THE

ONTARIO WATER RESOURCES

COMMISSION

INDUSTRIAL WASTES SURVEY

of

DENISON MINES LIMITED

Elliot Lake, Ontario

1968

A

Report On

An Industrial Wastes Survey

of

DENISON MINES LIMITED

Elliot Lake, Ontario.

August 7, 12, 1968.
and
September 20, 1968.

by

Division of Industrial Wastes
ONTARIO WATER RESOURCES COMMISSION

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REPORT

Ontario Water Resources Commission

Municipality Elliot Lake, Ontario. Date of Inspection August 7, 12 and Sept.20/68

Re: DENISON MINES LIMITED

Field Inspection by J. R. Hawley Report by J. R. Hawley

Denison Mines Limited was visited in order to update Ontario Water Resources Commission information concerning the Denison operation. Mining, milling, and waste treatment procedures were reviewed.

This report is intended only as a preliminary report, since a further detailed report concerning the tailings areas at Denison Mines Limited will be issued at the conclusion of the next field season.

SUMMARY

Denison Mines Limited must assume complete responsibility for all waste discharges resulting from past, present, and future mining and milling operations. This responsibility will outlive the mine itself.

It is quite probable that, in the future, as the tailings masses on the Denison property mature, their rate of production of acids and soluble salts will be accelerated. As a result, future modifications of the waste treatment facilities may be required. In particular, neutralization facilities may be required on the new dam across Stollery Lake in order to treat the Stollery Lake overflow. Alternately this neutralization could probably be carried out at the Long Lake overflow structure.

In the future, the Ontario Water Resources Commission will regard the Stollery Lake overflow at the new dam as the "final effluent" of the

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Denison operation. It is here that all future "final effluent" samples will be taken for chemical and radiological analysis.

At the present time the Ontario Water Resources Commission regard the wastes being discharged from Denison Mines Limited as "periodically acceptable" for discharge to a natural watercourse.

DETAILS OF SURVEY

Denison Mines Limited was visited on August 7, 1968, August 12, 1968, and September 20, 1968. Samples of the main process and waste streams were taken on August 12, 1968. A further sample, taken by officials of Denison Mines Limited and dated September 26, 1968, was received at our Toronto offices.

Personnel Participating

August 7, 1968. - Mr. R. C. Stewart - OWRC
Mr. K. H. Shikaze - OWRC
Mr. J. R. Hawley - OWRC

Mr. J. A. Abramo,
Mill Superintendent,
Denison Mines Limited,
Elliot Lake, Ontario.

August 12, 1968 - Mr. J. R. Hawley - OWRC
Mr. J. A. Abramo - Denison Mines Limited.

September 20, 1968 - Mr. J. R. Hawley - OWRC
Mr. T. J. Dafoe - OWRC

Mr. J. A. Abramo - Denison Mines Limited.

General Introduction

The Denison Mine is located on the west side of Quirke Lake and is approximately 11 miles by road north of the town of Elliot Lake. The property itself is bordered on the north by Quirke, on the east by Panel and Can-Met, and on the south by Spanish American and Stanrock. The 88 Denison claims staked in 1954 are located in townships 144 and 150 in the Elliot Lake district and cover an area of approximately 3,900 acres. Denison Mines Limited also hold the former Can-Met uranium property - - a 35 claim group.

NOTE: Additional information concerning the Can-Met operation forms Appendix A of this report.

Initial surface diamond drilling on the Denison property, coupled with information on the adjoining properties, outlined an orebody containing in excess of 100 million tons grading more than 0.12 per cent U_3O_8.

No. 1 shaft, collared in March 1955 and bottoming at 1,856 feet, was completed in April 1956. By November of 1957, production from No. 1 shaft had reached the initial production target of 3,000 tons per day. Sinking at No. 2 shaft began in February of 1956 and was terminated (in mid-June of 1957) at a depth of 2,776 feet. By February 1958, production from No. 2 shaft had reached 1,000 tons per day and by October 1958 the production total (No. 1 and No.2 shafts) had reached the initial objective of 180,000 tons per month. The two shafts, 2,600 feet apart, are connected by an incline drive. Breakthrough between shafts took place on December 21, 1957.

Mining is by the room-and-pillar method, with 67 per cent extraction on an advance basis. In the early stages, a considerable tonnage was

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mined by the use of drillmobiles, front-end loaders, and other mobile equipment. Now, virtually all mining is carried out by the use of airleg drills and mechanical scrapers. Ore is loaded either directly onto conveyors or into diesel trucks and electric shuttle cars for haulage to the ore passes.

The basic mine plan features the following: two vertical production shafts 2,600 feet apart; primary conveyorways, either in ore or in waste; strike haulage roads or conveyorways; rectangular rooms and pillars (a strip of these with the adjacent haulage road constitutes a production panel); and service roads that provide access between the shafts and haulage roads.

> NOTE: In several sections of the mine, uranium is being recovered utilizing the principles of bacterial leaching.

In 1966, a 6,200 foot drive was started in order to join the Denison and Can-Met workings. The drive continued in 1967 and 1968. To December 31, 1967, the advance totalled 2,200 feet.

In December 1968 the drive was completed on schedule and, as such, now provides access to the eastern section of Denison's orebody. Because of the drive, Denison's ventillation system is also improved.

Denison Mines Limited is the owner of the worlds largest uranium treatment plant. The plant has a rated capacity of approximately 6,000 tons per day. The first 3,000 ton unit of the mill started operations on May 16, 1957 and the second unit became operational during the fourth quarter of 1957. In 1965-66, a plant modernization programme was begun and a new

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leaching circuit was added. A pachuca plant replaced the original wood-stave leaching agitators (mid-year 1966). The twelve steel units and associated equipment are housed in a new building which is attached to the original structure. Construction began on an yttrium recovery circuit during 1966 and production of yttrium began early the following year. The yttrium circuit is located in the area made available by the removal of the leaching agitators.

Sales Contracts

1) Original Consolidated Denison - - Can-Met contract worth approximately $280.9 million with Eldorado Mining and Refining completed November, 1963.

2) August 1962 - December 1963, sold 5,565,923 pounds of U_3O_8 to the United Kingdom at $4.38 per pound.

3) December 1963 - June 1964 sold 2,700,000 pounds of U_3O_8 to the Canadian government at $4.44 per pound.

4) July 1964 - July 1965, sold 1,200,000 pounds of U_3O_8 to the United Kingdom (balance of Gunner contract) at $4.188 per pound.

5) September 1965, received Canadian government contract from Eldorado for 15,000,000 pounds U_3O_8 at $4.90 per pound deliverable 250,000 pounds per month from July 1, 1965, to July 1, 1972.

6) December 1966, Mitsui and Co., Tokyo appointed exclusive sales agent for Japanese markets; 1967, contracts signed with 8 Japanese power companies for delivery of minimum 21,000,000 pounds U_3O_8 over a period of 10 years.

7) Early 1968, contracts signed with 2 private West German companies for delivery of 250 and 150 tons U_3O_8 respectively.

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8) Late 1966, sales contract signed for approximately 300,000 pounds yttrium oxide annually with Michigan Chemical Co. and Molybdenum Corp. of America.

Company Structure

On March 24, 1960, Consolidated Denison Mines Limited amalgamated with Can-Met Explorations Limited. The amalgamated structure is known as Denison Mines Limited.

The following are subsidiaries and associated companies of Denison Mines Limited (1968):

i) Lake Ontario Cement Ltd. (Denison holds 53% interest)
ii) Vespar Mines Ltd. (Denison holds 38.6% interest)
iii) Lakehead Mines (Denison holds approx. 895,885 shares)
iv) Concord Finance Corp. (wholly owned)
v) Denhay Holdings (Denison holds 50% interest)
vi) Denison Mines (European) Ltd. (wholly owned marketing subs.)
vii) Denison Mines (U.S.) Inc. (U.S. exploration subs.)
viii) Denison Mines (Quebec)
ix) Black Hawk Mining Ltd. (Denison holds 1,403,953 shares)
x) Argosy Mining Corp. (Denison holds 787,553 shares)
xi) Midepsa Industries (Denison holds 17-1/2% interest)
xii) Canada Cement Co. Ltd. (Denison holds 338,785 shares)
xiii) Mogul Mines Ltd. (Denison holds 792,950 shares)
xiv) Goldray Mines Ltd. (Denison holds 300,000 shares)
xv) Coulee Lead and Zinc Mines (Denison holds 502,000 shares)
xvi) Consolidated Rexspar Minerals and Chemicals Ltd. (Denison holds 800,000 shares)

In addition, the Petroleum Division of Denison Mines Limited operates oil and gas leases in Alberta and British Columbia. At December 31, 1967, the estimated gross proven reserves were 32,388,000 bbls. oil

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plus 7,758,000 bbls. probable and 25.8 billion cubic feet of natural gas.

Financial Background

CONSOLIDATED PROFIT AND LOSS STATEMENT

12 Months to December 31	$ 1968	$ 1967	$ 1966	$ 1965
Gross Operating Profit	5,516,444	4,740,307	4,859,245	5,563,316
Investment Revenue	8,106,355	5,272,288	2,852,059	4,552,346
Profit on misc. sales	-	10,033	462,752	810,999
Total Revenue	-	10,022,628	8,174,056	10,926,661
Mining Tax	480,000	100,000	70,000	90,625
Deprec. and Deplet.	1,039,083	628,024	410,237	509,472
Share of Subsid. Profits	-	610,744	793,719	1,090,674
Net Profit	12,771,720	9,905,348	8,487,538	11,417,238

Working Capital, Dec. 31	1968	1967	1966	1965
Total Current Assets	26,686,150	22,275,938	9,705,969	-
Total Current Liabilities	17,137,883	23,839,366	15,010,188	-

Geology - Mineralogy

All the orebodies in the Elliot Lake district (including the Denison orebody) are contained in a quartz-pebble conglomerate in the lower part of the Mississagi formation, the basal formation of the Bruce series. Some of the beds are low grade and/or too thin to be mined. The main orebodies appear to lie in pronounced depressions on the pre-Huronian surface. The thickest beds of conglomerate occur in the lowest parts of the depressions, and are underlain by green schists in contrast to pre-Huronian granitic rocks that underlie the thin parts of the conglomerate. The main

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ore-bearing conglomerate is not at the base of the Mississagi, and, at the Quirke and Nordic Mines at least, in addition to a continuous bed of ore grade, there are other conglomerate beds above and below - - - all in the Mississagi.

The main Denison orebody, as defined by surface drilling and later confirmed by underground development, is a tabular deposit underlying an area about 8,000 by 14,000 feet. The ore horizon, averaging 16 feet in thickness and grading 0.139 per cent U_3O_8, is found 0 to 100 feet above the pre-Huronian basement rocks. This horizon consists of an upper (up to 15 feet thick) and lower (up to 12 feet thick) zone of uraniferous quartz-pebble conglomerate separated by a 3-14 foot band of pebbly quartzite. In places the ore is 35 feet thick, but locally the beds pinch, swell, and branch, and the stratigraphic sequence throughout the mine is variable. Although the interbedded quartzite is low-grade material (about 0.02 per cent U_3O_8), the entire section is mined in most places as one bed. Where it exceeds 9 feet in thickness, it is usually left in place as a sill pillar. The highest-grade zones are nearly always found in beds in which the pebbles are closely packed. Abundant pyrite is also used as a guide to higher-grade ore. A second discontinuous bed of uraniferous quartz-pebble conglomerate was found about 100 feet above the main ore beds.

The depth of the ore is from 400 to 500 feet at the north end of the property and goes to 3,000 feet at the south. The ore horizon strikes south 65 degrees east with an average dip of 19 degrees south. The dip varies from 10-30 degrees south, being flatter at depth. The channel structure containing the uraniferous conglomerate beds crosses the Denison property

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from northwest to southeast and appears to continue into the joining properties of Panel, Can-Met, Stanrock and Spanish American.

The uranium and rare earth values in the ore are concentrated (as microscopic mineral grains) in the matrix of the quartz-pebble conglomerate. The matrix itself is composed essentially of fine quartz (SiO_2) grains, sericite ($(H,K) AlSiO_4$), and a small proportion of chlorite ($H_4Mg_3Si_2O_9$-$H_4Mg_2Al_2SiO_9$). Important gangue minerals of the Denison ore are pyrite (FeS_2) and pyrrhotite (Fe_7S_8) both of which occur as constituents of the matrix.

The radioactive ore minerals named in order of abundance are generally brannerite, uraninite, and monazite. One mine in the area reports that brannerite, uraninite and monazite constitute 60 per cent, 30 per cent, and 10 per cent respectively of the uranium content of the ore.

The following chemical analyses of the principal ore minerals are intended to be a guide to the provenance of the various contaminants found in the waste discharges from the operation:

Mineral: brannerite
Formula: $((U, Ca, Fe, Y, Th)_3 Ti_5 O_{16})$?

The mineral is essentially an oxide of titanium, uranium, and calcium, with minor yttrium, thorium, and ferrous iron.

