FEASIBILITY STUDY
KERR ADDISON MINES LIMITED
ADANAC MOLYBDENUM PROJECT

VOLUME III

FEASIBILITY STUDY

OF THE

ADANAC MOLYBDENUM PROJECT

PREPARED FOR

KERR ADDISON MINES LIMITED

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TABLE OF CONTENTS

I. INTRODUCTION

This report summarizes work accomplished by Kaiser Engineers on the proposed surface facilities for the Adanac project of Kerr Addison Mines Limited.

On August 28, 1970, Kaiser Engineers was requested to furnish capital and operating costs for a 15,000 ton per day molybdenite concentrator and ancillary facilities. Included in this work was a study to determine power costs. The report is comprised of two alternatives: the first is for conventional coarse and fine crushing and rod mill and ball mill facilities followed by flotation. The second alternative is for coarse crushing followed by primary autogenous grinding and ball milling before flotation.

The work accomplished by Kaiser Engineers has been based on the operation of a pilot plant at the Adanac mine site in northwestern British Columbia and on flowsheets provided by Britton Research Limited of Vancouver, B. C. Autogenous grinding tests are currently being run at Lakefield Research of Canada Ltd. in Lakefield Ontario.

Prior work accomplished by Kaiser Engineers consisted of providing foundation design and construction management services for the pilot plant.

II. SUMMARY

A. CAPITAL COSTS

From the criteria used for this project, and from the proposed flow-sheets, a 15,000 ton per day molybdenite concentrator, utilizing conventional crushing followed by rod mill and ball mill grinding, can be built for an estimated total capital cost of $54.3 million.

The alternative considered, utilizing primary autogenous grinding and followed by ball milling, can be built for an estimated total cost of $51.6 million.

B. DIRECT MILLING COSTS

The conventional mill can be operated at $1.18 per ton of mill feed; the alternative considered, utilizing autogenous milling, will cost $1.20 per ton of ore to operate.

C. DIRECT POWER COSTS

Direct power costs will be 1.57 cents per kwh. This is based on power developed from diesel equipment.

The purchased power cost may be revised pending the outcome of continuing negotiations with the Canadian Federal Government on reducing the delivered cost of purchased power at the British Columbia border.

III. GENERAL PROJECT DESCRIPTION

A. LOCATION

The Adanac mining claims are located at an elevation of approximately 4,700 feet in a volcanic area some 21-miles northeast of the town of Atlin in British Columbia. The entire plant site is above the level of vegetation except for Caribou grass and shrubs, but it is below the permafrost zone common to this area. The plant will be located on the valley slope to the north of the present course of Ruby Creek and will be at the roadhead of the new access road from the Surprise Lake junction to the site (see Drawing No. 7008-C-1000).

The town of Atlin is a small community. It is an R. C. M. P. station and has no facilities other than a service station and a small country store. The labour forces for the job would have to be imported from industrial areas in British Columbia and the Yukon depending on the craft jurisdiction. In any event, the job is isolated and quarters for workmen and their subsistence and transportation would have to be furnished.

B. ACCESS

The claims, at the time of this report, are accessible only by an unimproved road from Surprise Lake, which is approximately 11 miles from the town of Atlin. The road from Surprise Lake to Atlin and then along Lake Atlin to the Trans-Canada Highway is a secondary gravel surfaced road. From Atlin to the Highway is 61 miles.

The closest railhead is Whitehorse, with a possible railhead existing at Carcross in British Columbia, but this site, at the time of this writing, does not have proper unloading and storage facilities. The railhead at both Carcross and Whitehorse is the White Pass and Yukon narrow gauge railway which begins at Skagway. Other access into the area is via the Trans-Canada Highway beginning in the vicinity of Edmonton, Alberta. There is a small unimproved airport in the vicinity of Atlin which has no operational facilities in the way of beacons, landing control or guidance items. The airport is suitable only for small planes.

C. GENERAL PLANT

1. Process Buildings and Auxiliary Structures

a. Process Buildings

The crusher buildings will be structural steel, insulated metal sheathed buildings and will be heated by steam from the power plant. The mill building will be a structural steel frame with insulated metal sheathing on roof and sides and will house the mills, flotation, drying and packaging. Within the structure will be offices for operating personnel and the metallurgical laboratory. There will also be a change house and sanitary facilities for personnel. Conveyor galleries connecting the mills with the crusher will be covered and heated (see Drawings No. 7008-M-2005 and 2006).

b. Maintenance Shop and Warehouse

This building will consist of a structural steel frame with insulated metal sheathing for roof and sides and will contain a main gallery for the repair of the 100-ton trucks and other heavy equipment. The other half of the building will consist of a two-story warehouse and a two-story office structure with adjacent dry rooms and sanitary facilities. The plant purchasing office will be located in this building adjacent to the warehouse section. Offices for shift foremen, both in service and mine, will be located in the building. Adjacent to this warehouse, there will be a 60-ft wide, 120-ft long prefabricated metal-sheathed building that will serve as a construction warehouse. It will also contain office space for the construction contractor. It is not intended that this building will be part of the permanently heated plant (see Drawings No. 7008-A-1002 and 1003).

c. Administration Building and Laboratory

This building will consist of a wood frame with metal insulated sheathing consisting of coloured panels and sun shades and will house the administrative personnel of the project in the main structure and the analytical laboratory and operating personnel for the laboratory in a single story adjacent "L". The office building and laboratory will have their own central heating and air conditioning unit and lunch rooms, and sanitary facilities will be provided for all personnel. A vault at each level will provide storage space for engineering, accounting and laboratory documents. The basement area of the plant, in addition to having the heating unit, will have space for surveyors and other engineering personnel (see Drawing No. 7008-A-1001).

Power House

The power house will consist of a structural steel frame and either concrete block walls with built-up roofing or will be metal sheathed with insulated panels. The power house will house all auxiliary equipment such as air compressors, fire pumps and water treatment equipment. Offices for operating personnel and sanitary facilities will be provided (see Drawing No. 7008-M-7007).

2. Utilities

a. Tankage

Storage tanks will be provided on terraces above the plant site for fire water (300,000 gal.), reclaim water (500,000 gal.) and potable water (100,000 gal.). In addition there will be two 600,000-gal. diesel storage tanks plus day tanks as required for operation (see Drawing No. 7008-C-1003).

b. Underground Services

The plant will have an underground fire loop with hydrants at strategically located points and will also have an underground sewage effluent system draining to sewage lagoons which will be located to the east of the main plant and south of the construction area (see Drawing No. 7008-C-1003). Water mains from the storage tanks to the use areas will be underground. It is anticipated that all water lines except sewage effluent will be made of steel (see Drawing No. 7008-C-1005).

c. Utility Piping

Utility piping, consisting of steam, condensate, compressed air and fresh water, will be run through conveyor galleries to and from the power plant for steam and air and from the water main at the power house for fresh water (see Drawing No. 7008-C-1005).

d. Electrical

Power to the operating sections of the process plant will be from the power house to the conveyor galleries and thence to the load centres as required for power and light in the structures. Power from the crusher building to the maintenance shop will be underground. Power for the mine and the pumps at the reclaim pond and at the potable water pond will be distributed by pole line (see Drawing No. 7008-E-7000).

D. POWER PLANT

The power plant will be installed in a separate centrally located building which will contain the generators and waste heat boilers, the auxiliary boilers for plant heating, plant air compressor, water treatment plant and fire pumps. The main switchgear will be enclosed in the building for direct transmission of power to the load centres of the plant. A separate switchyard will be required for purchased power (see Drawings No. 7008-M-7006 and 7007).

IV. PROCESS DESCRIPTION

A. INTRODUCTION

Two basic alternatives were chosen for the 15,000-tpd milling process rate requirement:

1. 54-inch gyratory for primary crushing followed by conventional crushing and grinding (the rod-mill ball-mill system is proposed and discussed).

2. 66-inch by 84-inch jaw crusher for primary crushing followed by wet autogenous grinding and ball mills.

Drawing No. 7008-G-2000 illustrates the first alternative. The second alternative is shown as Scheme No. 2 in Drawing No. 7008-G-2100. The latter is subject to revision pending the results of autogenous tests being conducted at Lakefield Research of Canada Ltd.

The flotation section, following conventional grinding, is outlined in Drawing No. 7008-G-4000 and -4001 (Revision A). Drawing No. 7008-G-4000 will also serve as a basis for estimation of equipment sizes for Scheme No. 2.

Regardless of the type of crushing and grinding circuit, the milling process will basically be the same. The flowsheets prepared for preliminary engineering estimates are described below. Detailed process criteria can be found in Section IX.

The alternative flowsheets are preliminary concepts based on information obtained from the work done at the Adanac pilot mill, which was operated from August 3 to November 8, 1970. Kerr Addison Mines Limited operated the pilot plant. Britton Research Ltd. of Vancouver provided the technical control for testing the beneficiation characteristics of the molybdenum ore.

B. PROCESS DESIGN CRITERIA

The preliminary flowsheets were designed to conform to the following guidelines:

Estimated average grade of ore	First 4 years, 0.25% MoS_2, decreasing to 0.20% MoS_2
Mining, ore and waste	3 shifts, 7 days or 20 operating shifts per week
Primary crushing	10 hours per day, 7 days per week (minutes of meeting with Chapman, Wood, and Griswold, November 5-6, 1970)
Secondary and tertiary crushing	3 shifts, 7 days/week chosen with provision for estimated maintenance
Milling rate	15,000 tpd, 350 days = 5,250,000 tpy
Milling recovery	95% MoS_2, minimum
Concentrate grade	92% MoS_2, minimum (dry basis)
Ratio of concentration	368 to 1

C. PERFORMANCE SPECIFICATIONS

1. Capacity

a. Daily Capacity of Crushing Plants

For both major alternatives, involving either the 54-inch-gyratory or the 66-inch by 84-inch jaw crusher for primary crushing, the plant will need to handle 15,000 dry, short tpd of open-pit ore. The ore will be assumed to contain about 5% moisture.

b. Availability of Crushing Plants

(1) Primary Crushing

Routine ore production will be confined to 10 hours per day. Primary crushing will be operated to accommodate this schedule.

(2) Secondary and Tertiary Crushing

Secondary and tertiary crushing will be provided for three shifts. Crusher availability is based on allowing 2 hours per day and one 8-hour shift per week for preventative maintenance. This will provide an average of 20.8 available hours per day or 87% of total time. To allow for contingencies, actual operating time is taken as 95% of available time, or 19.8 hours per day. Exhibit 1 in Section IX tabulates these calculations.

c. Hourly Crushing Rate

(1) Primary Crushing

For Scheme No. 1, the dumping cycle at the primary crushing station is estimated at 90 tons every 3.5 minutes over a continuous 10-hour period. The average feed rate to the primary crusher will be 1,540 dry, short tph. With one jaw crusher in Scheme No. 2, 20 hours per day operation is required.

The 54-inch gyratory crusher capacity is rated up to 1,770 tph. The capacity of a single 66-inch by 84-inch jaw crusher is rated at 876 tph. Preceded by a fixed grizzly scalping out minus 4-inch material, estimated at 10%, the capacity of one jaw crusher station will reach about 973 tph.

Exhibit 2 in Section IX shows the size distribution curves of various crushed products extrapolated from the data obtained from the pilot plant crushing plant. (Exhibit 2 is based on data shown in Exhibits 3 and 4.)

(2) Secondary and Tertiary Crushing, Scheme No. 1

The design capacity based on crushing for three shifts with about 83% overall availability will be 755 dry, short tph.

The capacity of a secondary crusher ranges from 900 to 1,025 tph, depending on the discharge setting.

A single tertiary crusher, equipped with an extra coarse bowl, has an approximate total capacity (net finished product plus circulating load) of 600 tph.

d. Daily Capacity of Grinding Plants

Both the conventional and the autogenous grinding plants must grind 15,000 dry, short tph.

e. Availability of Grinding Plants

Availability of the Scheme No. 1 or the conventional grinding circuit is based on operating 95% of the time. This provides for an average of 8-hours downtime for each mill each week.

Experience with autogenous grinding mills has indicated that their operating time will be 75% to 85% of all time at initial startup. Availability will be based on 85%.

f. Hourly Grinding Rate

The design capacity based on 95% availability is 660 tph or 330 tph for each conventional grinding circuit. Based on 85% availability, design capacity for the autogenous system is 735 tph, or about 370 tph for each of two lines.

2. Description of Ore and Products

a. Raw Ore to Crushing Plants

The feed for both crushing plant alternatives will consist of open pit material with a maximum lump size estimated to be 4 feet by 4 feet. Surface moisture is estimated to be 5%.

Alteration zones, containing clay minerals or "gouge," are known to exist. The amount of "gouge" material is unknown.

b. Crushing Plant Product, Rod-Mill--Ball-Mill Feed

The gyratory crusher will produce a minus 6-inch product, of which about 15% will be minus 3/4-inch. The minus 3/4-inch material by-passes the fine crushing plant, being conveyed directly to the mill bins. The 6-inch by 3/4-inch material is directed to a stockpile for reclaiming to the secondary crusher. Removal of the fine material from the ore stream will lessen freezing hazards in the stockpile. Ball mills will follow rod mills to produce the objective grind. An approximate size distribution of rod-mill feed is shown in Exhibits 5 and 6 in Section IX.

c. Crushing Plant Product, Wet Autogenous Mill Feed

The crushing plant incorporating the 66-inch by 84-inch jaw crusher will provide about 14.5% of minus 3/4-inch fine material and minus 8-inch plus 3/4-inch coarse material which will be proportioned and blended for autogenous-mill feed.

d. Grinding Plant Product

Both grinding plants are designed to produce a flotation feed product containing about 15% plus 65 mesh and 47% minus 200 mesh. Exhibits 7, 8, and 9 in Section IX show the size distributions of ball-mill discharge and cyclone products from the Adanac pilot plant.

3. Description of Crushing and Grinding Alternatives

a. Alternative No. 1 (See Drawing No. 7008-G-2000)

(1) Primary Crushing Plant

Open-pit ore will be delivered to the surface crushing plant at the rate of 15,000 tpd via 100-ton or larger trucks. Blasting is expected to yield a maximum lump size measuring 4 feet by 4 feet by 4 feet. Trucks will be able to dump from one side. The primary crusher will be a minimum of 3,000 feet from the open pit. Trucks will dump directly into the 54-inch gyratory crusher with an Open Side Setting (O.S.S.) of 6 inches.

The crusher product will fall into a 250-ton surge hopper, equipped with one 60-inch wide, 26-foot long pan feeder which will discharge to a 54-inch belt conveyor. All materials-handling equipment following the primary crusher will be designed to accommodate the rated capacity of the crusher, that is, 1,770 tph.

A dust control exhaust system, rated at 54,200 cfm, will be included in the crusher building.

The primary crusher product will be delivered via a 54-inch conveyor to one extra-heavy-duty 8-foot by 20-foot double-deck primary screen. The top deck will be stepped manganese rails with 2-inch openings. The bottom deck will have 3/4-inch slotted or rectangular openings. The primary screen undersize will be transported via a system of 30- and 42-inch conveyors, a 48-inch tripper conveyor and tripper to four fine-ore bins having a total live capacity of 10,000 tons which allows about 15 hours of grinding operation. To prevent freezing of the fines during the winter, a hot air heating arrangement will be included.

Screen oversize will be transported via a 48-inch belt conveyor, equipped with a weightometer for inventory, to a coarse-ore stockpile. Total storage will be 25,000 tons which will provide material for about 2-1/2 days of operating time.

Bulldozers may be used to increase dead storage, and also to reclaim dead storage during primary crusher downtime. Because of climatic conditions, the estimate includes a provision for a covered coarse-ore stockpile, based on Drawing No. 7008-M-2007. Covered storage may require a smaller stockpile.

(2) Secondary Crushing

Four 36-inch by 60-inch mechanical-type vibrating feeders will reclaim and feed coarse ore via a 42-inch belt conveyor, equipped with a tramp iron detector and magnet, to one secondary crusher with Closed Side Setting (C. S. S.) of 1-3/4 inches.

Crusher capacity at this setting ranges from 900 to 1, 025 tph.

Secondary crusher discharge will be sent via a 48-inch conveyor to a feed bin which will be able to split feed to two heavy-duty 8-foot by 20-foot double-deck secondary screens. The top deck will have 1-1/2-inch openings; the bottom deck will have 3/4-inch slotted openings.

(3) Tertiary Crushing

Two tertiary crushers with C.S.S. of 5/8-inch, will operate in closed circuit with the two 8-foot by 20-foot double-deck secondary screens. Product from the tertiary crushers will discharge and join secondary-crusher discharge upon a common 48-inch conveyor and a pair of 48-inch scissor conveyors. The material will discharge into the feed bin and thereafter be fed to the secondary screens, via two 60-inch by 96-inch mechanical-type vibrating feeders. Each feeder is rated at 1, 200 tph.

Primary- and secondary-screen undersize, minus 3/4 inch, will join and then pass through a system incorporating a 42-inch conveyor, a 48-inch tripper conveyor, weightometers and samplers. The 48-inch tripper conveyor and a tripper will transport the minus 3/4-inch material to four fine-ore bins, described previously.

Dust-control equipment will be provided at all major dust points in the secondary- and tertiary-crushing circuits, as well as in the screening and conveying systems. The exhaust system is rated at 104,300 cfm which includes the requirements for coarse-ore reclaim. For the fine-ore bins, 36,600 cfm will be provided. Drawing No. 7008-M-2010 illustrates the dust-control system.

Exhibit 10 in Section IX lists bulk densities of assorted crushed and granular products. These values relate to estimating requirements for crushers, screens, conveyors, and the coarse ore stockpile.

(4) Grinding

Slot feeders, 12-feet long by 10-inch to 26-inch wide taper, will discharge minus 3/4-inch feed from the fine-ore bins via two 30-inch mill-feed conveyors equipped with weightometers and samplers. Conveyors will transport this material at 330 tph to two identical grinding circuits. Each grinding circuit consists of a 13-1/2-foot by 20-foot rod mill and one 13-1/2-foot by 28-foot ball mill.

The rod mills will be fed with tube feeders integrated with the 30-inch mill-feed conveyors. Conveyor speed regulation will be controlled by a belt scale. The belt scale will weigh, record, and control a preset feed rate to each rod mill. Discharge from each rod mill will join ball-mill discharge in a common pump box from which the pulp will be pumped via two 14-inch by 12-inch pumps, to four 30-inch cyclones which are in closed circuit with the ball mill. The primary cyclones will be operated at 5 psi.

For each ball mill, an extra cyclone will be included as standby. There will be standby pumps.

A bond work-index value of 19.0 kw/hr per ton was used to estimate the horsepower requirements of the mills. Exhibit 11 in Section IX shows these calculations.

Cyclone underflow, at 70% solids by weight, will discharge directly into the ball mill. Ball-mill discharge at 65% solids, will be sent to a pump box to join rod-mill discharge for feed to the cyclones. Cyclone overflow, at 37% solids (range 35 to 40% solids) will be pumped with two 14-inch by 12-inch pumps to the flotation circuit.

Exhibit 12 in Section IX shows the cyclone grinding-mill material balance calculations and tabulates the flows used for estimating the size and number of pumps and cyclones. Exhibit 13 is the schematic for the conventional grinding circuit and shows the location of the flows numbered in the corresponding material-balance tabulations.

Exhibits 5, 6, and 7 in Section IX show the approximate size distribution of ball-mill discharge, cyclone overflow, and cyclone underflow. All have been derived from recent pilot-plant data. The size distributions of the cyclone products are plotted in Exhibit 14.

Exhibit 15 summarizes preliminary estimates of steel consumption for the size-reduction flowsheets.

Process water, milk-of-lime and other reagents will be added to the ore in the grinding-mill cyclone circuit. Reagent consumption, their addition rates and points of addition are given in Exhibit 16.

A small grinding mill handling about 9 tpd of coarse (minus 1/4 to minus 1/2-inch) bulk quicklime, may be required to prepare milk-of-lime slurry for flotation. An alternate method is to slurry finely-ground hydrated lime. Tanks and piping will be required to store, prepare and feed milk of lime and other flotation reagents. Exhibit 17 summarizes the estimates for storing, handling and feeding reagents.

Since the ore is very abrasive, consideration will be given to using rubber linings used to reduce wear in pumps and cyclones. A suitable rubber, such as neoprene, will be required due to the addition of Shell Carnea 21 (a light-grade marine oil) in the grinding circuit.

(5) Piping and Instrumentation

Piping and instrumentation is shown in Drawing No. 7008-P-3001. Drawing No. 7008-P-2001 relates to the crushing area.

b. Alternative No. 2 (See Drawing No. 7008-G-2100)

(1) Primary Crushing Plant

The purpose is to provide minus 3/4-inch fine material and minus 8 plus 3/4-inch coarse material which will be blended for autogenous-mill feed. The equipment suggested in this flowsheet is based on the average capacity of 755 tph and is subject to revision pending the completion of autogenous testing at Lakefield Research of Canada Ltd.

Open-pit material will be delivered at the rate of 15,000 tpd via 100-ton trucks to a dump pocket. The dump pocket will have a live storage capacity of 200 tons. Trucks will dump from one side. Material will discharge from the dump pocket onto a 72-inch by 30-foot pan feeder, capable of transporting maximum-size ore lumps. The pan feeder will discharge material over a fixed-rail grizzly with spacings set at 4 inches.

Grizzly oversize material will be fed into a 66-inch by 84-inch jaw crusher set at 8 inches.

Grizzly undersize, estimated to be 10% from Exhibit 2 in Section IX, will discharge directly upon a 48-inch by 24-foot pan feeder, which in turn will deliver material to a 48-inch conveyor. Crusher product will also discharge upon the 48-inch pan feeder. A dust-control exhaust system, rated at 73,950 cfm, will be included in the crusher building.

(2) Primary Screening

The 48-inch conveyor will discharge the combined products over one 6-foot by 16-foot double-deck screen. The top screen deck will have 1-1/2-inch openings; the bottom deck will have 3/4-inch rectangular openings. The jaw crusher is expected to produce about 14.5%, minus 3/4-inch product.

Screen oversize from the double-deck screens will be fed by a system of 48-inch conveyors to an outside coarse-ore stockpile. Total storage capacity will be 25,000 tons estimated for 1-1/2 days of operating. The comments on covered storage and the use of mobile equipment described for Drawing No. 7008-G-2000 also apply to this flowsheet.

The minus 3/4-inch screen undersize will be delivered by a system incorporating a 24-inch conveyor, a 24-inch tripper conveyor and tripper, to two fine-ore bins. The fine-ore bins will have a total live-storage capacity of 2,000 tons. To prevent freezing of the fines, provisions will be made for heating the ore through the slots.

Weightometers for inventory purposes, will be installed at the head of conveyors delivering product to fine- and coarse-ore storage.

Exhibit 10 in Section IX lists bulk densities of assorted, crushed products. These values relate to estimating requirements for crushers, screens, conveyors and the coarse-ore stockpile. Drawing No. 7008-P-2101 shows requirements for piping and instrumentation.

(3) Blending of Fine and Coarse Product for Autogenous-Mill Feed

Fine ore and coarse ore will be reclaimed from storage by their respective systems. Fine ore will discharge from the fine-ore bin via four 12-foot long by 10-inch to 26-inch wide taper-slot feeders onto two separate 48-inch mill-feed conveyors.

Coarse ore will be reclaimed via four 48-inch by 60-inch vibrating feeders and will be delivered to the two 48-inch mill-feed conveyors. This system will provide a blended feed, containing suitable proportions of fine and coarse fractions to feed the wet autogenous mills. Blending will be controlled by weightometers installed on the mill-feed conveyor at points just after fine and coarse ore discharge upon the mill-feed conveyors. Tramp iron detectors will be installed on conveyors discharging fine ore and coarse ore.

Blended feed will be sent to the grinding section via two 48-inch mill-feed conveyors, each handling 370 dry tph of new feed.

Dust control equipment will be provided for all major dust-points in the screening and conveying systems. Provisions are made for 18,150 cfm for conveyor transfer points, 17,850 cfm for the fine-ore bins, and 15,600 cfm for coarse-ore reclaim. Drawing No. 7008-M-2108 illustrates the dust-control system.

(4) Wet Autogenous Grinding

There will be two identical autogenous grinding circuits. Each grinding circuit will consist of a 26-foot by 10-foot wet-grate discharge mill, in closed circuit with two 6-mesh vibrating screens, and followed by a ball mill in closed circuit with five cyclones.

Each autogenous mill will be fed via a feeder conveyor integrated with the 48-inch mill-feed conveyor, whose speed will be controlled with a belt scale. The belt scale will weigh, record, and control a preset feed rate to each mill. Also, there will be a provision to control feed rate by the noise level within the mill. Water will be added to the mill to maintain proper pulp density, estimated to be about 75% solids.

