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September 1, 1977

Minerals Exploration Company,  
P.O. Box 50324,  
Tucson, Arizona 85703,  
U.S.A.

Attention: Mr. J.A. Abramo

Dear Sir:

Re: Anderson Uranium Mill


Pursuant to your request of July 20, 1977, we submit herewith our report "Metallurgical Summary and Preliminary Design Data for the Anderson Uranium Deposit".

This document provides the basis for the processing aspects of a project feasibility study which is to be prepared by others. Sufficient test work has not been completed to permit development of all the information needed for detailed engineering design. Where information has not been available from the laboratory studies, design parameters have been based on general experience with processing of other uranium deposits.

This report has been issued for use by Minerals Exploration Company, Union Oil Company and the engineering-contractor assigned to the project feasibility study. We request that it not be distributed to any other companies or individuals.

We appreciate this opportunity to be of service.

Yours very truly,

  
K.R. Coyne, P.Eng.

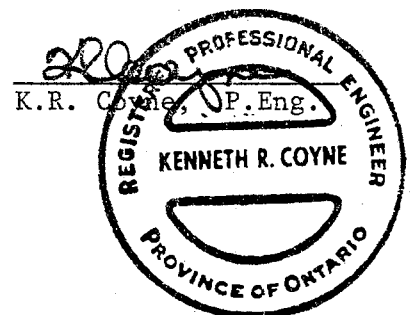
Distribution - 7 copies to J.A. Abramo  
                  - 3 copies to F.H. Buchella, Jr.

KRC/ec  
Encl.

Minerals Exploration Company

Metallurgical Summary and Preliminary Design  
Data for the Anderson Uranium Deposit

*M.E. Grimes*  
M.E. Grimes, P.Eng.



September, 1977  
Toronto, Ontario, Canada

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## 1. INTRODUCTION

### 1.1 Preface

A.H. Ross & Associates have been engaged since March, 1977 as metallurgical consultants to Minerals Exploration Company relative to recovery of uranium from the company's Anderson deposit in Arizona. The following activities have been undertaken and are summarized in this report:

Direction, interpretation and evaluation of the test work performed at Hazen Research, Inc., Golden, Colorado.

Selection of a suitable flowsheet.

Review of the test results from the viewpoint of recovery, capital cost, circuit alternatives, and optimization of operating cost.

Development of design data to serve as the basis for the process cost estimates for a project feasibility study.

In October, 1976, Hazen Research, Inc. were authorized to commence the current metallurgical studies on bulk samples from the deposit. The only other known prior investigation was a very limited series of amenability tests by the U.S. Bureau of Mines in late 1956.

The program at Hazen Research, Inc. generated data on alkaline leaching, acid leaching and grinding. The acid leach flowsheet was tested further including preliminary studies of pulp settling rates; a limited cyclic solvent extraction test yielded uranium distribution data and a small quantity of yellow cake was produced.

Sufficient test work has been completed to select a flowsheet and to specify the major process equipment for project feasibility purposes. Additional metallurgical studies are required to develop the information needed for detailed engineering design of the commercial plant. A further comprehensive research program would be required to determine the optimum

economic method for processing this deposit.

## 1.2 Evaluation Bases and Premises

The estimates of metallurgical recovery and reagent consumptions presented in this report have been based on an average mill feed grade of 0.07%  $U_3O_8$ .

For economic studies to select process conditions and evaluate flow-sheet alternatives, a selling price of \$42.00 per pound of  $U_3O_8$  in yellow cake was specified by Minerals Exploration Company.

The evaluations, design data, equipment sizing, etc., have been based on an average mill feed rate of 2,000 tons per day for 365 days per year or an annual processing rate of 730,000 tons.

## 1.3 Flowsheet

A general schematic flowsheet of the selected acid leach process is presented on page 1 - 4.

## 1.4 Reports of Metallurgical Investigations

A copy of the December 5, 1956 report from J.B. Rosenbaum, U.S. Bureau of Mines, Intermountain Experiment Station, Salt Lake City, Utah, Test No. AEC410 was available and provided results of investigations on a sample from Yavapai County, Arizona.

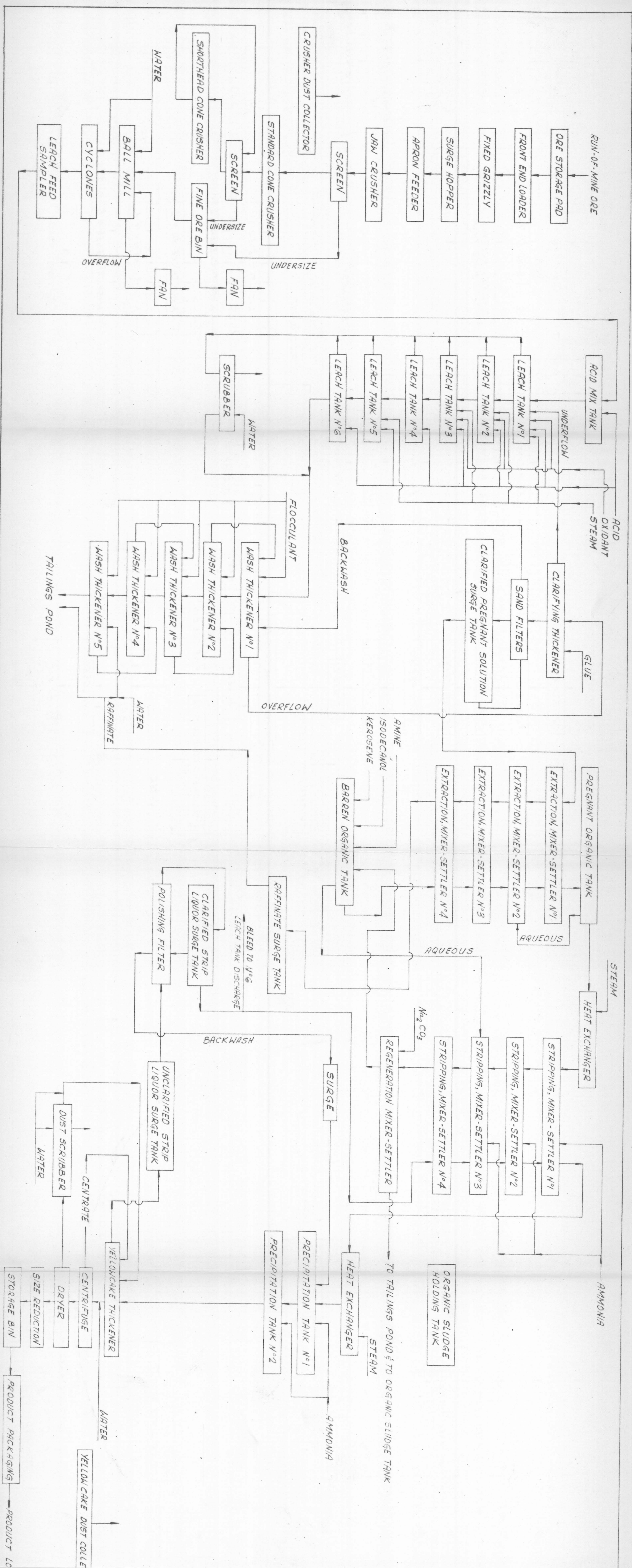
The results of the alkaline leaching investigations at Hazen Research, Inc. were submitted in their report "Alkaline Leaching of Anderson Mine Samples, August 15, 1977".

At the present time, a formal report has not been issued by Hazen Research, Inc. on the acid leaching studies. Data have been made available to A.H. Ross & Associates in the form of inter-office

memoranda, letters, etc. as follows:

Memorandum, R.B. Balderrama to J.E. Litz, December 10, 1976  
Memorandum, R.B. Balderrama to J.E. Litz, January 25, 1977  
Letter , G.C. Dohm, Jr. to J.E. Litz, February 14, 1977  
Memorandum, R.B. Balderrama to J.E. Litz, February 23, 1977  
Letter , J.E. Litz to A.H. Ross , March 3, 1977  
Letter , J.E. Litz to J.A. Abramo, March 30, 1977  
Telex , J.E. Litz to M.E. Grimes, April 19, 1977  
Letter , J.E. Litz to J.A. Abramo, April 25, 1977  
Letter , J.E. Litz to J.A. Abramo, May 3, 1977  
Letter , J.E. Litz to J.A. Abramo, May 27, 1977  
Letter, R.B. Balderrama to M.E. Grimes, June 24, 1977  
Letter , J.E. Litz to J.A. Abramo, July 1, 1977





2. SUMMARY

On the premise that the samples supplied for metallurgical testing are representative, it has been demonstrated that the Anderson uranium deposit is amenable to treatment in an acid leach flowsheet. The quantity of sample available for testing and the schedule specified for completion of the investigations have not allowed development of all the information required for detailed engineering of a processing facility. Similarly, the studies completed to date have not necessarily determined the economically optimum method for processing this deposit. Subject to the above limitations, the design data presented in this report have validity in a general context and are considered suitable for initial project feasibility purposes.

High extraction and recovery of uranium has been demonstrated under relatively severe leaching conditions and with high reagent consumptions. An overall recovery of 88.6 percent from mill feed assaying 0.07%  $U_3O_8$  has been estimated for the selected flowsheet when using the specified process conditions. Uranium losses, as a percent of mill feed, are estimated as follows:

Leaching	10.0%
Counter Current Decantation	0.7%
Solvent Extraction	0.2%
Unaccounted	0.5%
Total	<hr/> 11.4%

Annual production when processing 2,000 tons per day for 365 days per year would be 905,200 pounds  $U_3O_8$

The further metallurgical investigations, which are considered necessary prior to initiation of detailed engineering design of the milling facilities, include the following:

Testing of the variability, if any, of the deposit in respect to grinding, leaching and solid-liquid separation characteristics.

Detailed study of thickening and filtering of the leach pulp to permit selection of the best system and to optimize unit area requirements, soluble losses and flocculant requirements.

Continuous solvent extraction testing to verify that problems will not be encountered upon repeated (at least 10) cycle of solutions.

Additional batch type solvent extraction tests to develop unit settling areas and mixing retention times.

Preparation of sufficient yellow cake to verify that the product will meet all impurity specifications of the refineries.

Semi-autogenous grinding is a potential process alternative; however, consideration of this type of circuit will require development of data by a pilot plant test on a tonnage sample from the deposit.

### 3. REVIEW OF METALLURGICAL INVESTIGATIONS

#### 3.1 Deposit and Mineralogy

The Anderson uranium deposit is located in Yavapai County, Arizona, west of Wickenburg. Open pit mining is planned.

The deposit consists mainly of two ore horizons, referred to as the Middle Zone and the Lower Zone. These zones are separated from each other by a distance of some fifty feet. The zones outcrop in some places on the walls of the mesa under which most of the deposit lies. The host rock consists mostly of shale and siltstone, with both relatively hard and soft layers. The zones are highly variable with respect to both uranium and carbonate content, with a higher carbonate content in the lower zone.

Analyses were performed by Hazen Research, Inc. in April, 1977 on all available drill core intervals in their possession, arising from field drilling programs in 1975 and 1976. A compilation of these analyses is given in Table 1 on the next page. The  $\text{CO}_2$  content varies from 0.03 to 32%  $\text{CO}_2$ , and  $\text{U}_3\text{O}_8$  from 0 to 0.5%, with no apparent correlation between them. Hazen Research, Inc. also described the ore as "a shale containing up to 35%  $\text{CO}_2$  (80% limestone). The average content is from 5 to 10%  $\text{CO}_2$ ".

A report dated December 5, 1956, from the U.S. Bureau of Mines, Salt Lake City on samples from the Anderson deposit described them generally as "uraniferous lakebed material". From a microscopic examination, it was further reported that "the sample consists mainly of altered volcanic glass and montmorillonite clay with quartz, opal, limonite, gypsum, calcite, chalcedony, hematite, muscovite, biotite, and feldspar present. The uranium mineral was identified as tyuyamunite". The major constituents by assay were:  $\text{U}_3\text{O}_8$  0.31%,  $\text{V}_2\text{O}_5$  0.20%,  $\text{CaCO}_3$  2.9%.