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Typical Analysis:

Component	Per Cent
CaO	2.9
BaO	0.3
SrO	0.1
PbO	0.2
FeO	2.9
$(Y, Er)_2O_3$	3.9
UO_2	10.3
UO_3	33.5
ThO_2	4.1
ZrO_2	0.2
SiO_2	0.6
TiO_2	39.0
CO_2	0.2
H_2O	2.0
Total	100.2

NOTE: The state of oxidation of the Fe and U is uncertain. The water content is probably due to alteration. Contains Helium.

Mineral: uraninite

Formula: UO_2

NOTE: The natural material is always more or less oxidized, and the actual composition lies between UO_2 and U_3O_8 $(U_{3-4}O_8)$ with U^4 usually predominant.

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Typical Analyses:

Component	Per Cent		
	1	2	3
CaO	1.01	0.35	1.72
MnO	0.03	0.14	0.13
PbO	10.95	11.69	16.71
MgO	0.08	0.06	0.06
$(Y, Er)_2O_3$	2.14	0.73	1.19
$(Ce, La)_2O_3$	1.88	0.06	0.28
La_2O_3	-	-	-
CeO_2	-	-	-
UO_2	39.10	-	-
U_3O_8	-	86.16	64.86
UO_3	32.40	-	-
ThO_2	10.60	0.10	13.94
ZrO_2	-	-	-
SiO_2	0.19	0.21	0.37
Al_2O_3	0.09)	0.35	0.12
Fe_2O_3	0.43)		0.75
CO_2	-	-	-
H_2O	0.70	-	not det.
Insol.	0.15	-	-
Rem.	0.31	-	-
Total	100.06	99.85	100.13

1 - Wilberforce, Haliburton Co., Ontario. Remnant is He and other gases.

2 - Lac Pied des Monts, Saguenay district, Quebec.

3 - Winnipeg River area, southeastern Manitoba.

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NOTE: Thorium dioxide values in uraninite specimens seem to fall within the range 0.00% - 13.94%. Generally speaking, the thorium dioxide value will be much less than 3.00%.

Mineral: monazite

Formula: (Ce, La, Y, Th)(PO_4)

Typical Analyses:

Component	Per Cent
CaO	0.35
MgO	0.02
Ce_2O_3	22.63
(La, Nd)$_2$O$_3$	34.63
Y_2O_3	4.66
Fe_2O_3	0.08
Al_2O_3	0.10
ThO_2	7.32
SiO_2	1.54
P_2O_5	27.89
H_2O	0.40
Ign. Loss	-
Rem.	0.65
Total	100.27

Accessory minerals in the ore beds include zircon ($ZrSiO_4$), leucoxene (titanium rich alteration product), anatase (TiO_2), rutile (TiO_2), feldspars and titaniferous magnetite (Fe_3O_4 + Ti). Sulphides present in

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variable but generally minute quantities include pyrrhotite ($Fe_{1-x}S$) ($x = 0 \longrightarrow 0.2$), chalcopyrite ($CuFeS_2$), galena ($PbS$), sphalerite ($ZnS$), molybdenite ($MoS_2$), marcasite ($FeS_2$ - like pyrite), and cobaltite ($CoAsS$).

The mineralogical characteristics of the ore conglomerates in the Elliot Lake district are remarkably uniform. The ratio of pebbles to matrix has been estimated to be 65:35, and the composition of the conglomerates of the Quirke Lake Trough has been estimated to be as follows:

Quartz pebbles	40-65%
Sulphides	2-8 %
Sericite	5-20%
Matrix quartz	15-25%

Variations in the proportions of the radioactive minerals are shown by the wide variation in the ratio of thorium to uranium: Th/U varies from bed to bed, ranging from 0.1:1 to 5:1. However, the uranium content of the orebodies is quite constant and averages between 2.0 and 2.5 lbs. U_3O_8 per ton. In a typical ore conglomerate, the quartz pebbles are closely sized and range from 0.5 to 1.5 or 2 inches in diameter. They are usually moderately rounded to sub-angular and are loosely to tightly packed. The pebbles consist mainly of silica; white and grey pebbles are predominant, but pebbles of pink quartz, chert and jasper also occur, as well as occasional pebbles of feldspar and granite. The matrix consists of poorly sorted granules and silt-sized particles of quartz, sericite, feldspar and some chlorite. In addition, pyrite, radioactive minerals and other minerals are present here. Except for rare occurrences in fine pyritized fractures in the pebbles, most of the radio-activity is present in the matrix, and the radioactive minerals

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usually occur in the more richly pyritized areas of the matrix.

Brannerite, uraninite and monazite are responsible for most of the radioactivity in the conglomerate. Brannerite, a metamict uranium titanate, is believed to be the most abundant ore mineral in the conglomerates, although either uraninite or monazite may predominate locally. The brannerite occurs as discrete, irregular aggregates of fine prismatic crystals that are often intergrown with quartz as well as with anatase and/or rutile. The average size of the aggregate is approximately 65 to 100 mesh and the maximum size is 20 to 28 mesh. Some of the brannerite also occurs as fine stringers and disseminations in the quartz-sericite matrix.

Uraninite is present as subhedral grains, brecciated grains and remnant crystals that range in size from 48 to 325 mesh, with an average size of about 150 mesh. Some intergrowths of uraninite and brannerite are present.

Monazite, a rare-earth phosphate containing thorium and a minor amount of uranium, occurs as rounded grains with an average size of approximately 65 mesh and typically contains very fine inclusions of pyrite. The monazite is believed to account for most of the thorium and rare-earth contents of the radioactive conglomerate ores.

Several other radioactive minerals have been noted in the conglomerate ores, including zircon, hydrocarbon ("thucholite"), coffinite, pitchblende and uranothorite, but these minerals contribute only a very small proportion of the total radioactivity of the ores.

Pyrite is the main sulphide mineral in the matrix, pyrrhotite is less abundant and other sulphides are rare. The pyrite is present as

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euhedral to subhedral grains and in aggregates; the individual masses are usually between 14 and 100 mesh in size. The concentration of pyrite in the matrix varies considerably from place to place.

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Milling Procedures

NOTE: Appendix B of this report contains a detailed flowsheet (excluding crushing and grinding) of the Denison Mines Limited concentrator.

The Denison Mines Limited treatment plant utilizes the conventional acid-leach, ion-exchange, precipitation process for the recovery of uranium in the form of an ammonium diuranate (yellowcake) salt.

Crushing and Grinding

The crushing plant was first operated on April 1, 1957. Owing to the rugged nature of the terrain, it was necessary to build the crushing and treatment plants in separate locations.

Primary and secondary crushing is done on the surface in a building near the main production shaft. Run-of-the-mine ore is discharged to a coarse ore bin at the crusher house. The ore is fed (500-600 tons per hour) onto a crossbelt by vibrating pan feeders and proceeds to a 42" x 58" jaw crusher. From the jaw crusher, the ore passes through a 7-foot standard cone crusher and from there to a set of 5/8 inch 5' x 10' screens. Part of the ore is removed as fist-sized pebbles to be used in the grinding plant (in 1960, the Company converted to autogenous grinding units in its primary grinding plant. This change-over cut costs by eliminating the need for steel grinding balls and by reducing acid consumption in the leaching phase of uranium treatment). The oversize from the screens passes to a 7-foot short-head cone crusher which is in closed circuit with the screens. The screen undersize moves on, by conveyor, to the grinding plant, where it is stored in a 15,000-ton catenary fine-ore bin. The fine-ore bin discharges to two

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identical grinding circuits, each of which has a 3,000 ton per day capacity. *That is, from the fine-ore bin, constant weight feeders deposit the ore on belts that eventually feed it to two 10-1/2' x 14' rod mills. The rod-mill discharge flows by gravity to four 12' x 32' rake classifiers. The classifier sands are washed into four 10-1/2' x 13' ball mills, and the mill discharge is returned to the classifiers. The classifier overflow, which is approximately 35% solids (65% - 200 mesh), has a specific gravity of 1.280. This pulp is pumped through a 3,000 foot pipeline to the hydro-metallurgical plant, situated at an elevation of 225 feet above the grinding plant. The material enters the plant as a slurry, and is dewatered in the neutral thickeners, in preparation for leaching.

*This description refers to 1957 operation. The circuit was modified in 1960 to accommodate autogenous grinding.

Leaching

As stated above, as a prelude to leaching, the ground ore is dewatered in thickeners, where Separan is added from time to time to flocculate the slimes, thus aiding in the settling and subsequent filtering. More water is extracted on the neutral disc filters. The cake from these filters, with about 17% water, is discharged to a conveyor belt, then to repulp tanks where the cake is mixed with hot recycle solution to form a slurry having about 32% solution and 62% solids. The recycle solution is acidic, containing approximately 45 grams per litre of sulphuric acid (and 5-6 grams per litre of U_3O_8). From the repulp tanks, the slurry is pumped to the head of one of two identical sets of six steel pachucas (and associated equipment).

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The process of leaching uranium ore, as practiced by Denison Mines Limited, is by the use of sulphuric acid in an oxidizing medium.

Dilute sulphuric acid by itself will leach some of the values from the ore, but the extraction is low and an oxidizing agent is required to improve recovery. The oxidation process itself is slow. Air, blown through the mass, is the primary agent of oxidation. A constant temperature of approximately 170°F is attained by sparging raw steam into the pulp. A residence time of 40-50 hours is achieved by forcing series operation of the pachucas. Although air and steam are added to each pachuca, sulphuric acid is only added to the lead pachucas of each set. Sufficient acid (75-80 lbs per ton ore) is added at the head of leach to maintain about 45 grams per litre at the end of leach. Only part of the acid is consumed in leaching, but a large excess is required to maintain good uranium extraction. Because of this, acid disc filters are used to recover the recycle solution, which is re-used to acidify the fresh incoming ore. The solution left in the acid filter cake (15% moisture) is the only part of the leach liquor that is further processed. The remaining filtrate is the recycle solution.

> <u>NOTE</u>: During leaching, iron in solution is, initially, in the lower oxidation state as ferrous iron; it is then oxidized to the higher state and becomes ferric, and it is this ferric iron that oxidizes the uranium. The uranium, originally in the tetravalent state is oxidized to the hexavalent state. In the hexavalent state the uranium is soluble in dilute acids.

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Preparation for Ion-Exchange

After leaching, the pulp is diluted to about 55% solids. Lime is added to neutralize the free acid to about one gram per litre. The resultant pH is 2.0 (the optimum level for processing through the ion-exchange plant).

After lime addition, the pulp is pumped to the outside wash thickeners to separate the uranium-bearing solution from the barren rock particles. This solution ("unclarified pregnant solution") is clarified by being filtered through banks of clarifier leaves. The dewatered slurry is filtered and washed twice to remove any traces of uranium. The solids are then neutralized and discharged from the mill as tailings. The "clarified pregnant solution" (carrying dissolved iron and other elements) is pumped to the ion-exchange plant.

Ion Exchange

The clarified pregnant solution, containing not more than 3 parts per million suspended solids is pumped to the adsorption columns in the ion-exchange plant. The adsorption columns are filled with a finely divided artificial resin which adsorbs uranium ions. The discharge from these columns is termed "barren solution" and is either neutralized (with lime) and sent to waste, or is directed to a rare earth recovery circuit.

The reaction that takes place on the resin during adsorption may be represented by the following equation, in which R stands for resin:

$$2R_2SO_4 \quad + \quad UO_2\ (SO_4)_3^{-4} \quad \longrightarrow \quad R_4UO_2 \quad + \quad 2SO_4^{-2}$$

resin in sulphate form	uranium trisulphate complex	uranium on resin	sulphate going to waste

The uranium-bearing resin is moved to an elution column, where ammonium (nitrate + sulphate) solution, acidified with nitric acid, is pumped through the column and dissolves the uranium from the resin by replacing the uranium with the nitrate radical as follows:

$$R_4UO_2\ (SO_4)_3 + 4NO_3^{-1} \longrightarrow UO_2\ (SO_4)_3^{-4} + 4RNO_3$$

The resultant solution, called "strong eluate", contains a high concentration of uranium and very few impurities. This solution is pumped to the precipitation tanks.

Precipitation

Lime is first added to the strong eluate to achieve a pH of 1.8. This in turn forces the precipitation of calcium sulphate.

$$Ca(OH)_2 + H_2SO_4 \longrightarrow CaSO_4 + 2H_2O$$

Ammonia gas is then sparged into the tank raising the pH to 3.8. The result is the precipitation of ferric iron as the hydroxide.

$$Fe_2\ (SO_4)_3 + 6NH_3 + 6H_2O \longrightarrow 2Fe(OH)_3 + 3(NH_4)_2SO_4$$

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The calcium sulphate and ferric hydroxide solids are separated from the purified solution in a thickener. The thickener overflow, called "pure eluate", is pumped to the uranium precipitation circuit where ammonia gas is sparged into the solution until a pH of 6.8 is attained. At this pH, uranium is almost completely precipitated from solution as an ammonium diuranate ($(NH_4)_2 U_2O_7$) (a bright yellow solid) which is then pumped to the uranium thickener. Here, the precipitate settles out and is filtered, dried and packed in drums for shipment.