Each autogenous-mill discharge, at 75% solids, will be scalped over two 8-foot by 20-foot single-deck horizontal vibrating screens, having 6-mesh openings. The plus 6-mesh oversize may contain material as coarse as plus 1 inch. Grate arrangements of the mill will determine the topsize in the circulating load, which will be about 35%. Two 12-inch by 10-inch pumps, including one standby, will return screen oversize into the discharge end of the autogenous mill.

Exhibit 18 in Section IX shows an expected size distribution of autogenous-mill discharge based on experience elsewhere. The actual size distribution for Adanac ore will be available after completion of autogenous tests at Lakefield Research of Canada Limited.

The minus 6-mesh undersize, as a slurry, will be discharged consecutively into two pump boxes. Two 12-inch by 10-inch pumps will deliver the minus 6-mesh undersize to the second pump box, which will also receive ball-mill discharge. There will be an extra 12-inch by 10-inch pump for standby service. From the second pump box, three 14-inch by 12-inch pumps, including one standby, will feed four 30-inch cyclones in closed circuit with a 13-1/2-foot by 28-foot ball mill or pebble mill. The cyclones will be operated at 5 psi. For each ball mill, an extra cyclone will be included for standby service.

Exhibit 11 in Section IX shows the calculations for estimating the horsepower requirements for autogenous grinding.

Cyclone underflow, at 70% solids, will discharge into a 13-1/2-foot by 28-foot ball mill. Ball-mill discharge, at 65% solids, will feed into the pump box to join minus 6-mesh undersize. Cyclone overflow, at 37% solids (range 35 to 40% solids), will be pumped with two 14-inch by 12-inch pumps to the flotation circuit.

Exhibit 12 shows the cyclone autogenous-mill material-balance calculations and tabulates the flows used for estimating the size and number of pumps and cyclones. Exhibit 19 is a schematic for the autogenous grinding circuit and shows the location of the flows numbered in the corresponding material-balance tabulation.

Exhibit 15 summarizes the estimated steel consumption for this alternative flowsheet.

Reagents, added to this grinding circuit, are detailed in Exhibit 16. Reagent requirements for the autogenous grinding circuit are based on calculations in Exhibit 17. The addition of oil will require cyclones and pumps to be equipped with neoprene.

4. Flotation Flowsheet (See Drawings 7008-G-4000 and -4001)

Drawing No. 7008-G-4000 illustrates the equipment requirements and material balance quantities for flotation following conventional grinding. Drawing No. 7008-G-4001 (Revision A) is the corresponding schematic used to summarize requirements for pumps, piping, sampling and instrumentation.

Reagents required throughout the flotation section will be added as shown in Exhibit 20. Reagent consumption is based on calculations in Exhibit 17.

Exhibit 20 contains the material balance for flotation following conventional grinding. Exhibit 22 is a schematic of the flotation process and locates the flows enumerated in the material-balance tabulations. Both exhibits include tailings thickeners; however both can also be used to estimate flow quantities without thickeners.

The routing of products and flows within the flotation section is tentative. In the final cleaner flotation stages, all flotation cells will be equipped with double launders to allow operators to re-route flotation products as required. Product routing will depend on such factors as:

- Liberation achieved during regrinding
- MoS_2 assays of flotation products
- Recovery of MoS_2 slimes
- Recovery of flotation reagents
- Excessive accumulation of flotation reagents in circulating flows

This information can be extracted from data obtained from pilot-plant investigations on Adanac ore. This data is still being evaluated and will be supplied by Britton Research Ltd.

a. Rougher Flotation Circuit

The cyclone overflow from each grinding circuit will be pumped with two 14-inch by 12-inch pumps to a 10-foot diameter by 12-foot deep conditioner. Pulp density will be 37% solids, but is expected to vary from 35 to 40% solids. Each conditioner will provide 2.5 minutes of conditioning time. The balance of conditioning will occur in the pumps and cyclones of the primary grinding circuit.

Pulp pH will be 11.2 as a result of lime previously added in the primary grinding circuit. Dowfroth 250 or other suitable frother will be added to the conditioners.

If required, aeration will be included during conditioning.

Each conditioner will be equipped with automatic pulp samplers. From each conditioner, the pulp will be delivered by two 8-foot diameter by 5-foot deep distributors to two sections of rougher flotation cells. Each rougher flotation section will have twenty-four 300-cubic foot No. 120 Agitair machines. Retention time for rougher flotation will be about 20 minutes. Exhibit 23 in Section IX shows the calculations for estimating the requirements for flotation cells and conditioners.

Rougher flotation is expected to produce a concentrate at a ratio of concentration of 70 to 1, or about 1.5% of the flotation feed. Rougher concentrate grade is expected to be 15% MoS_2. Recovery of MoS_2 will be 96 to 97%.

The rougher flotation concentrate will be sampled and pumped to a 36-foot diameter by 10-foot deep thickener where additional lime will be added.

Thickener overflow will be produced at 100 US gpm and will be reclaimed for makeup water to the primary grinding mills and/or to the head of the rougher flotation section in order to recover frother and high-grade MoS_2 slimes. Exhibit 24 shows the calculations for estimating thickener requirements.

Rougher flotation tailings will be sampled and flowed either to two 260-foot diameter by 22-foot centre-depth tailings thickeners or direct to the tailings dam. A portion of the rougher flotation tailings will be routed to a pilot plant section to evaluate recovery of minor amounts of heavy by-product minerals.

b. First Cleaner Flotation

Thickener underflow (thickened rougher flotation concentrate) at 40% solids, will pass through an automatic sampler and will be collected in a sump and pumped to the first cleaner flotation section. Sodium sulphide will be added to the thickener sump to depress undesirable sulphide minerals. Sodium hydrosulphide or sodium cyanide will be likely substitutes for sodium sulphide.

The first cleaner flotation section will consist of twelve 100-cubic foot No. 60 Agitair machines which will provide 17 minutes retention time.

Concentrate from first cleaner flotation will be sent for regrinding, whereas tailings will be sent to cleaner scavenger flotation.

c. Cleaner Scavenger Flotation

The first cleaner-flotation tailings will be collected in a sump and pumped to a 7-1/2-foot diameter by 7-1/2-foot deep conditioner. Sodium sulphide and frother will be added to the conditioner, which will provide 5 minutes conditioning time. Pulp from the conditioner will be fed to cleaner scavenger flotation, which will also consist of twelve 100-cubic foot No. 60 Agitair machines.

Retention time will be about 20 minutes.

The cleaner scavenger tailing is expected to contain too high a concentration of sodium-sulphide reagent to allow its direct internal reclaim. It will be collected in a sump, passed through a magnetic flowmeter, and pumped to join the underflow from the tailings thickeners. There will be provision for sampling the cleaner scavenger tailings at the sump.

Cleaner scavenger concentrate will be collected in a sump and pumped back to the first cleaner flotation cells to join thickened rougher flotation concentrate.

d. <u>No. 1 Regrind Circuit - Regrinding First Cleaner Flotation Concentrate</u>

The first cleaner flotation concentrate will be collected in a sump and pumped to two 10-inch diameter cyclones of the first regrind circuit. There will also be a standby cyclone and pump. The cyclones will be operated at 11 to 12 psi and will be in closed circuit with a 5-foot by 10-foot grinding mill. Exhibit 25 in Section IX shows the estimates for the regrind mill and cyclones.

Consideration will be given to equipping the regrind mill with rubber liners. Grinding media will be either steel balls, flint or ceramic pebbles.

The pulp density of the cyclone feed is expected to be 20% solids, but will be adjusted to provide a cyclone underflow of 50% solids, which is the desired pulp density for the regrind mill.

Shell Carnea 21 oil, or a suitable substitute will be added to the cyclone underflow, which will be collected in a launder and fed to the regrind mill.

Mill discharge at 50% solids, will be collected in a sump and pumped to join first cleaner flotation concentrate as combined feed to the cyclones. Circulating load will be 250% of new feed. If required, consideration will be given to open-circuit regrinding. Other alternatives include using mechanical classifiers or Dutch State Mines (D.S.M.) curved screens instead of cyclones.

e. Second Cleaner Flotation

The cyclone overflow will be collected in a sump and then pumped to a 6-foot diameter by 4-1/2-foot deep distributor to join tailings from the final cleaner flotation stages. Pulp will be distributed between two sections of second cleaner flotation cells. In each section, there will be ten No. 21 Denver "Sub A" flotation cells each having 40-cubic feet of volume. Retention time will be about 13 minutes. The second cleaner tailings will be collected in a common sump box, pumped to join the thickened rougher flotation concentrate and will be fed to first cleaner flotation.

f. No. 2 Regrind Circuit - Regrinding Second Cleaner Flotation Concentrate

The second cleaner concentrate will be collected in a common sump box and then pumped to the No. 2 regrind circuit. For closed-circuit regrinding, the circulating load will be 250% of new feed. If required, open circuit regrinding will be used. The objective of regrinding the second cleaner concentrate is to produce a product 100% minus 200 mesh in order to achieve the required final MoS_2 grade. The second cleaner concentrate will be directed to 4-inch to 5-inch diameter cyclones in closed circuit with a 5-foot by 10-foot regrind mill. Inlet pressure will be 12 to 14 psi. There will also be a standby cyclone and standby pump.

Cyclone underflow, at 50% solids, will be collected in a launder and fed to the regrind mill. If required, Shell Carnea 21 oil will be added to the regrind mill. The mill discharge will be collected in a sump, pumped to join the second cleaner concentrate to provide cyclone feed.

Should the mill be equipped with rubber liners, the presence of Shell Carnea 21 must be considered. Grinding media will be either steel balls, flint, or ceramic pebbles. Consideration will be given to substituting the cyclones with either mechanical classifiers or D.S.M. curved screens.

Exhibit 25 in Section IX shows the estimates for the No. 2 regrind mill and cyclones.

g. Third Cleaner Flotation Stage

Cyclone overflow from the second regrind circuit will be collected in a sump, pumped to a 6-foot diameter by 6-foot deep conditioner, and passed to the third cleaner consisting of twelve No. 21 Denver "Sub-A" flotation cells.

The tailings from the third stage of cleaner flotation will join the tailings from subsequent final cleaning stages in a common sump, and will be pumped to the distributor ahead of the second cleaner flotation stage.

h. Fourth and Fifth Cleaner Flotation Stages

The third cleaner concentrate will be pumped to the head of the fourth stage of cleaner flotation consisting of six No. 21 Denver cells. Concentrate from fourth cleaning passes to the fifth cleaner bank of four No. 21 Denver cells.

Tailings from the third, fourth and fifth cleaner stages will join final cleaner tailings, and the combined tailings pumped to the distributor preceding the second stage of cleaner flotation.

i. Final Cleaner Flotation (Sixth to Ninth Stages)

Fifth cleaner concentrate will be collected in a sump and then pumped to final stages of cleaner flotation.

The final cleaner flotation stages will consist of a common bank of eight No. 21 Denver "Sub-A" flotation machines. The sixth, seventh, eighth, and ninth cleaner flotation stages will each consist of two No. 21 Denver "Sub-A" machines.

Sodium silicate and sodium sulphide will be added to the final stage of cleaner flotation. Concentrate from the fifth cleaner cells will be fed to the sixth cleaner cell. Concentrate from the sixth stage of cleaning will be delivered by launders and the self-pumping action of the Denver cells to successive final cleaning stages. The final cleaning stages will upgrade molybdenite concentrate to the desired marketable specifications shown in Exhibit 26.

Consideration will be given to providing all cleaner flotation cells with double launders in order to allow operators to reroute flotation products as required.

j. Reagents and Reagent Feeding

Exhibits 16 and 20 list the reagent consumption, dosages, and points of addition. Exhibit 23 summarizes the pertinent calculations.

For pH adjustments, lime will be added as milk of lime, which can be prepared in many of the following ways:

- Grind bulk quicklime (minus 1/4 to minus 1/2-inch pebbles) in a ball mill in closed circuit with a small classifier. This circuit will treat about 9 tons of bulk lime per day. Pump the classifier overflow, at 20% solids, to a lime storage tank.

- Slurry pulverized quicklime or slaked lime at 20% solids in a mixing tank. Pump the slurry to the lime storage tank in the concentrator.

- Slake quicklime in a suitable slaker capable of handling quicklime in pellet form.

Where lime will be added to the grinding mill, it may be fed as a dry solid by means of a slowly moving belt or disc-type feeder.

Shell Carnea 21, or some other suitable light grade marine oil, will be used as a molybdenite collector. It will be fed as a dispersion in a 5% aqueous solution of Arctic Syntex L or other emulsifying reagent. Molysperse 206 would be a substitute for the Arctic Syntex L.

Sodium sulphide (60 to 62% grade) will be used to depress undesired sulphide minerals such as chalcopyrite and pyrite. Sodium hydrosulphide, in liquid form, and sodium cyanide may be substitutes. Sodium sulphide will be prepared as a 10% aqueous solution.

Sodium silicate (40 degree Baume) liquid will be used as a siliceous gangue dispersant. Solution strength will be 10%.

Shell Carnea 21 and Dowfroth 250 will be used at full strength. Less expensive frother substitutes include Frothant M and Terpene SW.

Arctic Syntex L, sodium sulphide, sodium silicate, and lime will be prepared in mixing tanks equipped with suitable mixing motors. Prepared reagent solutions will be pumped to their respective storage tanks.

Shell Carnea 21 and Dowfroth 250 will be used at full strength and will be kept in storage tanks.

Exhibit 27 summarizes the requirements for mixing tanks, mixer motors and storage tanks for the different reagents. Tank sizes are based on reagent requirements for 24 hours of continuous operation.

All reagents will be pumped to their respective head tanks. The head tanks will be 4 feet in diameter and 3 feet deep. Overflow from the head tanks will return by gravity to the proper reagent storage tank.

All reagent feeding may be done with flowrators. Alternatively, cup and disc-type feeders can be used for frothers and impulse feeders for water soluble and emulsified reagents. Impulse feeders will be timer-controlled solenoid valves on pressurized looped pipelines.

For lime slurry, an excess volume will be continually delivered to lime feeders. Overflow will return by gravity to the lime storage tanks. Variable speed motors with rheostats will control the rate of addition from the feeders. Feeder discharge in the grinding circuit will be distributed by pumps and launders.

k. Dewatering of Final Molybdenite Concentrate

(1) Thickening

The final molybdenite concentrate will be collected in a sump and pumped to a 20-foot diameter by 10-foot deep thickener. Requirements for the final concentrate thickener are given in Exhibit 24. To aid settling, a flocculant, such as Superfloc 127 or Separan (American Cyanamid Co) will be added if required. Estimated dosage will be 0.003 pounds per ton of concentrate, added as a 1/2% strength solution.

Thickener overflow at 17 US gpm will be sampled, and passed through a magnetic flowmeter. Because an excessively high concentration of sodium sulphide is expected in the overflow, it will join cleaner scavenger flotation tailings and tailings thickener underflow in a common collection box and be sent to the tailings pond.

The thickened final molybdenite concentrate, at 40 to 50% solids, will be discharged into a sump and then sent to filtration, or alternatively, to a leaching section.

A leaching section may be required to decrease the amount of Pb, Bi, Fe and Cu in the molybdenite concentrate to make an acceptable product. Tests to determine the optimum leaching conditions are in progress. When this work becomes definitive, consideration will be given to including a mild HCl-$FeCl_3$ leaching section. Examples of molybdenite leaching sections include those installed at Brenda Mines Ltd., Peachland, B.C. (Noranda Mines Ltd.) and at Endako Mines Ltd., Endako, B. C. (Placer Development Ltd.)

(2) Filtering

The final MoS_2 concentrate will be collected in a sump and pumped to a 6-foot diameter, 4-disc filter having a capacity of 50 tons per day. Exhibit 28 shows the calculations for estimating filter requirements. The filtrate at 7 US gpm will either be used as makeup water for the final concentrate thickener or will be combined with fresh process water. The filter will have suitable vacuum and air-pressure equipment, as well as provision to recirculate overflow to the thickener.

The moisture content of the filter cake is expected to be 35%. Filtering tests on pilot-plant molybdenite concentrate, in the presence of Superfloc 127 flocculant (0.002 pounds per ton of concentrate), have yielded a moisture content of 21.1 to 30.2%.

l. Drying of Final Molybdenite Concentrate

Filter cake, at 35% moisture, will be conveyed to a suitable drying unit. Examples for consideration include the Holo-flite dryer, a gas-fired infrared dryer used by Brenda Mines Ltd., or a 10-foot diameter, 4-hearth Nicols-Herreshoff multiple-hearth dryer used by Endako Mines Ltd. The estimate is based on the Holo-flite dryer.

Exhibit 29 summarizes the estimated equipment requirements for a Holo-flite processor using either 150 psig steam or Monsanto's Therminol FR-1 or FR-Low Temp as the heat-exchange agent.

Daily capacity of the dryer will be 50 tons. As specified in Exhibit 26, the dried product is expected to contain 1-1/2 to 2% residual moisture, or 4 to 8% combined collector oil and residual moisture. (7% is typical.)

m. Packaging of Molybdenite Concentrate

The final dried MoS_2 concentrate will be elevated and discharged into a 50-ton capacity surge bin, and then packaged in polyethylene-lined 33 US gallon drums each holding 500 pounds net weight. Palletizing will be employed with four drums shipped per wooden pallet. Capacity of the packaging plant will be 50 tpd.

Exhibit 26 summarizes the controls and specifications that will apply to the marketing of molybdenite concentrate.

n. Tailings Disposal

(1) Including Tailings Thickeners

If tailings thickeners will be required, tailings from each rougher flotation section will be collected in their respective sumps, pass through automatic samplers, and be pumped to two 260-foot diameter by 22-foot centre-depth tailings thickeners, if thickeners are used. Thickener underflows, at 65% solids, or plant tailings will be sent to a common collection box to join other waste products (cleaner scavenger flotation tailings and final concentrate thickener overflow). Exhibit 24 shows the calculations for estimating tailings thickener requirements.

Cleaner scavenger flotation tailings will be passed through an automatic sampler and a magnetic flowmeter. Along with overflow from the final concentrate thickener, cleaner scavenger tailings will join the thickened rougher flotation tailings in a common collection box, where the combined final tailings will be diluted to 58.8% solids. The combined final tailings will be 2,874 US gpm and will flow by gravity via a wood stave tailings pipe to a 400-acre tailings pond. Minimum pipe diameter will be 16-inches inside diameter.

The cost estimate will be based on a 24-inch inside diameter wood stave pipe, which would accommodate a 19,000-tpd plant.

Initial elevation drop to the tailings pond will be 350 feet over a horizontal distance of 12,000 feet to the base of the rear or downstream tailings dam.

Cylindrical steel drop boxes along the tailings line will be installed as required. Drop boxes will be spaced to compensate for pipe friction losses. Maximum gradient between drop boxes will be 0.5%.

In the future, as the top of the downstream tailings dam reaches an estimated elevation of 4,450 to 4,475 feet, it will be necessary to pump all the tailings to the disposal area. During summer months, conventional spiggoting will be used for raising the tailings dam height.

Exhibit 30 is a summary analyzing the requirements for two tailings disposal systems. One includes tailings thickeners, and the other is without tailings thickeners. For each of the two systems, a list of considerations and design factors is included.

(2) No Tailings Thickeners

If tailings thickeners will be excluded, rougher flotation tailings will pass through automatic samplers and will join in a common collection box. Overflow from the final concentrate thickener and cleaner scavenger tailing will also join rougher flotation tailings at the collection box, which will provide a convenient sampling point for the assorted waste products.

The combined tailings, at 35% solids, will be discharged to the tailings pond at a rate of 5, 853 US gpm via the 24-inch ID wood stave pipe.

o. Water Supply

(1) Including Tailings Thickeners

Overflow from both tailings thickeners will be reclaimed for use in the milling circuit. Total available overflow from thickening will be 2, 998 US gpm. Reclaimed overflow will be pumped through a 16-inch inside diameter pipe to a 42-1/2-foot diameter by 48-foot high process water tank. Storage capacity will be 500, 000 US gallons, which represents 2 hours operation at the maximum calculated figures. Lineal pumping distance will be 1, 250 feet against a static head of 148 feet. Pump brake horsepower will be 168 for wood stave pipe and 175 for mild steel pipe.

The total amount of water required for milling will be 4, 969 US gpm. Reclaimed water, which will be recycled within the milling circuit, will include: overflow from rougher concentrate thickener (100 US gpm), filtrate from filter (7 US gpm), and overflow from tailings thickeners (2, 998 US gpm).

Total reclaimable water will be 3, 105 US gpm to which 1, 864 US gpm of new water will be added to meet milling requirements. The additional water will be obtained from a fresh water pond, which will be developed in the reservoir formed at the toe of the upper tailings dam. This reservoir system will permit pumping 1, 922 US gpm for

10 months. Lineal pumping distance to the process-water storage tank will be about 9, 500 feet against a static head of about 228 feet. Pump brake horsepower will be about 220 for a 14-inch inside diameter mild steel pipe. Specific details of fresh water requirements will be furnished by others. Water supply estimates are shown in Exhibit 31.

(2) No Tailings Thickeners

Along with the tailings discharged to the tailings pond, there will be a total of 4, 842 US gpm of water. Assuming that the sands in the tailings pond will settle to 70% solids, the maximum amount of recoverable process water will be 3, 715 US gpm. Lineal pumping distance to the process water storage tank will be 12, 000 feet against a static head of 498 feet. Minimum pipe diameter will be 18 inches, ID. For wood stave pipe, pump brake horsepower will be 749 and for mild steel pipe 822.

Since the concentrator will require a total of 4, 969 US gpm of water, of which 3, 715 US gpm is recoverable, 1, 254 US gpm of fresh water will be required. This fresh water will be available from the water reservoir previously described. Lineal pumping distance from the water reservoir to the process water storage tanks will be about 9, 500 feet. The makeup water will be pumped against a static head of about 228 feet through 12-inch ID pipe. Pump brake horsepower will be about 140.

Details of pumping requirements for 1, 254 US gpm of fresh water will be supplied by others.

p. Sampling

Automatic samplers will be used on principal products such as the flotation feed, rougher flotation tailings, cleaner scavenger flotation tailings, and overflow from the final concentrate thickener.

All piping that will deliver flows into sumps and pump boxes will be arranged to enable physical ease of sampling of products.

Drawing No. 7008-G-4001 (Revision A) shows the location of sampling points.

q. Instrumentation and Control

Drawing No. 7008-G-4001 (Revision A) along with the flow quantities and parameters shown in Exhibit 21 summarizes the requirements for estimating pumps and piping. This drawing defines and locates instrumentation requirements and control points. Magnetic flowmeters will be used on cleaner scavenger flotation tailings and overflow from the final concentrate thickener. Pulp level controls will be installed in the flotation machines.

r. Pilot Plant Test Area

There will be a moderate space allowance (approximately 30 feet by 30 feet) for a pilot-plant section to treat a portion of rougher flotation tailings in order to evaluate recovery of minor amounts of heavy byproduct minerals. Basic equipment will include four-rougher and two-cleaner Humphreys spiral concentrators. The space provided for a pilot test area may also include most of the Adanac pilot mill flotation equipment to evaluate reagents, flotation operating conditions, and to check the flotation cleaning stages.

s. Metallurgical Laboratory

A metallurgical laboratory will be located at the Adanac project site. Basic equipment will include grinding mills, flotation machines, filters, drying ovens, balances, and sizing devices. There will be provision for sample preparation, sample storage and dust control.

V. POWER PLANT STUDY

A. INTRODUCTION

The power study was initiated by Kerr Addison Mines in September 1970 to review and recommend power supply options for the Adanac Molybdenum Project. The study objective was to obtain project power with the lowest overall capital and operating expense. A number of types of power plants and fuels were reviewed:

1. Thermal plant operating on coal or Bunker C oil
2. Diesel plants of the heat exchange type
3. Purchased power from the Yukon grid of the Northern Power Commission
4. Low-head hydro
5. Natural gas as a fuel
6. The possibility of direct high voltage energy transmitted from the coast region cross country to the plant
7. Small nuclear generating plants

The final schemes considered for the project power supply were a coal fired thermal plant with standby diesel generator, a 20-mw diesel generating plant with jacket and flue heat recovery from waste heat boilers and a direct 13.8-kw connection from the Northern Canada Power Commission grid in the Yukon including a steam unit at the process plant for the heating of process water and auxiliary building heat.

Primarily due to economic factors, the following schemes were initially investigated but not considered in the final analysis:

1. Low-head hydro. This scheme proved to be not feasible due to the unsuitable location of the plant site and the small volume of run-off into a reservoir.

2. Natural gas as a fuel. This scheme proved to be not feasible even though existing gas fields have been discovered to the north and east of the area. No refining or scrubbing facilities are in existence in these fields and the cost of transmission lines would be exorbitant.

3. High voltage energy cross country power transmission. This scheme was discarded due to the problem of low demand in comparison with the cost of transmitting voltage of this type cross country.

4. Nuclear power. This method was eliminated due to small size, high capital cost, safety factors and stringent B. C. pollution controls.