Table 1. ANALYSES OF DRILL CORE INTERVALS

<u>Drill Hole Number</u>	<u>Year Drilled</u>	<u>Interval Feet</u>	<u>% U<sub>3</sub>O<sub>8</sub></u>	<u>% CO<sub>2</sub></u>	<u>Used in Composite</u>
7C	1975	19-21	0.181	0.13	
	1975	98-101	NA	0.03	
13C	1975	126-128	0.032	8.8	
	1975	131-138	0.017	5.9	
16C	1975	305-309	0.150	11.4	
17C	1975	131-134	0.023	8.4	
	1975	148-150	0.031	0.31	
	1975	199-205	0.032	0.01	
26C	1975	630-638	0.042	24.9	
	1975	720-724	0.108	0.22	
	1975	737-740	0.026	18.7	
49C	1975	612-621	0.041	9.5	
	1975	631-637	0.032	4.2	
51C	1975	406-408	0.081	0.33	
	1975	439-442	0.036	23	
	1975	445-447	0.093	0.30	
	1975	464-468	0.357	0.12	
79C	1975	45-57	0.028	8.6	
	1975	63-64	0.021	8.2	
113C	1975	299-303	0.034	0.07	
	1975	339-344	0.044	0.1	
119C	1975	32-32	0.076	0.06	
	1975	113-115	0.030	16.7	
	1975	119-124	0.041	25	
	1975	131-132	0.228	22.7	
135C	1975	383-389	0.042	0.15	
	1975	459-473	0.098	8.44	
149C	1975	382-386	0.047	0.36	
	1975	408-411	0.070	0.15	
222C	1976	109-113	0.050	32	No. 1
	1976	116-117	0.032	11	No. 1
229C	1976	123-125	0.063	13	No. 1
	1976	144-150	0.030	24	No. 1
244C	1976	77-79	0.020	0.20	No. 1
254C	1976	123-125	0.014	0.07	No. 1
273C	1976	36-38	0.128	0.03	No. 1
	1976	134-141	0.033	3.2	No. 1
	1976	175-177	0.043	0.04	No. 1
274C	1976	55-57	0.029	0.08	No. 1

Table 1. ANALYSES OF DRILL CORE INTERVALS

<u>Drill Hole Number</u>	<u>Year Drilled</u>	<u>Interval Feet</u>	<u>% U<sub>3</sub>O<sub>8</sub></u>	<u>% CO<sub>2</sub></u>	<u>Used in Composite</u>
275C	1976	47-53	0.101	12.9	No. 2
289C	1976	80-82	0.062	9.0	No. 2
	1976	138-156	0.028	6.7	No. 2
	1976	260-262	0.054	0.31	No. 2
337C	1976	424-427	0.043	15	No. 2
390C	1976	279-281	0.58	0.13	No. 2
412C	1976	129-135	0.241	0.71	No. 2
	1976	141-143	0.056	9.2	No. 2
431C	1976	561-566	0.066	17	No. 2
	1976	577-579	0.029	13	No. 2
	1976	587-589	0.032	4.0	No. 2
184C	1976	473-483	0.046	4.5	No. 3
	1976	505-509	0.043	21	No. 3
	1976	531-533	0.042	4.7	No. 3
	1976	660-662	0.075	0.21	No. 3
	1976	671-673	0.073	5.7	No. 3
434C	1976	529-531	0.029	0.10	No. 3
	1976	536-538	0.025	0.06	No. 3
435C	1976	98-101	0.078	0.04	No. 3
	1976	130-132	0.047	0.03	No. 3
	1976	178-180	0.032	3.6	No. 3
436C	1976	220-223	0.075	0.09	No. 3
	1976	272-274	0.017	2.1	No. 3
444C	1976	343-345	0.335	0.18	No. 3
	1976	365-367	0.026	0.07	No. 3
	1976	412-414	0.028	8.6	No. 3
	1976	423-425	0.039	0.17	No. 3
	1976	563-567	0.045	0.49	No. 3

## 3.2 Ore Samples

### 3.2.1 Bulk Composites

The first samples of ore on which testwork was conducted by Hazen Research, Inc. were received by them in December, 1976. Two bulk samples were received which were intended to be representative of the Middle and Lower ore zones of the deposit. They are reported to have been made up from bulk sampling at or near the surface of outcrops at three locations around the deposit. Chemical analyses of the two composites were as follows:

	<u>Middle Zone</u>	<u>Lower Zone</u>
U <sub>3</sub> O <sub>8</sub> % , by assay	0.039	0.043
U <sub>3</sub> O <sub>8</sub> % , by calculation	0.045	0.050
V <sub>2</sub> O <sub>5</sub> %	0.07	0.14
Mo %	0.001	0.001
CO <sub>2</sub> %	0.75	3.20
Fe %	3.05	3.06

Subsequent testwork on these samples proved them to be amenable to both acid and alkaline leaching, with much lower than average CO<sub>2</sub> content and little extractable organic. They appear to have been quite oxidized, and non-typical of ore to be processed.

### 3.2.2 1975 Drill Cores

Core samples obtained during a 1975 drilling program provided the next set of samples for testing by Hazen Research, Inc.

Composites of intervals were prepared for fourteen drill holes. On intervals where the cores were crushed, 3/4 of the coarse reject and all of the fine reject were used to prepare individual composites. In cases where the cores were split, 1/4 was held in reserve and the remaining core plus rejects were



used for individual composites. Except for hole AM-1G, which was no longer available, analyses of the core intervals referred to herein are shown in Table 1.

The drill hole composites were analyzed as follows:

<u>Hole No.</u>	<u>% U<sub>3</sub>O<sub>8</sub></u>	<u>% CO<sub>2</sub></u>
AM-1C	0.025	13.4
AM-7C	0.047	0.24
AM-13C	0.027	9.2
AM-16C	0.035	9.6
AM-17C	0.041	1.9
AM-18C	0.063	15.5
AM-26C	0.115	14.3
AM-49C	0.075	17.3
AM-51C	0.055	3.3
AM-79C	0.018	12.0
AM-113C	0.045	0.63
AM-119C	0.125	13.3
AM-135C	0.086	5.0
AM-149C	0.055	0.34
Arithmetic mean	0.058	8.3

### 3.2.3 1975 Composite

A composite of five drill holes from the fourteen 1975 holes was made from analytical pulp rejects. This material was nominally minus 100 mesh. The holes used were AM-1C, AM-26C, AM-51C, AM-135C, and AM-149C. Analysis of the composite showed:

U <sub>3</sub> O <sub>8</sub>	0.058%
V <sub>2</sub> O <sub>5</sub>	0.083%
CO <sub>2</sub>	10.0 %



### 3.2.4 1976 Mining Sequence Composites

Following instructions in a letter from Minerals Exploration Company of February 14, 1977, three new composites were made up by Hazen Research, Inc. from new drill cores received on December 1, 1976 and January 17, 1977. The drill cores were obtained by drilling at the property in the period November, 1976 to January, 1977. The three composites No. 1, 2 and 3 were said to be representative respectively of "early", "middle" and "late" mining periods of about equal duration.

The drill cores and intervals used in preparing the three composites are identified in Table 1.

One quarter of each drill core sample was saved and the remaining drill cores and their rejects were used to make the three composites. Each composite was crushed to pass 6 mesh and portions were split-out for head samples. The balance of the samples was ground to give a nominal 28 mesh grind, and provided the feed for subsequent testwork.

The composites were analyzed as follows:

	<u>Composite No. 1</u>	<u>Composite No. 2</u>	<u>Composite No. 3</u>
Chemical, % $U_3O_8$	0.039	0.054	0.058
$V_2O_5$	0.06	0.09	0.11
Mo	0.002	0.007	0.006
$CO_2$	11.4	9.7	5.5

These composites were used for all testwork to June, 1977.

### 3.2.5 June Composite

In May, 1977, a decision was made to employ acid leaching for further testwork and to conduct cyclic tests to obtain essential

flowsheet information. To undertake this work, it was decided at a meeting at Hazen Research, Inc. on June 2, 1977, to prepare a new composite from all drill core material on hand, corresponding to the "middle" and "late" mining periods, which had proven more refractory to processing than the "early" period sample. Additionally, the new composite was to be made up by judicious selection of core intervals to approximate, if possible, the predicted long term mill head grade of 0.071%  $U_3O_8$ .

The resulting composite was used for the cyclic test program in June, 1977. It analyzed as follows:

$U_3O_8$	0.058%
$V_2O_5$	0.107%
Mo	0.007%
$CO_2$	7.54 %

It had not been possible to obtain the 0.071% grade without rejecting too much of the remaining material.

### 3.3 Process Selection

#### 3.3.1 Alkaline Leaching Process

An extensive alkaline leaching program was undertaken. On the original bulk composites and on the 1975 drill hole composites, a similar number of both alkaline and acid leach tests were conducted, with the aim of choosing one of these process routes for the balance of the investigational program. These initial tests indicated a slightly higher uranium extraction with acid leaching, but an apparent significant reduction in reagent costs with alkaline leaching. Consequently, an interim decision was made to concentrate efforts on the alkaline process.

Most of the additional alkaline leaching testing was done on the three 1976 hole composites, representing early, middle and later mining periods. From this work, it was shown that a uranium extraction of about 89% could be obtained under the following conditions:

Grind	minus 65 mesh
Retention time	16 hours
Temperature	80°C
Oxidant	Oxygen
Sodium carbonate concentration	160 - 200 lb/ton
Sodium bicarbonate concentration	40 lb/ton
Equivalent sodium carbonate consumption	About 60 lb/ton
Cupric sulfate	2 lb/ton
Ammonia	2 lb/ton

Although the leaching conditions noted would give lower reagent costs than an acid leach circuit, a major problem became apparent because of the presence of substantial quantities of soluble organic material in the leach liquor. Filtration was rendered very difficult and the resultant leach liquors were black in

colour. Precipitation of uranium with sodium hydroxide appeared also to be adversely affected. Because of the economic necessity of closed circuit recycle of solutions, the potential build-up of organics under operating conditions would pose a serious problem. In order to remove the organics, tests were made of surfactants and flocculants, of additions of magnesia, of activated carbon adsorption, of flotation, and of roasting of the ore. The primary measurement of effectiveness was light transmittance of the treated solution after clarification.

Of the methods tested, only adsorption on activated carbon and roasting of the ore were effective at possibly acceptable costs. At least 100 gpl of carbon was needed (400 lb/ton of ore) to give a satisfactory removal but, in cyclic tests, removal decreased after two cycles, indicating that either a once-through system would be needed or calcining-regeneration of spent carbon. Regeneration by calcining would require treatment of a volume of carbon similar to that of the whole ore.

Roasting of whole ore at 350°C or higher was effective in eliminating the alkaline soluble organics, but a reduction in uranium extraction in subsequent leaching could not be avoided. At the lowest temperature, 350°C, the extraction was 90.7% compared to 92.6% without roasting under otherwise identical leaching conditions, in the case of a composite from 1975 drill cores. Later tests on 1976 composite No. 3 gave a much greater reduction at 350°C, from about 88% on unroasted ore to 82% on roasted ore. From a process cost viewpoint, roasting of the whole ore would be more economical than the cost of carbon adsorption plus roasting of the carbon.

### 3.3.2 Acid Leaching Process

The near optimum conditions for an acid leach process, as developed or reported in subsequent sections of this report, are:

Grind	Minus 28 mesh
Retention time	10 hours
Temperature	80°C
Sulfuric acid	500 lb/ton
Sodium chlorate	12 lb/ton
Solvent loss - amine	0.132 lb/ton
- isodecanol	0.198 lb/ton
- kerosene	0.275 gal/ton
Soda ash	0.273 lb/ton

### 3.3.3 Organics Removal in Acid Leaching

In discussing the effect of organics on oxidant consumption in Section 3.4, the conditions used to remove organics were given. In essence, ore is floated, the flotation concentrate is roasted, the calcine is added to the flotation tails, and acid leaching is conducted on the mixture. Although a reduction of some two-thirds was achieved in chlorate consumption, a decrease in uranium extraction of 1 to 2% was indicated.

There was no apparent effect on solvent extraction when the samples were leached without prior removal of organics, although the possibility must be recognized that a problem could develop in sustained operation. Consequently, flotation has not been included in the selected flowsheet. An organic regeneration step has been provided in the solvent extraction circuit as a precautionary measure.