The precipitation reaction is as follows:

$$6NH_3 + 6H_2O + 2UO_2SO_4 \longrightarrow (NH_4)_2 U_2O_7 + 2(NH_4)_2 SO_4 + 3H_2O$$

The uranium-free solution overflowing the uranium thickener, called "weak eluate", is pumped to make-up tanks, where nitric acid is added to make a fresh eluate solution, which is used to dissolve the uranium from the next batch of uranium-loaded resin in the elution columns.

<u>NOTE</u>: Going into the drier, the yellow cake contains about 50% moisture, 94% of which is removed in the drying process. The temperature in the drier is held at 280-300 degrees Fahrenheit, and is under instrument control. The dry precipitate is discharged into a 20,000 lb. storage bin.

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Operating Schedule: (1968)

Mine:	No. of employees	374
	Hours per day	2 eight-hour shifts
	Days per week	5-6
Mill:	No. of employees	118
	Hours per day	24
	Days per week	7

Production

Year	Tons of Ore Broken	Tons of Ore Milled	Avg. Tons Milled per day	% Recovery	Pounds U_3O_8 Produced	Avg. lbs. U_3O_8 per ton
1957	621,444	612,676	2,676	89.71	1,353,947	2.63
1958	1,897,870	1,861,229	5,101	92.37	4,239,761	2.46
1959	2,189,316	2,046,250	5,672	93.55	4,916,108	2.56
1960	2,021,643	2,013,846	5,787	93.04	4,911,761	2.70
1961	2,043,946	2,033,483	5,827	93.18	5,379,168	2.85
1962	1,808,011	1,828,993	5,680	93.11	4,844,259	2.88
1963	1,661,397	1,586,600	4,444	94.72	5,078,760	3.34
1964	1,232,742	1,275,384	3,573	95.57	3,950,364	3.14
1965	-	889,391	2,624	-	2,561,164	-
1966	-	981,709	2,922	-	2,748,602	-
1967	-	1,219,461	3,416	-	3,549,000	3.07
1968	-	1,316,000	4,000	-	3,843,000	3.07

NOTE: Yttrium recovery began early in 1967. Output is estimated at approximately 200,000 pounds per year.

Ore Reserves (January 1967)

Orebody stated to contain 300,000,000 lbs U_3O_8

At end 1967 readily accessible reserves - - - 3,800,000 tons.

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Water Supply - Sources and Quantities

a) Benner Lake Supply

Water for approximately 30 homes in the townsite is pumped continuously from an infiltration well located at Benner Lake.

Pumping rate: 166 imperial gallons/minute

Benner Lake Supply - At Pumphouse

pH	Chloride as Cl	Iron as Fe	Alkalinity as $CaCO_3$	Hardness as $CaCO_3$
7.3	14	0.40	14	42

b) Shaft No. 1 Supply

All water required in the shaft #1 area is withdrawn from Quirke Lake.

Pumping rate: 498 imperial gallons/minute

Quirke Lake Supply #1 - At Pumphouse

pH	Chloride as Cl	Iron as Fe	Alkalinity as $CaCO_3$	Hardness as $CaCO_3$
7.0	6	0.45	10	80

c) Mine Supply (principal)

Shaft No. 2 and concentrator areas are supplied with water from Quirke Lake.

Pumping rate: 1,743 imperial gallons/minute

Quirke Lake Supply #2 - At Shop Building

pH	Chloride as Cl	Iron as Fe	Alkalinity as $CaCO_3$	Hardness as $CaCO_3$
7.6	7	0.70	17	104

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NOTE: Most of the mining activity in the next few years will be concentrated in the northeast area of the property. At present (Nov.1, 1968) mining process water is supplied to this block from No.1 shaft. In order to accomplish this, a long delivery line, four main booster pumps in series, and a system of storage sumps has been set up to negotiate the difference in elevation of nearly 1,060 feet between the No.1 shaft station and the most northerly headings.

It has been proposed by Denison Mines Limited to discontinue this system and supply the mine with process water by gravity from Quirke Lake via a drill hole on Puhky Island. The proposed flow rate is 800 gpm.

Water Consumption

An insite into the present water consumption by the mine-mill complex can be gained from the flows quoted under the title "Water Supplies - Sources and Quantities".

Originally Denison Mines Limited discharged mine water (seepage, process water etc.) directly to Quirke Lake.

Mine Water #2 Shaft - June 25, 1959

Solids				Iron as Fe		Hardness as $CaCO_3$
Total	Susp.	Diss.	pH	Total	Ferrous	
604	90	514	4.5	13	4.0	195

In 1961, Denison Mines Limited diverted the mine water from Quirke Lake and directed it to the grinding circuit.

In 1964, water lines in the mill were rerouted so that up to

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1.5 million gallons per day of water originally discharged to the tailings area was being re-used in the mill in place of fresh water pumped from Quirke Lake. This action effectively removed approximately 2,240 imperial gallons per minute from the tailings watershed.

At the present time the <u>maximum</u> anticipated overflow from Long Lake is in the range of 2,500 imperial gallons per minute (this figure includes flow due to spring run-off). An additional 200 imperial gallons per minute will probably be contributed from the east Stollery Lake drainage area.

Denison Mines Limited reports a normal flow from the tailings area of 2.35 million imperial gallons per day.

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TAILINGS DISPOSAL - CHRONOLOGICAL HISTORY

1958

The original intention at Denison Mines Limited was that tailings from the mill were to discharge to Bear Cub Lake (Jousey Lake) and the effluent from there was to go to Long Lake (Denison Lake) and thence to Stollery Lake and on to the upper Serpent River. However, until the main tailings line was completed to this site, another Lake (Norman) nearby was used. The effluent from this lake discharged into what is now known as Smith or Williams Lake. Williams Lake in turn discharged into a small creek that followed another route to the Serpent River. The secondary tailings area was used for more than five months before the tailings line into the Bear Cub Lake area was in place and being used.

1959

The tailings line (a 14" diameter wooden stave pipe) continued to discharge into Smith Lake. A large portion of the waste flowed to the west but a dam was to be completed across the centre of Smith Lake in order to force the tailings to back up and fill Bear Cub Lake and the low surrounding area. This area was to become the permanent tailings basin until maximum capacity was reached. A dam had been completed at Bear Cub Lake to help retain the mill wastes.

1960-1961

Late in 1960 the capacity of Bear Cub Lake as a tailings basin had been reached and tailings began to enter Long Lake which lies south of Bear

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Cub Lake. The tailings line was relocated to a point just east of Smith Lake, and, as a result, the tailings now began to flow via a gulley to Bear Cub Lake and thence into the eastern extremity of Long Lake for settling.

In June of 1961, the tailings dam at the western end of Bear Cub Lake failed near the decant well and a considerable volume of settled tailings escaped to the centre portion of Long Lake. Flow is designed to go to the eastern end of Long Lake.

> <ins>NOTE</ins>: Without exception during the period 1960-1965, all analytical data secured by the Ontario Water Resources Commission concerning the Denison Mines Limited tailings area effluent must be considered non-representative. The sampling point has generally been the Serpent River as it cuts the Denison Road. In order to reach this point, the tailings area wastes have to pass through approximately 4,000' of Long Lake, 2,000' of Stollery Lake and 1,500' of the Serpent River itself. Stollery Lake is fed both by Cinder Lake and by Dunlop Lake.
>
> Although the Serpent River sample point may serve as a general indicator of conditions prevailing in the area, it is not a specific indicator and therefore it is not a good indicator.

1964

Construction of a rock dam between Long Lake and Stollery Lake was initiated late in 1964. The dam was designed to raise the level of Long Lake by 7.0 feet and hence provide an increased settling area.

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Because of the necessity of clarifying surface rights in the Long Lake area, the dam and sluiceway were not fully completed until 1966.

In October 1964, a small earth dam across a narrow bay at the west end of Long Lake was completed. The dam diverted the outflow of Cinder Lake directly into Stollery Lake and therefore assured that the natural drainage of approximately 1,000 acres would not pass over the tailings in Long Lake.

During 1964, 1965, and part of 1966, barren and solid wastes from the mill were directed to the Long Lake basin.

1966

On June 1, 1966, barren effluent from the mill which was formerly discharged to the clear water area of Long Lake was diverted to discharge directly to Stollery Lake. A small earth dam was constructed in the bush north of Long Lake to effect the barren effluent diversion.

By October 1966, barium chloride treatment facilities had already been constructed at the Long Lake decant system.

The Company then proposed to build a 1,100' long dam across Stollery Lake to settle any precipitated radium compounds from the Long Lake decant. The depth of water in the construction area is 20-21 feet.

1967

Late (November 13) in 1967 the solid tailings effluent and barren effluent were recombined to discharge to Long Lake. Barium chloride treatment continued.

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1968

The combined mill effluent continued to be discharged to the Long Lake basin. Construction of the Stollery Lake dam commenced and was well underway by September 1968. Completion of the project is expected sometime in 1969.

NOTE: Appendix C of this report contains analytical data (1958-1967) relevant to the waste discharges at Denison Mines Limited.

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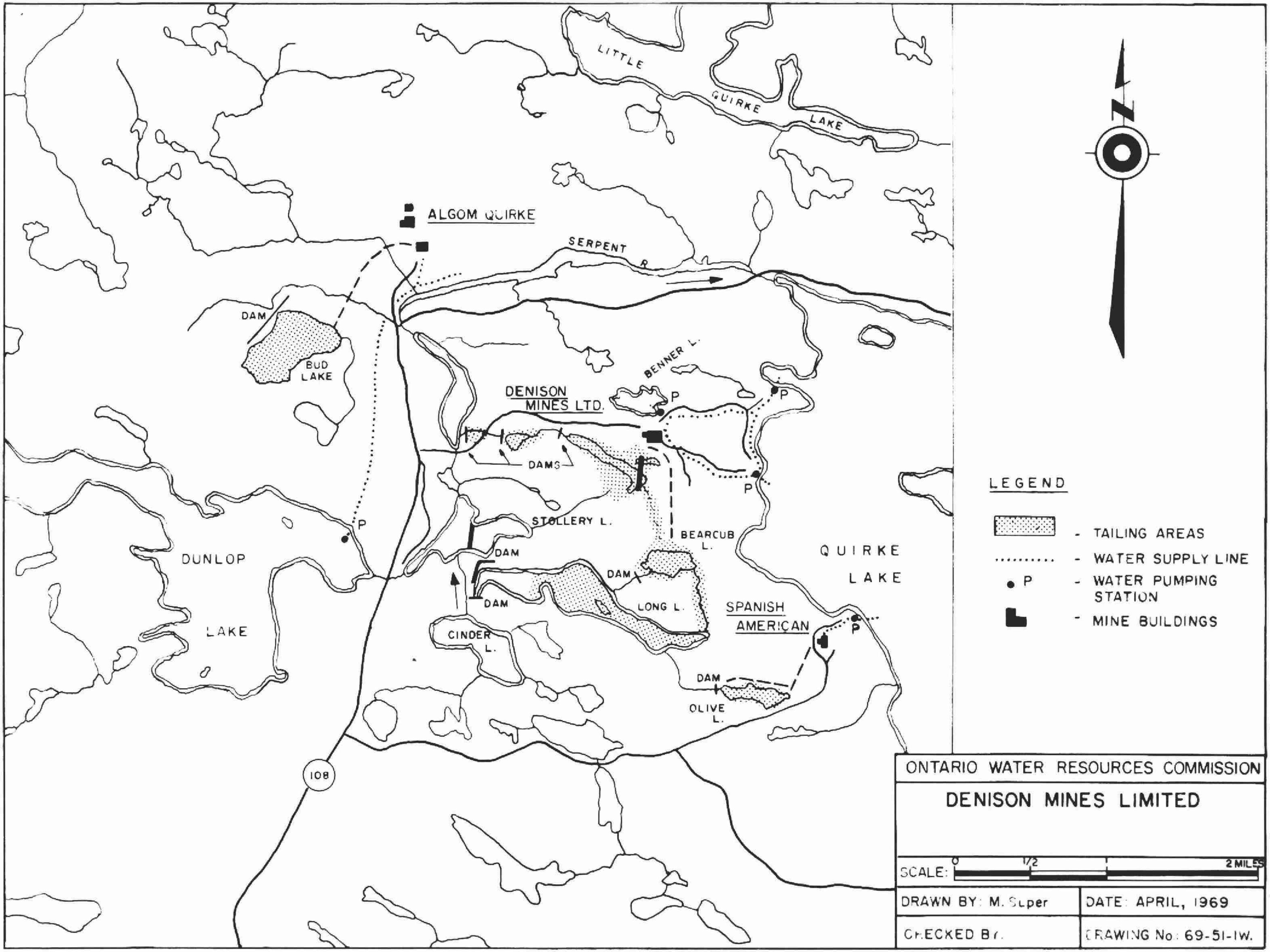
LITTLE QUIRKE LAKE
ALGOM QUIRKE
SERPENT R.
DAM
BUD LAKE
BENNER L.
DENISON MINES LTD.
P
P
DAMS
P
STOLLERY L.
P
DAM
BEARCUB L.
DUNLOP LAKE
DAM
LONG L.
QUIRKE LAKE
DAM
CINDER L.
SPANISH AMERICAN
P
DAM
OLIVE L.
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LEGEND
- TAILING AREAS
- WATER SUPPLY LINE
• P - WATER PUMPING STATION
- MINE BUILDINGS
ONTARIO WATER RESOURCES COMMISSION
DENISON MINES LIMITED
SCALE: 0 1/2 1 2 MILES
DRAWN BY: M. Super
DATE: APRIL, 1969
CHECKED BY.
DRAWING No: 69-51-IW.