The power plant will be installed in a separate centrally located building which will contain the generators and waste-heat boilers, the auxiliary boilers for plant heating, the plant air compressor, water treatment plant and fire pumps. The main switchgear would be enclosed in the power building for direct transmission to the load centres of the plant. A separate switchyard will be required for the scheme using purchased power (see Drawings No. 7008-M-7006, -7007, -7010 and -7011.

B. SUMMARY

Three feasible project power supply schemes were investigated and are outlined below:

1. Cost Summaries

The following are the order-of-magnitude cost summaries of the three schemes:

	Scheme 1 (Coal-Thermal)	Scheme 2 (Diesel)	Scheme 3 (Purchased Power)
Capital cost	$7,831,000	$5,522,000	$3,468,000
Annual owning and operating cost	$4,994,000	$3,669,000	$6,705,000

From the cost summaries, it is concluded that Scheme 2, diesel generation, represents the most economical method of providing power to the proposed mine site. Therefore, diesel generation is the power supply system on which the Adanac Molybdenum Project feasibility study and cost estimates should be based. Schematic and plan drawings for Schemes 2 and 3 are presented in Section IX.

Discussions presently being held between the Northern Canada Power Commission, B.C. Hydro, and Kerr Addison should be continued, particularly to explore the possibilities of a reduction in the presently quoted purchased power costs.

A general outline of scheme requirements is presented on the next page.

2. Scheme 1-Coal Thermal

This scheme provides for the generation of electricity using steam boilers and a single turbine generator set. The steam boilers would be fired with coal transported from the existing Anvil Coal Mines at Carmacks.

Equipment required is as follows:

Boilers	2 @ 270,000 lb/hr each
Turbine-generator:	1 @ 20 mw
Standby-generator (diesel):	1 @ 2-1/2 mw
Transformers and switchgear	

3. Scheme 2-Diesel

This scheme provides for the generation of electricity using engine-generator sets and recovering waste heat. The engines would operate on No. 2 fuel oil, supplied to storage tanks on site.

Equipment required is as follows:

Engine-generators:	4 @ 5 mw each
Waste heat boilers:	4 - 1 for each engine generator
Auxiliary boilers:	1 @ 1,000 hp each
Standby diesel	1 @ 75 kw
Transformers and switchgear	

4. Scheme 3-Purchased Power

This scheme provides for the supply of electricity by the Northern Canada Power Commission and the provision of plant heat from an on-site boiler plant. Equipment required for scheme 3 is listed on the next page.

Transmission line	Approximately 50 miles from the B.C. border to the Adanac project
Boiler plant	3 @ 1,000 hp each
Standby generator (diesel)	1 @ 2-1/2 mw
Transformers and switchgear	

C. GENERAL DESCRIPTION OF FEASIBLE SCHEMES

1. Scheme 1 - Coal Thermal

Steam would be generated in 2 - 270,000 lb/hr boilers at a pressure of 665 psia and 750 F and fed to a single condensing type turbine operating at 665 psia inlet pressure and condensing at 3 inches Hg. The steam turbine would be directly connected to a generator with a capacity of 20 mw generating at 4,160 volts.

Fuel for the plant would be coal, produced at the existing Anvil Coal Mine at Carmacks, and delivered onto a stockpile at the mill site, a distance of approximately 225 miles. From the stockpile, coal would be fed by conveyor into two hoppers and from each hopper into a pulverizer from which point it would be mixed with combustion air and blown into the boilers. Ash would be removed by truck to a disposal area, or possibly to tailings. The stockpile would contain approximately 4,000 tons of coal which is sufficient for operation for approximately 10 days.

A standby diesel generator with a capacity of 2-1/2 mw would provide èmergency power in the event of failure of the main generating plant.

2. Scheme 2 - Diesel

Four 5-mw diesel generator sets would produce power at 4,160 volts for distribution to the mill and mine equipment. Waste-heat boilers on each diesel engine, supplemented by one 1,000-hp boiler, fired by No. 2 oil, supply 25,000 lb/hr of steam at 150 psig for heating purposes.

Fuel for the plant would be diesel oil delivered into storage tanks at the mill site. Storage of fuel would be sufficient for operation for one month. Two storage tanks would be included. Each would measure 60-feet in diameter by 40-feet in height. The combined storage capacity would be 40,000 barrels of diesel oil.

A standby diesel generator with a capacity of 75 kw would provide emergency power in the event of failure of the main generating plant.

3. Scheme 3-Purchased Power

Power would be supplied at the B.C. - Yukon border by Northern Canada Power Commission and from the border to the mine site by B.C. Hydro and Power Authority. The scheme would include a 50-mile transmission line from the Yukon border to the mill site, with transmission cables supported by H-frame timber structures. Three 1,000 hp boilers located at the mill site would provide 75,000 lb/hr of steam at 150 psig for heating purposes.

A standby diesel generator with a capacity of 2-1/2 mw would provide emergency power in the event of failure of the main generating plant. Fuel for the heating plant would be diesel oil delivered into storage tanks at the mill site. Fuel storage capacity would be sufficient for operation for one month. One storage tank would be used. It would measure 60 feet in diameter by 40 feet in height and would hold 20,000 barrels of diesel oil.

D. STUDY CRITERIA

1. Fuel Cost Data

Coal	\$27.00 per ton delivered
Diesel oil	0.29 per imperial gallon delivered
Electricity	0.03 kwh at the B. C. - Yukon border

2. Power Requirements

Peak load:	22,050 kw (See Table V-1)
Daily load:	469,666 kwh (See Table V-2)
Annual load:	469,666 x 365 days = 171,428,100 kwh

3. Heating Requirements

Hourly load:	41.6×10^6 Btu/hr (See Table V-3)
Annual load:	99.6×10^9 Btu

4. Heating Values

Coal	11,800 Btu per lb
Diesel oil	18,400 Btu per lb

Table V-1 lists the connected load requirements for the feasibility study.

TABLE V-1

SUMMARY OF CONNECTED HORSEPOWER

Item	kw/hp (Rod mill-ball mill arrangement)
Concentrator power	22,131 hp
Boiler house auxiliary	1,900
TOTAL hp	24,031 hp
24,031 hp =	17,927 kw
Mine load	3,200 kw
Shops and warehouse lighting	1,000 kw
Mill lighting	1,500 kw
Office, powerhouse & site lighting	875 kw
TOTAL CONNECTED kw	24,502 kw
TOTAL BASED ON 0.9 DEMAND FACTOR	22,052 kw

TABLE V-2

ELECTRICAL POWER USAGE ANALYSIS*

Plant Area	Hours Running	Load Factor	hp/kw	kwh/day
Primary crushing	20	0.75	1,115 hp	12,480
Fine crushing	20	0.90	2,053 hp	27,540
Grinding	24	0.90	14,720 hp	237,170
Flotation	24	0.90	3,910 hp	63,000
Concentrate handling	24	0.90	333 hp	5,380
SUBTOTAL			22,131 hp	345,570
Mill lighting	24	0.75	1,500 kw	27,000
Shop lighting	24	0.60	1,000 kw	14,400
Office lighting	8	0.60	75 kw	360
Site lighting	24	0.60	500 kw	7,200
Powerhouse lighting	24	0.85	300 kw	6,120
SUBTOTAL			3,375 kw	55,080
Boiler house auxiliaries	24	0.90	1,900 hp	30,616
Mine load	24	0.50	3,200 kw	38,400
TOTAL kwh/day				469,666
Line losses				
B.C. border to mine				6,600
TOTAL kwh/day (NCPC)				476,266

*Usage is based on the running hours of the plant and the load factor of the various sections.

TABLE V-3

PLANT AREA HEATING REQUIREMENTS

PROCESSING PLANT

Item	Btu/hr.
Ore - 15,000 tpd heated from - 20 F to 35 F	13,800,000
Primary crushing	4,760,000
Secondary crushing	3,960,000
Concentrator building	5,890,000
Transfer towers	14,000
Conveyors galleries	1,000,000
Fine ore bins	692,000
Total	30,116,000
From waste heat boiler	21,000,000
Auxiliary heat required	9,100,000

ANCILLARY REQUIREMENTS

Item	Btu/hr.
Maintenance and warehouse bldgs	9,400,000
Administration bldg	540,000
Boiler house	1,000,000
Fuel oil heating	600,000
Total	11,540,000

Total Heat Required	41,656,000
Available waste heat	21,000,000
Auxiliary heat required	20,600,000

TABLE V-4

CAPITAL COST COMPARISON

Scheme 1 - Coal Thermal

Line/Item	Cost
1. Two - 270,000 lb/hr coal-fired steam boilers complete with all necessary trim, forced and induced draft fans, dust collector, pulverizer, coal blowers, soot blowers, instrumentation and controls, erection and startup services.	
COST COMPLETE, INCLUDING APPLICABLE TAXES	$2,600,000
2. One - 15 mw steam turbine generator complete with evaporator, exciter, all switchgear and controls	
COST COMPLETE, LESS TAXES	1,250,000
3. One - Deaerator 270,000 lb/hr capacity	20,000
4. One - Cooling Tower 160 x 10^6 Btu/hr	120,000
5. Feed Pumps	10,000
6. Chemical Treatment	25,000
7. Crane	60,000
8. TOTAL - MACHINERY	$4,085,000
9. Taxes on items 2 - 7 (5% on 12%)	261,000
10. Freight and handling, 20% of lines 2 - 7	297,000
11. Machinery installation, 25% of lines 2 - 7	371,000
12. Piping, ductwork, auxiliaries, 20% of line 8	817,000
13. TOTAL - MECHANICAL	$5,831,000
14. Substation and standby generator	$ 400,000
15. TOTAL - MECHANICAL AND ELECTRICAL	$6,231,000
16. Boiler house structure	940,000
17. Stack	150,000
18. Coal conveyors	300,000
19. Ash handling equipment	150,000
20. Coal hoppers	60,000
21. TOTAL BUILDING COST	$1,600,000
22. TOTAL COST	$7,831,000

TABLE V-4 (Cont)

Scheme 2 - Diesel

Line/Item	Cost
1. Four 5 mw diesel engine generator sets complete with waste heat recovery boilers, day tanks, air start system, lube oil system, starter panel, all necessary switchgear and controls.	
COST COMPLETE, LESS TAXES	$2,279,000
2. Transfer pumps and heaters	15,000
3. Crane	60,000
4. Auxiliary boilers and controls	195,000
5. Deaerator	5,000
6. Chemical treatment	8,000
7. TOTAL MACHINERY	$2,562,000
8. Taxes on lines 1 - 6 (5% on 12%)	451,000
9. Freight and handling, 20% of line 7	512,000
10. Machinery installation, 25% of line 7	641,000
11. Piping, duckwork, auxiliaries, 20% of line 7	512,000
12. TOTAL - MECHANICAL	$4,678,000
13. Substation	142,000
14. TOTAL - MECHANICAL AND ELECTRICAL	$4,820,000
15. Engine room structure	432,000
16. Engine bases	120,000
17. Tanks and miscellaneous steel	150,000
18. TOTAL BUILDING COST	$ 702,000
19. TOTAL COST	$5,522,000

TABLE V-4 (Cont)

Scheme 3 - Purchased Power

Line/Item	Cost
1. Three steam generators for 31,000 lb/hr, complete with all necessary switchgear and controls	$ 225,000
2. Transfer pumps and heaters	15,000
3. Deaerator	5,000
4. Chemical treatment	5,000
5. TOTAL - MACHINERY	$ 250,000
6. Taxes on lines 1 - 5 (5% on 12%)	44,000
7. Freight and handling, 20% of line 5	50,000
8. Machinery installation, 25% of line 5	63,000
9. Piping, ductwork, auxiliaries, 20% of line 5	50,000
10. TOTAL - MECHANICAL	$ 457,000
11. Substation and standby generator	750,000
12. TOTAL - MECHANICAL AND ELECTRICAL	$1,207,000
13. Boiler house structure	126,000
14. Crane	60,000
15. Tanks and miscellaneous	75,000
16. TOTAL BUILDING COST	$1,468,000
17. Transmission line	2,000,000
18. TOTAL COST	$3,468,000

TABLE V-5

ANNUAL OPERATING COST COMPARISON

Item		Scheme 1 Coal Thermal	Scheme 2 Diesel	Scheme 3 Purchased Power
Coal		$3,430,000		
Diesel Oil			$2,487,000	$ 458,000
Purchased electricity *				5,215,000
B. C. Hydro charge, 5%				261,000
Labour		135,000	90,000	135,000
Maintenance and material		13,000	45,000	5,000
Supervision		20,000	15,000	15,000
Supplies		5,000	33,000	1,000
Other expenses		5,000	22,000	1,000
TOTAL OPERATING EXPENSE		$3,608,000	$2,692,000	$6,091,000
DIRECT OPERATING COST		2.105	1.570	3.500
GROSS INVESTMENT		7,831,000	5,522,000	3,468,000
CAPITAL CHARGES				
Interest	8.5%			
Taxes	4.0%			
Insurance	0.2%			
Depreciation	5.0%			
	17.7%	1,386,000	977,000	614,000
TOTAL ANNUAL EXPENSES		$4,994,000	$3,669,000	$6,705,000
TOTAL COST/kwh		2.913	2.140	3.857

*$0.031 kwh

VI. CONSTRUCTION SCHEDULE

The construction schedule has been based on criteria that include the scope of the project, the isolated location of the plantsite, severity of the climate and hence the restricted time available for plant construction. The schedule reflects the need for close coordination between plant construction and mine stripping operations as an important part of the project.

Lead time sufficient to enable construction forces to move in at the proper time is included in the schedule. Manpower peak in summer and winter work under cover is based on best available delivery time of major equipment and structural steel.

The energizing of the plant in the late summer of 1972 is a critical milestone for both the plant completion and for mining operations.

KAISER ENGINEERS

PROJECT TITLE Adanac Molybdenum Project

CLIENT Kerr Addison Mines Ltd LOCATION Atlin, B.C.

Summary Scheme #2 - Autogeneous

JOB NO. 7008-4

DATE 1-15-71 BY B Hosel

COST ACCOUNT	DESCRIPTION	QUANTITY	UNIT L	EU	M	SC	LABOR 00	BURDEN 10	EQUIPMENT USAGE 20	MATERIAL 30	SUBCONTRACT 40	50	EQUIPMENT 60	TOTAL
	Construction Costs													
	Direct Costs													
	General Site & Utilities						56 500	17 300	9 000	42 000	2 388 500		152 100	2 665 400
	Crushing & Stockpile Area						485 500	102 200	72 600	471 400	1 308 200		1 165 600	3 605 500
	Grinding						606 200	127 500	99 000	552 000	2 058 900		3 549 700	6 993 300
	Flotation, Drying & Packaging						128 500	27 400	37 600	—	370 500		1 099 000	1 663 000
	Leaching System										300 000			300 000
	Plant Service Bldgs & Equip.						209 300	43 500	24 200	273 900	1 259 100		379 400	2 189 400
	Power Plant						97 000	20 600	15 150	94 500	889 500		2 825 850	3 942 600
	Mobile Equipment												270 000	270 000
	Piping						734 100	140 400	146 900	836 400	790 700			2 648 500
	Electrical										1 660 000		455 000	2 115 000
	Instrumentation										314 000			314 000
	Test and Checkout						50 000	10 000	5 000	15 000	20 000			100 000
	Sub Total						$ 2 367 100	$ 488 400	$ 409 450	$ 2 285 200	$ 11 359 400		$ 9 876 650	$ 26 786 700
	Taxes													
	Escalation													
	Premium Time													
	Travel In & Out													
	Total: Direct Costs													
	Indirect Costs													
	Contractors Field Overhead													
	Construction Plant													
	Construction Camp													
	Total: Indirect Costs													
	Total: Construction Costs													
	Engineering, Supervision & Procurement													
	Sub Total													
	Contingency													
	Engineer - Contractor Fee													
	Total: Estimated Cost													

KAISER ENGINEERS

PROJECT TITLE Adanac Molybdenum Project

CLIENT Kerr Addison Mines, Ltd LOCATION Atlin, B.C.

15,000 TPD Feed

JOB NO. 7008-4

DATE 1-19-71 BY B. Hosch

COST ACCOUNT	DESCRIPTION	QUANTITY	UNIT L	UNIT EU	UNIT M	UNIT SC	LABOR 00	BURDEN 10	EQUIPMENT USAGE 20	MATERIAL 30	SUBCONTRACT 40	50	EQUIPMENT 60	TOTAL
	Construction Costs													
	Direct Costs													
0100 000	General Site & Utilities						56,500	17,300	9,000	42,000	2,388,500		152,100	2,665,400
0200 000	Crushing & Stockpile Area						782,300	164,400	105,400	809,300	2,253,300		2,037,300	6,152,000
0300 000	Grinding						579,800	122,200	85,600	552,000	2,052,900		2,440,700	5,833,200
0400 000	Flotation, Drying & Packaging						128,500	27,400	37,600		370,500		1,099,000	1,663,000
0500 000	Leaching System										300,000			300,000
0600 000	Plant Service Bldgs & Equip.						209,300	48,500	24,200	273,900	1,259,100		379,400	2,189,400
0700 000	Power Plant						97,000	20,600	15,150	94,500	889,500		2,825,850	3,942,600
0800 000	Mobile Equipment												270,000	270,000
0900 000	Piping						760,900	145,500	152,200	851,700	790,700			2,701,000
1100 000	Electrical										1,613,500		455,000	2,068,500
1200 000	Instrumentation										314,000			314,000
1300 000	Test and Check-Out						50,000	10,000	5,000	15,000	20,400			100,400
	Sub Total						$ 2,664,300	$ 550,900	$ 434,150	$ 2,638,400	$ 12,252,400		$ 9,659,350	$ 28,199,500
	Taxes									460,000	300,500		INCL	760,500
6100 000	Escalation											4,080,000		4,080,000
6200 000	Premium Time						1,010,000	212,000	70,000		1,918,000			3,210,000
6300 000	Travel-In & Out							600,000						600,000
	Total: Direct Costs						$ 3,674,300	$ 1,362,900	$ 504,150	$ 3,098,400	$ 14,470,900	$ 4,080,000	$ 9,659,350	$ 36,850,000
	Indirect Costs													
7100 000	Contractors Field Overhead & Profit													3,410,000
7200 000	Construction Plant													230,000
7300 000	Construction Camp													3,250,000
	Total: Indirect Costs													$ 6,890,000
	Total: Construction Costs													$ 43,740,000
8100 000	Engineering, Supervision & Procurement (Includes Fee)													3,700,000
	Sub Total													$ 47,440,000
9100 000	Contingency													6,900,000
	Total: Estimated Costs													$ 54,340,000

KAISER ENGINEERS

PROJECT TITLE Adanac Molybdenum Project
CLIENT Kerr Addison Mines, Ltd LOCATION Atlin, B.C.
Covered Stockpile Alternate - Building Only

JOB NO. 7008-4
DATE 1-20-71 BY [illegible]

COST ACCOUNT	DESCRIPTION	QUANTITY	UNIT L	UNIT EU	UNIT M	UNIT SC	LABOR 00	BURDEN 10	EQUIPMENT USAGE 20	MATERIAL 30	SUBCONTRACT 40	[illegible]0	EQUIPMENT 60	TOTAL
	Stockpile Area													
	Direct Costs													
	Building - 280' x 94' x 48' = 26,320 SF													
	Excavation	4,400 CY									16,500			16,500
	Back Fill	2,500 CY									8,100			8,100
	Concrete	1,210 CY					60,500	12,700	6,800	104,100				184,100
	Structural Steel	260 T									175,500			175,500
	Miscellaneous Steel	12 T									10,800			10,800
	Roofing + Siding - corr. metal	44,400 SF				1.50					66,600			66,600
	General Construction	Allow					5,000	1,000	800	3,200	5,000			15,000
	Sub Total						65,500	13,700	7,600	107,300	282,500			476,600
	Taxes									5,400	7,000			12,400
	Escalation											4[illegible],700		47,700
	Premium Time						24,200	5,100	1,200	–	1,100			31,600
	Travel-In & Out							11,000						11,000
	Total: Direct Costs						$ 89,700	$ 29,800	$ 8,800	$ 112,700	$ 290,600	$ 4[illegible],700		$ 579,300
	Contr. Field Overhead + Profit; Construction Plant + Camp; Engr, Supervision & Procure. (Incl Fee); Contingency													$ 275,700
	Total: Estimated Costs													$ 855,000

VII. CAPITAL COSTS

This capital cost estimate is for a turnkey job in accordance with the estimate criteria that follow. The design, procurement, expediting and construction would be accomplished by Kaiser Engineers personnel.

A. CAPITAL COST ESTIMATE SUMMARY

1. Column Description

Labor 00: includes straight time actual craft labor cost up to general foreman.

Burden 10: includes welfare, insurance, and miscellaneous fringe elements on Labor 00.

Equipment Usage 20: includes the cost of renting construction equipment and maintaining same.

Material 30: includes the actual cost of material such as concrete, embedded metal, rebar, tiewire, forms, culverts, grouting, and other materials that become a part of the plant.

Subcontract 40: consists of those elements that appear will be subcontracted such as structural steel (fabricated and erected), architectural items such as roofing and siding, heating and ventilating, lath and plaster, floor tile, acoustics, and electrical installation.

Equipment 60: represents equipment purchased and installed.

2. Line Numbers and Description Columns (Direct Costs)

Direct costs include equipment purchase, freight, British Columbia tax, materials costs, equipment rental for construction, labor and burden to install.

Line 3, General Site & Utilities: includes soils investigation, site surveys, cut and fill, culverts, underground disposal lines, miscellaneous structures and plant storage tanks.

Line 4, Crushing & Stockpile: includes structures and numbered equipment in the 2000 series.

Line 5, Grinding: includes numbered equipment in the 3000 series. (The mill building contains Line 5 and Line 6 equipment.)

Line 6, Flotation & Product Recovery: includes numbered equipment in the 4000 series.

Line 7, Leaching: is a single line item as an allowance specified by the client.

Line 8, Plant Service Building: includes repair shops, machine shops and warehouse, offices for mine and mill, change rooms and equipment including cranes.

Line 9, Power Plant: includes structure, power generating equipment, auxiliaries, plant heating units, and distributing switchgear.

Line 10, Mobile Equipment: is as specified by the client.

Line 11, Piping: includes all distribution lines for air, water and steam, and all process lines carrying product between flotation heads and tails.

Line 12, Electrical: includes plant distribution to mine and mill and to reclaim, at line voltage per single line diagram. Motor control centers and switchgear are a part of this item as are all conduit and wire.

Line 13, Instrumentation: is an allowance specified by the client.

Line 14, Test & Checkout: includes that effort required to operate equipment on a dry-run basis, to effect calibration as required, and to test interlocks and panels.

Line 17, Taxes: include fixed taxes on materials and subcontract elements required on construction.

Line 18, Escalation: is calculated from the direct cost of the crafts concerned with the job and their existing contracts. The escalation is carried on through mid-summer of 1973. Escalation is also based on the construction schedule and its extended program for high peaks in summer and low peaks in winter. There is also an escalation of 5% on equipment and 10% on contractors' field overhead. Engineering, supervision and procurement are escalated at 10%. Premium time is estimated at 19%.

Line 19, Premium Time: Premium time is based on 20 hours per week for the crafts directly involved in the Labor 00 column, for their burden in the Burden 10 column and for operators of contractors' equipment in the Equipment Usage 20 column.

Line 20, Travel--In-Out: is based on existing union contracts which allow a craftsman hired in Vancouver to request home leave or return to his place of hire at intervals of 30 to 60 days depending on the craft. We have used a time period of 42 days to calculate this figure.

3. Line Numbers and Description Columns (Indirect Costs)

Indirect costs include items which support the direct costs but are not a part of the physical plant.

Line 25, Contractors' Field Overhead: includes salaries and burden for supervisors, engineers, accounting, purchasing and other costs required to construct the plant. Move-in expenses and travel expenses are included for field staff personnel.

Line 26, Construction Plant: includes the tools, shops, and consumable supplies required for construction including utilities and power.

Line 27, Construction Camp: covers the cost of providing a catered mess hall, living quarters, and recreation facilities. Utilities for this item are a part of Line 26.

4. Line 33, Engineering, Supervision and Procurement: covers costs of producing drawings, specifications, purchase and expediting of equipment, project costs, communications and travel for engineering personnel.

B. ESTIMATE CRITERIA

1. The estimate is based on the following schedule:

a. Engineering will commence during March, 1971, and will be substantially complete by June, 1972.

b. Construction will start approximately June 1, 1971, and will be ready for startup by July 15, 1973.

2. Design is based on a standard one shift work week. Construction is based on a one shift 60-hour week.

3. The estimate is based on labour and material prices as of January, 1971.

4. Escalation has been included as a line item based on the schedule shown above. Labour increases beyond the contract expiration dates of April 1, 1972, are estimated at 35¢ per hour increases for each 6-month period.

5. No allowances are included for possible increased costs due to shortage of skilled craft labour or possible delay in material and/or equipment deliveries.