3.3.4 Comparative Costs of Acid and Alkaline Leaching

In Section 3.4, the methods of estimating costs of process variables are discussed in relation to acid leaching. The same methods are applicable also to alkaline leaching. The significant addition in the case of alkaline leaching is inclusion of a cost for roasting of the whole ore. The estimates include a capital cost component plus operating costs. The tabulation below presents a comparison of the two circuits. Only those cost components that are expected to differ between the two circuits are included.

	<u>Acid Leaching</u>		<u>Alkaline Leaching</u>	
	<u>Condition</u>	<u>Cost \$/ton</u>	<u>Condition</u>	<u>Cost \$/ton</u>
Tons Ore Per Day	2,200		2,200	
U <sub>3</sub> O <sub>8</sub> - assay %	0.07		0.07	
- % extraction	90		88	
- lb/day	2,810		2,750	
Grinding	minus 28 mesh	0.52	minus 65 mesh	0.75
Retention time	10 hours	0.10	16 hours	0.16
Temperature	80°C	* 0.32	80°C	1.12
Sulfuric acid	500 lb/ton	6.25	-	-
Sodium chlorate	12 lb/ton	2.34	-	-
Oxygen	-	-	-	0.50
Sodium carbonate	0.273 lb/ton	0.02	60 lb/ton	3.00
Cupric sulfate	-	-	2 lb/ton	0.50
Ammonia	0.75 lb/ton	0.07	2 lb/ton	0.18
Solvent loss		0.28		-
Roasting of ore			2 hours, 350°C	4.30
		9.90		10.51
Value of Uranium loss		5.88		7.06
Total		15.78		17.57

\* Heat of reaction reduces sensible heat requirements in this case.

The direct operating comparison shows a small difference in favour of acid leaching. The value of uranium loss, at the present high uranium selling prices, makes the 2% higher recovery a major factor in favour of the acid process. Capital cost for an alkaline leach plant is usually about 10% greater than for an acid leach plant without the addition of roasting. Inclusion of the latter would add significant extra capital in the alkaline case.

### 3.3.5 Conclusions

The principal differences between the acid and alkaline circuits may be summarized as follows:

The cost comparisons favour the acid leach process.

There is a greater effect of extracted organics in the alkaline circuit as compared to the acid circuit, and a greater potential for build-up in recycle solutions.

About two percent greater uranium extraction is indicated for the acid circuit.

It is possible that the above conclusions would be modified as the result of a more extensive research program including studies of preconcentration, dual acid and alkaline circuits, other reagents for organics removal, and further optimizing of the factors so far studied. However, based on the data now available, it is concluded that the acid leach process should be selected at this time.

### 3.4 Metallurgical Test Work - Selected Flowsheet

#### 3.4.1 Grinding

The early stages of testwork had indicated that alkaline leaching at a minus 65 mesh grind seemed a preferred process; hence, the only grindability test was directed to this process. A Bond Work Index number of 12.8 was obtained in grinding nominal minus 6 mesh material to nominal minus 65 mesh.

Screen analyses of the 1976 hole composites at a nominal minus 28 mesh grind are given below:

<u>Mesh Size</u>	<u>Composite No. 1 % Passing</u>	<u>Composite No. 2 % Passing</u>	<u>Composite No. 3 % Passing</u>
28	100	100	100
35	89.4	94.3	93.8
48	74.2	85.9	80.9
65	63.7	76.5	71.6
100	54.1	68.0	62.7
150	46.0	60.4	54.3
200	37.6	52.4	45.3
270	32.5	46.3	39.6
400	27.0	38.3	32.1

#### 3.4.2 Leaching

##### 3.4.2.1 Relevant Data

The leaching test work by Hazen Research, Inc. on the Anderson deposit includes 100 acid leach tests. Some 38 of the acid leach tests were on the initial bulk composite samples. Uranium extractions greater than 95% were obtained under moderate leaching conditions. However, all subsequent tests on drill cores have shown that the bulk composites were far more amenable to leaching than drill core material, and also



substantially different in composition. In particular, it is postulated that the bulk composites were more highly oxidized than drill core material, to an extent that would not occur during production operations. Because of these factors, the acid leach work on the bulk composites has not been included in this review.

The remaining test data which are deemed to be valid in assessing acid leaching conditions are found in the 62 acid leach tests on drill cores obtained in 1975 and 1976.

#### 3.4.2.2 Amenability Tests on 1975 Hole Composites

The 14 drill hole composites prepared from 1975 drill cores were each leached under a fixed set of conditions previously found to be satisfactory in leaching of the bulk composites. The leaching conditions were: minus 28 mesh grind, 33% solids, 60°C, 16 hours, with  $H_2SO_4$  added to control pH at 1.0, and sodium chlorate to provide greater than minus 500 mv emf for the first 4 hours. The results are given in Table 2 in order of increasing  $CO_2$  content of the heads.

The wide variability of composition and of response to leaching between the different hole composites is apparent. The high average acid consumption and rather poor uranium extraction compared to most U.S. uranium milling experience is also evident. A significant correlation between acid requirement and  $CO_2$  content of the ore is shown by the data, as expected. Calculations show no correlation between uranium content and  $CO_2$  content of the samples. Acid consumption figures are calculated from solution assays and material balances. Even so, they correlate reasonably well with acid addition and indicate that, on the average, about

Table 2. COMPARATIVE ACID LEACHING OF 1975 HOLE COMPOSITES

Hole Number	Head		Chlorate Addition lb/ton	Acid Addition lb/ton	Acid Consumed lb/ton	Residue % U <sub>3</sub> O <sub>8</sub>	Reported % Extraction
	% U <sub>3</sub> O <sub>8</sub>	% CO <sub>2</sub>					
AM - 7C	0.047	0.2	3.0	170	106	0.003	95.0
- 149C	0.055	0.3	11.3	60	40	0.013	77.3
- 113C	0.045	0.6	2.7	161	96	0.005	89.6
- 17C	0.041	1.9	0.7	151	124	0.001	98.0
- 51C	0.055	3.3	3.3	305	241	0.012	80.0
- 135C	0.086	5.0	12.0	247	229	0.015	82.4
- 13C	0.027	9.2	1.6	536	470	0.002	94.1
- 16C	0.035	9.6	4.2	597	557	0.006	84.8
- 79C	0.018	12.0	4.8	837	783	0.003	82.9
- 119C	0.125	13.3	5.4	743	686	0.022	80.8
- 1C	0.025	13.4	2.7	807	723	0.004	85.3
- 26C	0.115	14.3	4.0	801	778	0.027	75.0
- 18C	0.063	15.5	5.7	776	743	0.017	72.7
- 49C	0.075	17.3	6.0	724	717	0.026	66.2
Arithmetic Mean	0.058	8.3	4.8	494	450	0.011	83.2

90% of the acid added is consumed, or conversely that little, if any, excess free acid is required. Chlorate requirements do not appear to correlate with other variables.

The leaching conditions gave satisfactory uranium extraction on some holes but poor results on others. The reason for the different response was not evident. It was decided to conduct further test work on composite samples to average some of the variability.

#### 3.4.2.3 Amenability Tests on 1976 Composites

During the program undertaken in April, 1977, acid leach amenability tests were conducted on the three 1976 composites. The test conditions were as follows: minus 28 mesh, 50% solids, 60°C, 6 hours,  $H_2SO_4$  addition to control pH at 1.0, and sodium chlorate to provide greater than minus 500 mv emf for the first 4 hours. These conditions differ from the amenability tests on the 1975 cores principally in leaching for 6 hours rather than 16, and with 50% solids rather than 33. Results of the tests are given in Table 3.

The average uranium extraction in these tests was substantially higher than the average on the 1975 hole composites, in spite of a shorter leach period. Acid addition and consumption were quite similar to those for the 1975 holes for the same  $CO_2$  content. A major difference was the much larger chlorate addition in these latter tests. It was postulated by Hazen that higher organics content in the ore in the 1976 cores was primarily responsible for the increased chlorate requirement. It is possible also that there may have been a difference in experimental technique. It appears that the higher chlorate addition was primarily responsible for the higher uranium extractions.

#### 3.4.2.4 Flotation-Calcine-Leach of 1976 Composites

As discussed in Section 3.3, removal of organics by flotation and calcination of the flotation concentrate, followed by leaching of the combined calcine and flotation tail has been tested, but has not been included in the selected flowsheet. Nonetheless, the leaching data arising from the three tests are of interest in interpreting the leaching behaviour of

Table 3. ACID LEACHING OF 1976 COMPOSITES

<u>Composite</u>	<u>Direct Leach</u>			<u>Flotation - Calcine - Leach</u>		
	<u>No. 1</u>	<u>No. 2</u>	<u>No. 3</u>	<u>No. 1</u>	<u>No. 2</u>	<u>No. 3</u>
<u>Head Assay</u>						
% U <sub>3</sub> O <sub>8</sub>	0.039	0.054	0.058	0.039	0.054	0.058
% CO <sub>2</sub>	11.4	9.7	5.5	11.4	9.7	5.5
<u>Flotation Concentrate</u>						
% by Wt.						
% U <sub>3</sub> O <sub>8</sub>				7.4	10.3	8.3
% U <sub>3</sub> O <sub>8</sub> Distribution				0.033	0.019	0.039
				7.1	3.6	6.5
						5.7
<u>Leaching</u>						
Chlorate Addition lb/t	13.0	23.0	27.0	4.8	7.9	8.2
Acid Addition lb/t	633	667	287	693	630	504
Acid Consumed lb/t	585	562	245	618	564	394
Free Acid gpl	19.5	35.2	3.7	12.8	11.4	22.6
Residue % U <sub>3</sub> O <sub>8</sub>	0.004	0.004	0.008	0.004	0.005	0.006
% U <sub>3</sub> O <sub>8</sub> Extraction	88.9	92.5	86.8	90.2	82.5	87.3
						86.7

the material, and so have been included herein. Each of the three composites were treated in the same way.

Flotation was performed on the minus 65 mesh rejects from the Bond Work Index study, using kerosene, Dowfroth 250, and acid to give a pulp of pH 1.5. The flotation concentrate was calcined for two hours at 500°C and then added to the flotation tail for acid leach tests. The results are given in Table 3.

The most noticeable difference between these tests and the straight acid leach tests was a reduction of chlorate requirements by two-thirds. It is believed that calcination of the flotation concentrate provided most of the oxidation required. Other evidence, such as leach solution colour, leads to the conclusion that calcination oxidizes organic material which otherwise consumes chlorate. An average increase of 60 - 80 lb/ton in acid requirement on the calcined material is also evident. Residue assays are not significantly different between the tests with and without flotation.

It appears from these tests and the earlier work that chlorate requirements are dictated primarily by organics content of the ore.

#### 3.4.2.5 Leaching of the June Composite

A total of 43 leaching tests were conducted on this composite, giving rise to pulps and liquors for other test work. The data from these tests has been arranged in various matrices to permit comparisons so far as possible between tests under different parameters, e.g., chlorate addition, and also to compare to data from other samples. Having identified the desirable

factor levels for satisfactory uranium extraction, cost factors were applied to determine the apparent near optimum conditions.

Based on a minimum acceptable uranium extraction of 90%, the following conclusions were drawn with respect to the leaching parameters for the June composite. The minimum chlorate addition should be 12 lb/ton. A leaching time of 8 hours at 80°C is acceptable, so long as acid and chlorate additions are not both at minimum levels. A temperature of 60°C would not be recommended. Acid requirement appears to some extent to be dependent on the level of other factors. Higher temperature and longer time both increase acid consumption, but the data does not establish if uranium extraction is necessarily increased also. There appears to be no advantage to excess free acid, hence, a level of about 15 gpl free acid, or 30 lb/ton of ore at 50% solids seems suitable in final leach liquor. On this basis, acid addition of 470 to 500 lb/ton is acceptable.

Eight tests were selected which met the required conditions, and relative cost comparison for the tests are summarized in Table 4 on the next page.

Only those parameters which differ between tests are included in the cost calculations. The unit costs are based on the following:

Leaching Time - an estimate of capital cost based on rubber lined steel tanks with agitators at 3½ year write-off.

Leaching Temp.- both sensible heat requirement and heat loss from convection and vaporization are included and cost based on steam at \$3.50/1,000 lb.

