983.031	.387	3.796	83.299	87.645	87.536	92.445	
4_GR RD T	942.000	45.940	.008	25.000	.015	.100	6.600	4.259	6.481	4.653	
5_J RD MID	337.000	16.435	.021	75.000	.023	.100	6.198	4.571	3.555	1.665	
6_J RD T	278.500	13.582	.016	70.000	.019	.090	3.903	3.526	2.427	1.238	
C:1456000000	1557.500	75.957	.012	43.865	.017	.098	16.701	12.355	12.464	7.555	
C:CAL.FEED	2050.500	100.000	.056	269.668	.106	.987	100.000	100.000	100.000	100.000	
TEST AL-10			ASSAY					DISTRIBUTION			
PRODUCTS	WEIGHT	WEIGHT %	U %	RA-226	TH %	S %	U %	RA-226	TH %	S %	
1_PY RD CON	116.300	3.127	.220	3600.000	.340	27.220	12.237	43.523	7.021	93.616	
2_GR CL CON	702.000	18.872	.120	585.000	.340	.040	40.289	42.690	42.382	.830	
3_HI RD CON	72.400	1.946	1.090	73.500	2.980	.640	37.742	.553	38.311	1.370	
C:123000000	890.700	23.945	.212	937.096	.555	3.638	90.268	86.766	87.715	95.817	
4_GR RD T	761.000	20.459	.005	45.000	.042	.050	1.826	3.560	5.675	1.125	
5_HI RD T	2068.000	55.596	.008	45.000	.018	.050	7.912	9.674	6.610	3.058	
C:1450000000	2829.000	76.055	.007	45.000	.024	.050	9.732	13.234	12.285	4.183	
C:CAL.FEED	3719.700	100.000	.056	258.617	.151	.909	100.000	100.000	100.000	100.000	
TEST AL-12			ASSAY					DISTRIBUTION			
PRODUCTS	WEIGHT	WEIGHT %	U %	RA-226	TH %	S %	U %	RA-226	TH %	S %	
1_PY RD CON	197.000	4.173	.220	1875.000	.370	21.010	17.439	28.177	13.390	92.977	
2_GR CL CON	309.000	6.545	.330	1120.000	.750	.190	41.029	26.400	42.572	1.319	
3_HI RD CON	449.000	9.511	.150	675.000	.360	.200	27.099	23.119	29.693	2.017	
C:123000000	955.000	20.229	.223	1066.524	.488	4.490	85.567	77.696	85.655	96.313	
4_GR RD T	2416.000	51.176	.009	40.000	.015	.040	8.457	7.372	6.657	2.171	
5_HI RD T	1350.000	28.596	.011	145.000	.031	.050	5.975	14.932	7.688	1.516	
C:1450000000	3766.000	79.771	.010	77.639	.021	.044	14.433	22.304	14.345	3.687	
C:CAL.FEED	4721.000	100.000	.053	277.678	.115	.943	100.000	100.000	100.000	100.000	
TEST AL-13			ASSAY					DISTRIBUTION			
PRODUCTS	WEIGHT	WEIGHT %	U %	RA-226	TH %	S %	U %	RA-226	TH %	S %	
1_PY RD CON	197.000	4.177	.220	1175.000	.370	21.010	16.020	19.359	13.404	92.782	
2_GR CL CON	309.000	6.552	.370	1120.000	.750	.190	42.261	28.944	42.616	1.316	
3_HI RD CON	481.000	10.199	.150	850.000	.340	.210	26.669	34.193	30.073	2.264	
C:123000000	987.000	20.929	.233	999.397	.474	4.355	84.950	82.496	86.093	96.362	
4_GR RD T	2416.000	51.230	.009	35.000	.015	.040	7.770	7.072	6.664	2.166	
5_HI RD T	1313.000	27.841	.015	95.000	.030	.050	7.280	10.432	7.243	1.472	
C:1450000000	3729.000	79.071	.011	56.126	.020	.044	15.050	17.504	13.907	3.638	
C:CAL.FEED	4716.000	100.000	.057	253.541	.115	.946	100.000	100.000	100.000	100.000	

Pyrite Flotation, Tabling with Desliming and High Intensity Magnetic Separation of 48-mesh Ground Ore