6. Startup and training of operating personnel are not included.

7. The following items are not included:

a. Spare parts

b. Owners cost for engineering, general services and expenses

c. Process and potable water reservoirs and piping to site storage facilities.

d. Access road to plant site and road to the mine

e. Tailings dam, reclaim piping and equipment

f. Land acquisition costs

g. Mine and mine equipment

h. Communications - other than in-plant

i. Owner furnished soil analysis, survey and test work

j. Allowance for work stoppage due to union conflicts

C. SUMMARY OF WORK TO BE PERFORMED

The following is a summary of the work to be performed by Kaiser Engineers in relation to the preliminary engineering and the capital and operating cost estimates for direct cost items:

1. Soils investigation for plant equipment and structures

2. Plant surface drainage and grading including plant roads and parking areas and including culverts as required. (Access road to plant and the road to the mine are not included.)

3. All buildings in the plant area including heating, ventilating, plumbing and lighting. (Pump houses are not included.)

4. All process plant equipment including piping, electrical, instrumentation and controls in accordance with the published flowsheets and the equipment lists

5. Powerplant including diesel generators, fuel storage, boiler section for plant heat and distribution lines to all plant load centres, the mine, the tailings dam and the water dam

6. Water storage and distribution at the process plant including fire water system

7. Tailings disposal line to tailings pond

8. Sanitary sewage system for the plant including the lagoons

9. Construction-camp site preparation, including grading and drainage, sanitary sewage system, water, fire protection and water storage and electrical

10. Gasoline and diesel oil dispensing system at the plant for mobile operating equipment

11. All shop equipment listed on the equipment list

12. Equipment for both metallurgical and analytical laboratories

13. Plant mobile equipment

14. Alternate cost on covered stockpile

The following estimate is for conventional and autogenous process alternatives and a covered stockpile building alternative.

VIII. OPERATING COSTS

A. BASIS FOR ESTIMATE

Operating costs are based on an average of 15,000 dry tons of ore per day for 350 operating days per year, or 5,250,000 dry tons per year. Costs for both the conventional milling and autogenous milling are tabulated and summarized. In each case, direct milling costs are shown separately from indirect operating costs which include power and services. These indirect costs will be distributed between the concentrator and the mine.

B. FREIGHT COSTS

Total freight cost applied to steel and reagent cost is $70.00 per ton. The breakdown is as follows:

(Whitepass and Yukon route) From Vancouver, B. C. to Whitehorse, Yukon $2.90 to $3.00 per cwt.

Trucking costs from Whitehorse to the Adanac site (135 miles @ 8¢ per ton mile) = $11.00 per ton.

Freight costs will rise by approximately 6% per year.

C. POWER CONSUMPTION AND COSTS

Power consumption is based on the power study described in Section V of this report. The power cost used for the operating cost estimate corresponds to power scheme No. 2 (diesel power generation). This cost is 1.570¢ per kwh. Power consumption and costs are summarized in Table VIII-2.

D. REAGENT CONSUMPTION AND COSTS

Reagent consumption and cost data are based on the Adanac Pilot Plant results. These are summarized in Exhibits 16, 17 and 20 in Section IX. Reagent costs represent vendors' estimates. The vendors have recommended an estimated price escalation of 3% to 5% per year. Reagent consumption and costs are summarized in Table VIII-5.

E. STEEL CONSUMPTION AND COSTS

Steel consumption for crushers, grinding mills and other items such as feeders and screens are based primarily on the average of plant data representing operating experience. For steel balls, the most recent Adanac Pilot Plant value of 1.3 pounds per ton of ore is used. Steel costs represent the vendor's estimates. Steel consumption and costs are summarized in Table VIII-6.

F. FUEL COSTS

Fuel costs have been proportioned according to the estimated heating demand of various operations summarized in Table V-3 of the power study. Fuel costs are summarized in Table VIII-7.

G. LABOUR COSTS

Labour costs are based on the guidelines supplied in the manning tables by Chapman, Wood, and Griswold. Direct milling costs are compiled separately from indirect labour costs. Tables VIII-8 to VIII-12 and 15 to 17 summarize the labour costs.

H. SUPPLEMENTARY OPERATING COSTS

Kaiser Engineers calculated milling power costs on the basis of a "19" work index and $0.0157 per kwh power cost. This figure is based on diesel plant power generation at the mine site.

For a comparison, Kaiser Engineers was requested to calculate equivalent figures for a power cost of $0.0350 per kwh, which is estimated to be the cost of purchased power, using a smaller work index of "18".

The higher power rate of $0.0350 per kwh, even with the somewhat smaller power requirement resulting from use of the smaller work index, results in an increased power cost of $2,395,000, or $0.456 per ton of ore milled. Also, the direct milling power is only reduced some 640 hp by the change in work index.

Details of the calculations are shown in Table VIII-13.

I. DIRECT OPERATING COSTS NOT INCLUDED AS PART OF THIS STUDY

1. Labor and Staff Salaries for Services

These include the proportionate share of labor and staff salaries for plant and general services plus administrative and management services by persons in auxiliary services.

2. Maintenance Supplies

These involve supplies other than grinding media, etc.

3. General Plant Overhead Charges

These are charges, not including labor, to operate and maintain auxiliary facilities and corporate management. These charges could include: power, heat, office supplies, telephone and telegraph, staff travel, yard and road maintenance, materials, home office, cars and plant vehicles, lab supplies, medical and safety supplies and maintenance supplies.

TABLE VIII-1

SUMMARY AND COMPARISON OF DIRECT MILLING COSTS OF PROCESS ALTERNATIVES*

Item	Annual Cost Alt 1	Annual Cost Alt 2	Cost/Ton of Ore Alt 1	Cost/Ton of Ore Alt 2
Power	$2,047,350	$2,175,350	$0.3898	$0.4142
Reagents	829,400	829,400	0.1580	0.1580
Steel	2,062,000	2,048,550	0.3928	0.3902
Heating	122,850	122,850	0.0234	0.0234
Labour	1,145,900	1,145,900	0.2183	0.2183
TOTAL	$6,207,500	$6,321,550	$1.1823	$1.2041

*Alternative No. 1, Rod Mill-Ball Mill operation
Alternative No. 2, Autogenous operation

TABLE VIII-2

SUMMARY AND COMPARISON OF CONCENTRATOR POWER COSTS OF PROCESS ALTERNATIVES

Item	Total HP Alt 1	Total HP Alt 2	Cost/Year Alt 1	Cost/Year Alt 2	Cost/Ton Alt 1	Cost/Ton Alt 2
Primary crushing	1,115	1,308	$ 68,600	$ 115,843	$0.0131	$0.0221
Fine crushing	2,053	--	151,530	--	0.0289	--
Grinding	14,720	17,365	1,303,000	1,535,500	0.2481	0.2925
Flotation	3,910	3,910	346,100	346,100	0.0659	0.0659
Concentrate handling	333	333	29,340	29,340	0.0056	0.0056
SUBTOTAL	22,131	22,916	$1,899,000	$2,027,000	$0.3616	$0.3860
Lighting	1,500 kw	1,500 kw	148,350	148,350	0.0282	0.0282
TOTAL			$2,047,350	$2,175,350	$0.3898	$0.4142

Power cost = $0.0157/kwh
Power costs based on a work index of 19 and a power cost of $0.0350 are summarized as a supplement in Table VIII-13.

TABLE VIII-3

CONCENTRATOR POWER COST CALCULATIONS
ALTERNATIVE NO. 1 (ROD MILL-BALL MILL OPERATION

Process	Cost
PRIMARY CRUSHING	
(1. 115 hp) (0. 746) = 832 kw	
(832 kw) (350 days/yr) (20 hr/day) (0. 751f) ($0. 0157/kwh) =$68, 600	
$68, 600 ÷ 5, 250, 000 tpy =	$0. 0131/ton
FINE CRUSHING	
(2. 053 hp) (0. 746) = 1. 532 kw	
(1. 532) (350) (20) (0. 90) (0. 0157) = $151, 530	
$151, 530 ÷ 5, 250, 000 =	$0. 0289/ton
GRINDING AND CLASSIFICATION	
(14, 720 hp) 0. 746 = 10, 981 kw	
(10, 981) (350) (24) (0. 90) (0. 0157) = $1, 303, 000	
$1, 294, 000 ÷ 5, 250, 000 =	$0. 2481/ton
FLOTATION	
(3, 910 hp) (0. 746) = 2, 917 kw	
(2, 917) (350) (24) (0. 90) (0. 157) = $346, 100	
$346, 100 ÷ 5, 250, 000 =	$0. 0659/ton
CONCENTRATE HANDLING	
(333 hp) (0. 746) = 248 kw	
(248) (350) (24) (0. 90) (0. 0157) = $29, 430	
$29, 430 ÷ 5, 250, 000 =	$0. 0056/ton

	Cost/yr	Cost/ton
TOTAL	$1, 898, 660	$0. 3616

TABLE VIII-4

CONCENTRATOR POWER COST CALCULATIONS
ALTERNATIVE NO. 2 (AUTOGENOUS OPERATION)

Process	Cost
CRUSHING	
(1, 308 hp) (0.746) = 976 kw	
(976) (350) (24) (0.90) ($0.0157) = $115, 843	
$115, 843 ÷ 5, 250, 000 =	$0.0221/ton
GRINDING	
(17, 365 hp) (0.746) = 12, 937 kw	
(12, 937) (350) (24) (0.90) (0.0157) = $1, 535, 500	
1, 535, 500 ÷ 5, 250, 000 =	$0.2925/ton
FLOTATION	
(39.0 hp) (0.746) = 2, 917 kw	
(2, 917) (350) (24) (0.90) (0.0157) = $346, 100	
$346, 100 ÷ 5, 250, 000 =	$0.0659/ton
CONCENTRATE HANDLING	
(333 hp) (0.746) = 248 kw	
(248) (350) (24) (0.90) (0.0157) = $29, 340	
29, 340 ÷ 5, 250, 000 =	$0.0056/ton

	Cost/Yr	Cost/Ton
TOTAL	$2, 027, 000	$0.3860

TABLE VIII-5

CONCENTRATOR FLOTATION REAGENT COSTS*

Reagent	Lb/Ton of Ore	Cost/Lb Delivered (Cents)	Cost/Ton of Ore (Cents)
Lime	1.100	5.1	5.610
Arctic Syntex L	0.010	45.0	0.450
Sodium sulphide	0.400	13.0	5.200
Sodium silicate	0.050	6.1	0.305
Shell Carnea 21	0.220	8.0	1.760
Dowfroth 250	0.065	38.0	2.470
Superfloc **	-	203.0	0.003
TOTAL	1.845		15.798

ANNUAL COST (5,250,000 tpy) (0.15798) = $ 829,400

ANNUAL TONNAGE (5,250,000 tpy) (1.845) = 9,686,000 lb
= 4,843 tons

* Same for both process alternatives
**Consumption = 0.005 lb/ton of concentrates

TABLE VIII-6

CONCENTRATOR STEEL CONSUMPTION AND COSTS OF PROCESS ALTERNATIVES*

Item	Lb/Ton of Ore Alt 1	Lb/Ton of Ore Alt 2	Cost/Lb Delivered (Cents) Alt 1	Cost/Lb Delivered (Cents) Alt 2	Cost/Ton of Ore (Cents) Alt 1	Cost/Ton of Ore (Cents) Alt 2
Primary crusher	0.010	-	43.0	-	0.43	-
Cone crushers	0.050	-	49.9	-	2.495	-
Jaw crushers	-	0.040	-	43.0	-	1.720
Rod mill liners	0.052	-	33.5	-	1.742	-
Ball mill liners	0.055	0.055	33.5	33.5	1.842	1.842
Autogenous mill liners	-	0.40	-	33.5	-	13.400
Rods	1.020	-	10.5	-	10.710	-
Balls	1.300	1.300	16.2	16.2	21.060	21.060
All other items	0.083	0.083	12.0	12.0	1.000	1.000
TOTAL	2.570	1.878			39.279	39.022

Annual cost (Alt 1) (5,250,000) ($0.39279) = $2,062,000
Annual cost (Alt 2) (5,250,000) ($0.39022) = $2,048,550
Annual tonnage (Alt 1) (5,250,000) (2.57) = 13,492,000 lb = 6,746 tons
Annual tonnage (Alt 2) (5,250,000) (1.878) = 9,859,500 lb = 4,930 tons

*Alternative No. 1 Rod Mill-Ball Mill operation
Alternative No. 2 Autogenous operation

TABLE VIII-7

CONCENTRATOR HEATING COSTS*

Item	Btu
Fuel and labour costs (auxiliary boiler)	\$2.50/million Btu
From Table V-3:	
Waste heat boiler output	21,000,000 Btu/hr
Auxiliary heat boiler requirements	9,100,000 Btu/hr
Plant area heating requirements**	30,100,000 Btu/hr
Assume a process plant requirement of 9.1×10^6 Btu for 7-1/2 months from auxiliary boiler	
(30) (24) = 720 hr/mo	
(720) (7.5) (9.1) (\$2.50) =	\$122,850/yr
Cost/ton = 122,850 ÷ 5,250,000 =	\$0.0234
Ancillary heating costs to be distributed by CWG	
From Table V-3:	
(720) (7.5) (11.54) (\$2.50) =	\$155,790
Cost/ton = 155,790 ÷ 5,250,000	\$0.0297

*Same for both process alternatives.
**All of waste heat credited to concentrator.

TABLE VIII-8

SUPERVISORY AND LABOUR COSTS FOR ADANAC CONCENTRATOR*

Classification	Number of Personnel	Base Cost/Yr	Fringe Benefits at 20%	Total Cost/Yr	Cost Per Ton of Ore
Supervision	10	$132, 000	$ 26, 400	$ 158, 400	$0. 0302
Research and assay office	12	115, 000	23, 000	138, 000	0. 0263
Milling labour	53	471, 600	94, 300	565, 900	0. 1078
Maintenance labour	23	236, 930	47, 300	283, 600	0. 0540
TOTAL DIRECT LABOUR	98	$954, 900	$191, 000	$1, 145, 900	$0. 2183

*Costs are the same for both process alternatives.

TABLE VIII-9

MILLING, ADMINISTRATION, AND SUPERVISORY COSTS FOR ADANAC CONCENTRATOR*

Classification	Number of Personnel	Unit Rate	Cost Per Month
Mill superintendent	1	$1,600	$ 1,600
Chief metallurgist	1	1,200	1,200
Mill foreman	1	1,200	1,200
Shift foreman	3	1,000	3,000
Crusher foreman	1	1,000	1,000
Repair foreman	1	1,000	1,000
Relief foreman	2	1,000	2,000
TOTAL	10		$ 11,000
Salaries per year			$132,000
Fringe benefits at 20%			26,400
TOTAL PER YEAR			$158,000
Cost Per Ton of Ore			$0.0302

*Costs are the same for both process alternatives.

TABLE VIII-10

COSTS FOR RESEARCH AND ASSAY OFFICES FOR ADANAC CONCENTRATOR*

Classification	Number of Personnel	Unit Rate	Cost Per Month
Chief chemist	1	$1, 100	$ 1, 100
Shift chemist	3	900	2, 700
Metallurgist	1	900	900
Technician	3	4. 43	2, 130
Sampler	3	4. 30	2, 065
Helper	1	4. 30	690
TOTAL	12		$ 9, 585
Salaries and wages per year			$115, 000
Fringe benefits at 20%			23, 000
TOTAL PER YEAR			$138, 000
Cost Per Ton of Ore			$0. 0263

*Costs are the same for both process alternatives.

TABLE VIII-11

OPERATING LABOUR COSTS FOR ADANAC CONCENTRATOR*

Classification	Number of Personnel	Unit Rate	Cost Per Month
Instrument operator	3	$5.37	$ 2,580
Crushing-screening operator	6	4.56	4,380
Grinding operator	3	4.56	2,190
Flotation operator	6	4.99	4,790
Filter operator	3	4.56	2,190
Leach operator	3	4.74	2,280
Packing operator	3	4.56	2,190
Tailings operator	3	4.43	2,130
Loading operator	1	4.74	760
Relief operator	10	4.70	7,520
Sampler	3	4.30	2,060
Reagent mixer	1	4.56	730
Helper	8	4.30	5,500
TOTAL	53		$ 39,300
Wages per year			$471,600
Fringe benefits at 20%			94,300
TOTAL PER YEAR			$565,900
Cost Per Ton of Ore			$0.1078

*Costs are the same for both process alternatives.

TABLE VIII-12

MAINTENANCE LABOUR COSTS FOR ADANAC CONCENTRATOR*

Classification	Number of Personnel	Unit Rate	Cost Per Month
Shift mechanic - 1st class	6	$5.62	$ 5,395
Shift mechanic - 2nd class	6	5.37	5,155
Shift mechanic relief	3	5.37	2,580
Shift electrician	3	5.62	2,700
Shift electrician relief	1	5.62	900
Instrument man	1	5.62	900
Helpers	3	4.30	2,060
TOTAL	23		$ 19,690
Wages per year			$236,300
Fringe benefits at 20%			47,300
TOTAL PER YEAR			$283,600
Cost Per Ton of Ore			$0.0540

*Costs are the same for both process alternatives.

TABLE VIII-13

SUPPLEMENTARY POWER COSTS COMPARISON
(Alternative No. 1 Rod Mill-Ball Mill Operation)

Item	WI = 19.0 ($0.0157/kwh) Per /Year	Cost /ton	WI = 18.0 ($0.0350/kwh) Per /Year	Cost /ton
Primary Crusher	$ 68,600	$0.0131	$ 152,880	$0.0291
Fine Crushing	151,530	0.0289	337,800	0.0634
Grinding	1,303,000	0.2481	2,783,590	0.5302
Flotation & Concentrate Handling	375,400	0.0705	837,460	0.1595
SUBTOTAL	$1,988,000	$0.4284	$4,111,730	$0.7832
Lighting	148,350	$0.0335	$ 330,750	$0.0630
TOTAL	$2,047,350	$0.4619	$4,442,480	$0.8462

WI = 18, power = $0.0350 Cost = $4,442,480

WI = 19, power = $0.0157 Cost = $2,047,350

Difference $2,395,000

$$\frac{2,395,000}{5,250,000} \times 100 = 45.6 \text{ cents per ton difference}$$

TABLE VIII-13 (Cont)
SUPPLEMENTARY POWER COST CALCULATIONS
(Alternative No. 1 Rod Mill-Ball Mill Operation)

WI = 18.0 Power Cost = $0.0350/Kwh

PRIMARY CRUSHING

(1,115 hp) (0.746) = 832 kw

(832) (350) (20) (0.75) ($0.0350) = $ 152,880

FINE CRUSHING

(2,053 hp) (0.746) = 1,532 kw

(1,532) (350) (20) (0.90) ($0.0350) = 337,800

GRINDING

hp = 14,720 - 650 = 14,100

(14,100 hp) (0.746) = 10,520 kw

(10,520) (350) (24) (0.90) ($0.0350) = 2,783,590

FLOTATION AND CONCENTRATE HANDLING

(4,243 hp) (0.746) = 3,165 hp

(3,165) (350) (24) (0.90) ($0.0350) = 837,460

$4,111,730

cost/ton of ore = $0.7832

LIGHTING

(1,500 kw) (350) (24) (0.75) ($0.0350) = $ 330,750

cost/ton of ore = $0.0630

TABLE VIII-13 (Cont)
SUPPLEMENTARY POWER CALCULATIONS

$$\text{kwh/ton} - \text{WI}\left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right)$$

WI = Work Index = 18.0

Scheme I*

Rod Mills	**F = 11,200	P = 1,190	Tonnage = 660 tph	hp = 3,109
Ball Mills	F = 1,190	P = 170	Tonnage = 660 tph	hp = 7,588

Scheme 2***

Autogenous Mills	F = 164,000	P = 1,190	T = 735	hp = 4,696
Ball Mills	F = 1,190	P = 170	T = 735	hp = 8,450

Comparison

Scheme 1	WI = 19.0	WI = 19.0	WI = 18.0	WI = 18.0
Rod Mills****	3,290 + 10%	3,620	3,109 + 10%	3,420
Ball Mills	8,030	8,030	7,588	7,588
TOTAL	11,320	11,650	10,697	11,008

Scheme 2	WI = 19.0	WI = 18.0
Autogenous Mills	4,960	4,690
Ball Mills	8,970	8,450
TOTAL	13,930	13,140

* Scheme 1 = Alternative No. 1 (Rod Mill-Ball Mill Generation)
** F = 80% passing size, feed
P = 80% passing size, product
*** Scheme 2 = Alternative No. 2 (Autogenous Generation)
**** Rod Mills have 10% added for broken rods

TABLE VIII-13 (Cont)
SUPPLEMENTARY POWER CALCULATIONS

ROD MILLS kwh/ton = $18\left(\frac{10}{\sqrt{P}} - \frac{10}{\sqrt{F}}\right)$ 660 tph

$$18\left(\frac{10}{\sqrt{1,190}} - \frac{10}{\sqrt{11,200}}\right) = \frac{180}{34.5} - \frac{180}{105.8} + 5.217 - 1.701 = 3.516$$

hp = kw (1.34) hp = 3.516 (1.34) = 4.711 hp/tph
hp = (660) (4.711) = 3,109.26

BALL MILLS

$$18\left(\frac{10}{\sqrt{170}} - \frac{10}{\sqrt{1,190}}\right) = \frac{180}{13.04} - \frac{180}{34.5} = 13.80 - 5.217 = 8.58$$

(8.58 kwh/t) (1.34) = 11.497 hp Hr/t

hp = (660) (11.497) = 7,588

ROD MILL	3,109	ROD MILL = $\frac{3,109 \times 100}{10,697}$ = 29.1% of Total
BALL MILL	7,588	
TOTAL	10,697	

AUTOGENOUS MILL

$$18\left(\frac{10}{\sqrt{1,190}} - \frac{10}{\sqrt{164,000}}\right) = \frac{180}{34.5} - \frac{180}{405} = 5.217 - 0.444 = 4.773$$

(4.773) (1.34) = 6.39
(735) (6.39) = 4,696 hp

Ball Mills hp = (735) (11.497) = 8,450 hp

TABLE VIII-14

INDIRECT OPERATING COST SUMMARY*

Item	Annual Cost	Cost Per Ton of Ore
Power		
Shops and warehouse lighting	$ 79,116	
Office lighting	1,975	
Site lighting	39,560	
Powerhouse lighting	33,321	
SUBTOTAL	$ 153,972	$0.0293
Power plant auxiliaries	168,210	0.0320
TOTAL AUXILIARY POWER	$ 322,182	$0.0613
Services		
Administration	$ 228,200	$0.0435
Home Office expense	36,000	0.0068
General engineering	95,000	0.0181
Plant services	592,100	0.1128
SUBTOTAL	$ 951,300	$0.1812
TOTAL	$1,333,390	$0.2539

*Indirect operating costs are the same for both process alternatives. These are kept separate from direct operating costs and will be divided between the concentrator and mine at Adanac by C W & G Ltd.

TABLE VIII-15

HOME OFFICE AND ADMINISTRATION COSTS FOR ADANAC CONCENTRATOR*

Classification	Number of Personnel	Unit Rate	Cost Per Month
Head Office			$ 3,000
Manager	1	$2,300	2,300
Assistant manager	1	1,800	1,800
Chief accountant	1	1,200	1,200
Purchasing agent	1	1,000	1,000
Payroll accountant	1	800	800
Stores accountant	1	800	800
Personnel director	1	1,100	1,100
Stores and shipping clerks	2	700	1,400
Warehousemen	4	650	2,600
Timekeeper	1	650	650
Stenos	4	450	1,800
Phone operator	1	400	400
TOTAL	19		$ 15,850
Salaries per year			$190,200
Fringe benefits at 20%			38,040
Home office per year			36,000
TOTAL PER YEAR			$264,240
Cost Per Ton of Ore			$0.0503

*Costs are the same for both process alternatives and will be divided between the concentrator and mine at Adanac by C W & G Ltd.

TABLE VIII-16

GENERAL ENGINEERING COSTS FOR
ADANAC CONCENTRATOR*

Classification	Number of Personnel	Unit Rate	Cost Per Month
Chief engineer	1	$1,400	$ 1,400
Geologist	1	1,200	1,200
Pit engineer	1	1,100	1,100
Surveyor	1	900	900
Surveyor helper	1	750	750
Draftsman	1	650	650
Technician	1	650	650
TOTAL	7		$ 6,600
Salaries per year			$79,200
Fringe benefits at 20%			15,800
TOTAL PER YEAR			$95,000
Cost Per Ton of Ore			$0.0181

*Costs are the same for both process alternatives and will be divided between the concentrator and mine at Adanac by C W & G Ltd.