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Sampling and Analysis

On August 12, 1968, the following grab samples were taken at the Denison mill:

1) Intake water from Quirke Lake
2) Recycle water
3) Barren solution to tails
4) Solids to tails

All analytical results are appended to this report. All results (except pH) are expressed in parts per million (ppm).

In addition, a sample of each of the above was sent to the Ontario Department of Health for radiological analysis (see Appendix D).

Analytical results based on a sample (Long Lake Overflow after $BaCl_2$ addition) taken by officials of Denison Mines Limited and dated September 26, 1968 are appended. The sample unfortunately was not large enough to permit radiological analysis.

WASTE LOADINGS (Gross)

Flow assumed - 2.35 mgd

NOTE: See Appendix C for analytical results "Top of Dam Between Long and Stollery Lakes".

The following waste loadings should be regarded only as a general indication of the quantity of the various contaminants that escape the Denison tailings areas. The concentrations (ppm) of the various substances mentioned below were arrived at by taking a straight arithmetic average of the available (1966-1968 - "Top of Dam Between Long and Stollery Lakes")

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analytical results. Erratic or unusual values were rejected.

The Commission realizes that, in the past, there have been intermittent periods of negligible flow from the Long Lake basin.

i)	Suspended solids	-	avg. conc.	8.7 ppm
			gross loading	204 lbs/day
ii)	Sulphates as $SO_4^{=}$	-	avg. conc.	1385 ppm
			gross loading	32,500 lbs/day
iii)	Total phosphorus	-	avg. conc.	.08 ppm
			gross loading	2 lbs/day
iv)	Total Kjeldahl nitrogen	-	avg. conc.	59.0 ppm
			gross loading	1380 lbs/day
v)	Iron as Fe	-	avg. conc.	4.9 ppm
			gross loading	115 lbs/day
vi)	Manganese as Mn	-	avg. conc.	1.4 ppm
			gross loading	30 lbs/day
vii)	Chlorides as Cl^{-}	-	avg. conc.	20.0 ppm
			gross loading	470 lbs/day

General Discussion

Since intensive field work with regard to the tailings areas at Denison Mines Limited is planned for the next field season, the analytical findings of this preliminary report will not be commented on in any great detail.

Radiological considerations are discussed in Appendix D. The high levels of "dissolved" gross Alpha and gross Beta radiation present in the barren solution indicates that, with regard at least to water pollution,

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the disposal of the barren solution must be handled with the utmost caution.

An indication of the potential pollution threat posed by the Denison Mines Limited tailings areas can be gained by the simple fact that, during the past three years, over 78 per cent of samples taken of the Long Lake overflow indicated a waste that was not suitable for discharge to a watercourse because of pH considerations. The low pH was due, in part, to the various methods of waste disposal that were being instigated on an experimental basis at the Denison property.* Recent analytical results are more encouraging.

* The Company reported that neutralizing the tailings to Long Lake (barren to Stollery) resulted in increased radium values in the Long Lake effluent. Thus, until barium treatment was provided, it was decided best not to neutralize the tailings, but to over neutralize the barren to Stollery so that the Radium would not be leached from the tails in Long Lake and the pH leaving Stollery would be acceptable.

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Conclusions

i) For the last three years (1966-1968) the suspended solids concentration of the Long Lake overflow has been acceptable for discharge to a natural watercourse during 96 per cent of the total time. During 1968, the suspended solids concentration was entirely acceptable for discharge to a natural watercourse.

ii) During the last three years (1966-1968) the overflow from Long Lake has been acceptable for discharge to a natural watercourse, with regard to pH, for only 22 per cent of the total time. During 1968, this acceptability factor rose to 40 per cent.

It is apparent that the overflow from Long Lake tends to be acidic in nature and as such the effluent from Long Lake is only periodically acceptable for discharge to a natural watercourse.

Remarks

Denison Mines Limited must assume complete responsibility for all waste discharges resulting from past, present, and future mining and milling operations. This responsibility will outlive the actual mine itself.

It is quite probable that, in the future, as the tailings bodies on the Denison property age, their rate of production of acids and soluble salts will be accelerated. The result could be a heightening of the already intolerable burden that presently faces the Quirke Lake environment.

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Recommendation

Denison Mines Limited should actively investigate the rehabilitation of the various sections of their tailings area. The limited extent of some of the tailings basins makes them ideal experimental plots for revegetation etc.

With this in mind, the Ontario Water Resources Commission will conduct an extensive field survey of the Denison tailings areas during the next field season. All drainage networks will be traced in detail. Tailings samples will be taken for nutrient analysis and will aid in the determination of the relative movement of contaminants within the tailings bodies themselves.

Prepared by:

J. R. Hawley
J. R. Hawley,
Mining Technician,
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Approved by:

R C Stewart
R. C. Stewart, P. Eng.,
District Engineer,
Division of Industrial Wastes.

JRH/vs

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ONTARIO WATER RESOURCES COMMISSION
CHEMICAL LABORATORIES

INDUSTRIAL WASTE ANALYSIS

All analyses except pH reported in p.p.m. unless otherwise indicated

1 p.p.m. = 1 mgm. / litre
= 1 lb./100,000 Imp. Gals.

Municipality: Elliot Lake **Report to:** J.R. Hawley * **c.c.** Chem. Lab.-*

Source: Denison Mines Limited

Date Sampled: Aug. 12/68 **by:** J.R.H.

br

Lab. No.	5-Day B.O.D.	Solids			pH at Lab.	Sulphate as SO_4 *	Iron as Fe		Nitrate as N	Nitrite as N	Hardness as $CaCO_3$	Alkalinity as $CaCO_3$	Acidity as $CaCO_3$	Chlorid as Cl
		Total	Susp.	Diss.			Tot.	Diss.						
T 2503		198	3	195	6.1	95	1.00	0.65	-	-	102	33	3	10
T 2504		4912	81	4831	8.3	1810	1.50	0.55	1.8	0.292	1812	113	0	64
T 2505		33222	46	33176	2.1	14800	36.5	5.00	0.07	0.017	2400	0	17750	3
T 2506		565180	-	-	3.4	1930	37000 *	** 0.02*	3.1	0.094	1844	118	280	41

* Performed in Chem. II

** Test performed on filter supernatant only

T 2503 1. Intake water from Quirke Lake Grab 10:40 a.m.

T 2504 2. Recycle water - Duplicate Grab 10:00 a.m.

T 2505 3. Barren solution to tails - duplicate Grab 10:30 a.m.

T 2506 4. Solids to tails Grab 10:50 a.m.

One sample of each with covering sheet sent to Radiation Protection Lab., Dept. of Health, 360 Christie St. at request of K. Shikaze.

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ONTARIO WATER RESOURCES COMMISSION
CHEMICAL LABORATORIES

INDUSTRIAL WASTE ANALYSIS

All analyses except pH reported in p.p.m. unless otherwise indicated

1 p.p.m. = 1 mgm. / litre = 1 lb./100,000 Imp. Gals.

Municipality: Elliot Lake **Report to:** J.R. Hawley * **c.c.** Chem. Lab.* /rd

Source: Denison Mines Limited

Date Sampled: Sept. 26/68 **by:**

Lab. No.	5-Day B.O.D.	Solids Total	Solids Susp.	Solids Diss.	pH at Lab.	Sulphate as SO_4	Iron Tot.	Iron Diss.					
T-3343		382	2	380	7.2	193	0.36	0.16					

Lab. No.	Nitrate as N.	Nitrite as N.	Hardness as $CaCO_3$	Alkalinity as $CaCO_3$	Chloride as Cl.	Acidity as $CaCO_3$
T-3343	35.	0.176	214	22	8	4

T-3343 1. Long Lake Overflow after B_aCL_2 addition.

APPENDIX A

CAN-MET EXPLORATIONS LIMITED

The Can-Met mine is on the south shore of Quirke Lake about 16 miles by road from the town of Elliot Lake.

Two shafts were sunk on the property. The first was a service shaft that bottomed at 2,126 feet while the second, a three-compartment shaft, terminated at a depth of 2,395 feet. Both shafts were connected by a haulageway at the 2,100 foot horizon.

A three thousand ton daily capacity mill was constructed on the property. Plant construction was begun on July 31, 1956, and completed on August 15, 1957. Owing to miscellaneous delays, production did not begin until October 21, 1957. The initial rate was 900 tons of ore per day. The first shipment of uranium precipitates was made in December (1957). Delays were encountered in shaft-sinking owing to water conditions and the scarcity of highly skilled shaft-sinking crews. Later, underground development was held up due to a badly faulted orebody.

Ore reserves were reported by the Company in 1959 to be 6,642,380 tons of partly proven ore with an average grade of 0.092% U_3O_8(10% dilution allowed for). The total tonnage, including all categories of ore, was reported to be 8,362,069 tons (Company prospectus, April 2,1958).

The Company received a contract from Eldorado for the sale of precipitates valued at $75,852,000. This was later increased to $79,670,000, and the delivery time was extended from March 31, 1962, to March 31, 1963.

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NOTE: In October 1959, Can-Met announced that underground rock conditions made cost-cutting procedures less effective and that a low treatment rate resulted in higher unit costs. Soon thereafter it was announced that ore reserves were insufficient to fulfill contract commitments, and on March 24, 1960, the Company amalgamated with Consolidated Denison Mines Limited to form a new company:- Denison Mines Limited.

Mining operations at Can-Met ceased in April, 1960 (milling ceased April 15, 1960), and the remainder of the Can-Met contract was transferred to Denison.

The mill was sold in June 1962.

Geology

The Can-Met deposit is on the north limb of the Quirke Lake syncline. It is bordered on the west by Denison, on the north by Panel and on the south by Stanrock. The depth of the ore (which does not outcrop on the property) is from 1,500 feet in the north to 2,000 feet near the shafts. The average strike is north 40 degrees west, with dips ranging between 0 and 15 degrees south. The average thickness of the ore is 17 feet, but widths of up to 25 feet have been encountered. Extensive faulting occurs throughout the orebody.

The orebody, which consists of two beds of uraniferous quartz-pebble conglomerate, is in one of the fringe areas of the main ore channel situated on the north side of the Quirke Lake Trough. The beds rest on the basement rocks at the north end and 70 feet above the basement at the south end.

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A sample of ore sent to the Mines Branch in 1956 was found to contain brannerite, monazite, and uraninite. Possible by-products were reported by a Company official to be thorium and rare-earth minerals. The ratio of uranium to thorium was said to be 2:1. Large pockets of pyrrhotite, pyrite, and some chalcopyrite have been encountered in the ore bed.

> NOTE: The ore was recovered utilizing the conventional room-and-pillar method of mining. As such, 30% of the ore remains in place and can probably be recovered at a future date. The pillars are about 20 feet square.

Milling

The treatment plant went into operation on October 21, 1957, at an initial rate of 900 tons per day. The first shipment of yellowcake was made on December 9, 1957 and in September 1958 the mill reached its capacity of 3,000 tons of ore per day.

The plant was of the conventional acid-leach, ion-exchange, precipitation type. During the early part of 1959, recovery was 93%; mill heads were 0.103% U_3O_8. The precipitate was shipped directly to the United States.

All tailings were discharged to the tailings basin on Stanrock property.

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Production

The following, taken from the Company's annual report for 1958, is a summary of production for the period October 16, 1957, to September 30, 1958:

Tons ore broken	- 541,296
Tons ore hoisted	- 531,898
Dry tons milled	- 535,466
Average daily tonnage	- 1,530
% extracted	- 91.71
% recovered	- 90.83
Pounds U_3O_8 produced	- 826,250
Average mill head (%U_3O_8)	- 0.092

To September 30, 1959, the mill handled 1,477,160 tons of ore and recovered 2,495,709 pounds of U_3O_8. Revenue from May 1, 1958 to September 30, 1959 totalled $23,517,448. Production figures covering the period October 1, 1959, to April 1960 are not available. Can-Met did however report that, up to the end of 1959, a few months prior to closure, it had produced 4,719,000 pounds of U_3O_8 out of its contract total of 7,710,600 pounds.