To find out if a grind coarser than 65 mesh could give satisfactory results, and at the same time to generate less slimes and reduce the grinding and overall preconcentration costs, a procedure was designed whereby ore was ground to minus 48 mesh, followed by pyrite flotation, slime-deck tabling, desliming and Sala high intensity magnetic separation. The Sala was operated by applying the conditions given in Tables 7 and 8. Desliming of the minus 200-mesh fraction of the pyrite rougher tailing was done directly from the Deister slime table using the slime deck not only as a concentrator but also as a deslimer. As in Test AL-16A, the slimes produced were added to the uranium concentrate. This procedure (Test AL-17) which is presented in Figure 4, although conducted at a relatively coarse grind for flotation, gave high uranium, radium, thorium, and pyrite recoveries. The combined (pyrite and uranium) concentrate comprised 49.7% of the ore by weight and had a sufficiently high uranium grade for either the sulphuric or the hydrochloric leaching process. This procedure also gave preconcentration tailings with very low uranium, radium, thorium and sulphur contents. The results are given in Table 12.

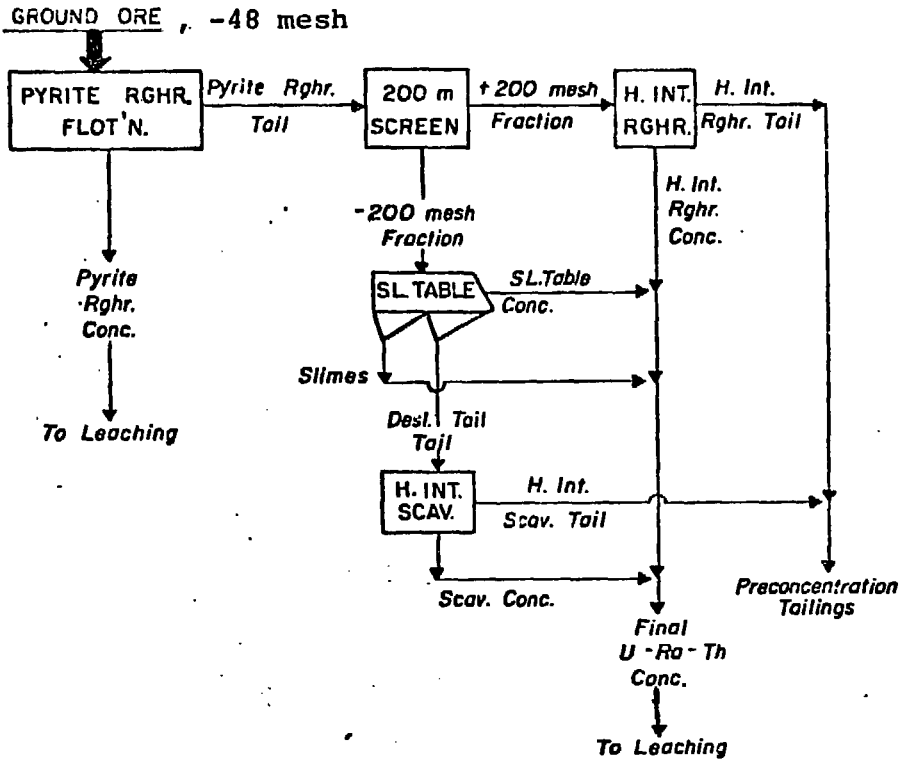


Fig. 4 - Flowsheet of Pyrite Flotation, Tabling with Desliming and High Intensity Magnetic Separation

TABLE 12

Results from Pyrite Flotation, Tabling with Desliming and High Intensity Magnetic Separation of 48-mesh Ground Ore

TEST AL-17	ASSAY						DISTRIBUTION			
	PRODUCTS	WEIGHT	WEIGHT %	U %	RA-226	TH %	S %	U %	RA-226	TH %
1-PY RO CON	78.500	4.002	.210	800.000	.360	20.400	14.969	15.435	10.247	88.825
2-TA SLIMES	115.200	5.873	.046	230.000	.100	.210	4.812	6.512	4.177	1.342
3-SL TA CON	21.800	1.111	.490	1760.000	.920	.270	9.700	9.430	7.272	.326
4-HI SC CDN	131.000	6.679	.085	320.000	.200	.190	10.111	10.303	9.500	1.391
5-HI RO CON	620.000	31.608	.102	335.000	.260	.140	57.425	51.050	62.946	4.815
C:1234500000	966.500	49.274	.111	390.361	.269	1.804	97.017	92.731	94.142	96.689
6-HI RO T	760.000	38.746	.003	25.000	.016	.060	2.001	4.670	4.409	2.529
7-HI SC T	235.000	11.981	.005	45.000	.017	.060	.982	2.599	1.449	.782
C:1670000000	995.000	50.726	.003	29.724	.016	.060	2.983	7.269	5.858	3.311
C:CAL.FEED	1961.500	100.000	.056	207.422	.141	.919	100.000	100.000	100.000	100.000

The screen analysis of the combined preconcentration tailing of this test (AL-17) was as follows:

Mesh	+65	+100	+150	+200	+270	+325	-325
% Weight	166	28.0	16.9	9.8	6.6	2.8	19.3

This screen analysis indicates that the preconcentration tailing produced is suitable for mine backfill.