TABLE VIII-17

PLANT SERVICES COST FOR ADANAC PROJECT*

Classification	Number of Personnel	Unit Rate	Cost Per Month
Plant engineer	1	$1,600	$ 1,600
Electrical superintendent	1	1,400	1,400
Shop foreman	1	1,100	1,100
Electrical foreman	1	1,100	1,100
Surface foreman	1	1,000	1,000
Safety engineer	1	1,000	1,000
Medical officer	1	1,000	1,000
First aid attendant	4	800	3,200
Security guard	8	700	5,600
Carpenter	2	5.62	1,800
Electrician	1	5.62	900
Machinist	2	5.62	1,800
Pipefitter	1	5.62	900
Mechanic - 1st class	1	5.62	900
Rigger	2	5.37	1,720
Heating plant operator	4	4.96	3,175
Powerplant operator	4	4.96	3,175
Dozer operator	1	4.77	765
Grader operator	1	4.77	765
Welder	1	4.77	765
Truck driver	4	4.74	3,035
Maintenance helper	4	4.74	3,035
Surface labourer	2	4.30	1,380
TOTAL	49		$ 41,115
Salaries and wages per year			$493,400
Fringe benefits at 20%			98,700
TOTAL PER YEAR			$592,100
Cost Per Ton of Ore			$0.1128

*Costs are the same for both process alternatives and will be divided between the concentrator and mine at Adanac by C W & G Ltd.

IX. PROJECT CRITERIA

The criteria in this section are divided into two parts, process and design.

The process criteria have been developed from various sources including observation of the pilot plant results at the Adanac site, on information supplied by others in the consulting group and by vendors on specific equipment items. Calculations and data based on these criteria are presented as technical exhibits that are referenced in Section IV to supplement the process description.

The design criteria have been written specifically for the Canadian Building Code and are intended as a guide for design and detailed engineering. The criteria conforms with that of the Canadian codes and professional groups.

PROCESS CRITERIA

Exhibit No.	Title
1	Calculations for Crushing and Screening Plants
2	Screen Tests of Adanac Pilot Plant Products and Extrapolated Curves
3	Size Distribution of Jaw Crusher Product, Adanac Pilot Plant
4	Size Distribution of Ball Mill Feed, Adanac Pilot Plant
5	Size Distribution of Ball Mill Discharge, Adanac Pilot Plant
6	Estimated Rod Mill Feed Size Distribution
7	Extrapolated Size Distribution of Minus 3/4-in. Rod Mill Feed
8	Size Distribution of 10-in. Cyclone Overflow, Adanac Pilot Plant
9	Size Distribution of 10-in. Cyclone Underflow, Adanac Pilot Plant
10	Bulk Densities, Assorted Products From Pilot Plant
11	Grinding Circuit Calculations
12	Summary of Material Balance of one Rod Mill - Ball Mill Cyclone Circuit
13	Schematic of One Rod Mill - Ball Mill Cyclone Circuit
14	Screen Tests, Adanac Pilot Plant Flotation Products
15	Estimated Steel Consumption for Suggested Size Reduction Flowsheets

EXHIBIT 1

CALCULATIONS FOR CRUSHING AND SCREENING PLANTS

I. CALCULATIONS FOR ESTIMATED AVAILABILITY OF CRUSHING PLANT

A. Primary Crusher

1. 54-in. Gyratory Crusher

Proposed open-pit production	90 tons dumped every 3-1/2 min for 10 hr/day
Average feed rate to crusher	$\frac{(60)\,(90)}{3\text{-}1/2} = 1,540$ tph
Tons fed per 10-hr day	(1, 540) (10) = 15, 400

Rated capacity of 54-in. gyratory with open-side-setting of 6 in.

Allis-Chalmers	1, 770 tph (equipment representative's estimate)
Nordberg	1, 440 tph (page 4, Nordberg Bulletin 318B)
Traylor	1, 790 tph (Traylor Bulletin TCB-1B page 30)

2. 66-in. by 84-in. Jaw Crusher

Feed rate, as shown in (1) above	1, 540 tph

Rated capacity of 66-in. by 84-in. jaw crusher set at 8 in.

Allis-Chalmers*	876 tph

*Allis-Chalmers Canada Ltd. Bulletin A-CC 1296

Traylor Type "H"* 1,050 tph

If a fixed rail grizzly with 4-in. spacings is placed before the jaw crusher, grizzly undersize will be 10% from Exhibit 2.

Capacity of combined grizzly and jaw crusher will be increased to:

Allis-Chalmers $\frac{876}{0.9} = 973$ tph

Traylor Type "H" $\frac{1,050}{0.9} = 1,167$ tph

B. Secondary and Tertiary Crushers

1. For Preventative Maintenance

Allow 2 hr per day
one 8-hr shift per week

Available operating time per week will be (7 days/week) (3 shifts/day) (8 hr/shift) minus (7 days/week) (2 hr/day) minus (8 hr/week) = 146 hr/week

Total hr/week = (24 hr/day) (7 days/week) = 168

Available operating time % = $\frac{146}{168} \times 100 = 87\%$

or 20.8 hours per day

2. For Contingencies

Allow 95% if available operating time

Actual daily operating time will be

(24 hr) (0.87) (0.95) = 19.8 hr

Hourly capacity of crushing plant $\frac{15,000}{19.8} = 755$ tph

*Traylor Bulletin TCB-4, Page 21

II. SUMMARY OF SCREEN AREA CALCULATIONS*

Alternative & Operation	C.A.C.	Tyler
DRAWING NO. 7008-G-2000		
Primary Screens after primary gyratory	@ 755 tph, recommend one 5-ft by 16-ft double-deck screen, 2-1/2-in. openings over 3/4-in. slotted openings. To handle the estimated capacity of crusher @ 1,500 tph recommend one 8-ft by 20-ft unit, with similar double decks. Required area: Top deck = 66 sq ft Bottom deck = 55 sq ft	@ 755 tph, required area = 84 sq ft or 8.98 tph/sq ft. Recommend one 6-ft by 14-ft sloped @ 25°. To handle the estimated top capacity of crusher @ 1,500 tph recommend two 6-ft by 14-ft units.
Secondary Screens	@ 755 tph reclaim from coarse ore storage, recommend one 8-ft by 20-ft double-deck unit with 1-1/2-in. openings over 3/4-in. slotted openings. Screen area required = 155 sq ft	@ 860 tph (755 tph plus circulating load from tertiary crushers) required area = 224 sq ft or 3.83 tph/sq ft. Recommend two 8-ft by 14-ft units sloped @ 23°.
DRAWING NO. 7008-G-2100		
Primary Screens after crusher	@ 755 tph or @ estimated top capacity of jaw crusher = 973 tph. Recommend one 8-ft by 20-ft double-deck screen with 2-1/2-in. openings over 3/4-in. slotted openings.	@ 755 tph, same estimates and recommendation as for Drawing No. 7008-G-2000. To handle the estimated top capacity of 965 tph including 10% scalping with a grizzly, recommend one 6-ft by 14-ft @ 25° or two 6-ft by 14-ft preferred. one 6 ft by 16 ft or one 8 ft by 14 ft
Wet Vibrating Screens in closed circuit with wet autogenous grinding mills	@ 735 tph total feed plus estimated 35% plus 6-mesh oversize (circulating load determined to be 63.5% or 468 tph). Screen area required = 445 sq ft. Three 8-ft by 20-ft screens have (3) (150) = 450 sq ft. Recommend four 8-ft by 20-ft screens.	@ 370 tph plus estimated 35% plus 6-mesh oversize, or approximately 500 tph per line. Recommend two 8-ft by 20-ft single-deck units.

*Estimates by Canadian Allis-Chalmers and W. S. Tyler Co. Representatives. Estimates based on data in Exhibit 2.

III. CALCULATIONS FOR TONNAGE DISTRIBUTION ON CONVEYORS, SCREENS, CRUSHERS, AND FEEDERS

A. DRAWING NO. 7008-G-2000

1. 54-in. Conveyor 8-ft by 20-ft Primary Screen

Average tonnage = average feed rate to primary crusher, or 1, 540 tph

Maximum tonnage = rated capacity of 54-in. primary crusher set at 6 in. or 1, 770 tph

2. 30-in. Conveyor Accepting Minus 3/4-in. Primary Screen Undersize

Average tonnage @ 100% screen efficiency @ 13% minus 3/4-in. undersize (curve 3 in Exhibit 2)

$$= 1,540 \text{ tph} \left(\frac{13}{100}\right) = 200 \text{ tph}$$

Maximum tonnage $= 1,770 \text{ tph} \left(\frac{13}{100}\right) = 230 \text{ tph}$

3. 48-in. Conveyor Accepting Plus 3/4-in. Primary Screen Oversize

Average tonnage = 1, 540 - 200 = 1, 340 tph

Maximum tonnage = 1, 770 - 230 = 1, 540 tph

4. 42-in. Conveyor for Coarse-Ore Stockpile Reclaim

Average tonnage based on feeding fine-ore bins @ 1, 770 tph for 10 hours with 13% minus 3/4-in. undersize, with balance of milling tonnage drawn from coarse-ore stockpile.

Average tonnage to 10, 000 ton capacity fine-ore bin from primary crusher and primary screen

$$= 1,770 \text{ tph} \left(\frac{13}{100}\right)(10 \text{ hr}) = 2,300 \text{ tons}$$

Balance of tonnage required from coarse-ore stockpile

$$= 10,000 - 2,300 = 7,700 \text{ tons}$$

Average daily operating time for secondary and tertiary crushers (Item A-2) will be 19.8 hours.

Average tonnage drawn from coarse-ore stockpile

$$= \frac{7,700 \text{ tons}}{19.8 \text{ hours}} = 389 \text{ tph}$$

Maximum tonnage based on 1,200 tph, the rated capacity of a secondary crusher set at 2 in.

5. <u>48-in. Conveyor Collecting All Secondary and Tertiary Crusher Discharge</u>

Average tonnage based on 84% minus 1-1/2-in. product and 42% minus 3/4-in. product in secondary crusher (curve 6 in Exhibit 2) and crusher manufacturer's data for estimating circulating load = 35%.

Average tonnage from coarse-ore stockpile (Item 4)
= 389 tph

Average tonnage through both tertiary crushers, based on 84% passing 1-1/2-in. product, 42% passing 3/4-in. product, and circulating load of 35% to tertiary crushers
= 306 tph

<u>TOTAL AVERAGE TONNAGE</u> = 389 + 306 = 695 tph

Maximum tonnage based on same distribution of minus 1-1/2-in. and minus 3/4-in. product and circulating load of 35%.

Maximum tonnage from coarse-ore stockpile based on rated capacity of secondary crusher
= 1,200 tph

Maximum tonnage through both tertiary crushers
= 940 tph

<u>TOTAL MAXIMUM TONNAGE</u> = 1,200 + 940 = 2,140 tph

6. Two 48-in. Scissor Conveyors Discharging All Secondary and Tertiary Crusher Product Into Hopper Before Secondary Screening

Average tonnage = as for item (5)
= 695 tph

Maximum tonnage = as for item (5)
= 2, 140 tph

7. 8-ft by 20-ft Secondary Screens

Minus 3/4-in. product, based on 84% minus 1-1/2-in product, and 42% minus 3/4-in. product from crushers, and 35% circulating load.

TOTAL AVERAGE TONNAGE OF MINUS 3/4-IN. PRODUCT
= 389 tph

TOTAL MAXIMUM TONNAGE OF MINUS 3/4-IN. PRODUCT
= 1, 200 tph

8. 42-in. and 48-in. Conveyors Collecting All Minus 3/4-in. Product

Average tonnage, based on 200 tph from primary crusher and screening, with balance drawn from coarse-ore stockpile.

From primary crusher and screening, 200 tph for 10 hr fills fine-ore bin with 2, 000 tons.

Therefore, balance required from coarse-ore stockpile
= 10, 000 - 2, 000 = 8, 000 tons in 19. 8 hr or 404 tph

TOTAL AVERAGE TONNAGE = 200 + 404 = 604 tph

Maximum tonnage based on primary and secondary crushers both operating at rated capacity:

Primary Crushing = 230 tph

Secondary and tertiary crushing
= 1, 200 tph

TOTAL MAXIMUM TONNAGE = 1, 430 tph

B. DRAWING NO. 7008-G-2100

1. Grizzly Feed

Average feed rate based on calculated hourly capacity of crushing plant (Item A-2)

$$= 755 \text{ tph}$$

Maximum feed rate based on 10% of minus 4-in. undersize scalped out by grizzly (curve 1, Exhibit 2)

$$= \frac{755}{0.90} = 973 \text{ tph}$$

2. Jaw Crusher Feed

Average feed rate @ 90% of grizzly oversize

$$= (755)\ (0.9) = 680 \text{ tph}$$

Maximum tonnage = rated capacity of jaw crusher, or 876 tph

3. 48-in. Conveyor Accepting Discharge from Grizzly and Jaw Crusher

Average tonnage = 755 tph

Maximum tonnage = 973 tph

4. 6-ft by 16-ft Primary Screen

Tonnages of oversize and undersize based on 14.5% minus 3/4-in. product and 90% screening efficiency.

Minus 3/4-in. undersize to fine-ore bins (two 24-in. conveyors):

Average tonnage $= (0.90)\left(\frac{14.5}{100}\right)(755)$

$= 99$ tph

Maximum tonnage $= (0.90)\left(\frac{14.5}{100}\right)(973)$

$= 127$ tph

Minus 8-in. plus 3/4-in. oversize to coarse-ore stockpile (two 48-in. conveyors):

Average tonnage = 755 - 99 = 656 tph

Maximum tonnage = 973 - 127 = 846 tph

5. Feeders, Fine-Ore Bins

Feed proportioned according to production of minus 3/4-in. undersize and plus 3/4-in. oversize.

Average tonnage (each of four feeders)

$$= \frac{99}{4} = 25 \text{ tph}$$

Maximum tonnage (each of four feeders)

$$= \frac{127}{4} = 32 \text{ tph}$$

6. Feeders, Coarse-Ore Stockpile

Average tonnage (each of four feeders)

$$= \frac{656}{4} = 164 \text{ tph}$$

Maximum tonnage (each of four feeders)

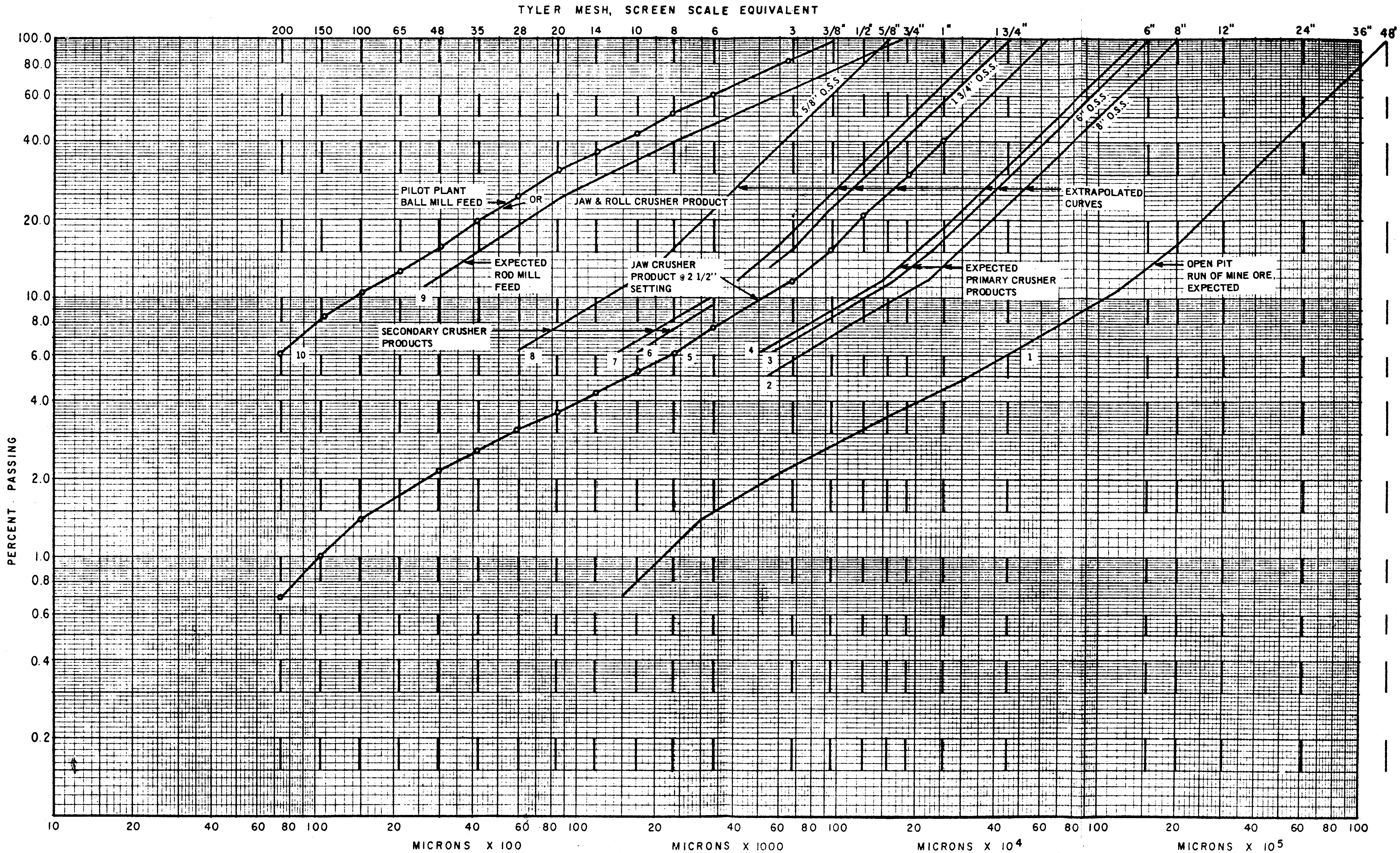
$$= \frac{846}{4} = 212 \text{ tph}$$

IV. SUMMARY OF CONVEYOR CAPACITIES FOR 100-POUND/CU FT MATERIAL*

Capacity tph

Belt Width, Inches	Belt Speed in Feet Per Minute 300	400	600
24	405	540	810
30	675	900	1, 350
42	1, 365	1, 820	2, 730
48	1, 815	2, 420	3, 630
54	2, 355	3, 140	4, 710

*Link-Belt Book 3216, Table 5, page 71, Class B Loading, Idler troughing angle 35 degrees.



SCREEN TESTS OF ADANAC PILOT PLANT PRODUCTS & EXTRAPOLATED CURVES

EXHIBIT 2

EXHIBIT 3

SIZE DISTRIBUTION OF JAW CRUSHER PRODUCT ADANAC PILOT PLANT*

Tyler Mesh	Wt % Retained	Wt % Retained Cumulative	Wt % Passing Cumulative
Approx minus 2-1/2 plus 1 inch	59.5	59.5	40.5
minus 1 plus 1/2 inch	19.5	79.0	21.0
minus 1/2 plus 3/8 inch	5.6	84.6	15.4
minus 3/8 inch plus 3 mesh	3.7	88.3	11.7
plus 6	4.0	92.3	7.7
plus 8	1.5	93.8	6.2
plus 10	1.1	94.9	5.1
plus 14	0.8	95.7	4.3
plus 20	0.7	96.4	3.6
plus 28	0.5	96.9	3.1
plus 35	0.5	97.4	2.5
plus 48	0.5	97.9	2.1
plus 65	0.4	98.3	1.7
plus 100	0.3	98.6	1.4
plus 150	0.4	99.0	1.0
plus 200	0.3	99.3	0.7
minus 200	0.7	100.0	-
TOTAL	100.0		

*Crusher setting approximately 2-1/2 in. No further adjustment possible (See Exhibit 2)

EXHIBIT 4

SIZE DISTRIBUTION OF BALL MILL FEED
ADANAC PILOT PLANT*

Tyler Mesh	% Retained	% Retained Cumulative	% Passing Cumulative
Minus 1/2 plus 3/8 inch	4.20	4.2	95.8
Minus 3/8 inch plus 3 mesh	12.5	16.7	83.3
Plus 6	22.2	38.9	61.1
Plus 8	9.9	48.8	51.2
Plus 10	7.7	56.5	43.5
Plus 14	7.2	63.7	36.3
Plus 20	6.0	69.7	30.3
Plus 28	5.2	74.9	25.1
Plus 35	5.3	80.2	19.8
Plus 48	4.2	84.4	15.6
Plus 65	2.8	87.2	12.8
Plus 100	2.3	89.5	10.5
Plus 150	2.2	91.7	8.3
Plus 200	2.2	93.9	6.1
Minus 200	6.1	100.0	-
TOTAL	100.0		

*Belt sample of crushing plant (jaw crusher and rolls) product = ball mill feed, 8 to 4 shift, No. 118, September 29, 1970 (see Exhibit 2)

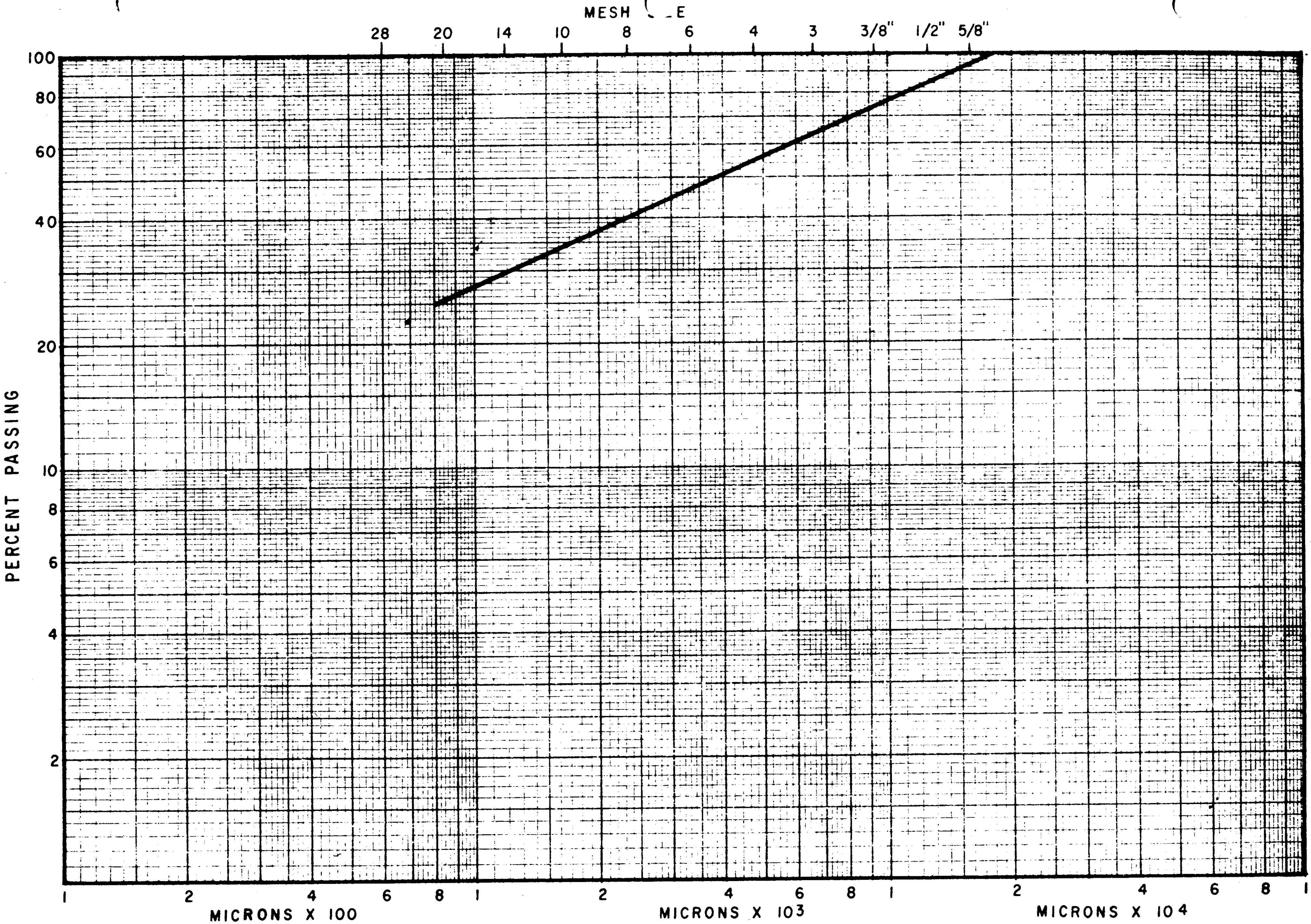
EXHIBIT 5

SIZE DISTRIBUTION OF BALL MILL DISCHARGE ADANAC PILOT PLANT*

Tyler Mesh	% Retained	% Retained Cumulative	% Passing Cumulative
8	0.5	0.5	99.5
10	0.8	1.3	98.7
14	1.3	2.6	97.4
20	2.6	5.2	94.8
28	4.5	9.7	90.3
35	9.2	18.9	81.1
48	16.2	35.1	64.9
65	17.5	52.6	47.4
100	10.9	63.5	36.5
150	8.6	72.1	27.9
200	6.3	78.4	21.6
325	4.6	83.0	17.0
Minus 325	17.0	100.0	-
TOTAL	100.0		