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APPENDIX B

KEY

MILL FLOW SHEET

DENISON MINES LIMITED

Designation	Function
A-1	tank
A-2	thickener
A-3	thickener
A-5	thickener
A-6	thickener underflow storage
A-7	tank
A-8	distributor

NOTE: Recycle water from A-7

a) directed to grinding plant
b) used as make-up water for lime slurrying
c) used as wash water for 2nd stage filtration
d) used for reagent make-up

Designation	Function
B-1	distributor
B-2 (3)	disc filter
B-6 (7)	disc filter
B-12(13)	repulper
B-16(17)	repulper
C-1	pachuca
C-2	pachuca
C-3	pachuca
C-4	pachuca
C-5	pachuca
C-6	pachuca

Designation	Function
C-9	pachuca
C-10	pachuca
C-11	pachuca
C-12	pachuca
C-13	pachuca
C-14	pachuca
	<u>NOTE</u>: air and steam are blown into all pachucas
	<u>NOTE</u>: recycle pulp from C-6(14) to C-1(2)(9)(10) has been abandoned at least on a temporary basis.
C-18	tank
C-24	tank
C-25	tank
C-26	tank
	<u>NOTE</u>: Lime added to C-24 to achieve pH of 2.0
C-35	pump box
D-1	thickener
D-2	thickener
D-4	unclarified pregnant storage
D-5	cyclones (2)
E-27	tank
E-28	tank
E-29	tank

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Designation	Function
F-24	clarified pregnant storage
F-25	wash solution
F-26	fresh eluant
F-27	strong eluant
	NOTE: after resin transferral to adsorption from elution, the first 30,000 gallons becomes "royal barren" solution and is sent to tank A-1.
G-1	thickener
G-2	thickener
G-3	tank
G-4	tank
G-5	tank
G-7	tank
G-8	tank
G-9	tank - NH_3 added to achieve pH 6.8 (final ppt'n.)
G-11	tank - lime added to achieve pH 1.8 (sulphate control)
G-12	tank
G-13	tank - NH_3 added to achieve pH 3.8 (iron control)
G-16	tank
G-34	tank
H-1	filter
H-2	filter
H-3	filter

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REAGENT CONSUMPTION

DENISON MINES LIMITED

1958 — Concentrator + Grinding Plant

Reagent	Daily Consumption lbs/ton ore
sulphuric acid	85.9
sodium chlorate	3.6
nitric acid	4.41
magnesium oxide	1.26
calcium oxide	49.6
glue	.46
Separan	.05
steel balls	2.53*
steel rods	1.55
water	1,500 tons/day in concentrator only.

*NOTE: primary grinding plant converted to autogenous grinding units in 1960.

1960-61 — Concentrator

Reagent	Total Daily Consumption
sulphuric acid (93%)	180 tons
burnt-lime rock (92%)	90 tons
nitric acid (60%)	19 tons
sodium chlorate (99%)	2.5 tons
ammonia (liquid)	2 tons
bone glue	0.7 tons

NOTE: In 1968 the daily production of U_3O_8 was 10,000 - 12,000 lbs. The ore grade was apparently under 3.0 lbs. U_3O_8 per ton. An average ton of ore consumed 76 lbs. H_2SO_4 and 6.6 lbs. HNO_3.

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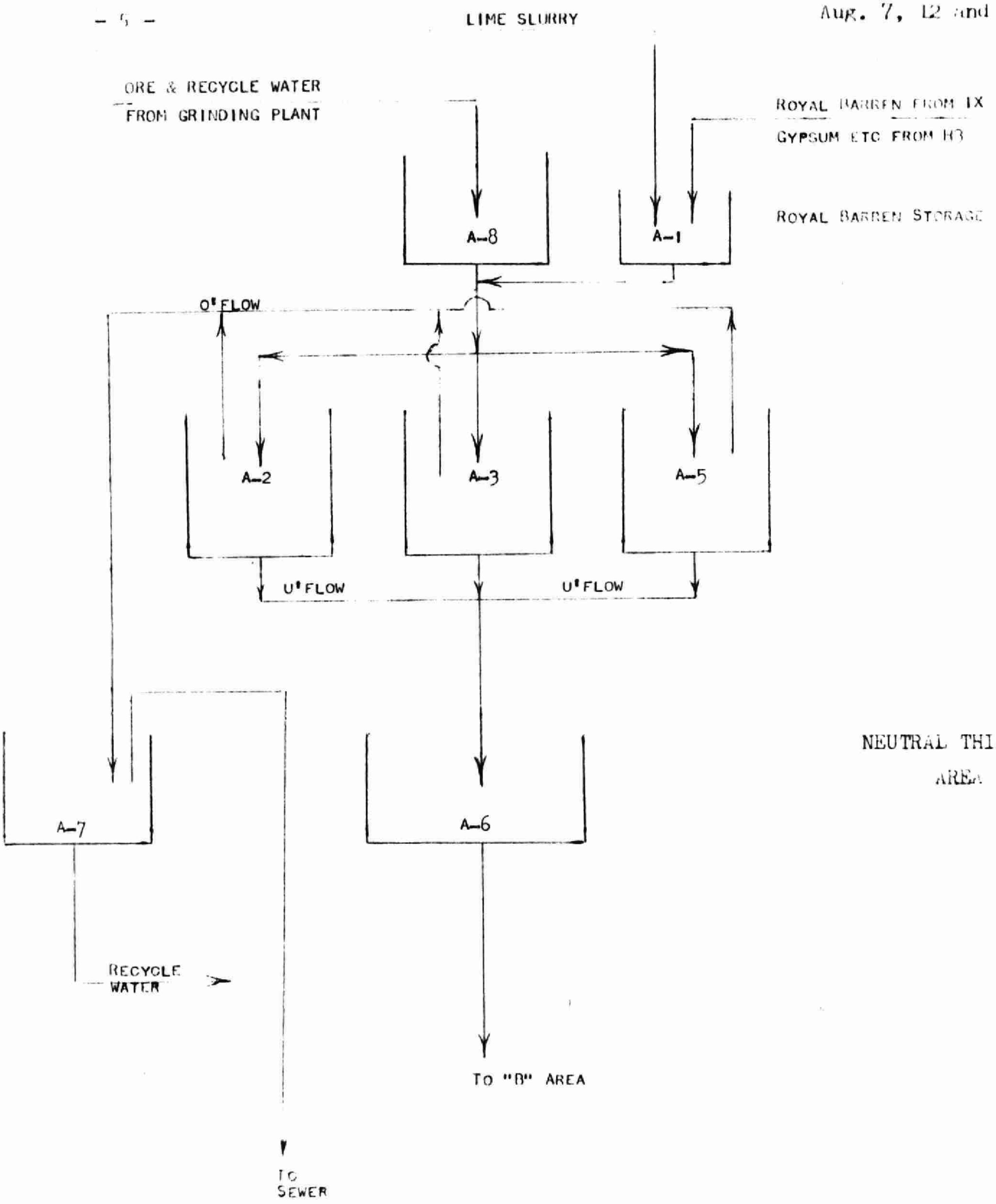

NEUTRAL THICKENER AREA

Mill Circuit
Denison Mines Limited

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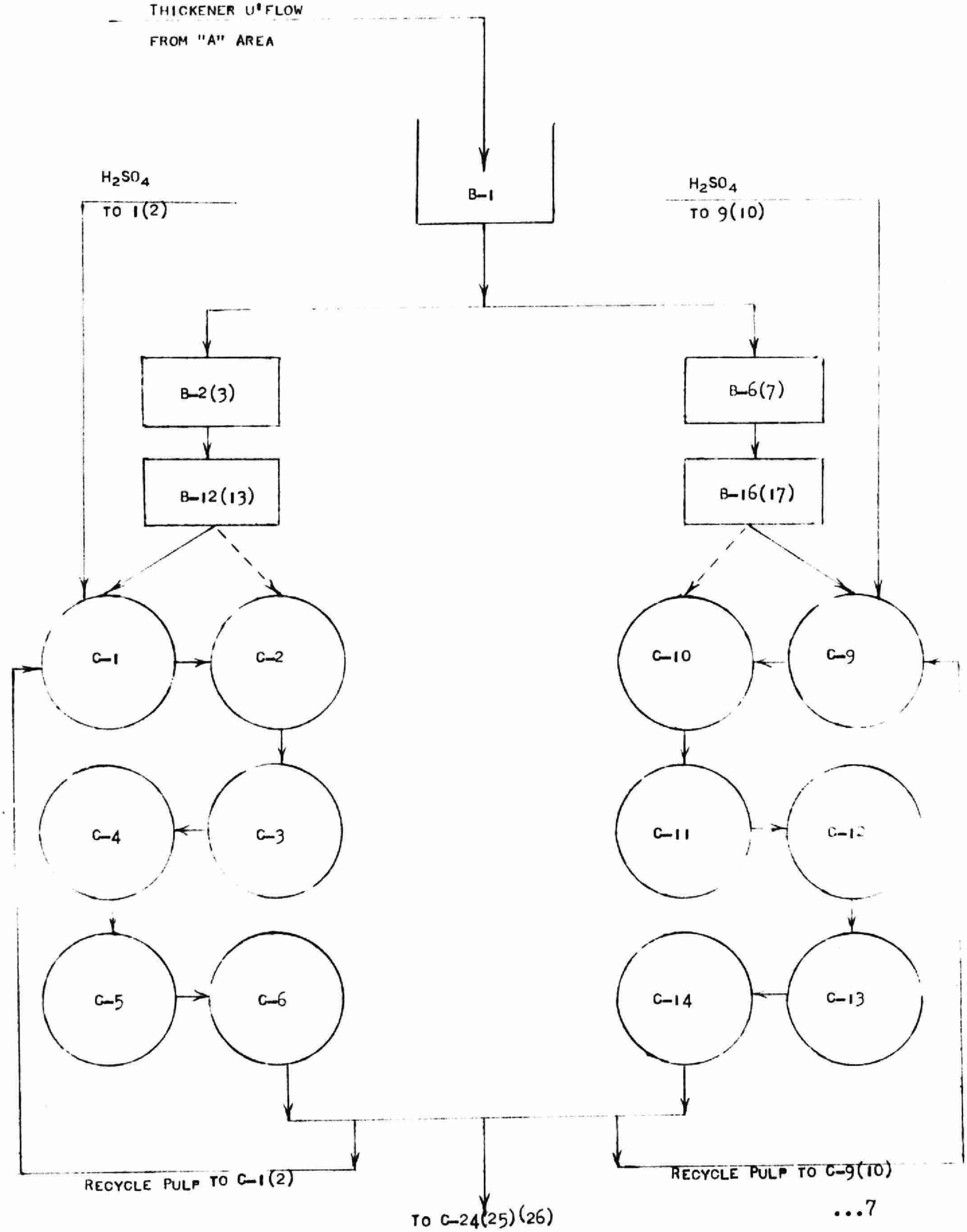
ACIDIFICATION CIRCUIT