Acid Cost - \$25.00/ton

Sodium Chlorate - 19.5¢/lb

Residue Value -  $U_3O_8$  content at \$42/lb  $U_3O_8$ Table 4 - Comparative Costs for Selected Tests

HRI Test No.	Temp. °C	Time Hr.	Acid Addition lb/ton	Chlorate Addition lb/ton	Production Cost \$/ton	Residue Value \$/ton	Total Cost \$/ton	Residue % $U_3O_8$
-120	80	8	500	12	9.72	4.96	14.67	0.0059
-121	80	8	500	20	11.27	4.79	16.06	0.0057
-118	80	8	530	16	10.87	4.45	15.32	0.0053
-119	80	8	530	16	10.87	4.79	15.66	0.0057
-124	80	12	470	12	9.38	4.70	14.08	0.0056
-122	80	12	470	16	10.16	4.70	14.86	0.0056
-123	80	12	470	16	10.16	4.28	14.44	0.0051
-125	80	12	500	12	9.76	4.54	14.29	0.0054

It was concluded from consideration of the comparison in Table 4 and the probable effect of deviations caused by ore changes and unit cost changes, that the preferred choice of conditions for design would be 500 lb/ton acid, 12 lb/ton chlorate, 80°C, for 10 hours. The time is an average from Table 4.

It should be stressed that the quantities cited above are for the average composition of the ore as represented by the June composite. Judging by the limited data comparisons for acid leaching of the No. 1, 2 and 3 composites of 1976 drill cores, a substantially higher acid consumption can be expected from processing of the earlier mined ore. Allowing for some blending, it would still appear that about 650 lb/ton of acid may be required.



### 3.4.3 Acid Pulp Thickening

#### 3.4.3.1 Test Data

Hazen Research, Inc. reported their interpretation of thirteen settling tests in graduate cylinders on pulp samples from various leach tests. All leach tests were on samples at the nominal minus 28 mesh grind with varying temperatures, retention time, acid and oxidant additions. The flocculants included Nalco 6WM-867, Nalco 5WM-537, Superfloc 16, Polyhall 1080, Jaguar MDD and Dow MG-200. The data indicates that Dow MG-200 yielded the best results which are shown below. The Hazen interpretation is reported to be based on the Kynch Method of unit area calculation without any scale-up factor.

<u>Hazen Test No.</u>	<u>Leaching Conditions</u>			<u>Settling Results</u>	
	<u>Time Hours</u>	<u>Temp. °C</u>	<u>Acid lb/ton</u>	<u>MG-200 lb/ton</u>	<u>Unit Area ft<sup>2</sup>/ton/day</u>
1009-137	3	135	500	0.16	4.5
1009-147	8	80	500	0.15	7.0
1009-148	8	80	500	0.18	5.3
1009-149	8	80	500	0.20	6.2

Based on the above limited data and applying a suitable scale-up factor, a unit thickener area of 8.5 sq ft/ton/day is suggested for preliminary design purposes. Flocculant addition rate is estimated at 0.20 lb/ton in No. 1 CCD thickener with a total consumption in a 5 stage CCD circuit of 0.43 lb/ton.

The pulp settling information generated to date on samples from the Anderson deposit is not suitable for detailed engineering design purposes. A comprehensive series of tests on representative pulp samples leached under near optimum conditions will be required to

provide the data for selection of the most economic CCD thickening circuit. It is recommended that this type of test work be performed by representatives of equipment suppliers, such as the Eimco Division of Envirotech Corporation.

#### 3.4.3.2 CCD Thickener Stages

Based on the above data, a 155 foot diameter thickener would be suitable for a mill design rate of 2,200 tons per day. The average underflow density is estimated at 53 percent solids. The total direct and indirect cost for a 155 foot thickener, including pumps and controls, is estimated at \$1,723,000. Based on a pregnant solution to ore ratio of 3 to 1, the value of the soluble uranium loss with a varying number of stages of CCD has been estimated as follows:

<u>Stages</u>	<u>U<sub>3</sub>O<sub>8</sub> Loss of Soluble Feed</u>	<u>Annual Value of Soluble Loss</u>
6	0.45%	\$176,000
5	1.04%	\$402,000
4	2.41%	\$931,000

Excluding maintenance and additional flocculant costs, the simple payback of additional capital based on value of uranium loss is estimated as follows:

Payback on 5 versus 4 stages:

$$\frac{\$1,723,000}{\$931,000 - \$402,000} = 3.3 \text{ years}$$

Payback on 6 versus 5 stages:

$$\frac{\$1,723,000}{\$402,000 - \$176,000} = 7.6 \text{ years}$$

Based on the above, a 6 stage CCD cannot be justified and a 5 stage circuit is marginally economic on the basis of a 3 to 1 pregnant solution to ore ratio. A 4 stage CCD would be indicated if a higher wash ratio is used. However, during the initial one to two years of plant operation, it is not unusual for one or more thickeners to be out of service for extended periods with a resulting significant increase in soluble loss. Therefore, it is considered prudent to use a 5 stage CCD thickener circuit.

#### 3.4.3.3 Pregnant to Ore Ratio

The soluble loss from a 5 stage CCD system with varying amounts of wash, equivalent to pregnant to ore ratios in the range of 3.0 to 3.75, have been calculated. The capital cost of the clarification-solvent extraction circuit corresponding to the changes in flow rates with varying pregnant to ore ratios has been estimated. The results of the economic analysis are shown below including the simple payback in years on additional capital.

Change in Preg. to Ore Ratio	Reduction in Loss of Soluble $U_3O_8$ in Feed	Annual Value of Change in Soluble Loss	Capital Cost Increase	Payback in Years
3.00 to 3.25	0.21%	\$81,000	\$153,000	1.9
3.25 to 3.50	0.15%	\$58,000	\$147,000	2.5
3.50 to 3.75	0.12%	\$46,000	\$144,000	3.1

From the above, it is concluded that the near optimum pregnant solution to ore ratio would be 3.75 to 1.

#### 3.4.4 Solvent Extraction

Because of the large number of leaching tests undertaken in examining both acid and alkaline flowsheets, there remained a limited supply of sample to produce liquors from the cyclic solvent extraction tests which concluded the test program. In fact, continuous tests were carried out on four days for 2 to 3 hours each, processing 12-18 litres of aqueous feed in each test. The leach liquors were prepared from leach pulps produced during the parallel leaching test program.

Four stages of extraction and four stages of stripping were employed. The solvent was 3% Alamine 336 plus 2% isodecanol, with kerosene as diluent. Ammonium sulfate plus ammonia was used for solvent stripping.

The feed liquors to solvent extraction were in the range of 0.161 to 0.174 gpl  $U_3O_8$ . Average raffinate  $U_3O_8$  of 0.0002 gpl was experienced in the last two cycles, when pregnant organic contained 1.79 and 2.08 gpl  $U_3O_8$ , and stripped organic 0.045 and 0.03 gpl  $U_3O_8$ . Favourable extraction equilibrium was indicated by the close approach of the second stage aqueous discharge to the final raffinate concentration. However, if the organic loading was well below its saturation value, this would also enhance the apparent equilibrium.

The organic loadings reached in these continuous tests are substantially lower than usually experienced with mill pregnant liquors and 3% amine solvent. Usually an operating level of about 2.5 gpl  $U_3O_8$ , with a saturation value of 3.0 gpl is experienced. Also, the favourable extraction equilibrium suggests that higher loading should be possible. On the other hand, the present leach liquors were considerably lower in uranium concentration than the usual mill pregnant liquor, which would tend to lower the saturation level to some extent. Additionally, saturation tests were not reported, and uranium balances were not good on the tests. Because of these uncertainties, a solvent loading of greater than 2.0 gpl  $U_3O_8$  for a 3% amine solvent does not appear to be justified.

An accidental entry of excess ammonia during the first stripping cycle caused uranium precipitation in the stripping circuit so that equilibrium was not obtained during the limited test period.

No evidence was observed of unusual emulsions or crud in the extraction circuit. This is also true with respect to the stripping circuit but, in this case, the presence of uranium precipitate could have obscured other evidence.

Required mixing time for extraction only was determined. Uranium transfer from aqueous feed to organic was measured in four batch tests and 99.5% of equilibrium was obtained in 15 seconds.

Phase separation rate was measured for extraction only in dynamic tests with dispersion band widths at  $\frac{1}{2}$  inch intervals from  $\frac{1}{2}$  to 3 inches, with organic to aqueous flow ratio of 1.6/1. The required settler area, excluding any scale-up factor, was calculated at  $0.60 \text{ ft}^2/\text{gpm}$  of total flow at  $\frac{1}{2}$  inch dispersion width to  $0.34 \text{ ft}^2$  at 3 inch dispersion width.

At least ten cycles in the continuous solvent extraction test unit would be required to confirm the test data and to detect any slow building impurity or emulsion problem. The data obtained to date is only adequate for general flowsheet selection.

#### 3.4.5 Precipitation

Uranium precipitation was performed only on the strip liquors from the last two cycles of the continuous solvent extraction tests. Because of limited quantities of feed, only a few grams of  $\text{U}_3\text{O}_8$  were available in strip liquor. Precipitation was done at pH 7 with ammonia gas. The following analyses were reported on oven-dried product:

$\text{U}_3\text{O}_8$	81.4 %
$\text{V}_2\text{O}_5$	0.16 %
$\text{P}_2\text{O}_5$	0.025%
Mo	0.059%

The product meets the concentrate feed specifications for the Allied Chemical Company, Kerr-McGee Corporation and Eldorado Nuclear Limited refineries for the elements analyzed.

3.5 Estimate of Reagent ConsumptionsPer day

Based on 2,200 TPD of ore

Sulfuric acid (100% basis)

550 tons

Sodium chlorate

26,400 pounds

Flocculant (Dow MG-200)

946 pounds

Sodium carbonate

600 pounds

Glue

220 pounds

Amine

290 pounds

Isodecanol or Tridecanol

435 pounds

Ammonia - ore grade 0.07%  $U_3O_8$ 

1,647 pounds

- ore grade 0.10%  $U_3O_8$ 

2,352 pounds

Kerosene

605 gallons

#### 4. DESIGN DATA

##### 4.1 Millsite Conditions

Site	Owner's property in Yavapai County, Arizona.
Elevation	Approximately 1,975 feet above sea level.
Temperature - mill water	65° - 70°F
- ore	20° - 90°F

##### 4.2 Ore Characteristics

Specific gravity	2.45
Bulk density of crushed wet ore	
- compacted	100 lb/cu ft
- loose	80 lb/cu ft
Moisture (mine run) - assumed	10% maximum
Angle of repose of crushed ore	35 degrees

## Ore composition (dry basis)

Typical Analysis of Composite Samples

	<u>No. 1</u>	<u>No. 2</u>	<u>No. 3</u>
<u>Chemical:</u>			
U <sub>3</sub> O <sub>8</sub> - %	0.039	0.054	0.058
V <sub>2</sub> O <sub>5</sub> - %	0.06	0.09	0.11
CO <sub>2</sub> - %	11.4	9.7	5.5
Mo - %	0.002	0.007	0.006

X-Ray Fluorescence:

Cu - %	0.014	0.021	0.014
Zn - %	0.020	0.026	0.022
As - %	0.001	0.011	0.012
Fe - %	1.4	1.7	1.7
Ni - %	0.040	0.086	0.014
Ru - %	0.011	0.014	0.021
Ba - %	0.072	0.077	0.099
Sr - %	0.060	0.120	0.110
Ti - %	0.110	0.066	0.130
Zr - %	0.024	0.036	0.034
Mn - %	0.051	0.025	0.044
Y - %	0.006	0.006	0.006
Pb - %	-	0.016	0.010
Cr - %	-	0.007	0.044

4.3 Production Criteria

Operating days per year 365

Plant feed rate (dry basis)

Average	2,000 tpd
Design	2,200 tpd
Average annual rate	730,000 tpy



## Ore grade

Average	0.07% $U_3O_8$
Design	0.10% $U_3O_8$

Overall recovery 88.6 %

## Uranium losses

Ore leaching	10.0 %
Soluble loss - CCD	0.7 %
- Solvent Extraction	0.2 %
Unaccounted loss	0.5 %

 $U_3O_8$  production

Pounds per operating day

Average 2,480

Design 3,898

Pounds per year (average) 905,200

4.4 Ore Receiving and Crushing

## Conditions:

Ore pad	Gravel surfaced area for ore storage in 30 foot high piles
Feeding method	Front end loader from stockpile
Crushing circuit	Three stage with circuit closure on third stage
Crushing rate	198 dry tons per hour
Operating schedule	6 days per week 2 shifts per day 6.5 hours per shift
Maximum ore particle size	30 inch
Product size	all minus $\frac{1}{2}$ inch
Receiving hopper grizzly	24 in x 30 in grid openings
Primary crusher	48 in x 60 in jaw
Secondary crusher	5 ft diameter standard cone
Tertiary crusher	5 ft diameter shorthead cone
Coarse ore screen	Double deck with $\frac{1}{2}$ in openings on bottom deck
Fine ore screens	Rod deck type with $\frac{1}{2}$ in openings

General:

The cone crushers must be suitably protected from magnetic and non-magnetic tramp steel. Consideration should be given to use of an automatic lime marker in conjunction with the detection ahead of the secondary crusher.