Pyrite Flotation, Uranium Flotation, High Intensity Magnetic Separation and Tabling

Four tests were carried out applying the following procedure. Pyrite was floated in the usual manner and the radioactive minerals were floated using about 2 lb of unsaturated fatty acids per ton of ore as collector without frother addition. The flotation tailing produced was sized into plus and minus 200-mesh fractions by screening.

The plus 200-mesh fraction was tabled on a sand Deister table. The rougher concentrate produced was then cleaned on another Deister sand table. The tailing from the cleaner table was returned to the head of the rougher table. This tabling of the plus 200-mesh fraction of the flotation tailing produced table cleaner concentrate and rougher tailing.

The minus 200-mesh fraction of the flotation tailing was treated by high intensity magnetic separation at the conditions given in Table 8. The flowsheet of this test (AL-14C) is given in Figure 5 and the results in Table 13.

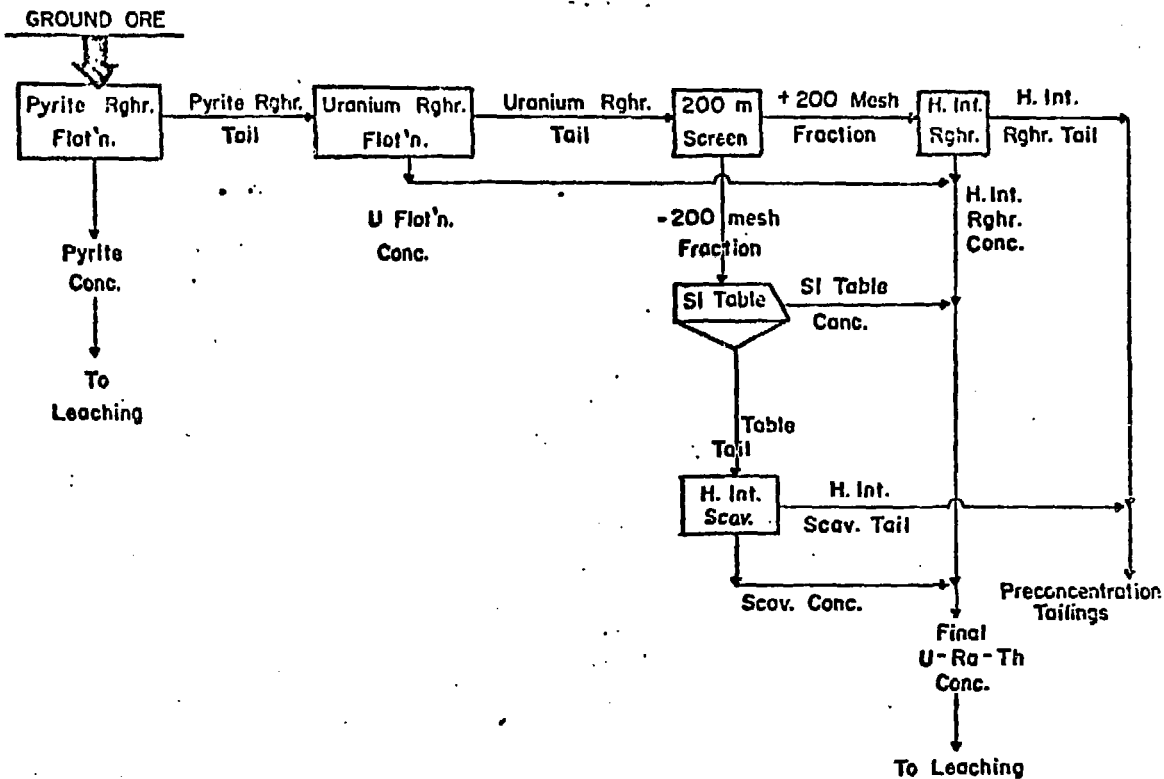


Fig. 5 - Flowsheet of Pyrite and Uranium Flotation, High Intensity Magnetic Separation and Tabling.