80% passing size = 400 microns
% solids = 67%

*Lateral development ore, 4 to 12 shift, No. 119, September 29, 1970



ESTIMATED ROD MILL FEED
SIZE DISTRIBUTION

EXHIBIT

EXHIBIT 7

EXTRAPOLATED SIZE DISTRIBUTION OF MINUS 3/4-IN. ROD MILL FEED*

Tyler Mesh	% Passing Cumulative
5/8 in.	95
1/2 in.	85
7/16 in.	80
3/8 in.	75
3 mesh	65
6 mesh	47
8 mesh	40
10 mesh	34.5
14 mesh	28.5
20 mesh	24

80% passing size = 7/16 in. or 11,200 microns

*See Exhibits 2 and 6

EXHIBIT 8

SIZE DISTRIBUTION OF 10-IN. CYCLONE OVERFLOW ADANAC PILOT PLANT*

Tyler Mesh	% Passing Cumulative
48	98.8
65	92.1
100	79.3
150	64.0
200	50.1
325	39.8
Minus 325	-

80% passing size = 150 microns
% solids = 40%

NOTE

During the initial continuous operation of the pilot plant in August 1970, the size distribution of the cyclone overflow ranged as follows:

43.6% to 49.9% minus 200 mesh
9.2% to 17.9% plus 65 mesh
MoS_2 recovery was consistently about 95%

*Lateral Development Ore, 4 to 12 shift, No. 119, September 29, 1970

EXHIBIT 9

SIZE DISTRIBUTION OF 10-IN. CYCLONE UNDERFLOW ADANAC PILOT PLANT*

Tyler Mesh	% Retained	% Retained Cumulative	% Passing Cumulative
8	0.8	0.8	99.2
10	0.9	1.7	98.3
14	1.8	3.5	96.5
20	3.3	6.8	93.2
28	5.4	12.2	87.8
35	11.2	23.4	76.4
48	19.5	42.9	57.1
65	20.1	63.0	37.0
100	10.6	73.6	26.4
150	7.0	80.6	19.4
200	4.7	85.3	14.7
325	3.3	88.6	11.4
Minus 325	11.4	100.0	-
TOTAL	100.0		

80% passing size = 460 microrons
Circulating load @ 65 mesh = 430%; @ 150 mesh = 425%
% solids = 68%

*Lateral Development Ore, 4 to 12 shift, No. 119, September 29, 1970

EXHIBIT 10

BULK DENSITIES
ASSORTED PRODUCTS FROM PILOT PLANT
ADANAC MOLYBDENUM PROJECT*

Items	Bulk Density (lb/cu ft)
Run of mine ore minus 8 plus 3/4 in. hand-picked pieces	75 to 86
Jaw crusher products (sized fractions)	
minus 2-1/2 in. plus 1 in.	76-1/4
minus 1 plus 1/2 in.	71.4
minus 1/2 in. plus 8 mesh	78.5
Ball-mill feed	
minus 1/2-in. ore.	
Wet sample from stockpile	99-1/4 at 4% moisture
Dry. 1-litre beaker sample. Sample from 8 to 4 shift, No. 118, September 29, 1970	88-1/4
minus 8-mesh fraction only	
Wet	86-1/4 at 4-1/2% moisture
Dry	87.8
minus 1/2 in. plus 8-mesh fraction (dry)	88.1
minus 3/4-in. lab roll-crusher product	98.5 at estimated 2-1/2 % moisture
Final MoS_2 concentrate **	90.2 at 17.4% moisture
Dry filter-cake density (cake thickness 1/2 in.)	110
Ore fines 99.4% to 325 mesh	46.8

* From field trip to site, September 28 - October 2, 1970

**Brenda Mines Ltd. used 90 lb/cu ft for gas-fired infrared dryer

Bulk density of crushed ore with fines will be taken to be: 100 lb/cu ft

for closely sized crushed ore: 95 lb/cu ft

EXHIBIT II

GRINDING CIRCUIT CALCULATIONS

Bond work index, WI: 18 to 20 (Preliminary grindability tests by Noranda); Use 19.

Grind objective: 15% plus 65 mesh

47% minus 200 mesh

80% passing size, final product: 170 microns

Total feed rate @ 95% of available time is $\left(\frac{15,000}{24}\right)\left(\frac{1}{0.95}\right) = 660$ tph

I. ALTERNATIVE 1: DRAWING NO. 7008-G-2000

A. ROD-MILL HORSEPOWER

Feed: 100% passing size, 3/4-in.

Estimated 80% passing size (Exhibits 2 & 6), 7/16-in. of F = 11,200 microns

Product: assume 100% passing size 8 mesh

Estimated 80% passing size 14 mesh or P = 1,190 microns

Reduction Ratio, Rr, $= F/P = \frac{11,200}{1,190} = 9.4$

from Allis-Chalmers Catalogue: GAC6061 (1953) Pages 1-4

$$WI/W = 19/W = \left(\frac{\sqrt{Rr}}{\sqrt{Rr}-1}\right)\left(\frac{P}{100}\right) = 5.12$$

$$W, \text{ kwh/ton} = \frac{19}{5.12} = 3.71$$

Consumed hp = (3.71) (660) (1.34) = 3,281

Add 10% for broken rods:

TOTAL CONSUMED hp = 3,281 + 328 = 3,609

Therefore, choose 1 unit @ 3,609 hp or 2 units @ 1,805 hp each

Rod Mill size (Allis-Chalmers Bulletin 07B1193B)

13 ft by 18 ft requires 1,400 hp, installed

Estimate 13-1/2 ft by 20 ft (2 units)

B. BALL MILL HORSEPOWER

Feed: or Rod Mill Discharge = 80% passing size, F, 1,190 microns

Product: (objective grind)

80% passing size, P, 170 microns

Using similar equation and calculations,

TOTAL CONSUMED hp = 8,013

Therefore, choose two units @ 4,000 hp each

Ball Mill size (Allis-Chalmers Bulletin 07B1192-02)

A 13-ft by 20-ft unit requires 2,100 hp, installed.

A 13-ft by 17-ft unit requires 1,525 hp, installed.

Estimate two units, each 13-1/2 ft by 28 ft

II. ALTERNATIVE 2: DRAWING NO. 7008-G-2100

A. Autogenous Mill Horsepower

Feed or Jaw Crusher Product:

From Exhibit 2, 100% passing size 8 in.

Curve 2 estimated 80% passing size 6-1/2 in. or

F = 164,000 microns

Product: Assume similar to rod-mill discharge

80% passing size, P = 1,190 microns

$$\text{Feed rate @ 85\% availability} = \left(\frac{1,500}{24}\right)\left(\frac{1}{0.85}\right)$$

= 735 tph or 370 tph for each line

Total feed rate = 370 tph

Using similar equation and calculations -

TOTAL CONSUMED hp = 2,498 for each line

Estimate two mills, each 26 ft by 10 ft and each requiring 3,000 hp

B. BALL MILL HORSEPOWER

Feed or Autogenous Mill product 80% passing size, F, is 14 mesh or 1,190 microns

Product (objective grind)

80% passing size, P, 170 microns

Feed rate to ball mill governed by 85% availability of autogenous unit.

$$\text{Feed rate} \left(\frac{15,000}{24}\right)\left(\frac{1}{0.85}\right) = 735 \text{ tph or } 370 \text{ tph for each line}$$

TOTAL CONSUMED hp = 8,984

Therefore, select two mills, each requiring a consumed hp of 4,500

Iron Ore of Canada at Carol Lake uses Allis-Chalmers

13-1/2-ft by 28-ft overflow ball mills drawings 2,500 hp.

The Griffiths Mine in Ontario uses 14-ft by 28-ft pebble mills, each

drawing 2,000 hp, after autogenous grinding.

Choose two ball mills, one after each autogenous mill, and each

drawing 2,600 hp.

III. GRINDING MEDIA REQUIREMENTS

A. 13-1/2-ft by 20-ft rod mill - 203 tons rods

B. 13-1/2-ft by 20-ft ball mill - 154 tons balls

C. 13-1/2-ft by 28-ft ball mill

estimate $\frac{(154)(28)}{(20)} = 216$ tons

EXHIBIT 12

SUMMARY OF MATERIAL BALANCE OF ONE ROD-MILL-BALL-MILL CYCLONE CIRCUIT*

Item	1	2	3	4****	5	6	7****	8	9
	330 tph		330 tph	825 tph	1,155 tph		825 tph		300 tph
Solids, dry tpd	7,920		7,920	19,800	27,720		19,800		7,920
Water, tpd	417**	2,223	2,640	10,662	21,972	8,670	8,486	2,176	13,485
Water tons solids/day	3,046		3,046	7,615	10,662		7,615		3,046
TOTAL WATER & WATER TONS, TPD***	3,463	2,223	5,686	18,277	32,634	8,670	16,101	2,176	16,531
US gpm, pulp	576	370	946	3,042	5,431	1,443	2,680	362	2,751
% solids	95		75	65	55.8		70		37
sp gr solids	2.6		2.6	2.6	2.6		2.6		2.6
sp gr pulp			1.86	1.67	1.52		1.76		1.29

* See Exhibit 13 Drawing No. 7008-G-2000

** Surface Moisture estimated to be 5%

*** 32 cu ft of water equals 1 ton or 1 water-ton.

$$\text{Fluid tons} = \frac{\text{dry tons of solids}}{\text{specific gravity of solids} = 2.6} = \frac{\text{cu ft}}{32}$$

**** Circulating load = 250%; (330) (2.5) = 825 tph

KAISER ENGINEERS

SUMMARY OF MATERIAL BALANCE OF ONE AUTOGENOUS-MILL BALL-MILL CYCLONE CIRCUIT*

Item	1	2	3	4	5	6	7	8***	9	10***	11	12
	370 tph		500 tph	130 tph		370 tph	370 tph	925 tph	1,295 tph	925 tph		370 tph
Solids, dry tpd	8,880		12,000	3,120		8,880	8,880	22,200	31,080	22,200		8,880
Water, tpd	467**	3,186	4,000	347	9,027	3,651	3,651	11,954	24,632	9,512	2,442	15,120
Water tons of solids, tpd	3,415		4,615	1,200		3,415	3,415	8,538	11,954	8,538		3,415
TOTAL WATER & WATER TONS, TPD	3,882	3,186	8,615	1,547	9,027	7,066	7,066	20,492	36,586	18,050	2,442	18,535
US gpm pulp	646	530	1,433	257	1,503	1,176	1,176	3,410	6,089	3,004	406	3,085
% solids	95		75	90		71	71	65	55.8	70		37
sp gr solids	2.6		2.6	2.6		2.6	2.6	2.6	2.6	2.6		2.6
sp gr pulp			1.86	2.24		1.78	1.78	1.67	1.52	1.76		1.29

* See Exhibit 19 & Drawing No. 7008-G-2100

** Surface Moisture estimated to be 5%

*** Circulating Load = 250%: (370)(2.5) = 925 tph

I. CYCLONE REQUIREMENTS

Based on hydraulic capacity data (US gpm) shown in the previously summarized material balances * and considering the objective grind, the estimates** for cylones are:

Four 30-in. cyclones, each capable of handling about 1,500 US gpm of pulp, for each grinding circuit.

This recommendation applies to both alternatives shown respectively in Drawings No. 7008-G-2000 and 7008-G-2100.

II. TYPCIAL MATERIAL BALANCE CALCULATION

A. Item 1 Rod-Mill-Ball-Mill Circuit

Surface moisture content 5%

Solids, dry tpd 7,920

Water, tpd @ 5% moisture: $\frac{x}{7,920+x} = \frac{5}{100}$

$$\text{Water, tpd} = X = 417$$

$$\text{Water, US gpm} = \frac{417}{6.0086} = 69.5$$

B. Item 3 Rod-Mill-Ball-Mill Circuit

Solids, dry tpd 7,920

% solids 75

Water, tpd $\frac{7,920}{0.75} - 7,920 = 2,640$

Water tons solids, tpd $\frac{7,920}{\text{sp gr of solids}} = \frac{7,920}{2.6}$

$$= 3,046$$

---

* Column 5 for Rod-Mill-Ball-Mill Cyclone Circuit
Column 9 for autogenous-Mill-Ball-Mill Cyclone Circuit

** Reference: "Practical Application of Liquid Cyclones in Mineral Dressing Problems" by D. T. Tarr - Krebs Engineers. Estimates confirmed by Krebs Cyclone representative.

Total tons, water & water tons, solids, tpd $= 2,640 + 3,046 = 5,686$

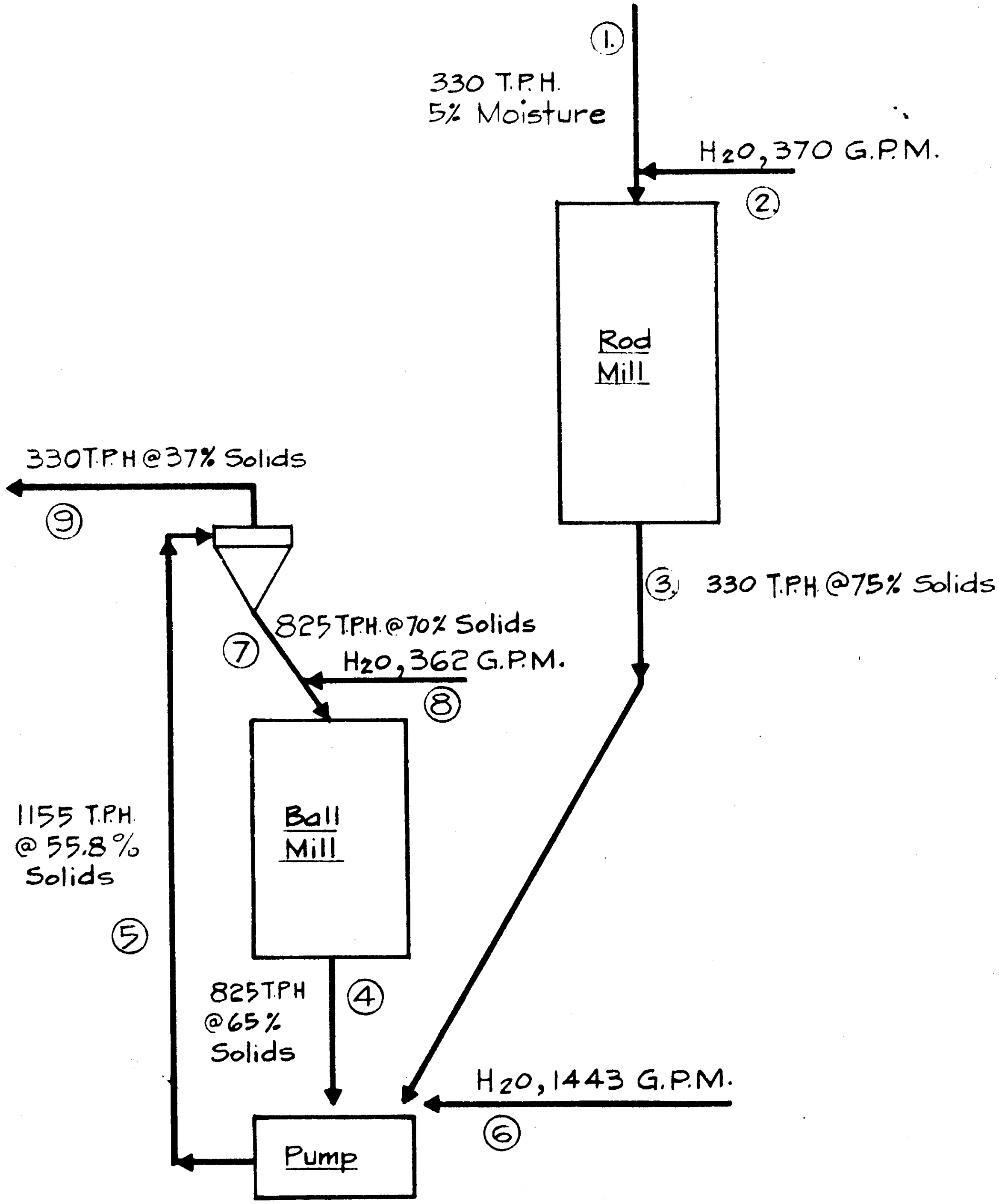
US gpm pulp $= \frac{5686}{6.0086} = 946$

i.e. Tons water/24 hr $=$ (US gpm) (6.0086)

Specific gravity of pulp

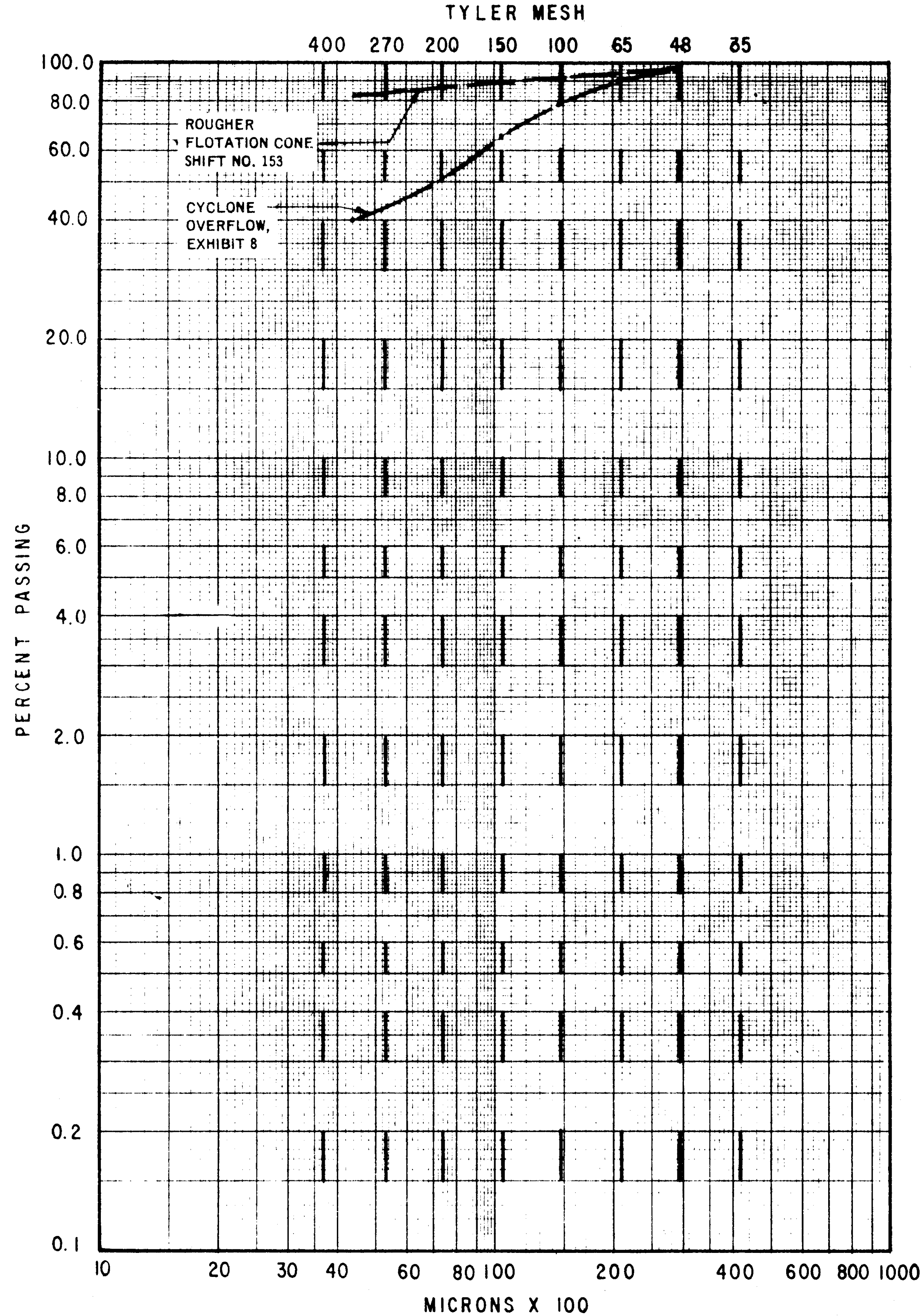
$$= \frac{100}{\frac{\%\ \text{solids}}{\text{sp gr solids}} + \frac{\%\ \text{liquid}}{\text{sp gr liquid}}}$$

$$= \frac{\qquad}{\frac{27}{2.6} \quad \frac{27}{1}} = 1.86$$



SCHEMATIC OF ONE ROD MILL –
BALL MILL – CYCLONE CIRCUIT
(MATERIAL BALANCE SHOWN IN EXHIBIT 12)

EXHIBIT 13



SCREEN TESTS, ADANAC PILOT PLANT FLOTATION PRODUCTS

EXHIBIT 14

EXHIBIT 15

ESTIMATED STEEL CONSUMPTION FOR SUGGESTED SIZE REDUCTION FLOWSHEETS SUMMARY OF PRELIMINARY ESTIMATES

	lb/ton*	Cost** $/ton	Remarks
Drawing No. 7008-G-2000 Scheme 1	2.57	0.393	Two-lines each with 13 ft by 20 ft rod mill + 13-1/2 ft by 28 ft ball mill.
Drawing No. 7008-G-2100 Scheme 2	1.878	0.390	Wet autogenous grate & ball mills, Two-line grind; minus 8 in. feed.

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* Sources of Information
Estimates based on literature data representing operating experience with other abrasive ores and also Adanac's pilot plant experience of 1.3 lb/ton of ore for forged steel ball consumption.

** Based on vendors' estimates for delivered prices to Whitehorse, Y. T. mining area (New Imperial Mines, Anvil Mining Corp. Ltd.)

for Ni-hard liners - 28¢/lb
Ni-hard balls 1 in. diameter - 16¢/lb   FOB Whitehorse
Ni-hard balls 3 in. diameter - 12¢/lb

Trucking freight - $11.00/ton or 0.55¢/lb
(Whitehorse to Adanac mine site)

I. CONVENTIONAL ROD MILL-BALL MILL GRINDING DRAWING NO. 7008-G-2000 (Revision A)

Item	lb/ton of ore	Delivered Cost Adanac Site, (¢/ton of ore)
Gyratory crusher*	0.010	0.430
Cone crushers*	0.050	2.495
Liners: Rod mills**	0.052	1.742
Ball mills**	0.055	1.842
Grinding media:		
Rod mills	1.020	10.710
Ball mills***	1.300	21.060
All other items	0.083	1.000
TOTAL	2.57	39.279

* Manganese steel
** Ni-hard
*** Forged steel

II. PRIMARY WET AUTOGENOUS GRINDING FOLLOWED BY BALL MILLING. DRAWING NO. 7008-G-2100 (Revision A)

Item	lb/ton of ore	Delivered Cost Adanac Site, (¢/ton of ore
Jaw crusher*	0.04	1.720
Liners:		
Autogenous mills	0.40	13.400
Ball mills	0.055	1.842
Grinding Media:		
Ball mills (3" dia.)	1.300	21.060
All other items	0.083	1.000
TOTAL	1.878	39.022

* Manganese steel

III. APPROXIMATION OF METAL CONSUMPTION OF CASTINGS (RODS, BALLS EXCLUDED) FROM ABEX REPORT*

Metal consumption (lb) = (tons/yr)(0.2054)+(Abrasive Nature)(49,608) + (% passing 200 mesh)(0.83) - 169,661

Abrasive nature of ore ranked from 1 to 6; use 6 (very hard)

(5,250,000)(0.2054)+(6)(49,608)+(47)(83)

- 169,661 = total lb of metal castings consumed

1,078,350 + 297,648 + 3,901 - 169,661 = 1,210,238

$$\text{lb/ton} = \frac{1,210,238}{5,250,000} = 0.23$$

For comparison from Item 1:

Total steel wear	=	2.57 lb/ton of ore
Steel wear, ball & rods	=	2.32 lb/ton of ore
Net wear, castings	=	0.25 lb/ton of ore

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* "Metal Consumption in the Mechanical Process of Crushing and Grinding at Metal Mines in Canada", Abex Industries of Canada Ltd. March, 1966, page 11.

EXHIBIT 16

REAGENT CONSUMPTION RATES IN GRINDING CIRCUIT FOR ADANAC MOLYBDENUM ORE

(See Exhibit 17)

Reference	Point of Addition	Reagent	Total Consumption: lb per short ton	US gpm	cc/min
Drawing No. 7008-G-2000: Conventional Milling	Feed to each rod mill	Lime*	0.5	2.93	-
		Shell Carnea 21 oil	0.1	0.15	555
		Arctic Syntex L**	0.005	0.13	500
Drawing No. 7008-G-2100: Autogenous Milling	To each pump box collecting screen under-size and ball mill discharge	Lime*	0.5	3.285	-
		Shell Carnea 21 oil	0.1	0.16	622
		Arctic Syntex L**	0.005	0.15	560

* Added as lime slurry, 20% solids

** Added as 5% aqueous solution; 10% solution possible if immersion heater used. Molysperse 206 will be a possible substitute.