Suitable dust suppression and collection equipment is to be provided.

The concrete floors are to be sloped and a sump provided so that periodic wash down can be done.

Facilities should be included for removing or breaking oversize that will not pass the jaw crusher grizzly.

The fine ore screens should be sized to handle wet and sticky ore.

4.5 Fine Ore Storage and Grinding

## Conditions:

Fine ore bin	3,000 ton live capacity of same general design as used by Petrotomics Company, Shirley Basin, Wyoming and Kerr McGee Nuclear Corporation, Grants, New Mexico
Grinding circuit	Single stage ball mill in closed circuit with cyclones
Grinding rate	91.7 dry tons per hour
Operating schedule	7 days per week 3 shifts per day 8 hours per shift
Product size	Nominal minus 28 mesh and 80% passing 300 microns
Pulp densities	
Mill discharge	70% solids (w/w)
Cyclone overflow	55% solids (w/w)
Work index (Bond)	12.8
Ball mill	11 ft by 11 ft grate discharge with 800 hp motor
Cyclones	Two 20 inch diameter (1 operating and 1 spare)

General:

Access to the top of the ore bins is to be via the feed belt gallery and an alternate route is to be provided.

The tripper is to be automatically controlled from bin level indicators.

Venting of the ore bins with a fan is required to avoid radon accumulation.

The feeders under the bins are to be 60 in wide belts of the same design as used at the uranium mills of Kerr McGee, Petrotomics, Exxon and Continental Oil.

An integrating weightometer is required on the conveyor feeding the ball mill; a Merrick type "E" is suggested.

A ball storage area and equipment for adding balls to the mill are to be provided.

A fan is to be installed to vent the grinding mill at the discharge end in order to remove radon gas.

4.6 Leaching

## Conditions:

Pulp density (leach feed)	55% solids
pH	1.5
emf (terminal)	minus 500 mv
Temperature	175°F
Acid consumption (100%) - average	500 lb/ton
- design	650 lb/ton
Chlorate consumption - average	12 lb/ton
- design	16 lb/ton
Retention time	10 hours (nominal)

## General:

A mechanically agitated acid mix tank is to be provided at the head of the leach circuit. The tank is to be 22 ft diameter by 22 ft high with 8 ft of freeboard to allow for foaming. A large portion of the total acid requirement will be added to this tank via manual control.

The maximum size of leach tank is to be 22 ft diameter by a nominal 22 ft high. Install 6 tanks in a single train and leave space only for future addition of a 7th tank. Allow 6 ft, 4 ft and 2 ft of freeboard in tanks 1 and 2, 3 and 4, 5 and 6 respectively.

The tank outlets are to be baffled to overflow pulp drawn from the lower portion of the tank.

Mechanical type agitators are to be used. It is suggested that the leach vessels be insulated and located outdoors.

Provision to bypass any leach tank is to be included.

Acid addition is to be automatically controlled to No. 1 leach tank based on pH measurement. The acid lines are to terminate above the pulp level and be sufficiently remote from the tank walls to prevent

damage to the tank. Facilities are to be available to add acid to all leach tanks. To reduce over concentration, acid is to be added at 4 points in the acid mix tank and the first leach vessel.

Sodium chlorate will be added to the leach pulp as 40% solution; facilities are to be available to add chlorate to the first three vessels. The chlorate lines are to terminate above the leach pulp level.

The acid mix tank and leach tanks are to be covered and vented to a common wet scrubber.

Provision is to be included for heating of leach pulp to 175°F by steam sparge into the first two leach vessels. Steam addition to make up for heat losses is required on the third and fifth tanks.

All distribution piping is to be located above the leach vessels.

4.7 Counter Current Decantation

## Conditions:

## Design basis

number of stages	5
unit area	8.5 sq ft/tpd
size	155' diameter

## Flocculant

type	Dow MG-200
dosage - No. 1 thickener	0.20 lb/ton
- total CCD	0.43 lb/ton
concentration - stock solution	0.3%
- to thickeners	0.03%; dilution with raffinate or overflow solution

## Underflow pulp density

for flowsheet	53%
for pump design	47%

Pregnant liquor flow 1,375 gpm

## Wash solution

raffinate	65% to 100%
water	35% to 0%

## General:

Use two underflow lines per thickener.

Normal pumping of thickener underflows is to be done with diaphragm pumps; each pump will have the capacity to handle full flow at 47% solids.

Centrifugal pumps will be provided as spares, one for each diaphragm pump.



The underflow pumps are to be located to give minimum suction head.

Interstage pumping will be accomplished by directing the diaphragm pump discharge and the thickener overflow solution into a pumpbox, with subsequent advance of the mixture to the next stage with a centrifugal pump. The pump box should be as tall as possible to permit a variable head on the pump to compensate for flow fluctuations.

Piping is to be arranged so that any thickener may be bypassed.

Piping and pumps transporting thickener feed will be designed to handle pulp at 13% solids to permit recycle of overflow solution.

Motorized rake lifting mechanism and remote torque indication are to be provided.

Thickeners are to be provided with underflow tunnels with good drainage characteristics; they can be about 7 ft high by 4 ft wide.

The thickener general arrangement should be such that the length of suction lines to the diaphragm pumps is minimized.

The thickener tank walls should extend 1 foot above the solution level to decrease wave action due to winds and prevent solution loss over the tank walls.

The piping arrangement should permit recycle, advance and bypass of underflow solids plus recycle, advance and bypass of overflow solution.

The minimum recommended slope for thickener feed launders is  $\frac{3}{8}$  inch per foot.

Since most flocculant solutions degrade with time, it is advantageous to design for daily mixing of fresh flocculant. This also applies to glue make-up.

Provision should be made for staged flocculant addition (not less than 5) along the thickener feed launder with a final addition into the center well.

4.8 Clarification

## Conditions:

Feed rate (design)	1,375 gpm
Clarifying thickener-surge tank	
Tank size - diameter	65 ft
- surge working volume	330,000 gallons
Solids - feed solution	150 - 250 ppm
- overflow solution	35 - 80 ppm
- underflow pulp	5% (maximum)
Glue - usage	0.10 lb/ton
- concentration	7.2 gpl
Clarifying filters	
Type	downflow pressure sand filters
Effluent solids	10 ppm
Size of units	10 ft diameter
Number of units	2 operating, 1 spare
Clarified pregnant liquor storage	
Tank working volume	330,000 gallons

## General:

Underflow from the clarifying thickener-surge tank reports to No. 1 leach tank with provision to bypass to No. 1 CCD thickener.

The surge capacity in the clarifying thickener will be provided by additional tank volume above the solution withdrawal point. Withdrawal will be by an exterior ring main around the mid-section of the tank with connections at three points.

Glue addition is to the center well of the clarifier-surge vessel.

A tunnel is not needed for the underflow from the clarifier-surge vessel; the underflow line may be located inside and on the tank bottom.

Clarifying filter backflush solution is to be returned to No. 1 CCD thickener with provision to go to No. 2 thickener. Clarified pregnant solution is to be used for backwashing.

Provision should be made for pH adjustment of pregnant solution in the clarifier by manual addition of acid.

4.9 Solvent Extraction

## Conditions:

## Pregnant liquor

Design flow	1,375 gpm
Average grade	0.168 gpl $U_3O_8$
Maximum grade	0.240 gpl $U_3O_8$
$U_3O_8$ content, maximum	3,960 lbs/day
Specific gravity	1.05

## Extraction

Number of stages	4
Solvent	
Concentration	2.5 v/v% Alamine 336 or Adogen 364 2.5 v/v% Isodecanol or Tridecanol 95 v/v% Kerosene
Specific gravity	0.80
Specific heat	0.46 BTU/lb/°F

Organic loading	1.80 gpl $U_3O_8$
Advance organic flow	183 gpm
Recycle organic flow	1,580 gpm
Mix tank working volume	2,510 gallons
Settler area	1,800 sq ft per stage

## Stripping

Number of stages	4
Strip solution	150 gpl $(NH_4)_2 SO_4$ with pH controlled at 4.0 - 4.3 with gaseous ammonia

## Strip liquor

Grade	20 gpl $U_3O_8$
Flow	16.5 gpm
Specific gravity (No. 1 stage)	1.12

Pregnant organic flow	183 gpm
Aqueous recycle flow	33 gpm
Pregnant organic temperature	86°F
Mix tank working volume	945 gallons
Settler area	160 sq ft per stage
$NH_3$ requirement for neutralization	0.4 lb/lb $U_3O_8$

## Regeneration

Number of stages	1
Barren organic flow	183 gpm
Regeneration liquor	
Composition	5% $Na_2CO_3$
Advance flow	10 gpm
Aqueous recycle flow	90 gpm
Mix tank working volume	945 gallons
Settler area	160 sq ft per stage

## SX circuit tanks

Barren organic working volume	23,500 gallons
Pregnant organic working volume	23,500 gallons
Raffinate surge	60 feet minimum diameter x 9.5 feet high
Organic sludge holding working volume	33,500 gallons

## General:

The views of the Fire Insurance Underwriter should be obtained on the location of the solvent extraction circuit relative to the general plant layout.

Maximum overflow crest height on settler overflow weirs is to be

1.0 inches.

Provision is to be made for solvent recovery from the raffinate surge tank on an intermittent basis. The organic will be removed by skimming through an overflow pipe and will be returned to the No. 4 extraction mixer.

A single rectangular type solvent extraction circuit can be used. Organic depth in the settlers is to be 15 inches. Aqueous and organic overflow weirs, extending the full width of the settlers, are to be used.

A continuous bleed is to be taken from the bottom of the barren and pregnant organic surge tanks and returned to No. 3 strip mixer and to No. 2 extraction mixer respectively. When solvent regeneration is in operation, then the bleed from the barren surge tank should go to No. 2 extraction mixer.

A 'picket fence' baffle is to be located across the feed end of the settlers. A head loss of  $\frac{1}{4}$  -  $\frac{1}{2}$  inch across the baffle should be used to size the pickets.

Make-up kerosene, alcohol and amine are to be added to the barren organic tank by separate pumps. Barrel type pumps are suitable for the alcohol and amine.

Mix tank working volume excludes freeboard and any space under the false bottom below the impeller.

The speed of the mixer impellers should be variable and the bottom clearance should be adjustable by raising and lowering the shaft.

The pregnant and barren organic surge tanks are to be steel with FRP lining; they are to be covered and vented to atmosphere.

Provision is to be made for the introduction of approximately 15 cfm of air with the ammonia feeding the strip mixers.

A Teflon tube bundle or carbon block heat exchanger is suitable for heating the organic.

The purpose of the organic sludge holding tank is to assist in the removal of interfacial crud from the settlers should this become necessary and for possible recovery of uranium from the spent  $\text{Na}_2\text{CO}_3$  regeneration solution. The tank should be equipped with an agitator mechanism, pipe lines for sulfuric acid, ammonia and process water. An arrangement is needed for decanting this tank for supernatant disposal to tailings and for discharge of the yellow cake sludge to leaching and/or precipitation. Space should be provided for the possible future installation of a separate acidification tank, precipitation tank and thickener for processing spent  $\text{Na}_2\text{CO}_3$  solution.