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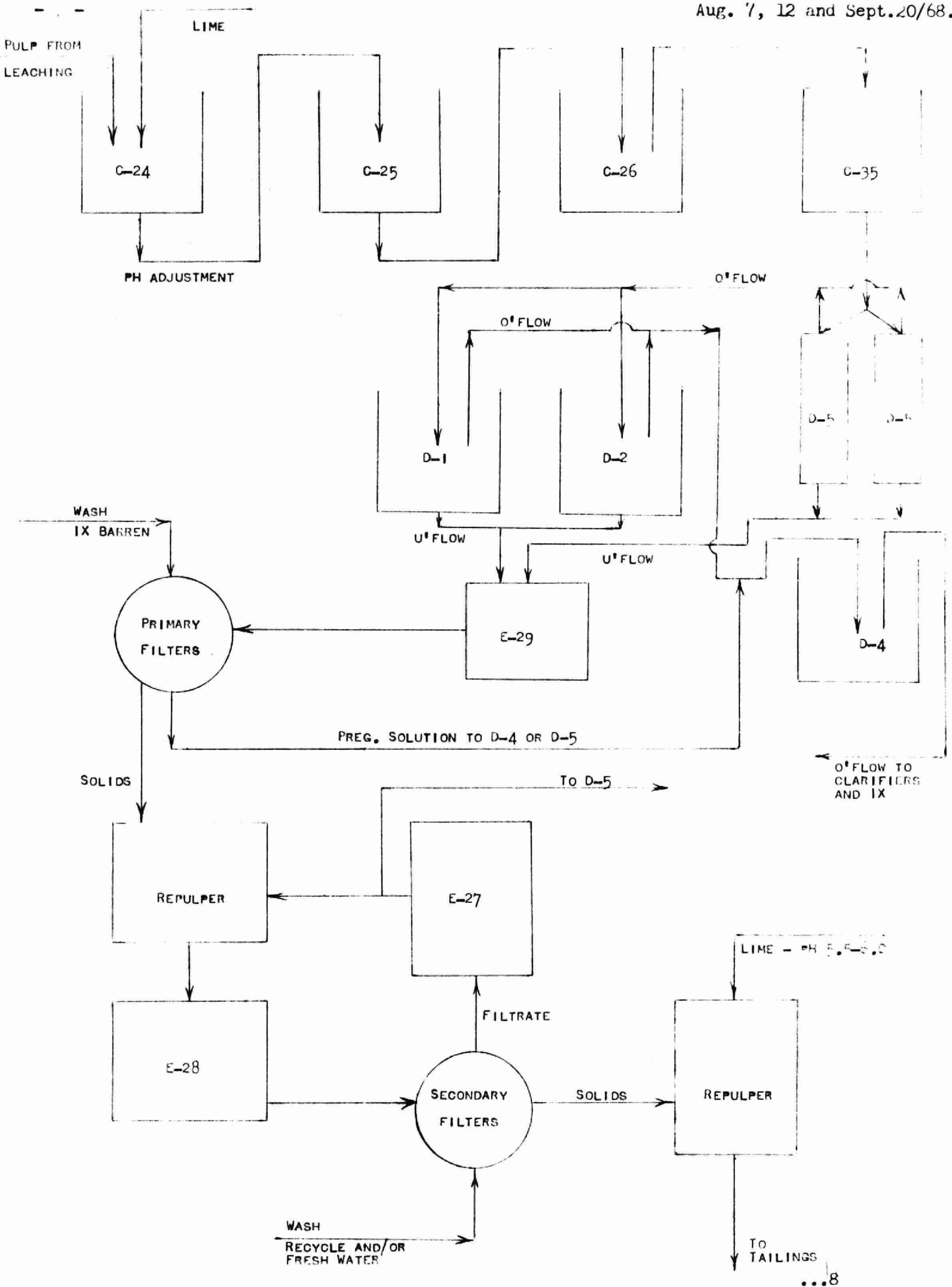
LIME
PULP FROM LEACHING
C-24
C-25
C-26
C-35
PH ADJUSTMENT
O'FLOW
O'FLOW
D-5
D-5
D-1
D-2
WASH
IX BARREN
U'FLOW
U'FLOW
PRIMARY FILTERS
E-29
D-4
PREG. SOLUTION TO D-4 OR D-5
O'FLOW TO CLARIFIERS AND IX
SOLIDS
TO D-5
REPULPER
E-27
LIME - PH 5.5-6.0
E-28
FILTRATE
SECONDARY FILTERS
SOLIDS
REPULPER
WASH
RECYCLE AND/OR FRESH WATER
TO TAILINGS

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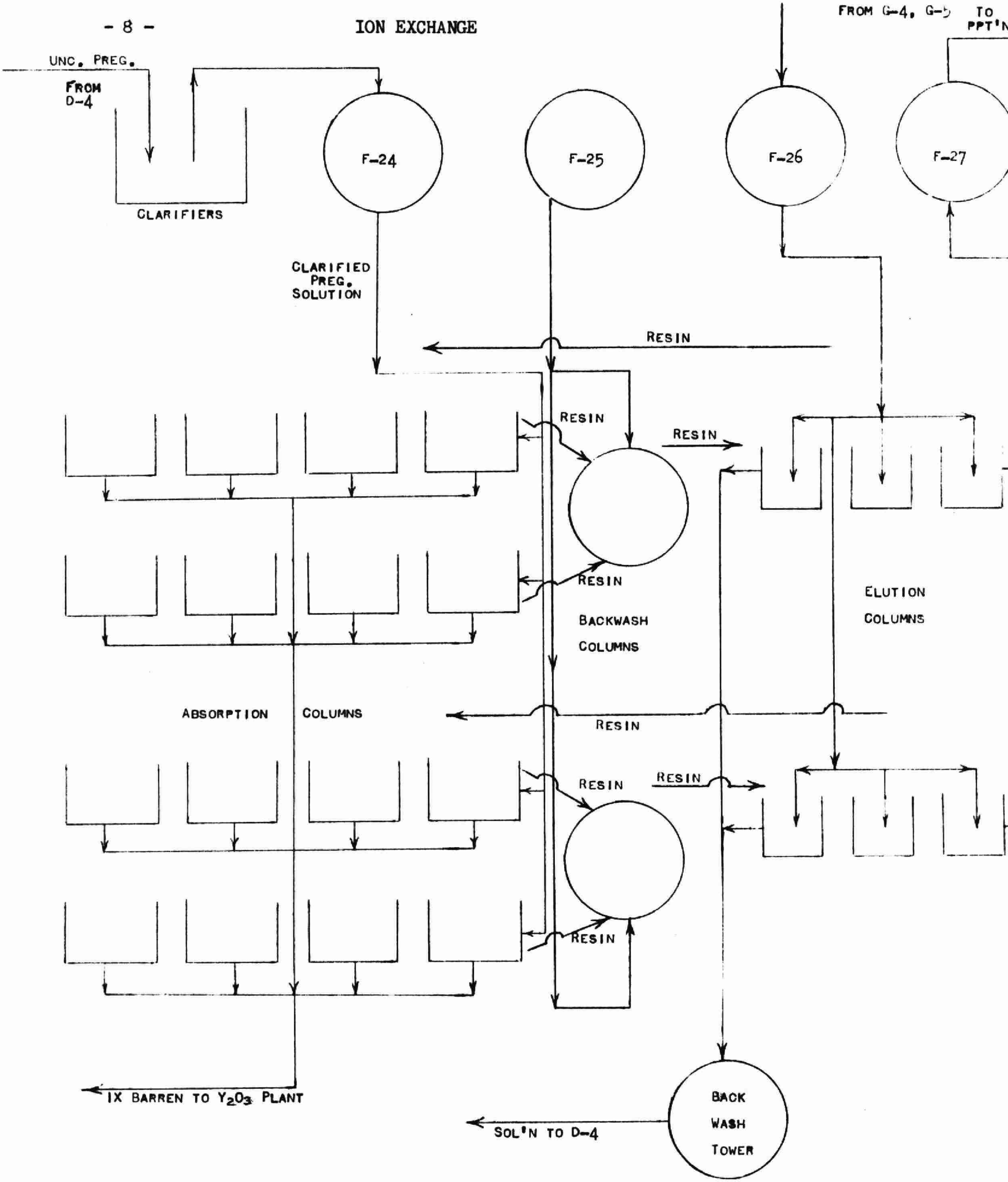
ION EXCHANGE



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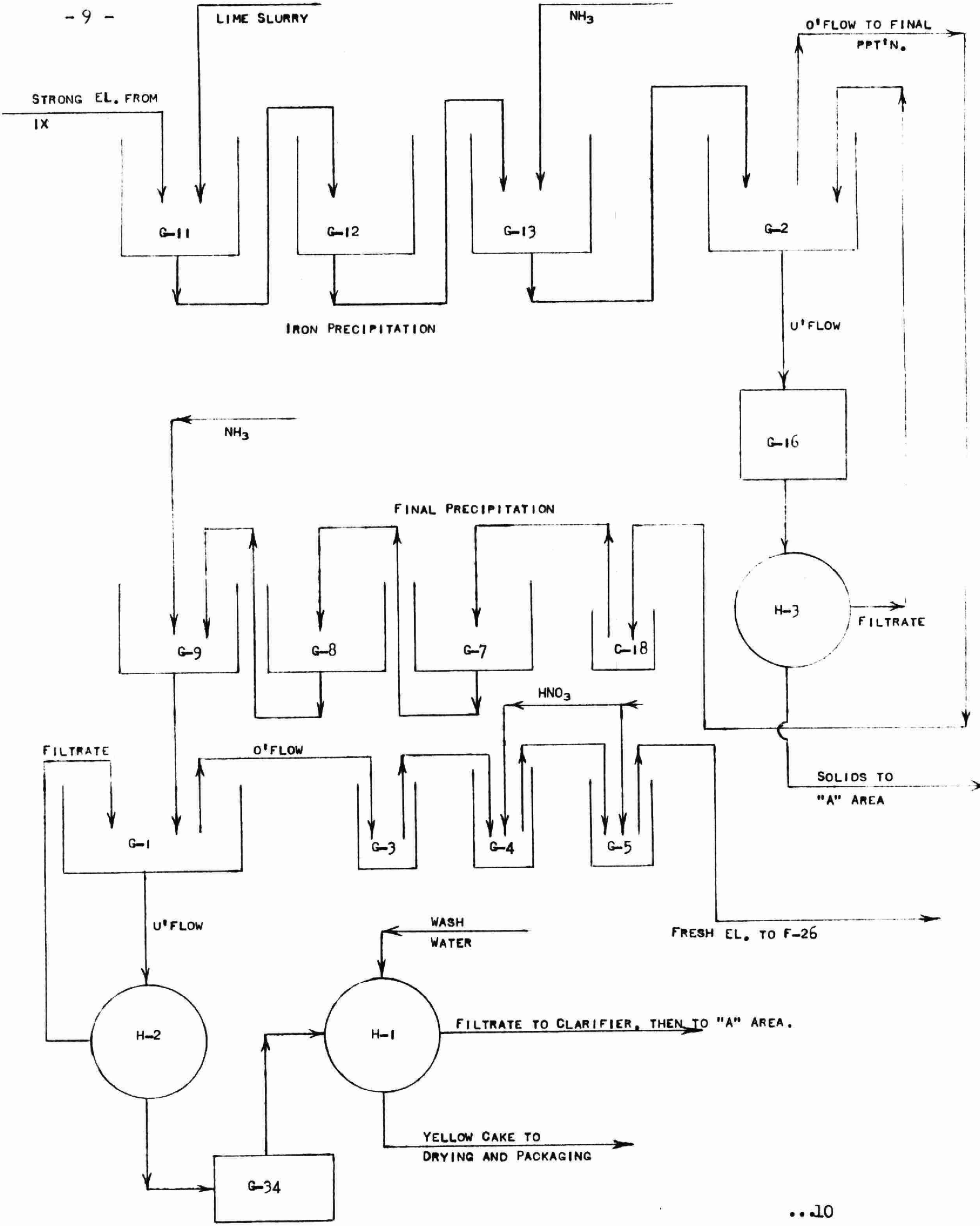

STRONG EL. FROM
IX
LIME SLURRY
NH$_3$
O'FLOW TO FINAL
PPT'N.
G-11
G-12
G-13
G-2
IRON PRECIPITATION
U'FLOW
G-16
NH$_3$
FINAL PRECIPITATION
H-3
FILTRATE
G-9
G-8
G-7
C-18
HNO$_3$
FILTRATE
O'FLOW
G-1
G-3
G-4
G-5
SOLIDS TO
"A" AREA
U'FLOW
WASH
WATER
FRESH EL. TO F-26
H-2
H-1
FILTRATE TO CLARIFIER, THEN TO "A" AREA.
YELLOW CAKE TO
DRYING AND PACKAGING
G-34

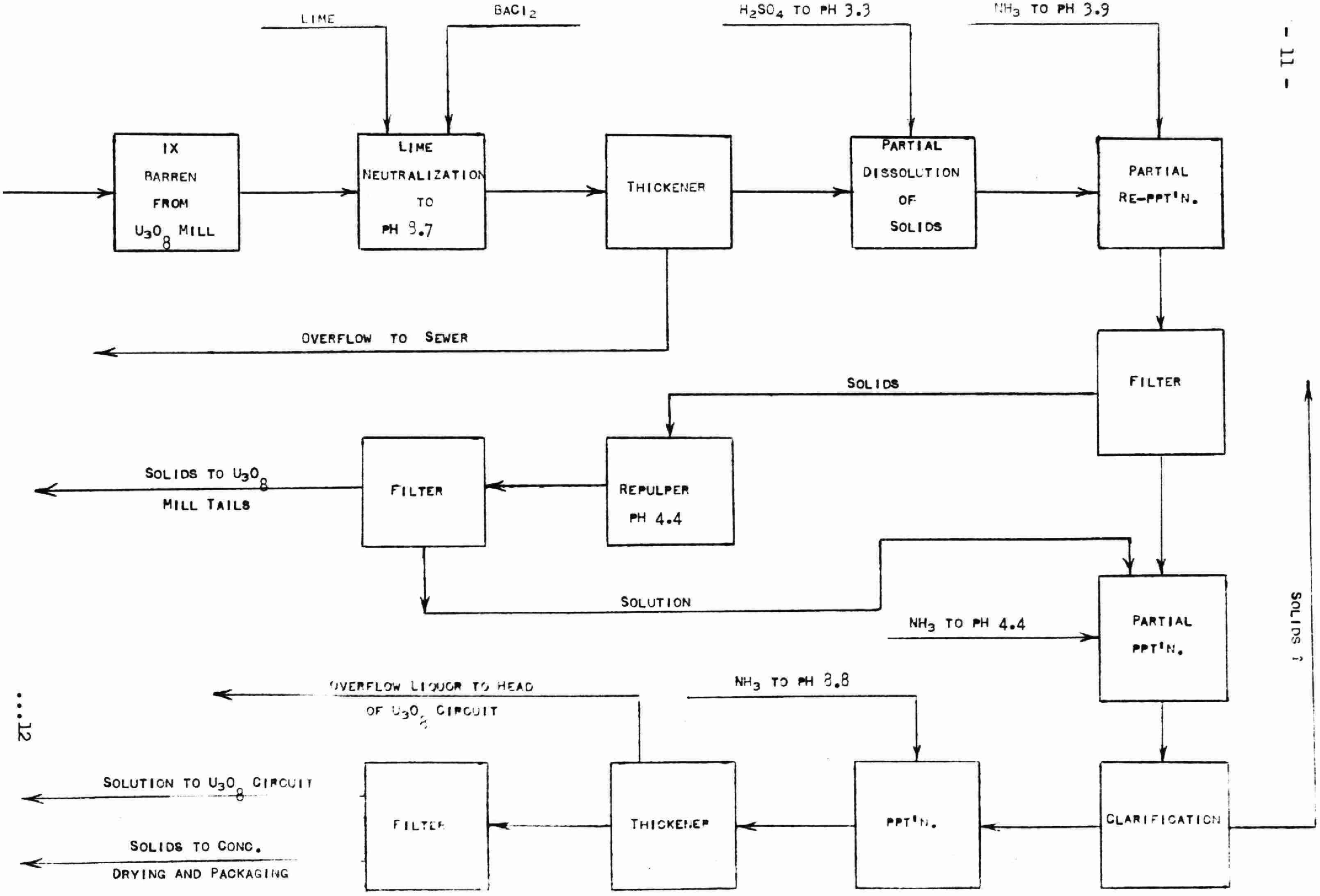
...10

YTTRIUM RECOVERY CIRCUIT

In late 1966, as previously stated in the main body of the report, Denison Mines Limited signed a sales contract with the Michigan Chemical Co. and with Molybdenum Corp. of America for the delivery of approximately 300,000 lbs. of yttrium oxide annually. As a result, an yttrium oxide recovery circuit was constructed in the area made available by the removal of the original set of leaching agitators. Yttrium recovery was first scheduled to begin in January 1967. Production was initially reported to be 200,000 lbs. annually.