TABLE 13

Results from Pyrite and Uranium Flotation, High Intensity Magnetic Separation and Tabling

TEST AL-14C	ASSAY					DISTRIBUTION				
	PRODUCTS	WEIGHT	WEIGHT %	U %	RA-226	TH %	S %	U %	RA-226	TH %
1-PY RO CDN	145.900	3.685	.220	840.000	.320	22.380	13.558	11.744	7.610	88.465
2-U RO CDN	130.600	3.298	.086	345.000	.370	.330	4.744	4.310	7.876	1.168
3-GR RO CDN	510.000	12.800	.160	775.000	.420	.100	34.467	37.874	34.912	1.382
4-NI RO CDN	702.000	19.750	.120	475.000	.330	.280	39.637	35.594	42.061	5.932
C1123400000	1548.500	39.614	.139	595.673	.362	2.201	92.405	89.529	92.459	96.947
5-GR RO T	1530.000	38.043	.007	35.000	.014	.040	4.353	5.158	3.509	1.667
6-NI RO T	833.000	21.543	.009	65.000	.029	.060	3.243	5.313	4.032	1.307
C1560000000	2391.000	60.386	.000	45.703	.018	.047	7.595	10.471	7.541	3.053
CICAL.FEED	3959.500	100.000	.040	263.366	.155	.932	100.000	100.000	100.000	100.000

High Intensity Magnetic Treatment of Pyrite Concentrate

The pyrite obtained by flotation as a separate concentrate always contained some uranium, radium and thorium. To extract these radioactive metals and make the pyrite available for burial or for sulphuric acid production, the pyrite concentrate should be separately leached. To find out if the separate leaching of the pyrite concentrate could be avoided, the removal of the uranium, radium and thorium by high intensity wet magnetic separation was tested. For this purpose, the pyrite concentrate was first sized on 200-mesh screen. The minus 200-mesh fraction was then subjected to a rougher and cleaner Sala high intensity separation; the rougher at 9,000 gauss and the cleaner at 6,000 gauss. The results for this fraction are given in Table 14.

TABLE 14

Results of Removal of Uranium, Radium* and Thorium from Pyrite Concentrate

TEST AL-19	ASSAY					DISTRIBUTION		
	PRODUCTS	WEIGHT	WEIGHT %	U %	TH %	S %	U %	TH %
1_PY CL CDN	148.300	48.464	.370	.550	26.580	81.249	79.952	70.162
2_PY CL T1	38.200	12.484	.100	.160	15.760	5.656	5.991	10.716
C:1200000000	186.500	60.948	.315	.470	24.364	86.906	85.944	80.878
3_PY CL T2	119.500	39.052	.074	.120	8.990	13.094	14.056	19.122
C:CAL.FEED	306.000	100.000	.221	.333	18.360	100.000	100.000	100.000

*Radium analysis not carried out.

Because results for the minus 200-mesh fraction were not satisfactory, the separation of the plus 200-mesh fraction by the Sala was not carried out.

SUMMARY OF RESULTS AND DISCUSSION

The application of bulk flotation alone, i.e., flotation of pyrite and radioactive minerals together (Table 2), tabling alone (Table 3), tabling and pyrite flotation (Table 5), did not meet the objective of the investigation.

The high intensity magnetic treatment of the unsized ore by the Sala separator alone, applying rougher and scavenger treatment (Table 4, Test AL-9), gave a good combined high intensity magnetic concentrate comprising 47.56% of the ore by weight and assaying 0.113% U (2.26 lb) per ton of concentrate with 94.5% uranium recovery. The radium and thorium recoveries were in the expected range, 87.9% Ra-226 and 92.5% Th, but the pyrite recovery was only 80%, considerably lower than desired. The scavenger tailing had low uranium, radium and thorium contents but unacceptable sulphur content (0.35% S), indicating that pyrite has to be removed by flotation prior to the application of other methods to meet the goal of less than 0.1% S in the preconcentration tailings.

The application of a procedure consisting of pyrite flotation followed by high intensity magnetic separation of sized flotation tailing gave satisfactory results (Table 7, Test AL-11). These results are summarized as follows:

Products of Test AL-11	Wt %	Assays				Distribution %			
		U	Ra	Th	S	U	Ra	Th	S
Py. & U Conc.	32	0.18	868	0.4	2.7	95.0	93.4	91.0	96.5
Preconc. Tails	68	0.004	24	0.017	0.04				

It should be realized that the concentration of minerals by high intensity magnetic separation is a fairly simple process, however, it involves some expenses due to the capital cost of the equipment, electrical power and water requirements. Cost of the uranium concentration from a South African gold-uranium ore is quoted to be 69 South African cents⁽¹⁾ per ton of ore or about one Canadian dollar per ton of ore.