Provision should be made for manual addition of sulfuric acid to No. 3 extraction settler aqueous overflow for pH adjustment of barren organic from the regeneration mixer-settler.



4.10 Yellow Cake Precipitation and Washing

## Conditions:

Precipitation tanks	Design for two tanks in series with working volume of 750 gallons per tank. Allow 18 inch freeboard in each tank.
Precipitation temperature	140° - 160°F
pH (both tanks)	7.3 - 8.2
Ammonia required	0.20 lb/lb $U_3O_8$ approximately
Yellow cake	
Specific gravity	5.7
Specific heat	0.25 BTU/lb/°F
Washing thickener	
Underflow	20% solids for pump design 33% solids for flowsheet
Overflow	200 ppm solids
Diameter	18 ft, cone bottom
Barren liquor sand filter	
Feed rate	21.2 gpm
Type	Pressure sand filter; backwash with clarified barren
Size	2 ft diameter
Surge tanks	
Unclarified barren liquor	2,500 gallons
Clarified barren liquor	2,500 gallons
Filter backwash	150 gallons

## Centrifuge

Discharge density	60% solids
Centrate solids	2.0% of feed solids
Size	18 in by 28 in Bird solid bowl, or equivalent

## General:

The precipitation tanks are to be mechanically agitated. The tanks are to have baffles and the agitator designed to provide only mild agitation of the yellow cake slurry. The precipitation tanks should be covered and vented with natural draft.

The solution balance in the yellow cake thickening and centrifuging circuit is designed to produce dried yellow cake containing no more than 1.5% soluble  $\text{SO}_4$ . A circuit balance is shown on the next page.

Circuit bleed from the clarified barren strip surge tank is to tailings.

Backwash from the sand filter will be returned to the precipitation tanks via an elevated surge tank.

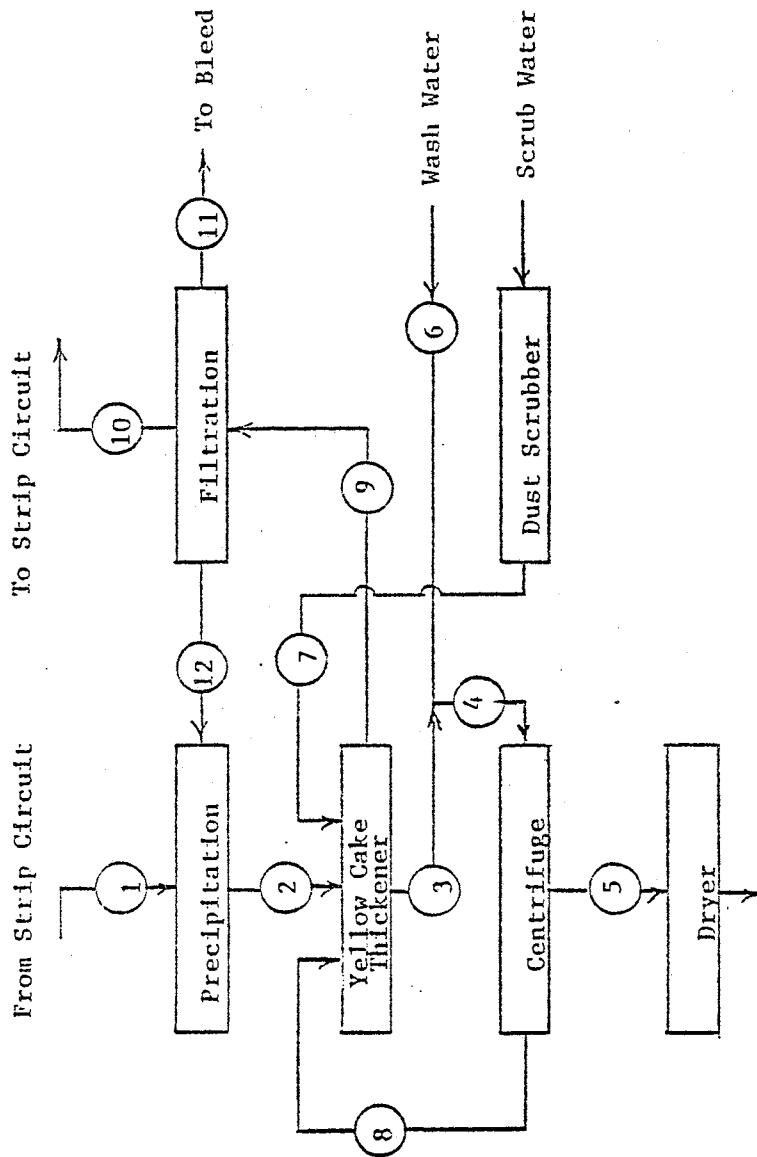
The straight side of the thickener is to be about 5 ft high, and the tank bottom is to have a slope of about 1:2. The feed well should be about 3 ft deep. No tunnel is to be used.

Potable water should be used for the wash circuit and the hose connections. The area floor sump should discharge to the thickener.

Flocculant is not ordinarily required in the precipitation thickener; if found necessary, piping may be added at a later date.

Feed to the centrifuge is to enter through a small open pot; dilution water is added at this point. The design must provide adequate access for sampling and volume measurements.

# Yellow Cake Washing Circuit - Soluble Sulfate Balance



Stream Number	1	2*	3	4	5	6	7	8	9	10	11	12
<b>SOLIDS</b>												
Yellow Cake - lbs/day	4,586	4,637	4,776	4,776	4,680	-	94	96	51	-	-	51
U <sub>3</sub> O <sub>8</sub> - lbs/day	3,898	3,941	4,060	4,060	3,978	-	80	82	43	-	-	43
<b>LIQUID</b>												
Flow - tpd	97.45	104.16	4.77	23.63	1.56	18.85	5.68	22.07	127.12	97.45	22.96	6.71
SO <sub>4</sub> - tpd	13.26	13.90	0.52	0.52	0.03	-	-	0.49	13.87	10.63	2.50	0.73
SO <sub>4</sub> - gpl	136.1	134.3	109.1	22.1	22.1	-	-	22.1	109.1	109.1	109.1	109.1

\* A total of 0.09 tons/day of SO<sub>4</sub> is lost in this stream as insoluble SO<sub>4</sub> when the yellow cake is precipitated.

The centrifuge discharge chute should be vented to the scrubber on the yellow cake dryer.

4.11 Yellow Cake Drying and Packaging

## Conditions:

## Dryer

## Dryer feed

for dryer design

40% solids

for flowsheet

60% solids

## Capacity, dry solids/day

4,680 lbs yellow cake at  
85%  $U_3O_8$ Moisture specification of dried  
product

less than 2%

## Drying temperature

Solids, maximum hearth

750°F capability

Furnace - for design

1,060°F gas temperature

- capability

1,200°F gas temperature

## Product size

 $\frac{1}{4}$  inch maximum

## Dust load from dryer to scrubber

2.0% by weight of dryer feed

## Packaging

## Bulk density of yellow cake

Loose

80 lb/cu ft

Compacted

120 lb/cu ft

## Storage capacity

Sized for packaging on  
one shift for 5 days  
per week.

## Drum capacity (55 gallon)

Approximately 900 lbs  
of yellow cake

General:

A multiple hearth dryer of the Herreschoff type is to be used for drying yellow cake. The approximate total hearth area is estimated to be 95 sq ft.

The centrifuge solids discharge can be delivered to the top hearth using a screw feeder.

The dryer is to be equipped with a high energy venturi type wet scrubber using water as the scrubbing medium; scrubber effluent will be sent to the yellow cake thickener. A separate dry dust collector is to be provided for the packaging area. Stacks from dust collecting equipment must be installed to permit measurement and sampling of discharge flows.

The product size reduction unit is to be enclosed and vented to the dry dust collector.

The product does not require cooling prior to discharge into the storage bin.

The dryer is to be designed to allow access to all hearths without dismantling the entire unit.

Full drums of yellow cake can be stacked a maximum of 3 high.

Provision should be made for vacuum cleaning in the dryer-packaging area.

The sump in the dryer-packaging area should discharge to the yellow cake thickener.

The dryer and drum packing station are to be in an enclosed area which is not to contain exposed pipes or ductwork. The walls and ceiling are to be sheathed to cover steelwork, etc. and to provide a smooth surface that can be readily cleaned. Floor trenches should not be used and the floor sump should be of minimum size.

## 4.12 Mill Sampling Facilities

### 4.12.1 Moisture in Mill Feed

A sampler of the type described below is to be used to cut a sample from the grinding mill feed belt. Sample frequency should be adjustable between once to twice per hour.

The automatic sampler cutter sits above the feed belt to the ball mill. The cutter is about 4" wide and slightly curved; it is bolted onto a rod that is attached to a drive shaft located about 2 ft above the belt. The drive motor is controlled by a timing switch. The intermittent sample is discharged to a teflon lined chute beside the conveyor belt. The chute discharges into a collecting barrel via a PVC pipe.

### 4.12.2 Mill Feed

An automatic sampler of the straight-line type is to be located on the leach feed pulp; this unit would operate continuously. The initial sample will be reduced by 2 stages of Vezin samplers. The third cut is to be stored in a small tank equipped with a mixer and will be further reduced at the end of each shift by a third Vezin cutter.

### 4.12.3 Leach Discharge

Access is to be provided to readily obtain a manual sample at the discharge from the leach circuit.

### 4.12.4 Mill Tailings

An automatic sampler of the straight-line type will be located on the CCD thickener circuit pulp discharge; this unit will operate continuously. The initial sample will be reduced by a second stage automatic sampler which will operate intermittently. Final sample volume will be approximately 10 quarts for each

8 hour shift.

Any raffinate sent to tailings disposal should enter the system downstream of the mill tailings sampler.

4.12.5 Yellow Cake

A sample will be cut by hand from the top of each drum of yellow cake.

4.12.6 Solution Samples

Solution samples are to be taken to provide an 8 hour composite sample of the following streams:

- Pregnant Liquor (SX feed)
- Raffinate
- Pregnant Organic
- Barren Organic
- Pregnant Strip Liquor
- Barren Strip Liquor
- Washing Circuit Bleed Stream
- Regeneration Aqueous Discharge

4.12.7 General

All samplers, with the exception of mill feed and yellow cake, will require corrosion resistant construction.

Sampling of process streams by the operator to control solvent extraction is in addition to those listed above.



#### 4.13 Main Installed Spare Pumps

This list of spare pumps is subject to change pending details of final flowsheet and review of general arrangement drawings.

- Feed to Cyclones
- Leach Feed
- Leach Discharge
- CCD Thickener Underflows (1)
- CCD Thickener Slurry Advance Pumps
- Tailings Discharge
- Clarifying Thickener Feed
- Sand Filter Feed
- Pregnant Liquor Feed to SX
- Feed to Raffinate Tank
- Discharge from Raffinate Tank
- Feed to Pregnant Organic Tank (2)
- Feed to Barren Organic Tank (2)
- Feed to Yellow Cake Precipitation
- Feed to Barren Strip Liquor Sand Filter
- Discharge from Clarified Barren Liquor Surge Tank
- Sulfuric Acid Feed to Process
- Sodium Chlorate Feed to Process
- Flocculant Feed Systems to Process
- Glue Feed System to Process

- (1) Centrifugal pump as spare to diaphragm pump.
- (2) No spare on discharge of tank on premise that the tank can be bypassed.

#### 4.14 Standby Power Requirements

Standby power should be provided to operate the following equipment during a failure of the normal electrical power supply.

Leach agitator drive; all to be connected, but only one at a time to be driven.

Leach discharge pump.

All CCD thickener drives and rake lifting devices.

Thickener underflow diaphragm pumps.

Slurry advance pump on each thickener.

Tailings pumps.

Solvent extraction area sump pumps.

Yellow cake thickener drive and underflow pump.

Yellow cake dryer cooling fan.

Plant lighting.

Instrument air.

Gland water pump.

Steam boiler.

#### 4.15 Instrumentation

##### 4.15.1 Qualification

This outline of requirements for process instrumentation represents recommendations with regard to applications where instruments can be used effectively to aid the operator in achieving efficient mill operation. It is subject to change pending finalization of the general arrangement and flowsheet circuits, and process equipment selection.

##### 4.15.2 Crushing

Recording ammeters on crusher motors.