The following is the original flowsheet (Dated May 18, 1967) of the yttrium recovery circuit:

...11



Y_2O_3 FLOWSHEET

...12

The above yttrium circuit was to remain in effect until August 1967 at which time further specific changes would be introduced to the circuit. However, due to delays in construction and due to process problems the required changes did not take place in August and the new process was not actually on stream until October of 1967. Until the month of December 1967 the barren effluent was only intermittently processed. In January of 1968 the circuit was shut down for most of the month because of operating difficulties. In February 1968 the circuit was kept operating for a reasonable percentage of the time. On August 12, 1968 Company officials indicated that the circuit had been down again for approximately two months.

The new direct process consists of removing some of the impurities from the yttrium bearing solution by partial neutralization with slurried lime. The precipitated impurities or solid tailings are removed by filtration, and the filtrate or prime solution is fed to a modified solvent extraction process. The organic phase consists of di-ethyl hexyl phosphoric acid (DEHPA) carried in kerosene. After stripping the yttrium from the prime solution, the loaded organic phase is stripped with diluted nitric acid. The loaded nitric acid (pregnant solution) is then treated with ammonia to precipitate the yttrium. The precipitate is separated by filtration and dried and packaged. The filtrate, because of contained yttrium values, is combined with the prime solution and fed back to the solvent extraction circuit. The prime solution, after the contained yttrium is removed, becomes raffinate or tailings solution, and is treated with lime. The solid tailings from the process are mixed with the solid tailings from the U_3O_8 circuit, and the combined tailings are treated with lime. The solution tailings are treated

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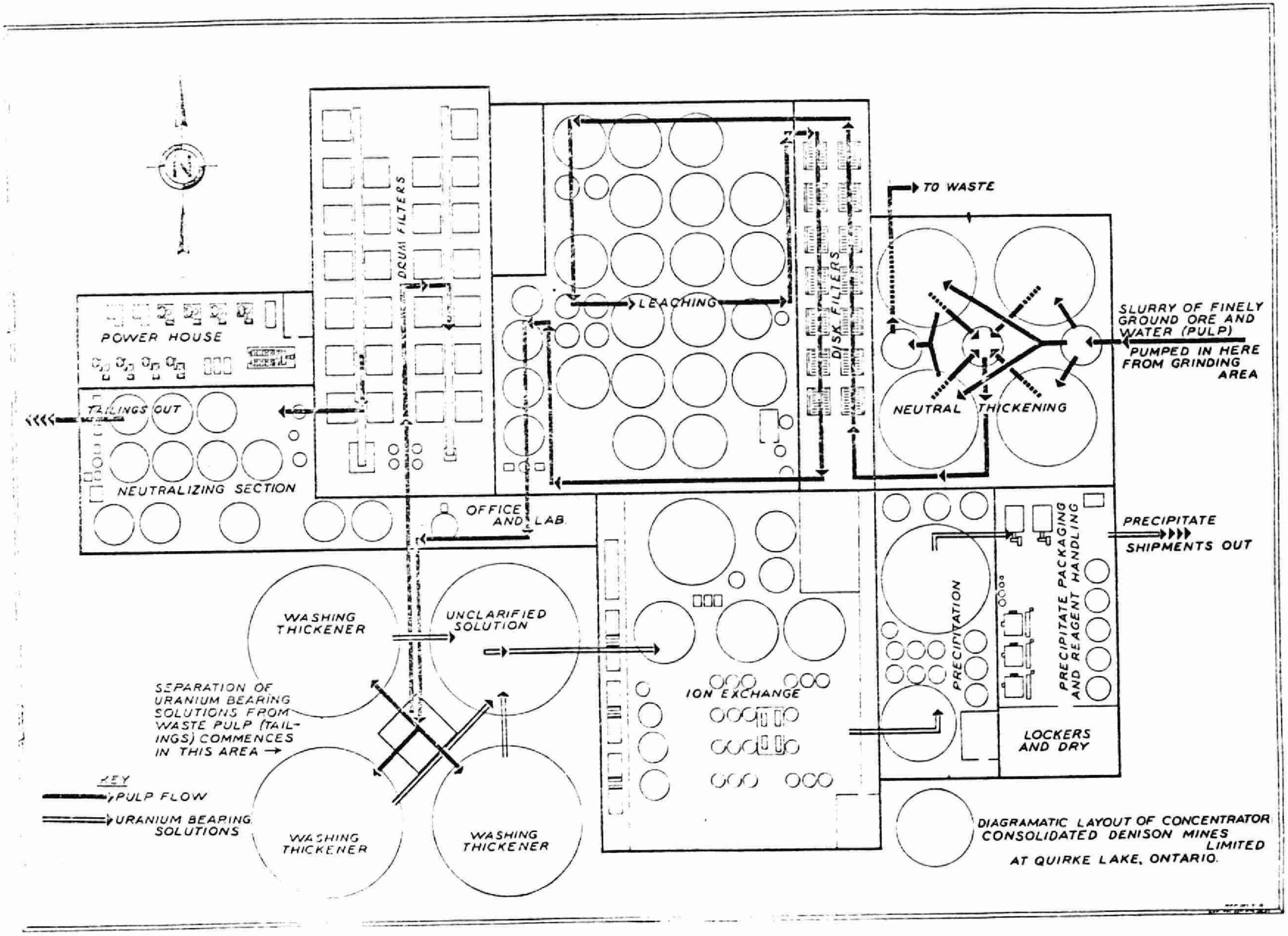
with lime and then combined with any barren solution coming from the U_3O_8 circuit before going to the tailings disposal area.

As might be expected yttrium oxide production during the year (1968) was less than scheduled because of the market oversupply problems experienced by the principal yttrium refining companies. Apparently, Denison Mines Limited commenced proceedings against Michigan Chemical Corp. claiming substantial damages for breach of contract for the purchase by Michigan Chemical Corp. of yttrium oxide concentrates.

...14



DIAGRAMATIC LAYOUT OF CONCENTRATOR
CONSOLIDATED DENISON MINES LIMITED
AT QUIRKE LAKE, ONTARIO.

(1957 - Before Concentrator Modification)

APPENDIX C

NOTE: All results in Appendix C (except pH) are expressed in parts per million (ppm)

Tailings Area Decant - Smith Lake Basin

DATE		SOLIDS				IRON			
Year	Month	Total	Susp.	Diss.	pH	Total	Fe^{++}	Acidity	Nitrates
1958	December	2,834	52	2,782	5.3	5.0	-	0.1	30.0
1959	March	3,520	126	3,394	5.4	1.6	-	4.0	-
1959	March	1,880	30	1,850	3.8	50.0	-	121	-
1959	April	2,962	20	2,942	5.1	12.0	-	30	-
1959	April	2,782	70	2,712	5.6	4.6	-	14	-
1959	April	1,378	46	1,332	4.9	154	60	220	-
1959	April	3,680	26	3,654	5.7	5.6	0.4	19	-
not stated	-	3,702	270	3,432	4.9	8.0	1.4	31	-
1959	May	3,480	230	3,250	4.7	30.	1.0	27	-
1959	May	3,902	30	3,872	4.7	1.4	0.0	6.0	-
1959	June	3,380	20	3,360	6.1	0.72	0.0	3.0	-
1959	June	3,726	22	3,704	6.2	0.88	0.0	1.6	-
1959	June	3,646	242	3,404	6.8	0.96	0.0	2.2	-
1959	June	3,748	38	3,710	5.8	2.4	0.0	2.0	70.0

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SERPENT RIVER AT DENISON ROAD

Year	Month	Day	Total Solids	pH	IRON Total	 Fe^{++}	Nitrates	Hardness as $CaCO_3$
1960	July	27	156	6.4	0.0	0.0	1.4	118
1960	Aug.	1	342	6.6	.16	0.0	2.8	214
1960	Aug.	10	762	6.6	0.0	0.0	4.8	528
1960	Aug.	22	270	6.5	0.0	0.0	2.7	176
1960	Aug.	29	894	6.8	0.0	0.0	7.5	510
1960	Sept.	8	418	6.4	.08	0.0	5.0	270
1960	Sept.	14	692	6.3	0.0	0.0	7.5	410
1960	Sept.	21	894	6.5	0.0	0.0	10	525
1960	Sept.	28	1,180	6.4	0.0	0.0	12.5	635
1960	Oct.	3	1,248	6.4	0.0	0.0	18.0	730
1960	Oct.	13	1,004	6.0	Trace	0.0	15.0	22?
1960	Oct.	19	714	6.4	Trace	0.0	10.0	440
1960	Oct.	24	1,218	6.8	0.0	0.0	20.0	800
1960	Nov.	3	592	6.5	0.0	0.0	10.0	385
1960	Nov.	7	920	6.5	0.0	0.0	15.0	585
1960	Nov.	17	528	6.4	0.0	0.0	8.0	324
1960	Nov.	21	396	6.5	.46	0.0	8.0	236
1960	Nov.	28	272	6.5	0.0	0.0	.15	172
1960	Dec.	7	556	6.4	0.0	0.0	10	380
1960	Dec.	12	214	6.4	0.10	0.0	2.8	130
1960	Dec.	22	398	6.5	0.20	0.0	4.4	252
1960	Dec.	30	852	6.3	0.44	0.0	13.0	500
1961	Jan.	3	778	6.7	0.4	0.0	9.0	490
1961	Jan.	19	728	6.8	0.32	0.0	8.0	560
1961	Jan.	23	698	7.6	0.24	0.0	9.0	500
1961	Jan.	30	634	6.1	0.44	0.0	9.5	430
1961	Feb.	6	738	6.4	0.64	0.50	9.0	530
1961	Feb.	14	998	5.7	0.56	0.20	11.0	590

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Serpent River at Denison Road (continued)

Year	Month	Day	Total Solids	pH	IRON		Nitrates	Hardness as $CaCO_3$
					Total	Fe^{++}		
1961	Feb.	23	1,072	4.9	0.64	0.0	10.5	860
1961	Mar.	3	1,044	4.9	0.92	0.0	9.0	690
1961	Mar.	6	986	4.7	1.0	0.0	14.0	690
1961	Mar.	10	984	5.1	1.08	0.4	1.2	700
1961	Mar.	24	1,114	4.6	0.96	0.2	13.0	640
1961	Mar.	25	30	5.9	0.0	0.0	0.18	22
1961	April	10	494	6.1	0.48	0.4	6.5	6.1
1961	April	24	706	5.9	0.64	0.0	11.5	5.9
1961	April	20	632	6.1	0.68	0.0	5.5	6.1
1961	May	5	436	6.4	0.44	0.0	6.5	240
1961	May	8	422	6.4	0.44	0.0	7.0	240
1961	May	18	286	5.9	1.0	0.0	1.2	152
1961	May	22	490	6.0	0.0	0.0	5.4	240
1961	May	31	654	8.5	Trace	0.0	5.8	250
1961	June	7	426	7.3	0.0	0.0	5.5	340
1961	June	15	936	6.4	0.0	0.0	7.2	590
1961	June	22	1,108	6.2	0.0	0.0	8.0	580
1961	June	28	1,000	6.3	0.16	Trace	13.0	630
1961	July	7	856	6.9	0.72	.68	9.0	420
1961	July	13	462	7.7	0.01	0.0	7.5	236
1961	July	19	448	6.6	0.16	0.0	6.3	270
1961	Aug.	9	166	6.8	Trace	0.0	2.5	80
1961	Aug.	14	568	6.8	0.1	0.0	8.5	320
1961	Aug.	28	240	6.7	0.1	0.04	6.2	220
1961	Sept.	6	690	6.1	0.23	0.02	0.7	244
1961	Sept.	13	1,110	6.4	0.92	0.17	18.0	652
1961	Sept.	20	510	6.3	0.57	0.30	7.5	276
1961	Sept.	25	310	6.4	0.54	0.38	4.0	176