EXHIBIT 17

FLOTATION REAGENT CONSUMPTION AND HANDLING

I. FLOTATION REAGENT DATA AND CALCULATIONS

A. FLOTATION REAGENT DATA

1. Lime

Feed as 20% solids slurry made from pulverized (minus 200 mesh) slaked or hydrated lime.

Specific gravity of slaked (hydrated) lime = 2.24
Specific gravity of burned or quick lime is 2.62

2. Arctic Syntex L

Prepared as 5% aqueous solution and 10% with immersion heater

Specific gravity of aqueous solution = 1.0

3. Sodium Sulphide

60 to 62% commercial grade prepared as 10% aqueous solution

Specific gravity of 10% aqueous solution = 1.115 (Lange's Handbook of Chemistry, 9th edition, page 1169)

4. Sodium Silicate

Grade 40° Baume prepared as 10% aqueous solution

Specific gravity = 13.33 lb/imp. gal for 40° Baume

Calculated specific gravity of 10% aqueous solution = 1.04

5. Shell Carnea 21

Light grade marine oil

Specific gravity = 0.9

6. Dowfroth 250

   Oily liquid

   Specific gravity = 1.0

7. Superfloc 127

   Prepared as 1/2% solution for addition to final concentrate thickener and filter

   Specific gravity of aqueous solution is 1.0

B. MIXING AND STORAGE TANK CALCULATIONS

1. Lime as 20% solids slurry

   Consumption @ 1.1 lb/ton for 15,840 tpd = 8.712 tpd

Solids, tpd		8.712
Water, tpd @ 20% solids		34.848
Water tons solids, tpd	$= \frac{8.712}{2.24} =$	3.889
TOTAL WATER & WATER TONS, TPD		38.737
US gpm, pulp	$= \frac{(38.737)}{6.0086} =$	6.447
Cu ft/hr	= (6.447) (8.0208) =	51.6
Cu ft/24-hr day	= (51.6) (24) =	1,240

   Choose 12 ft diameter by 12 ft deep tank with volume of 1,356 cu ft

2. Arctic Syntex L

   Consumption @ 0.01 lb/ton for 15,840 tpd = 158.4 lb

   @ 5% solution by weight, total solution $= \frac{158.4}{0.05} =$ 3,168 lb/day

Volume required per day @ sp gr of 1.0 $= \frac{3,168 \text{ lb/day}}{62.4 \text{ lb/cu ft}} =$

50.8 cu ft per day

US gpm $= \frac{\text{cu ft/hr}}{8.0208} = 0.264$

cc/min = (0.264) (3,785) = 1,000

Choose 4-1/2-ft diameter by 4-1/2-ft deep tank

3. Other Reagents

Similar calculations for the remaining reagents results in volume and consumption as follows:

Reagent	Volume cu ft per 24-hr day	Total consumption US gpm	Total consumption cc/min
Sodium Sulphide	911	4.73	
Sodium Silicate	122	0.634	2,400
Shell Carnea 21	62.1	0.322	1,220
Dowfroth 250	16.5	324.5	
Superfloc 127	1,282	6.68	

II. FLOTATION REAGENT STORAGE AREA BASED ON 45-DAYS STORAGE DRAWING NO. 7008-G-2000 and -4000

Reagent	Form, Shipping Wt and Volumes	Consumption lb/ton	Consumption lb stored for 45 days	Total Storage Volume Cu Ft
Lime, slaked	Pulverized (minus 200 mesh) 50-lb bags, 1-1/2 cu ft/bag	1.1	785,000	23,500
Arctic Syntex L	Powder, 50-lb bags, 3 cu ft/bag	0.01	7,125	428
Sodium Sulphide (60 to 62% grade)	Dryflaked, 450 lb per 45-Imp. gal drums*	0.4	285,000	7,400
Sodium Silicate (40^{o} Baume)	Liquid, 600 lb per 45-Imp. gal drum	0.05	35,600	697
Shell Carnea 21 (sp gr = 0.9)	Liquid, 405 lb per 45-Imp. gal drum	0.22	157,000	4,520
Dowfroth 250 (sp gr = 1.0)	Liquid, 450 lb per 45-Imp. gal drum	0.065	46,300	1,200
Superfloc 127	Powder, in 100-lb drums**	0.005 of concentrate	18,000	1,800
TOTAL				39,545

* Standard 45-Imp. gal drum dimensions: 23-in. diameter by 35-in. high; volume of four drums arranged in square on 4-ft by 4-ft pallet =

$$(4)\,(4)\,\frac{35}{12} = 46.8 \text{ cu ft}$$

**Approximate drum dimensions are 20-in. diameter by 30-in. high; volume of four drums arranged in a square on 4-ft by 4-ft pallet =

$$(4)\,(4)\,\frac{30}{12} = 40 \text{ cu ft}$$

EXHIBIT 18

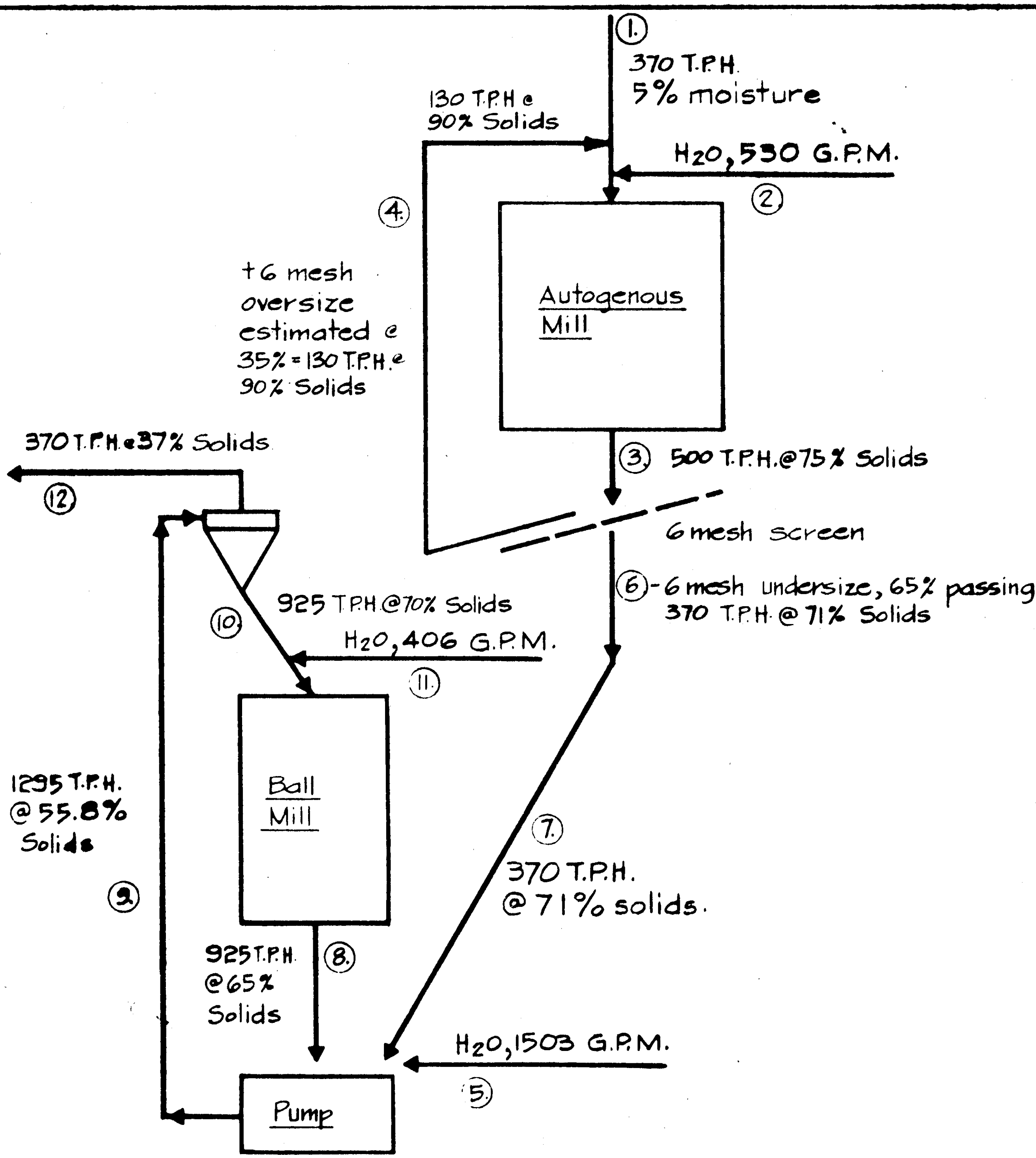
SIZE ANALYSIS OF A WET AUTOGENOUS MILL DISCHARGE

Tyler Mesh	% Passing Cumulative(1)	% Passing Cumulative(2)
Plus 1 in.	95.2	98.86
Plus 3/4 in.	85.1	97.68
Plus 5/8 in.	76.9	97.07
Plus 1/2 in.	70.0	96.58
Plus 3/8 in.	62.0	94.24
Plus 3 mesh	54.8	91.62
Plus 4	50.9	89.30
Plus 6	46.9	87.10
Plus 8	43.7	84.55
Plus 10	41.0	82.85
Plus 14	39.4	78.77
Plus 20	36.8	74.36
Plus 28	33.2	67.38
Plus 35	29.5	59.44
Plus 48	25.5	50.89
Plus 65	20.9	41.43
Plus 100	16.7	33.15
Plus 150	13.2	26.45
Plus 200	10.3	20.88
Plus 325	7.2	14.90
Minus 325	-	-

Data from a pilot plant unit operated (1) with full grate discharge to provide a coarse primary grind for a uranium pegmatite ore; (2) as for (1) but with sized feed.

Source of information: Mines Branch Investigation Report IR 59-21, Department of Energy, Mines & Resources, Ottawa, Canada, February 20, 1959, Table 20.

NOTE:
Actual size distribution for Adanac ore will be available after completion of autogenous tests at Lakefield Research of Canada Ltd.



SCHEMATIC OF ONE WET AUTOGENOUS MILL–
BALL MILL – CYCLONE CIRCUIT
(MATERIAL BALANCE SHOWN IN EXHIBIT 12)

EXHIBIT 19

EXHIBIT 20

REAGENT CONSUMPTION RATES IN FLOTATION CIRCUIT FOR ADANAC MOLYBDENUM ORE

(Drawings No. 7008-G-4000 & 4001 Revision A)

		Conventional Milling Total Consumption			Autogenous Milling Total Consumption		
Point of Addition	Reagent	lb per short ton	US gpm	cc/min	lb per short ton	US gpm	cc/min
To each 10-in. diameter by 12-ft deep thickener, rougher flotation	Dowfroth 250	0.03		150	0.03		168
Feed to No. 1 Regrind Mill	Shell Carnea 21 Oil	0.01		55.5	0.01		62.2
Rougher concentrate thickener	Lime*	0.1	0.6		0.1	0.7	
First cleaner flotation	Sodium sulphide**	0.1	1.2		0.1	1.3	
Feed to No. 2 Regrind Mill	Shell Carnea 21 Oil	0.01		55.5	0.01		62.2
Final cleaner flotation	Sodium sulphide***	0.2	2.4		0.2	2.7	
Final cleaner flotation	Sodium silicate***	0.05	0.5		0.05	0.6	
Scavenger flotation	Sodium sulphide	0.1	1.2		0.1	1.3	
Scavenger flotation	Dowfroth 250	0.005		25	0.005		28
Final concentrate thickener	Superfloc 127	0.003	4		0.003	4.5	
Filter	Superfloc 127	0.002	2.7		0.002	3.0	

(Calculations in Exhibit 17)

* Added as lime slurry, 20% solids
** 60 to 62% grade prepared as 10% aqueous solution
*** 40 degree Baume liquid prepared as 10% aqueous solution

EXHIBIT 21

MATERIAL BALANCE FOR FLOTATION FOLLOWING CONVENTIONAL GRINDING

Item	Callout Numbers On Schematic Of Flotation Process (See Exhibit 22)																			
	1	2	3	4	5	6	7	8	9	10	11	12	13	14	15	16	17	18	19	20
Solids, Dry tpd	15,840	15,600	240	-	240	240	-	365	125	-	438	313	125	165	80	-	280	200	80	40
Water, tpd	26,971	26,411	560	400	960	360	600	3,066	292	1,147	1,752	313	1,439	2,733	187	1,200	1,587	200	1,387	93
Water tons solids/day	6,092	6,000	77	-	77	77	-	126	29	-	102	73	29	39	18	-	62	44	18	9
Total water and water tons, tpd	33,063	32,411	637	400	1,037	437	600	3,192	321	1,147	1,854	386	1,468	2,772	205	1,200	1,649	244	1,405	102
US gpm, pulp	5,502	5,394	106	67	173	73	100	531	53	191	309	64	244	461	34	200	274	41	234	17
Percent solids	37	37.1	30	-	20	40	-	10.6	30	-	20	50	8.0	5.7	30	-	15	50	5.5	30
Sp gr, solids	2.6	2.6	3.1	-	3.1	3.1	-	2.9	4.3	-	4.3	4.3	4.3	4.2	4.5	-	4.5	4.5	4.5	4.5
Sp gr, pulp	1.29	1.30	1.26	-	1.16	1.37	-	1.07	1.30	-	1.18	1.62	1.06	1.05	1.30	-	1.13	1.64	1.04	1.30
Cu ft per hour, pulp	44,130	43,264	-	537	1,388	-	802	4,259	-	1,532	2,478	-	-	3,698	-	1,604	2,198	-	1,877	-
US gpm, water	4,489	4,396	-	67	-	-	100	-	-	191	-	-	-	-	-	200	-	-	-	

Item	Callout Numbers On Schematic Of Flotation Process (See Exhibit 22)																				
	21	22	23	24	25	26	27	28	29	30	31	32	33	34	35	36	37	38	39	40	41
Solids, Dry tpd	-	40	40	-	40	-	40		240	40	-	40	125	85	40	200	-	15,600	15,800	-	15,800
Water, tpd	67	160	60	100	21.5	38.5	1	20.5	2,774	93	67	160	2,706	2,546	1,294	2,681	18,011	8,400	11,018	4,310	6,771
Water tons solids/day	-	9	9	-	9	-	9	-	92	15	-	15	41	27	10	77	-	6,000	6,077	-	6,077
Total water and water tons, tpd	67	169	69	100	30.5	38.5	10	20.5	2,886	108	67	175	2,747	2,573	1,304	2,758	18,011	14,400	17,158	4,310	12,848
US gpm, pulp	11	28	11.5	16.6	5.1	6.4	1.7	3.4	480	18	11	29	457	428	217	459	2,998	2,398	2,856	718	2,138
Percent solids	-	20	40	-	65	-	98	-	8.0	30	-	20	4.4	3.2	3.0	6.9	-	65	58.8	-	70
Sp gr, solids	-	4.5	4.5	-	4.5	-	4.5	-	2.6	2.7	-	2.7	3.04	3.2	4.0	2.6	-	2.6	2.6	-	2.6
Sp gr, pulp	-	1.18	1.45	-	2.02	-	4.21	-	1.05	1.23	-	1.14	1.03	1.02	1.02	1.04	-	1.67	1.57	-	1.76
Cu ft per hour, pulp	88	225	-	133	-	51	-	27	3,850	-	88	-	-	-	-	-	24,046	-	-	5,759	-
US gpm, water	11	-	-	16.6	-	6.4	-	3.4	-	-	11	-	-	-	-	446	2,998	1,398	1,844	718	1,127

EXHIBIT 21 (Cont)

ASSUMPTIONS FOR MATERIAL BALANCE CALCULATIONS

1. Pulp density of flotation froths will be taken as 30% solids and then diluted to 20% solids for the launders.

2. The pulp density of the feed to the cyclones in the first regrind circuit will be 20% solids.

3. For the feed to the cyclones in the second regrind circuit, the pulp density will be 15% solids.

4. Pulp density in both regrind mills will be 50% solids.

5. The rougher flotation concentrate represents 1.5% of the flotation feed. To estimate the distribution of flows, its corresponding MoS_2 assay ranges from 10 to 23%, usually 10 to 15%.

6. The underflow of the thickener receiving rougher flotation concentrate will have a pulp density of 40% solids.

7. The scavenger flotation concentrate will join the thickener underflow mentioned in Item 6. The pulp density of the scavenger flotation tailing will be taken to be about 10% solids. It will be routed to the final tailings thickener underflow.

8. The MoS_2 assay of the first cleaner flotation concentrate is estimated to be about 30% which will be used to base distribution of solids in the flotation circuit.

9. A conditioner will be required between the first cleaner flotation and scavenger flotation. This conditioner will be sized on the basis of a retention time of 5 minutes.

10. A conditioner will be required before the third cleaning stage to allow for full depression of undesired minerals. A retention time of 5 minutes will also be used to estimate the capacity of this conditioner.

11. Specific gravity of solids for various products in the flotation process are calculated weighted averages based on data extracted from continuous operation of the Adanac pilot plant.

FLOW QUANTITIES FOR TAILINGS DISPOSAL SYSTEM WITH NO TAILINGS THICKENERS

Exhibits 21 and 22 include tailings thickeners. For a system without thickeners, rougher flotation and cleaner scavenger tailings will be combined. Exhibits 21 and 22 also provide information to estimate the particular flow quantities for a system without thickeners.

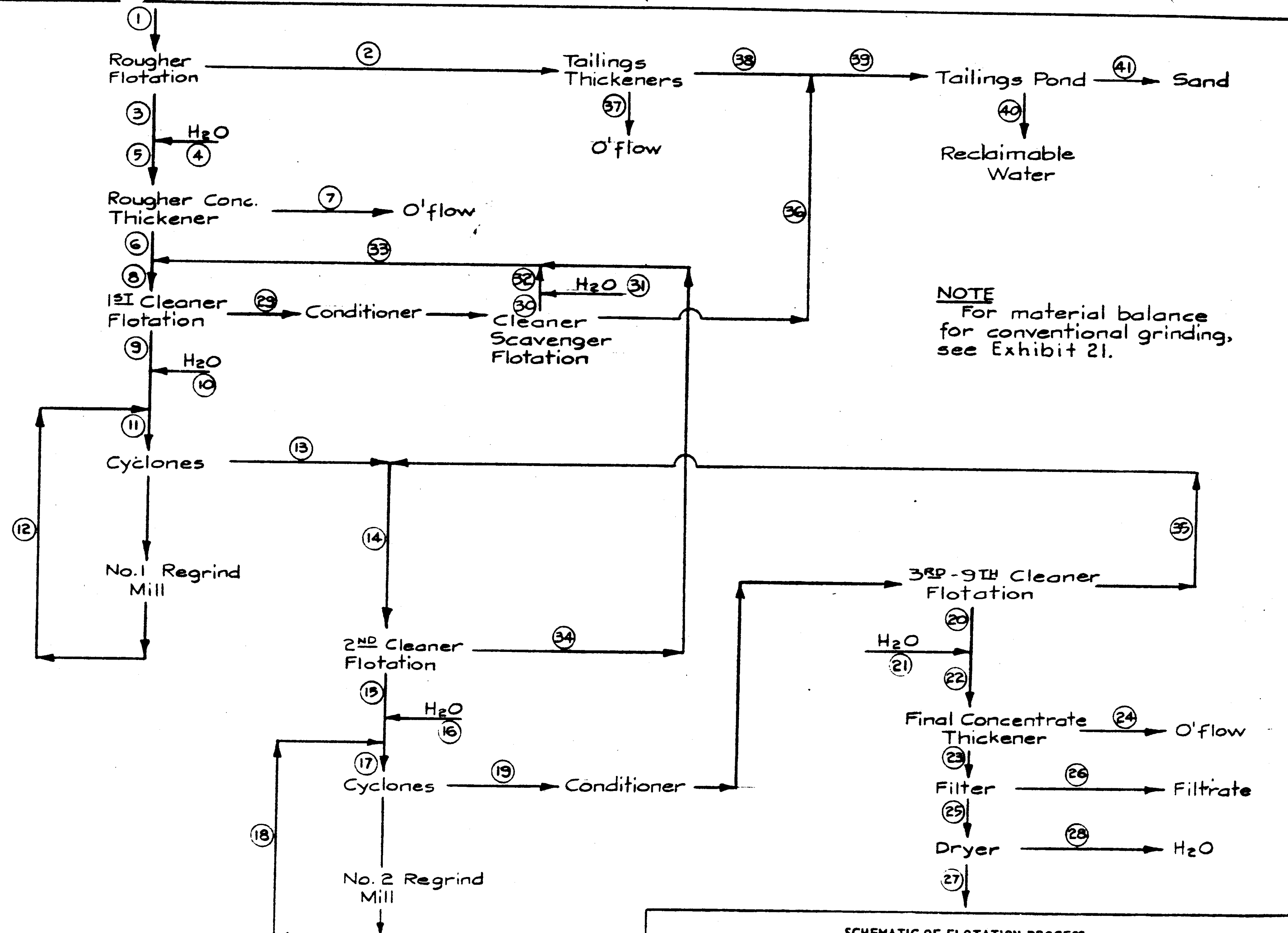
From Exhibit 21:

TOTAL DRY SOLIDS, TPD	= 200 + 15,600 =	15,800
TOTAL WATER, TPD	= 2,681 + 26,411 =	29,092
TOTAL WATER TONS SOLIDS, TPD	$= \frac{15,800}{2.6} =$	6,077
Total water and water tons, tpd	= 29,092 + 6,077 =	35,169
US gpm, pulp		5,853
% solids		35.2
Sp gr, solids		2.6
Sp gr, pulp		1.28
US gpm, water		4,842

Fresh make-up water requirements:

Total water discharged with combined flotation tailings = 4,396 + 446 =	4,842 US gpm
Water retained in tailings pond assuming sands settle to 70% solids and excluding any internal reclaim and estimates for seepage (see Exhibit 21) =	1,127 US gpm
TOTAL RECOVERABLE TAILINGS POND WATER =	3,715
Total water required for milling (see Exhibit 31) =	4,969 US gpm

Fresh water make-up required = 4,969-3,715 = 1,254 US gpm



SCHEMATIC OF FLOTATION PROCESS

EXHIBIT 22

EXHIBIT 23
FLOTATION & CONDITIONER RETENTION TIMES

I. ROUGHER FLOTATION RETENTION TIMES

No. of cells required: (48) (300) cu ft cells

Feed rate (Exhibit 21, Item 1): US gpm = 5,502
cu ft/hr = 44,130

Retention time $\frac{(48)\ (300)\ \text{cu ft}}{44,130\ \text{cu ft/hr}} = 0.326$ hr

$= (60)\ (0.326) = 19.6$ min

Applying similar calculations to the other flotation sections, and referring to Exhibit 21 and 22 the retention times are:

Flotation Section	Cell Volume (cu ft)	Feed Rate (cu ft/hr)	Retention Time (min)
Rougher	14,400	44,130	19.6
1st cleaner	1,200	4,259	16.9
Cleaner scavenger	1,200	3,850	18.7
2nd cleaner	800	3,698	13
3rd cleaner	480	1,877	15.3
4th & 5th cleaner	400	320	75
6th to 9th cleaner	320	281	114

II. ROUGHER FLOTATION CONDITIONER RETENTION TIMES

From Exhibit 21, Item 1:

Feed rate = 44,130 cu ft/hr

Volume of two 10 ft diameter by 12-ft deep conditioners

$$= \frac{(3.14)\ (10)\ (10)\ (12)\ (2)}{4} = 1,884 \text{ cu ft}$$

Retention time, each conditioner $= \frac{(1,884)\ (60)}{44,130} = 2.56$ min

III. CALCULATION OF CONDITIONERS FOR CLEANER FLOTATION

A. BEFORE CLEANER SCAVENGER FLOTATION

Feed rate, (Exhibit 21, Item 29) = 3,850 cu ft/hr

Required conditioning time = 5 min

Volume required $= \left(\frac{5}{60} \text{ hr}\right) (3,850 \text{ cu ft/hr})$

$= 321$ cu ft

A 7-1/2 ft diameter by 7-1/2 ft deep conditioner has a volume of 332 cu ft

B. BEFORE 3rd CLEANER FLOTATION

Feed rate, (Exhibit 21, Item 19) = 1,877 cu ft/hr

Volume required for 5 minutes conditioning $= \left(\frac{5}{60}\right) (1,877) =$ 156 cu ft

A 6 ft diameter by 6-ft deep conditioner has a volume of 169 cu ft

IV. AIR REQUIREMENTS FOR FLOTATION

No. 120 Agitair roughers flotation cells (300 cu ft capacity per cell)

48 cells @ 420 cfm/cell = 20,160 cfm @ 1-1/2 to 2 psig

No. 60 Agitair cleaners flotation cells (100 cu ft capacity per cell)

12 cells @ 140 cfm/cell = 1,680 cfm @ 1-1/2 to 2 psig

No. 60 Agitair cleaner scavengers flotation cells (100 cu ft capacity per cell)

12 cells @ 140 cfm/cell = 1,680 @ 1-1/2 to 2 psig

TOTAL AIR REQUIRED FOR AGITAIR CELLS = 23,520 cfm @ 1-1/2 to 2 psig

Denver No. 21 Sub-A flotation cells (40 cu ft capacity per cell)

50 cells @ 1/2 cfm/cu ft of cell = 1,000 cfm @ 1 psig, maximum

V. BLOWER REQUIREMENTS (AS PER DENVER EQUIPMENT CO. RECOMMENDATIONS)

Denver D-21, Sub-A type (cleaner flotation cells):

One Spencer 1007-H, 7-1/2 hp; 8-1/2 in diameter blast gate

Rating = 1,000 cfm @ 1 psig maximum

Agitair flotation cells:

Two Spencer 20,150 units, 150 hp each; 24-in. diameter blast gate

Rating = 21,360 cfm @ 1-1/2 psig

EXHIBIT 24

THICKENER ESTIMATES*

I. ROUGHER FLOTATION CONCENTRATE

Use overflow rate of 1.5 ft/hr, normally used to estimate clarifiers.