Sliming of the uranium-bearing minerals during grinding of uranium ores is a common occurrence because these minerals, including uranothorite, are softer than most of the other minerals in the ore. The recovery of these slimes by ore dressing methods has always been a problem. Because a matrix for the Sala separation suitable for slimes was not available at this time, the recovery of the uranothorite slimes by this separator could not be attempted.

The possibility of recovering these slimes by other means, such as thickening or cycloning, was tried. Figure 2 outlines how the Sala scavenger tailings, obtained from the minus 200-mesh fraction of the pyrite rougher tailing where most of the slimes remain, were deslimed by the thickening method. The slimes produced, comprising about 11% of the ore by weight, were added to the uranium preconcentrate. This procedure gave high uranium, radium, thorium and pyrite recoveries and resulted in the low contents of these metals and pyrite in the preconcentration tailing (0.003% uranium, 34 pCi/g radium-226, 0.01% thorium and 0.04% S, Table 10, Test AL-16A). The summarized results of this procedure without (AL-16) and with (AL-16A) desliming from a 65-mesh grind of the ore were as follows:

Products of Test. AL-16 & 16A	Wt %	Assays				Distribution %			
		U	Ra	Th	S	U	Ra	Th	S
<u>AL-16</u>									
Py., U. Conc.	43.3	0.134	413	0.30	3.3	94.8	87.5	95.7	98.1
Preconc. Tail	56.7	0.006	45	0.01	0.05				
<u>AL-16A</u>									
Py, U. Conc. & Slimes	46.9	0.111	423	0.25	3.1	96.9	91.6	95.7	98.6
Preconc. Tail	53.1	0.003	34	0.01	0.04				

A procedure where the ore was ground to minus 48 mesh to minimize the formation of slimes, followed by pyrite flotation, slime-deck tabling, desliming and high intensity magnetic separation of the sized pyrite flotation tailing is outlined in Figure 4. As in the previous procedure, the slimes obtained from the pyrite rougher tailing, comprising about 4% of the ore by weight, were added to the final preconcentrate. This procedure resulted in 97% uranium, 92.7% radium, 94% thorium and 96.7% pyrite recoveries in a combined (pyrite and radioactive) concentrate that comprised 49.7% of the ore by weight (Table 14, Test Al-17). The pyrite and uranium concentrates produced had sufficiently higher uranium grades for either sulphuric or hydrochloric acid leaching. The preconcentration tailing produced had very low uranium, radium, thorium and sulphur contents as shown in the following summarized results.

Products of Test AL-17	Wt %	Assays				Distribution %			
		U	Ra	Th	S	U	Ra	Th	S
Py, U. Conc. & Slimes	49.3	0.111	390	0.27	1.80	97.0	92.7	94.1	96.7
Preconc. Tail	50.7	0.003	30	0.016	0.06				

A procedure employing flotation of pyrite followed by flotation of uranium, radium and thorium minerals in combination with tabling and high intensity magnetic separation but without desliming (Figure 5) gave reasonable results (Table 13, Test AL-14C) but less favourable results than the procedures without uranium flotation (given in Figures 1,2,3, & 4, results in Tables 7,8,9, & 12, respectively). A comparison of these results indicated that flotation of radioactive minerals, which would be a relatively expensive operation, was unnecessary since it was not successful in eliminating the high intensity magnetic separation step. The summarized results of the procedures given in Figure 5 were as follows;

Product of Test AL-14C	Wt %	Assays				Distribution %			
		U	Ra	Th	S	U	Ra	Th	S
Py. & U. Conc.	39.6	0.139	595	0.362	2.28	92.4	89.5	92.5	96.9
Preconc. Tail	60.4	0.008	45	0.019	0.05				