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Serpent River at Denison Road (continued)

Year	Month	Day	Total Solids	pH	IRON Total	 Fe^{++}	Nitrates	Hardness as $CaCO_3$
1961	Oct.	4	456	6.6	0.58	0.29	7.5	256
1961	Oct.	11	688	6.4	0.51	0.46	-	360
1961	Oct.	23	204	7.4	0.43	0.43	-	136
1961	Oct.	30	164	6.4	0.76	0.38	2.5	88
1961	Nov.	1	604	5.8	0.60	0.22	10.0	324
1961	Nov.	10	654	5.8	0.92	0.22	10.0	360
1961	Dec.	14	234	7.7	0.44	0.27	3.8	130
1962	Jan.	11	200	6.0	0.43	0.22	4.0	90
1962	Feb.	19	280	5.9	0.38	0.22	4.0	100
1962	March	15	514	6.0	0.48	0.16	7.5	290
1962	April	13	532	6.2	1.60	0.65	13.0	292
1962	May	14	246	6.1	0.59	0.49	5.0	160
1962	June	21	254	6.3	0.10	0.0	20.0	144
1962	July	16	856	5.3	0.15	0.05	15.0	470
1962	Aug.	10	936	5.6	0.07	0.0	20.0	478
1962	Sept.	18	1,300	6.4	0.70	0.0	25.0	680
1962	Oct.	12	2,978	6.0	2.60	0.90	30.0	1,630
1962	Nov.	16	1,378	6.0	0.43	0.27	25.0	750
1962	Dec.	17	816	5.8	0.20	0.05	20.0	460
1963	Jan.	14	536	6.5	0.24	0.16	13.0	290
1963	Feb.	-	412	6.5	0.20	0.10	6.0	208
1963	March	-	624	6.4	0.10	0.08	10.0	500
1963	April	11	320	6.7	0.10	0.06	7.5	154
1963	May	-	222	6.6	0.11	0.09	5.0	130
1963	June	-	624	6.6	0.54	0.0	13.0	340
1963	July	3	590	6.7	0.12	0.0	13.0	332

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Serpent River at Denison Road (continued)

Year	Month	Day	Total Solids	pH	IRON Total	IRON Fe^{++}	Nitrates	Hardness as $CaCO_3$
1963	Aug.	-	1,848	3.1	0.24	0.0	40.0	940
1963	Sept.	16	2,138	6.2	0.10	0.05	50.0	1,090
-	-	-	-	-	-	-	-	-
1963	Nov.	15	2,528	6.0	0.10	0.03	63.0	1,270
1963	Dec.	11	2,240	5.6	0.68	0.30	50.0	1,150
1964	Jan.	7	1,358	5.6	0.95	0.55	25	240
1964	Feb.	12	550	6.9	0.40	0.20	10	1,520
-	-	-	674	6.3	0.08	0.08	20	370
1964	April	8	680	6.4	0.16	0.05	14	380
1964	May	15	216	6.9	0.10	0.00	4	116
1964	June	15	458	6.9	0.12	0.00	6	232
1964	July	-	300	7.5	0.22	0.09	4	146
1964	Aug.	12	3,290	5.6	4.25	0.62	12.5	1,720
1964	Sept.	9	792	6.6	0.49	0.20	14.0	500
1964	Oct.	-	210	6.2	0.20	0.10	3.0	130
1964	Nov.	11	168	6.9	0.20	0.15	2.0	90
1964	Dec.	-	52	6.8	0.30	0.21	0.3	16
1965	Jan.	-	150	7.3	0.11	0.07	1.8	80
1965	Feb.	-	160	6.4	0.18	0.10	3.0	92
1965	March	-	160	-	0.10	0.08	0.12	86
-	-	-	-	-	-	-	-	-
1965	May	-	102	6.8	0.17	0.08	1.2	56
-	-	-	-	-	-	-	-	-
1965	July	-	408	6.8	0.23	0.02	10.0	208
1966	Nov.	7	338	6.6	-	-	10.0	218

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Serpent River at Denison Road (continued)

					IRON			
Year	Month	Day	Total Solids	pH	Total	Fe^{++}	Nitrates	Hardness as $CaCO_3$
1966	Dec.	5	74	6.9	0.18	-	1.5	46
1967	Jan.	2	166	6.6	0.26	-	0.0	48
1967	Jan.	30	142	6.6	0.80	-	2.50	60
1967	Feb.	27	84	-	-	-	2.00	-
1967	March	27	218	6.7	0.11	-	3.00	136
1967	April	24	-	6.6	1.30	-	-	108
1967	May	22	176	6.4	0.11	-	0.55	62
1967	June	22	386	7.2	0.11	-	0.52	166
1967	July	13	566	6.9	0.5	-	3.0	240
1967	July	18	570	5.5	0.10	-	0.50	266
1967	Aug.	15	628	7.1	0.23	-	0.20	280
1967	Sept.	12	992	6.8	0.06	-	6.00	420
1967	Oct.	9	1,238	6.5	0.16	-	2.00	554
1967	Nov.	7	548	5.1	1.45	-	1.25	316
1967	Dec.	9	184	5.2	0.60	-	0.25	110

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TOP OF DAM BETWEEN LONG AND STOLLERY LAKES

			SOLIDS		PHOSPHORUS		NITROGEN AS N										
'EAR	NO.	DAY	TOTAL	SUSP.	TOTAL	SOLUBLE	FREE NH_3	TOTAL KJELDAHL	NITRITE	NITRATE	HARDNESS AS $CACO_3$	ACIDITY AS $CACO_3$	PH	SULPHATE AS SO_4	IRON AS FE	MANGANESE AS MN	CHLORIDES AS CI
966	10	10	2,848	15	0.04	.00	36.1	52.8	.40	40.0	1,590	30	5.0	1,564	1.7	1.25	-
966	11	7	2,560	9.0	0.36	.04	-	42.9	.60	25.0	1,500	48	-	-	2.31	1.75	-
966	12	13	1,628	4.0	0.0	.00	12.6	19.0	.05	8.0	910	161	3.7	948	27.5	1.50	9
1967	1	7	1,881	13.0	0.0	0.00	13.2	14.0	0.03	8.5	1,080	14.2	3.7	1,139	19.8	1.00	8
1967	1	30	1,790	8.0	0.0	0.00	16.4	18.0	0.01	10.0	1,000	-	3.4	1,072	20.3	0.4	8
1967	2	28	2,140	10.0	0.02	0.00	15.0	19.8	0.17	20.0	-	98.0	-	1,290	-	0.0	-
1967	3	27	1,988	5.0	0.00	0.00	18.8	20.0	0.20	17.5	1,300	67.0	3.9	1,175	3.37	1.5	19
1967	4	24	-	-	-	-	-	-	-	-	540	32.0	5.1	550	2.40	-	9
1967	5	22	2,566	6.0	0.28	0.27	49.2	52.8	0.25	17.5	1,388	70.0	4.1	1,240	3.0	2.17	20
1967	6	22	2,606	14.0	0.13	0.07	19.70	59.0	-	7.00	1,458	24.0	4.1	1,620	9.50	1.85	21
1967	7	18	2,736	14.0	0.08	0.05	-	4.10	-	10.5	1,520	18.0	4.5	912	2.85	0.23	21
1967	8	15	2,956	4.0	0.23	0.005	26.2	44.0	-	1.0	1,549	132.0	3.7	1,700	0.10	1.8	11
1967	9	12	2,956	6.0	0.07	0.01	70.0	79.2	-	20.0	1,580	172.0	3.8	1,720	1.15	1.7	23
1967	10	9	2,940	12.0	0.03	0.02	36.1	55.0	-	10.0	1,528	140.0	3.8	1,700	2.82	2.2	20
1967	12	8	2,968	19.0	0.10	0.02	41.0	59.4	-	10.0	1,486	16	4.0	1,680	6.80	1.5	21
1968	1	4	892	14.0	0.10	0.01	29.5	48.0	-	10.0	1,270	58	4.5	1,240	3.70	1.7	11
1968	1	30	3,082	5.0	0.03	0.02	98.5	105.0	-	30.0	1,400	132	4.2	1,320	2.00	2.21	27
1968	2	27	2,932	< 15	0.04	0.01	312.0	165.0	-	35.0	1,650	43	4.9	1,580	0.24	1.5	30
1968	3	26	2,054	6	0.06	0.01	52.5	79.0	-	12.5	1,156	38	4.7	1,080	2.80	1.03	15
1968	4	23	3,244	5	0.05	0.01	0.02	0.05	-	0.21	1,640	42	8.2	1,200	0.35	1.4	26
1968	6	17	3,136	2	0.05	0.01	16.0	25.0	-	6.5	1,774	6	6.0	1,750	0.35	2.15	29
1968	7	18	3,140	5	0.03	0.01	[illegible]	93.0	-	10.0	1,732	106	[illegible]	[illegible]	0.77	0.42	22
1968	8	13	3,446	< 15	0.04	0.01	120.0	130.0	-	11.0	1,770	74	[illegible]	1,950	0.15	2.4	30
1968	9	12	3,074	8	0.04	0.01	100.0	130.0	-	6.1	1,676	34	4.5	1,840	3.35	1.55	27
1968	10	8	2,868	7	0.04	0.02	[illegible]	95.0	0.98	74.0	1,700	20	4.1	2,000	0.12	1.15	25

APPENDIX D

RADIOLOGICAL RESULTS - DENISON MINES LIMITED

All samples requiring radiological analysis were submitted to the Occupational Health Laboratories of the Ontario Department of Health. The Radiation Protection Laboratory is located at 360 Christie Street in Toronto, Ontario.

Sample Number	Waste Stream
T-2503	Intake Water from Quirke Lake
T-2504	Recycle Water
T-2505	Barren Solution to Tails
T-2506	Solids to Tails

ALL SAMPLES WERE TAKEN ON AUGUST 12, 1968

	Sample Number			
	T-2503	T-2504	T-2505	T-2506
Dissolved solids ppm	167	2,986	31,300	2,860
Suspended solids ppm	3	70	6	790,000
Radium-226 (dissolved) pC_i/litre	16	380	1,250	1,600
Gross Alpha (dissolved) pC_i/litre	55	1,030	286,000	3,250
Gross Alpha (undissolved) pC_i/litre	64	300	100	-
Gross Alpha (on solids) pC_i/gram	-	-	-	2,600
Gross Beta (dissolved) pC_i/litre	64	1,430	545,000	3,800
Gross Beta (undissolved) pC_i/litre	63	310	180	-
Gross Beta (on solids) pC_i/gram	-	-	-	1,900

In relation to the contamination of water by radioactive wastes, particularly radium-226, it is of interest to note that most natural waters

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have a radium-226 content of less than 1 picocurie per litre (pC_i/litre) and many contain less than 0.1 picocuries per litre. When we realize that barren being discharged to waste at Denison contains 1,250 pC_i/litre dissolved radium-226 and the "solid" tails contain 1,600 pC_i/litre dissolved radium-226, the need for continuous and precise $BaCl_2$ treatment of the combined waste is accentuated. The high levels of "dissolved" gross alpha and gross beta radiation present in the barren solution indicates that, with regard to water pollution, the barren solution must be handled with the utmost caution.

The following statements are conclusions from the 1965 Deputy Minister's Report entitled "Report on Radiological Water Pollution in the Elliot Lake and Bancroft Areas - 1965":

1) In accordance with modern scientific knowledge and opinion, the Department considers that any unnecessary exposure to radioactivity should be kept to a minimum.

2) With respect to water quality standards for radioactivity, there is no single-figure value which distinguishes between safe and harmful levels of radioactive content.

3) The Department recommends that levels of from 30 to 10 picocuries of radium-226 per litre of water be adopted as the immediate or first-level objective to be attained in those lakes and streams where present levels are in excess of this range.

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4) The Department recommends that levels of from 10 to 3 picocuries of radium-226 per litre be adopted as the immediate objective to be attained in public drinking waters in the Elliot Lake and Bancroft areas.

5) The Department does not consider that such exposure to radium-226, as would be entailed in swimming or part-time use of lakes containing between 20 and 40 picocuries per litre, presents any hazard to health.

Since 1964, the Ontario Water Resources Commission has set the objective for radium-226 in drinking water supplies at 3 picocuries per litre, and we are also attempting to reduce industrial effluents to this objective.