Tilt switches to detect plug-ups at critical transfer points.

A weightometer on the conveyor to the fine ore bins to indicate crushing feed rate.

##### 4.15.3 Grinding

Remote control of the ore feeder vari-speed drives from the grinding control panel.

The feeders under the fine ore bins and water to the ball mill are to be controlled by the weightometer on the ball mill feed belt which is to be both indicating and totalizing.

An indicating-recording wattmeter for the ball mill motor.

Flowmeters on the water supply to the feed and discharge of the grinding mill.

A gamma density recorder-controller to control water addition to the cyclone feed.

4.15.4 Leaching

pH recorder-controller for No. 1 and No. 2 leach tanks to provide automatic adjustment of acid addition to maintain constant pH regardless of changes in ore type or throughput.

pH indicator for Nos. 3 and 5 leach tanks. Flow rotameters on acid addition lines to the acid mix tank and Nos. 3, 4, 5 and 6 leach tanks; adjustment of flow will be made manually.

Totalizing flow indicator on total acid addition to leaching.

Manual control of sodium chlorate addition through a flow meter and control valve to leach tanks No. 1 and 2.

Multi point emf recorder with electrodes in leach tanks No. 1, 2, 3 and 5.

Totalizing flow indicator on total sodium chlorate addition to leach.

Temperature recorder-controller for steam addition to leach tank No. 1 and 2.

Temperature recorder on leach tanks No. 3 and 5 where steam addition is by manual control.

Flow totalizer on total steam supply to leaching.

4.15.5 CCD Thickening and Clarification

Flow recorder-controller for raffinate recycle to CCD wash. Maintain set volume of recycle raffinate to provide desired CCD wash ratio.

Flow recorder-controller for fresh water wash to CCD.

Dependent upon the type of distribution system selected, it may be desirable to have measurement of flocculant flow to each thickener to facilitate manual adjustment of flocculant additions.

Torque recorders and motorized lifting devices for the thickener rake mechanisms. Push button controls for the lifting devices should be located both on the thickeners and in the control room for the CCD area.

Gamma gauge density and magnetic flowmeter recorders on each thickener underflow are recommended.

Bubbler type level indicator in the clarifying thickener-surge tank.

Flow indicator-controller on clarifying thickener underflow to leach or CCD alternative.

The instrumentation for the clarifying sand filters (flow-meters, switches and relays for automatic operation) are to be provided by the supplier of the filters.

A means of measuring solution clarity, such as an on-stream turbidimeter, should be provided.

#### 4.15.6 Solvent Extraction and Precipitation

Flow recorder-controllers on the following process streams to the solvent extraction circuit:

- Pregnant solution feed
- Barren organic to extraction
- Pregnant organic to stripping
- Barren ammonium sulfate to stripping
- Soda ash regeneration

Temperature indicator-controller on pregnant organic to stripping. Maintain pregnant organic at set temperature by automatic adjustment of steam addition to heat exchanger.

pH recorder-controllers for stripping. Maintain set pH in No. 1, No. 2 and No. 3 stripping stages by automatic adjustment of ammonia addition to each of these 3 strip mixers.

Ammonia is to be introduced into the mixers through type 316 stainless steel ring spargers. Air would be bled into the ammonia streams to reduce local ammonia concentration and the possibility of local precipitation.

The pH measurements are made on a small bleed stream via an outlet through the wall of the mixer compartments.

Bubble type level indicators on each of the following tanks:

- Pregnant organic surge
- Barren organic surge
- Clarified pregnant liquor storage
- Unclassified barren liquor surge
- Clarified barren liquor surge

Orifice type flow integrator on raffinate discharge to tailings for metallurgical accounting purposes.

A pH recorder-controller is required for precipitation. It will maintain a set pH in No. 1 precipitation tank by automatic adjustment of ammonia flow. Manual addition facilities are required for No. 2 precipitation tank.

A temperature recorder-controller will maintain a set temperature in No. 1 precipitation tank by automatic adjustment of steam flow to the heat exchanger.

Controls for the ammonia storage and vaporizer system to be provided by the equipment supplier.

4.15.7 Yellow Cake Washing and Drying

Flowmeters on wash water to the centrifuge feed and to the dryer dust scrubber.

A small measuring pot with shut-off valve on the bottom discharge line of the feed to the centrifuge. The flow can be determined by the time required to fill the pot to a fixed level.

A variable speed pump to transfer yellow cake from the thickener to the centrifuge, with the control located at the centrifuge.

The dryer instrumentation, draft control, temperature recorders, alarm system, etc., is to be provided by the supplier of the dryer.

#### 4.16 Materials of Construction

##### 4.16.1 General

The selection of construction materials is governed primarily by such process chemical and temperature requirements as:

A leaching temperature of 175°F.

The addition of 12 lbs per ton of sodium chlorate to the leaching circuit.

Kerosene in solvent extraction.

Because early failure of bonded linings have sometimes been experienced, all lining contracts should specify that the lining contractor accept responsibility for the condition of the surface to be covered.

##### 4.16.2 Grinding

Pump tanks and launders in the grinding and neutral thickening areas should be of steel construction, lined with rubber. Linatex is widely used for wet abrasion applications because of the relative ease in building up worn areas.

##### 4.16.3 Leach Tanks

Rubber-lined mild steel vessels and agitators are suggested. Because the temperature (175°F) will blister and harden soft natural rubber, the following laminate is proposed:

1/16 inch Shore hardness 45, natural rubber (to the metal).

1/8 inch Shore hardness 90, natural rubber.

1/32 inch Shore hardness 45, natural rubber.

5/32 inch Shore hardness 60, chlorobutyl\* (to the slurry).

\*e.g. Goodyear LS-582, Blair 9261 P.E.



Leach tank covers may be of the same materials. Each tank will be provided with a fibreglass or polyethylene vent.

Steam injection pipes may be alloy 20, or carbon steel pipe covered and lined with hard rubber.

The outside of the leaching tanks should be painted with one priming coat and two coats of acid resistant paint.

#### 4.16.4 Thickeners

Thickener tanks in acid circuits have been successfully constructed from the following materials:

Wood stave

Rubber lined steel

FRP lined steel

FRP lined concrete bottom, wood stave wall

For the vertical shafts and rake arms, rubber-covered mild steel has been used successfully but 316 stainless steel is preferred. Stainless steel rake blades are recommended.

If wood stave construction is used for the thickener walls, the supporting bands and turnbuckles should be covered with polyethylene tubing for acid protection.

#### 4.16.5 Clarification and Solution Storage

The pressure sand filters should be rubber-lined mild steel, and 316 stainless steel is suggested for the internal distribution piping.

Solution holding tanks may be wood stave or mild steel lined with rubber or FRP. The raffinate tank should not be rubber lined.

#### 4.16.6 Solvent Extraction

FRP lined mild steel is suitable for storage tanks and for mixer-settlers. FRP bonded to concrete, self-supporting FRP and 316 stainless steel can be considered for the mixer-settlers.

Type 316 stainless steel is preferred for the wetted parts of the pumper-mixers.

Teflon and impervious graphite have been used for the steam heater on the pregnant organic stream. Stainless steel has failed on this service.

#### 4.16.7 Precipitation, Washing and Drying

The uranium precipitation tanks may be wood stave or FRP coated mild steel; the mechanisms should be mild steel, FRP or neoprene covered or 316 stainless steel.

Mild steel FRP lined is suggested for the strip liquor holding tanks.

The yellow cake thickener tanks can be FRP lined mild steel or wood stave. The thickener centre shafts and rake arms can be mild steel, coated with FRP or neoprene. Stainless steel is recommended for the rake blades and also is preferred for the rake arms.

The pregnant liquor heat exchanger should be of teflon or impervious graphite construction.

The wetted parts of the yellow cake centrifuge should be of type 316 stainless steel construction. Stainless steel is suggested for the screw of the dryer screw feeder.

The wet scrubber should be type 316 stainless steel.

#### 4.16.8 Chlorate Storage

The sodium chlorate tanks are of mild steel construction. Since corrosion is fairly severe, it is recommended that the inside of the tanks be painted with a phenolic resin coating.

#### 4.16.9 Pumps

Rubber lined pumps with 316 stainless steel trim are recommended for acid slurry service throughout the plant. Acid solution pumps may be of stainless steel or, where contact with organic solvent is not possible, may be rubber lined with stainless steel trim.

Type 316 stainless steel pumps are suitable in the solvent extraction circuit for both aqueous and organic. In order to reduce gland seal flows, vertical type pumps have been used where layout permits; otherwise, mechanically sealed horizontal pumps are employed.

In the precipitation circuit, the yellow cake slurry pumps may be Sandpiper type (air-operated diaphragm), O.D.S. or Moyno. Natural rubber staters are preferred for Moyno pumps, they should be over-sized (slow speed) and care must be taken to avoid interruption of feed to the pumps.

The sodium chlorate solution pumps should be of cast iron construction, self-priming and glandless. There must be no hydrocarbon based oil or grease lubrication.

Sulphuric acid pumps should be of mild steel construction and equipped with mechanical seals.

#### 4.16.10 Concrete Surfaces Subject to Acid Spillage

Concrete protection practice in acid-leaching uranium mills has varied from nil to almost complete coverage. If no protection is provided, it is probable that some resurfacing will be required after 3 - 7 years' operation.

At the Exxon mill, in the leach area, the bottom level of the CCD pumphouses, and around the SX mixer-settlers, the floor, column footings, pump bases and twelve inches of the outside wall were coated with an epoxy containing glass flakes. Two applications were used to provide a total thickness of 20 mils.

At the Elliot Lake operations in Canada, several types of coatings were tried. Denison Mines used asphalt in the leaching area and, after twenty years, the covering is still in service, although the concrete has broken up to some extent in the pH adjustment area. Epoxy paste, applied by trowelling, was used extensively at Denison and at other Elliot Lake mills. With this covering, it is essential that the concrete be completely dry before application and, in the sumps and lower floor areas, which were not adequately dried, the coating failed. In the subsequent expanded leaching installation at Denison, the floors, piers and walls up to a height of 18 inches were protected with a silica loaded epoxy compound.

The areas around ion exchange and precipitation at the Shirley Basin mill of Utah International are protected with a polyester quartz sand mixture. The only cracking has been along expansion joints. The floor and walls of the CCD thickener tunnels have been covered with FRP up to a height of eighteen inches.

#### 4.16.11 Piping

Carbon steel is the standard material for pulp slurries, unlined for neutral pulps, soft natural rubber lined for acid pulps. Victaulic couplings are usual but flanged connections should be

employed on thickener underflow lines.

Material conducting hose has been used in limited areas such as:

Where abrasion is particularly severe.

To obtain long radius bends.

To decrease pipe vibration.

To facilitate maintenance on pipe connections.

Where abrasion is not expected to be severe, high density polyethylene and ABS may be also considered for slurry lines. For security reasons, such materials should not be used on thickener underflow lines. FRP has low abrasion resistance and should not be used for slurry piping.

Acid solution lines may be of such material as PVC, FRP, ABS, rubber lined steel, etc.; the main criterion is installed cost. Failure of plastic lines in the manifolds around the sand bed filters, because of water hammer, has occurred. Rubber lined steel is suggested for this location.

PVC and FRP piping have often been used for the organic solvent. In the locations where a line breakage would drain the contents of a vessel, stainless steel is preferred. Buna-N rubber has satisfactory resistance to the organic and may serve for gaskets, sleeves, etc.

In the yellow cake precipitation-thickening area, PVC, FRP, rubber lined steel, etc. are satisfactory. Thickener underflow should be stainless or rubber lined steel.

For the tailings pump discharge, rubber lined steel, high density polyethylene, polypropylene and ABS have been used successfully.

Schedule 80 carbon steel pipe with welded fittings and flanged connections is commonly used for concentrated sulphuric acid. Small lines are readily blocked by iron sulphate and sizes below

1 inch should be avoided.

#### 4.17 Analytical and Testing Services

##### 4.17.1 Mine Control

Mine operation can be largely controlled using radiometric methods to delineate ore and waste both at the working faces and on the loaded trucks. Marginal material can be sampled for laboratory analysis to determine the final destination of the material.

The radiometric instruments require chemical analyses for their calibration.