The loss of the minus 500-mesh uranothorite fines, and particularly the minus 15 μ m slimes formed during grinding, was a major factor in limiting the uranium recovery to 95%. Because most of the uranothorite slimes remained in the pyrite rougher tailing, desliming of the flotation tailing was conducted prior to or after the high intensity magnetic separation step. The addition of these untreated slimes to the preconcentrate increased the overall weight of the preconcentrate. This was particularly pronounced when the ore was ground to minus 65 mesh and fines comprised about 11% of the ore by

weight. It should be noted that these were carefully controlled laboratory grinds. The grinding on the pilot plant or operating scale might produce more slimes than the laboratory grinding. It is interesting to note that the procedure given in Figure 4 for a 48-mesh grind of the ore produced a preconcentration tailing with a sulphur content of 0.06% S, indicating that although the 48-mesh grind is considered coarse for flotation, it appears that pyrite and the radioactive minerals were sufficiently liberated to be amenable for flotation or other concentration methods. The 48 mesh grind gave very low pyrite and acceptably low contents of the radioactive minerals in the preconcentration tailing. It is believed that, in both cases, this is due to the fact that pyrite and uranium, radium and thorium minerals are softer than most of the non-uranium-bearing minerals of the ore and, as a result, become ground fine enough to respond to separation techniques.

CONCLUSIONS

The procedures developed for the preconcentration of a low-grade uranium ore (0.057% U) with uranothorite as a uranium-bearing mineral, applying combinations of flotation, gravity concentration and high intensity magnetic separation, described in Figures 1, 2 and 4 with the respective results in Tables 9, 10 and 14, resulted in the recovery of:

96-98% of pyrite
95-97% of uranium
93-94% of radium
93-95% of thorium

The combined concentrates (pyrite and radioactive) comprised between 32% and 50% of the ore by weight depending on the procedure applied.

The preconcentration tailing produced from this ore comprised between 50% and 68% of the ore by weight depending on the preconcentration method used, and assayed:

0.04 - 0.06% pyrite
0.003- 0.004% uranium
24 - 34 pCi/g radium-226
0.01 - 0.015% thorium

Based on its physical composition, the preconcentration tailing seems suitable for mine backfill.

ACKNOWLEDGEMENTS

Acknowledgements for contributions to this investigation are extended to the Chemical Laboratory of CANMET, namely, L. Dalton and J. Hole and their technical groups, and J. Leung of the Ore Processing Laboratory who conducted large numbers of analyses. Acknowledgements are also extended to the CANMET Draftsmen for the illustrations and to L.E. McCulloch of the ore Processing Laboratory for the computerized calculations of the results of this investigation.

REFERENCES

1. Corrans, I.J. and Levin, J. "Wet high-intensity magnetic separation for the concentration of witwatersrand gold-uranium ores and residues"; Journal of South African Institute of Mining and Metallurgy; March, 1979.
2. "Battelle explores magnetic process for low-grade uranium ore"; Engineering and Mining Journal; August, 1979.
3. James, H.E. "Preconcentration of very low-grade ores to recover additional uranium"; South African Uranium Extraction Research and Development Activities. Abstract published by International Working Group on Uranium Extraction; Paris; June 11-13, 1979.
4. Raicevic, D. "Decontamination of Elliot Lake uranium tailings"; CANMET, Dept. of Energy, Mines and Resources; CIM Bulletin; August, 1979.
5. Haque, K.E. "Hydrochloric acid leaching of an Elliot Lake uranium ore"; CANMET, Dept. of Energy, Mines and Resources; Lab Report ERP/MSL 79-59(OP); 1979.
6. Seeley, F.G. "Problems in separation of radium from uranium tailings"; Oak Ridge National Laboratory, Oak Ridge, Tennessee. Published in Hydrometallurgy 2, by Elsevier Scientific Publishing Company; Amsterdam, The Netherlands.
7. Skeaff, J.M. "Chlorination of uranium ore for extraction of uranium, thorium and radium and for pyrite removal"; CANMET, Dept. of Energy, Mines and Resources; CIM Bulletin; August, 1979.
8. Campbell, A.D. "Vegetation as a means of stabilizing mine tailings, Eldorado Nuclear Limited, Eldorado, Saskatchewan, Proceedings of the Meeting of Canadian Uranium Producer's Metallurgical Committee"; Toronto; May 16-17, 1978.
9. Hughson, M.R. and Kaiman, S. "Mineralogical report on drill-core samples from the Agnew Lake property of Kerr-Addison Mines Ltd."; Mines Branch Investigation Report IR 67-41; 1967.
10. Laguitton, D. "I - Metcal, metallurgical calculations from laboratory and pilot plant investigation results" Dept. of Energy, Mines and Resources; Lab Report MRP/MSL 79-120(IR); 1979.
11. Raicevic, D. "Upgrading of a low-grade uranium ore from Agnew Lake Mines Limited"; Mines Branch Investigation Report IR 70-8; 1970.