Drill core from mine development would likely be analyzed both chemically and radiometrically.

##### 4.17.2 Process Control

The basic mill samples and their frequency of collection are listed below on a basis of milling 2,000 tons per day of ore:

<u>Sample</u>	<u>Frequency</u>	<u>Elements to be Analyzed</u>
Feed to Grinding	shift composite	H <sub>2</sub> O
Leach Feed	shift composite (possible); daily composite	U <sub>3</sub> O <sub>8</sub> , Mo, V <sub>2</sub> O <sub>5</sub> , CO <sub>2</sub>
Leach Discharge		
Liquid	shift or daily grab	emf, pH, Fe <sup>++</sup> , Fe <sup>+++</sup> , H <sub>2</sub> SO <sub>4</sub>
Solid	shift or daily grab	U <sub>3</sub> O <sub>8</sub>
CCD Underflow		
Liquid	daily composite	U <sub>3</sub> O <sub>8</sub>
Solid	daily composite	U <sub>3</sub> O <sub>8</sub>
Pregnant Solution	shift composite	U <sub>3</sub> O <sub>8</sub> , pH, turbidity
Raffinate	shift composite	U <sub>3</sub> O <sub>8</sub>
Pregnant Organic	shift composite	U <sub>3</sub> O <sub>8</sub>

<u>Sample</u>	<u>Frequency</u>	<u>Elements to be Analyzed</u>
Barren Organic Pregnant Strip	shift composite	$U_3O_8$
Liquor	shift composite	$U_3O_8$
Barren Strip		
Liquor	shift composite	$U_3O_8$
Product	daily composite	$U_3O_8$ , $SO_4^{=}$ , $H_2O$
Organic	weekly grab	amine, alcohol
Regeneration		
Aqueous	daily composite	Mo, $U_3O_8$ , pH
Washing Circuit		
Bleed	daily composite	$U_3O_8$
Tailings Liquor	weekly composite from shift grab	$U_3O_8$ , $V_2O_5$ , Mo, Fe, $NH_3$ , Ra, Se

#### 4.17.3 Product Control

While the lot size has not been specified, it is estimated that approximately 1 lot per two weeks will be produced. Therefore, two to three samples of yellow cake would be submitted monthly for a check on specification requirements.

Depending on the contract stipulations, the following elements would be determined:

$U_3O_8$ ,  $PO_4$ , Halides,  $SO_4$ , Fe,  $H_2O$ , Mo

Capability for analyzing for the following elements should exist, but their determination may not be routinely required:

$V_2O_5$ , F, As,  $CO_3$ , Th, Zr, Ca, Na, B, K



#### 4.17.4 Miscellaneous Services

Laboratory analytical services should be considered also for the following demands:

Additional exploratory work involving geological samples; diamond drill or churn drill samples for future mining requirements.

Miscellaneous mill samples such as dust collection system products for  $U_3O_8$  content, daily leach feed composite samples for screen analyses to control mesh of grind, provide stock solutions for mill operators' use, etc.

Miscellaneous samples generated by plant metallurgists such as screen assay test fractions for  $U_3O_8$  size distribution, leach test products, solvent extraction test products, etc.

Miscellaneous infrastructure samples such as potable and boiler water treatment plant analyses, sewage treatment plant analyses, plant operator urine analysis for albumin, sugar, and  $U_3O_8$  content. If the local hospital has the facilities, then the last item would probably be performed under medical auspices.

Miscellaneous plant effluent control including radium determinations.

#### 4.17.5 Metallurgical Testing

The laboratory facility should include space and equipment for routine testing of all unit operations in the mill flowsheet for optimization of process variables.

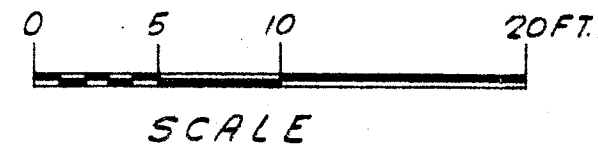
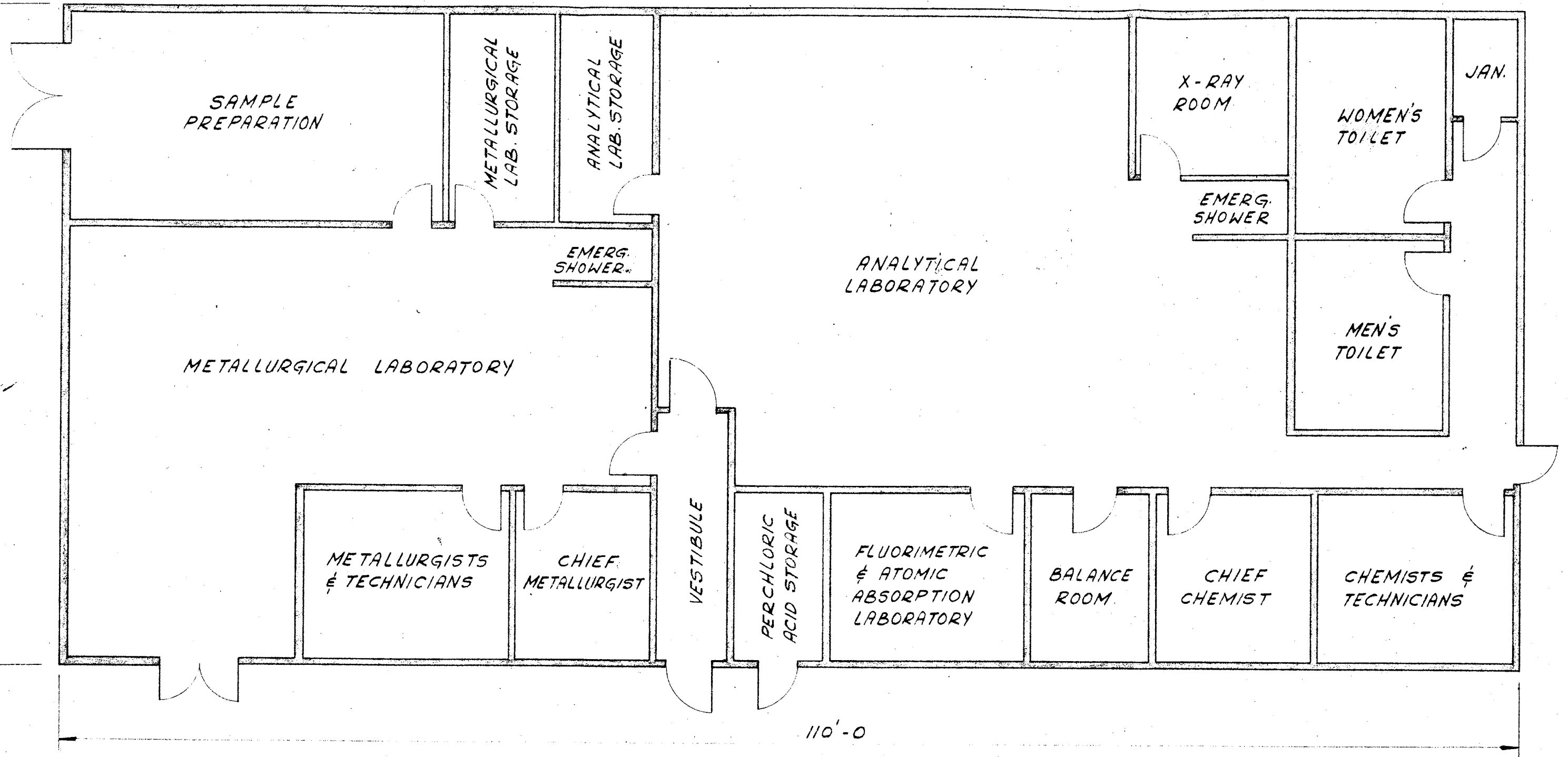
Testing of samples from other properties, amenability procedures for custom ores, etc. may also be required from time to time.

4.17.6 Facilities

Sample preparation facilities will be necessary since all mine samples, plant feed, tailings and product samples, plus any solid samples from experimental test work would have to be suitably dried, ground, and individually blended.

An operation of this magnitude requires a well equipped facility. The accompanying drawing shows the suggested layout of a suitable analytical and metallurgical laboratory plus offices.

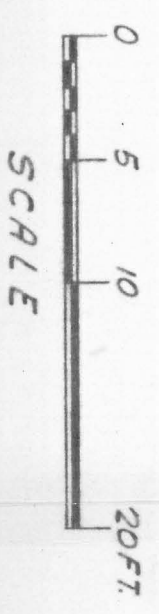
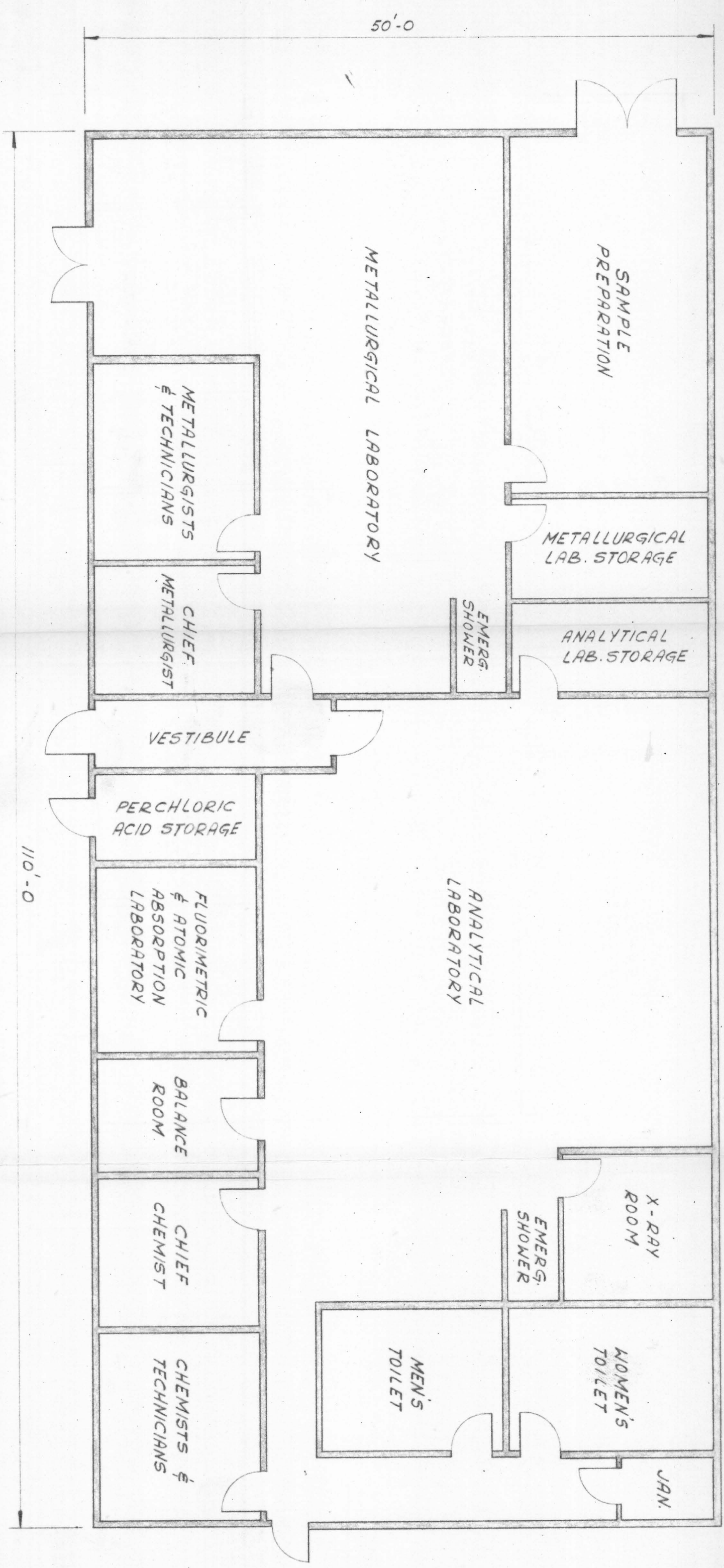
The arrangement assumes that an X-ray unit would be available for assaying. Although not necessary for routine mill analyses, an atomic absorption unit is very desirable for miscellaneous determinations, environmental water samples, etc.



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