From Exhibit 21, Item 5:

Feed to thickener:

US GPM = 173

Cu ft/hr = (173) (8.0208) = 1,388

Area required $= \dfrac{1,388}{1.5}$ = 925 sq ft

$D^2 = \dfrac{(4)(925)}{3.14} = 1,180\ (ft)^2$

D = 34.4 ft

Choose 36 ft diameter by 10 ft deep thickener**

hp required = 1-1/2

II. FINAL CONCENTRATE THICKENER

From Exhibit 21, Item 22:

Feed to thickener:

US GPM = 28 28

Cu ft/hr = (28) (8.0208) = 225

Use overflow rate of 1 ft/hr

Area required $= \dfrac{225}{1.0}$ = 225 sq ft

$D^2 = \dfrac{(4)(225)}{3.14} = 287\ (ft)^2$

D = 17 ft

Choose 20 ft diameter by 10 ft deep unit. **

*Data from Adanac pilot plant is still being evaluated.
**Denver Equipment Co. Bulletin No. T5-B7.

III. TAILINGS THICKENER*

Estimated unit area, sq ft/tpd = 6

Feed rate: Choose daily milling capacity based on autogenous milling = 17,760 tpd

Actual feed to thickener (Exhibit 21, Item 2)

17,494 tpd

Area required = (17,494) (6) = 104,964 sq ft

$$D^2 = \frac{(104,964)\ (4)}{3.14} = 133,700\ (ft)^2$$

D = 366 ft diameter for one thickener

For two thickeners:

$$D^2 = \frac{(52,482)\ (4)}{3.14} = 66,850\ (ft)^2$$

D = 260 ft

Choose two thickeners, each 260 ft in diameter. Recommended maximum centre depth = 22 ft

*From Dorr-Oliver-Long's estimate based on preliminary settling tests on Adanac ore. More recent tests on Adanac pilot plant samples are being evaluated.

EXHIBIT 25

EQUIPMENT REQUIREMENTS FOR REGRIND CIRCUIT

I. CYCLONES*

No. 1 Regrind Circuit

From Exhibits 21 and 22 (Item 11)

Feed rate = 309 US gpm of pulp @ 20% solids

Cyclones required: Two 10 in. diameter, each with 150 US gpm capacity

Inlet pressure = 11 to 12 psi

No. 2 Regrind Circuit

From Exhibits 21 and 22 (Item 17)

Feed rate = 274 US gpm of pulp @ 15% solids

Cyclones required: Four 6 in. diameter, each with 70 US gpm capacity

Inlet pressure = 12 to 14 psi

II. REGRIND MILLS

No. 1 Regrind Mill

From Exhibits 21 and 22 (Item 11)

Feed rate = 438 tpd

From Exhibit 14, estimate feed size to mill as 100% minus 35 mesh

Assume product size 100% minus 100 mesh

Assume ore hardness as medium soft

*Estimated from Figure II "Practical Application of Liquid Cyclones in Mineral Dressing Problems" by D. T. Tarr, Krebs Engineers.

Using Denver Equipment's Ball Mill slide rule, Bulletin No. B2-B34-A

hp = 100

Mill size = 5 ft diameter by 10 ft long

No. 2 Regrind Mill

From Exhibit 21 and 22 (Item 17)

Feed rate = 280 tpd

Assume feed size = product size from No. 1 regrind circuit, or 100% minus 100 mesh.

Grind objective is 100% minus 200 mesh.

Using the same method of approximation used for No. 1 regrind mill, a 5 ft diameter by 8 ft long mill, requiring 80 hp will be required.

Choose a 5 ft diameter by 10 ft long mill

III. GRINDING MEDIA REQUIREMENTS*

For each 5 ft diameter by 10 feet long mill, 24, 300 lb of steel ball charge will be required (based on 50% of mill volume and with weight of balls taken at 256 lb per cu ft.

Size of balls: 1-1/2 in., 2 in., and 3 in.

Ball charge, lb: 8, 100, 8, 100, and 8, 100

*Denver Equipment Co. Handbook, Page 705
Denver Equipment Co. Bulletin No. B2-B34-A page 29.

EXHIBIT 26

STUDY REQUIREMENTS FOR MARKETING OF MOLYBDENUM

Item	Requirement
Product	Molybdenite concentrate only, marketed as premium grade
Packing	Concentrate shipments in standardized containers, polyethylene lined 33 US gallon drums holding 500 lb net weight
	Palletized shipping, four drums per pallet
	No provision for briquetting

Grade Specifications	Dry Basis %
MoS_2	92.0 Minimum
SiO_2	6.0 Maximum
Fe	0.75 Maximum
Al_2O_3	0.50 Maximum
CaO	0.50 Maximum
MgO	0.50 Maximum
P	0.05 Maximum
Bi	0.06 Maximum
Pb	0.06 Maximum
Cu	0.10 Maximum
WO_3	0.06 Maximum
Moisture plus oil	4 to 8%, with 7% typical

EXHIBIT 27

MIXING & STORAGE TANK REQUIREMENTS FOR FLOTATION REAGENTS*

Tank Type	Reagent	Tank Dimensions Diameter	Length, ft	Mixing Motor
Mixing; 1 each for	Lime slurry	12	12	one 15 hp
	Sodium sulphide	11	11	one 5 hp
	Sodium silicate	5-1/2	5-1/2	one 3/4 hp
	Arctic syntex L	4-1/2	4-1/2	one 3/4 hp
	Superfloc 127	12	12	one 15 hp
Storage; 1 each for	Lime slurry	12	12	one 5 hp
	Sodium sulphide	11	11	one 5 hp
	Sodium silicate	5-1/2	5-1/2	one 3/4 hp
	Arctic syntex L	4-1/2	4-1/2	one 3/4 hp
	Shell carnea 21	4-1/2	4-1/2	
	Dowfroth 250	3	3	
	Superfloc 127	12	12	one 5 hp
Head	One for each of seven reagents	4	3	five 1/3 hp

*Tank dimensions are based on reagent requirements for 24 hr of continuous milling operation.

EXHIBIT 28

FILTER ESTIMATE*

I. CAPACITY

A. CONVENTIONAL MILLING:

15, 840 tpd @ 0. 25% MoS_2 feed grade and 92% MoS_2 concentrate grade = 43 tpd

B. AUTOGENOUS MILLING:

17, 760 tpd @ 0. 25% MoS_2 and 92% grade = 48. 2 tpd

Maximum expected production of concentrate will be 50 tpd

II. FILTER AREA

Estimated required unit filter area ** = 0. 01 tons of solid per sq ft per hr

Feed rate, tons per hour = $\frac{50}{24}$ = 2. 083

$$\text{TOTAL FILTER AREA REQUIRED} = \frac{2.083 \text{ tph}}{0.01 \text{ tons per sq ft per hour}}$$

$$= 208.3 \text{ sq ft}$$

A Denver disc filter, 6 ft diameter by 4 disc, has a filter area of 200 sp ft

*Adanac pilot plant filter data is being evaluated.

**Filtering experience with 90% minus 200 mesh molybdenite at Climax, Taggart's Handbook of Mineral Dressing, 1945, Table 3, page 16-07.

EXHIBIT 29

EQUIPMENT REQUIREMENTS FOR DRYING FINAL MOLYBDENITE CONCENTRATE*

I. PRODUCT DATA AND PROCESS OPERATING CONDITIONS

Product handled		= molybdenum concentrate
Feed rate		= 3, 330 lb/hr (dry basis)
Feed bulk density		= 110 lb/cu ft
Temperature,	initial	= 60 F
	final	= 212 F
Moisture content,	initial	= 30%
	final	= 2%
Specific heat	of solids	= 0. 25 Btu/lb/degree F
	of volatiles	= 1. 0 Btu/lb/degree F
Latent heat of evaporation:		970 Btu/lb

II. EQUIPMENT REQUIREMENTS

A. HEAT EXCHANGE AGENT: MONSANTO'S THERMINOL FR-1

Temperature,	initial	= 575 F
	final	= 525 F

Rate, = 150 US gpm of Therminol FR-1

Effective heating surface area required = 100 sq ft

Holo-flite Equipment:	Number of tiers	2
	Number of screws/tier	2
	Nominal screw diameter	12 in.
	Nominal screw length	16 ft
	Pitch of screws	5 in.
	Screw speed	1. 6 to 3. 2 rpm
	hp	1-1/2

Extra equipment: 1, 500, 000 Btu oil heater

*Western Precipitation Division (Joy Manufacturing Co.)
Proposal WP 1342-1033-AO, BO; filed under KE Job No. 7008
File No. 5. 02

B. HEAT EXCHANGE AGENT: 150-PSIG STEAM

Temperature,	initial	= 366 F
	final	= 366 F

Rate = 1,650 lb/hr of 150-psig steam

Effective heating surface area required: 200 sq ft

Holo-flite Equipment:		
	Number of tiers	= 3
	Number of screws/tier	= 2
	Nominal screw diameter	= 16 in.
	Nominal screw length	= 16 ft
	Pitch of screws	= 6 in.
	Screw speed	= 0.9 to 1.8 rpm
	hp	= 1-1/2

III. LAYOUT DIMENSIONS

Length	= 11.5 ft
Width	= 6.0 ft
Height	= 6.5 ft

EXHIBIT 30

ANALYSIS OF TAILINGS DISPOSAL SYSTEM

I. ANALYSIS INCLUDING TAILINGS THICKENERS

A. HYDRAULIC REQUIREMENTS

Total overflow available from both tailings thickeners	= 2, 998 US gpm
Linear pumping distance, estimated	= 1, 350 ft
Pumping velocity, assumed maximum for water	= 5 fps
Minimum pipe diameter, ID	= 16 in.

B. PUMPING REQUIREMENTS FOR RECLAIMED WATER

1. Wood Stave Pipe

Static head, base of thickeners to top of process water tank = 148 ft

Friction loss ("C" factor = 140; pumping velocity = 5 fps) = 5 ft/ 1, 000 ft

Total friction loss $= \dfrac{(1,250)\ (5)}{(1,000)} = 6.25$ ft

20% allowance for bends, elbows etc.

$$\frac{(1,250)\ (5)\ (0.2)}{(1,000)} = 1.25 \text{ ft}$$

Total dynamic head, ft = 155. 5 ft

Total dynamic head, psig = (155. 5)(0. 4335) = 67. 4

Brake horsepower (assume 70% efficiency) $= \dfrac{(2,998)\ (155.5)}{(3,960)\ (0.7)} = 168$

2. Mild Steel Pipe

Static head		= 148 ft
Friction loss ("C" factor = 100; pumping velocity = 5 fps)		= 9 ft/1,000 ft
Total friction loss	$= \frac{(1,250)(9)}{(1,000)}$	= 11.25 ft
20% allowance for bends, elbows etc. $\frac{(1,250)(9)(0.2)}{(1,000)}$		= 2.25 ft
Total dynamic head, ft		161.5 ft
Total dynamic head, psig = (161.5)(0.4335)		= 70
Brake horsepower (assume 70% efficiency)	$= \frac{(2,998)(161.5)}{(3,960)(0.7)}$	= 175

C. TAILINGS LINE

Thickener underflow @ 65% solids	= 2,398 US gpm
Cleaner scavenger flotation tailing @ 6.9% solids	= 459 US gpm
TOTAL PULP TO BE HANDLED @ 58.8% SOLIDS	= 2,857 US gpm
Distance	12,000 ft
Vertical drop	350 ft
Average grade $= \frac{(350)(100\%)}{(12,000)}$	= minus 2.9%

1. Wood Stave Pipe

Friction loss ("C" factor = 140, flow velocity = 5 fps)	= 5 ft/1,000 ft
Pipe diameter, ID	= 16 in.

2. Mild Steel Pipe

Friction loss ("C" factor = 100; flow velocity = 5 fps)	= 9 ft/1,000 ft
Pipe diameter, ID	= 16 in.

Maximum gradient between drop boxes would be 0.5%. Drop boxes would be spaced to compensate for friction losses.

II. ANALYSIS WITHOUT TAILINGS THICKENERS

A. HYDRAULIC REQUIREMENTS

Water contained in rougher flotation tailings @ 37.1% solids		= 4,396 US gpm
Water contained in cleaner scavenger flotation tailings @ 6.9% solids		= 446
TOTAL WATER TO TAILINGS POND		4,842
Water retained in tailings pond, assuming sands settle to 70% solids	minus	1,127
Maximum amount of water available for reclaim from tailings pond (internal reclaim excluded; seepage, rainfall and evaporation also excluded)		= 3,715 US gpm
Linear pumping distance, estimated		= 12,000 ft
Pumping velocity, assumed maximum for water		= 5 fps
Minimum pipe diameter, ID		= 18 in.

B. PUMPING REQUIREMENTS FOR RECLAIMED WATER

1. Wood Stave Pipe

Static head, toe of rear tailings dam to top of process water tank	= 498 ft

Friction loss ("C" factor = 140;
pumping velocity = 5 fps) = 4.25 ft/1,000 ft

TOTAL FRICTION LOSS = $\frac{(12,000)(4.25)}{(1,000)}$ = 51 ft

20% allowance for bends, elbows, etc.
$\frac{(12,000)(0.2)(4.25)}{(1,000)}$ = $\underline{10.2}$ ft

TOTAL DYNAMIC HEAD, FT = 559.2 ft

TOTAL DYNAMIC HEAD, PSIG =
(559.2) (0.4335) = 242

Brake horsepower = $\frac{(3,715)(559.2)}{(0.7)(3,960)}$ = 749
(assume 70% efficiency)

2. <u>Mild Steel Pipe</u>

Static Head = 498 ft

Friction loss ("C" factor = 100;
pumping velocity = 5 fps) = 8 ft/1,000 ft

TOTAL FRICTION LOSS =
$\frac{(12,000)(8)}{(1,000)}$ = 96 ft

20% allowance for bends, elbows, etc.
$\frac{(12,000)(0.20)(8)}{(1,000)}$ = $\underline{19.2}$ ft

TOTAL DYNAMIC HEAD, FT = 613.2 ft

TOTAL DYNAMIC HEAD, PSIG =
(613.2)(0.4335) = 266

Brake horsepower = $\frac{(3,715)(613.2)}{(0.7)(3,960)}$ = 822
(assume 70% efficiency)

C. TAILINGS LINE

Rougher flotation tailings, pulp @ 37.1% solids	= 5,394 US gpm
Cleaner scavenger flotation tailings; pulp @ 6.9% solids	= 459 US gpm
TOTAL PULP TO BE HANDLED @ 35% SOLIDS	5,853 US gpm
Distance	= 12,000 ft
Vertical drop	= 350 ft

$$\text{Average grade} = \frac{(350)(100\%)}{12{,}000} = \text{minus } 2.9\%$$

1. Wood Stave Pipe

Friction loss ("C" factor = 140; flow velocity = 5 fps	= 3.25 ft/1,000 ft
Pipe diameter, ID	= 22 in.

2. Mild Steel Pipe

Friction loss ("C" factor = 100; flow velocity = 5 fps)	= 6 ft/1,000 ft
Pipe diameter, ID	= 22 in

Maximum gradient between drop boxes will be 0.5%. Drop boxes will be spaced to compensate for friction losses.

III. CONSIDERATIONS AND DESIGN FACTORS AFFECTING COST ESTIMATE

A. INCLUDING TAILINGS THICKENERS

1. If the ground material at the site is suitably impervious to leakage, the base of the thickener may not have to be constructed of concrete.

2. Thickener underflow pulp must be at an initial temperature several degrees above freezing to compensate for heat losses along 12,000 ft of tailings line; otherwise, material may freeze before being ejected into the tailings pond.

   For wood stave pipe, heat loss is estimated to be between 1 F and 2 F per mile (determined at approximately minus 30 F). For larger pipelines, the temperature drop can be as low as 1/2 F per mile.

   Therefore for 12,000 ft of wood stave pipe, the initial temperature of the thickener underflow should be 35 to 37 F.

3. The retention time of water in a thickener may be about 2 hours, so heat losses would not be as severe for recovering reclaim water from thickener overflow as compared to recovering reclaim water from a tailings pond.

   Water can be retained in the tailings pond for a duration of several weeks. Hence, at the pump inlet, the temperature of the water will likely be 32 F.

4. If the cleaner scavenger flotation tailing is joined with the thickener underflow, the amount of reclaim water directly recoverable as thickener overflow will be 2,998 US gpm.

   Fresh water requirements would be:

Total water required for milling	= 4,969 US gpm
Reclaimed water:	
Overflow from tailings thickeners	= 2,998 US gpm
Overflow from rougher concentrate thickener	= 100 US gpm
Filtrate from filter	= 7 US gpm
TOTAL RECLAIMED WATER	= 3,105 US gpm
TOTAL FRESH WATER REQUIRED	= 1,864 US gpm

According to the minutes of the November 5-7, 1970 working session, the reservoir system can yield 1,600 Imp. gpm or 1,922 US gpm for 10 months.

5. If cleaner scavenger flotation tailing is not too contaminated with sodium sulphide, it will be directed to join rougher flotation tailings, and then fed to the thickener. The amount of reclaim water recoverable as thickener overflow will be 3,426 US gpm.

   TOTAL FRESH WATER REQUIRED
   WILL BE 1,436 US gpm

   To handle this larger amount of recoverable thickener overflow, pipe diameter would have to be increased from 16 in. to 18 in. Pumping brake horsepower would be increased by 25 bhp.

6. Tailings directed to thickeners will be relatively coarse. Handling coarse sands in thickeners will require each thickener to be equipped with a lifting mechanism, which is an expensive item.

   The alternative would be to pump all the tailings to cyclones and direct cyclone overflow to the thickeners. Coarse cyclone underflow would join thickener underflow.

   This alternative would require three 14 in. by 12 in. pumps (including one standby) feeding five 30-in. diameter cyclones (including one standby).

B. WITHOUT TAILINGS THICKENERS

For a tailings disposal system excluding tailings thickeners, the comments mostly relate to factors influencing the choice between wood stave pipe and steel pipe to return 3,715 US gpm of reclaim water 12,000 ft against a maximum static head of 498 ft.

1. Wood stave pipe has a rated working pressure of 600 ft (260 psig). The factor of safety is 4 to 1. For a system where there are intermittent pressure surges, i.e. water hammer, a working pressure of up to 750 ft (325 psig) is allowed. Water line will require check valves to prevent pumps from reversing; telemetering gauges will be required to meter flows to the process water storage tank.

2. Use of wood stave pipe at conditions near its working pressure may require closer spacing of wire rings resulting in increased costs approaching that of steel pipe.

3. Water at the tailings pond can be exposed to prevailing temperatures for up to several weeks. At below freezing temperatures, the temperature of the water at the pumping station will likely be 32 F, which must be maintained to prevent freezing.

4. To prevent water from freezing:

   a. The water pipe will have to be insulated and heated, or buried to keep heat losses down. Exposed steel pipe can have severe heat losses due to radiation and wind factor. For steel pipe, heat loss can be 5 Btu/sq ft/day. If the pipe is to be buried, the depth of frost line must be considered. A cover of 9 to 11 feet is used for water lines at Whitehorse, Yukon.

   b. Water at the inlet can be heated by steam. Steam source may either be the main plant, in which case a steam line will be required. Otherwise, a small steam plant located at the inlet will be required.

   c. To keep steel pipe heated, electrical or steam tracings may have to be located along the pipe.

   d. For wood stave pipe, steam tracings would be used to keep the water above freezing temperatures.

   e. Steam requirements to compensate for heat losses would be based on the amount of heat available from the latent heat of evaporation of steam.

5. A tailings system which excludes thickeners may require two tailings pipelines to prevent complete plant shutdown in the event of a blockage or other problems in the tailings line.

6. The possibility of having to include booster pumping stations, to accommodate high dynamic pumping heads, would have to be considered.

C. OTHER MISCELLANEOUS CONSIDERATIONS

1. In the event of line blockage or a line break, either tailings disposal system will require an area set aside to allow dumping of the tailings.

2. Pumps remotely located can be driven by independent diesel units. Otherwise, additional powerlines will be required to energize electric motors driving the pumps.

IV. EXHIBIT 30
(Supplement)

TAILINGS THICKENERS WITH UNDERFLOW AT 50% SOLIDS

Thickener underflow with a pulp density of 50% solids was requested. The influence on fresh water requirements for two possibilities are provided:

- No reclaim from tailings pond
- With reclaim from tailings pond

A. THICKENER UNDERFLOW

Solids, dry tpd	15, 600
Water, tpd	15, 600
Water tons solids, tpd	6, 000
TOTAL WATER AND WATER TONS, TPD	21, 600
US gpm, pulp	3, 595
% solids	50
Sp gr, solids	2. 6
Sp gr, pulp	1. 44
US gpm, water	2, 596

Fresh makeup water requirements:

TOTAL REQUIRED FOR MILLING	4, 969 US gpm
Overflow, tailings thickener	1, 800 US gpm
Overflow, rougher concentrate thickener	100
Filtrate from filter	7
TOTAL RECLAIMED WATER	1, 907
TOTAL FRESH WATER REQUIRED (IF NONE RECLAIMED FROM POND)	3, 062 US gpm

B. COMBINED THICKENER UNDERFLOW AND CLEANER SCAVENGER TAILINGS

Solids, dry tpd	15, 800
Water, tpd	18, 281
Water tons solids, tpd	6, 077
TOTAL WATER AND WATER TONS	24, 358
US gpm, pulp	4, 054
% solids	46. 4
Sp gr, solids	2. 6
Sp gr, pulp	1. 40
US gpm, water	3, 042

Fresh makeup water requirements:

Water retained in tailings pond (assuming sands settle to 70% solids, and excluding any internal reclaim and estimates for seepage and precipitation)	1, 127 US gpm
Maximum water available for reclaim from tailings pond	1, 915 US gpm
TOTAL FRESH WATER REQUIRED (if 1, 915 US gpm will be reclaimed from pond) = 4, 969 - (1, 907 + 1, 915) =	1, 147 US gpm

C. SUMMARY OF PUMPING AND PIPING REQUIREMENTS FOR TAILINGS DISPOSAL SYSTEM ALTERNATIVES

1. With Thickeners*

Thickener Underflow = 65% solids (no reclaim of 718 US gpm from tailings pond)

*In all cases, internal reclaim will be 107 US gpm: 100 from rougher concentrate thickener, and 7 on filtrate

Item	Pumping Head, Ft Static	Dynamic	Lineal Pumping Distance Ft	ID Pipe Inches	Water US gpm	Pump bhp
Thickener Overflow	178	191.5	1,250	16	2,998	208
Fresh water	228	324.9	9,500	14	1,864	220
TOTAL						428
Thickener underflow = 50% solids (no reclaim of 1,915 US gpm from tailings pond)						
Fresh water	228	296.4	9,500	18	3,062	327
Thickener Overflow	178	189.4	1,250	14	1,800	123
TOTAL ESTIMATED PUMP BHP						450
Thickener underflow = 50% solids (include 1,915 US gpm reclaimed from tailings pond)						
Fresh water	228	376.2	9,500	10	1,147	244
Pond reclaim	498	620.4	12,000	14	1,915	429
Thickener Overflow	178	189.4	1,250	14	1,800	123
TOTAL ESTIMATED PUMP BHP						796
2. No Thickeners						
Fresh water	228	302.1	9,500	12	1,254	137
Pond reclaim	498	613.2	12,000	18	3,715	822
TOTAL ESTIMATED PUMP BHP						959

3. In general, the total brake horsepower for pumping both fresh and reclaimed process water will be about the same regardless of whether thickener underflow pulp density will be 65 or 50% solids, provided water will not be reclaimed from the tailings pond.

   If water from the tailings pond will be reclaimed, and pumping distance = 12,000 ft, static head = 498 ft; the total bhp required will be approximately 800 for the case when thickener underflow pulp density will be 50% solids, compared to 428 bhp if thickener underflow pulp density will be 65% solids.

   For thickener underflow @ 65% solids, only 718 US gpm of water will be available at the tailings pond. This quantity is rather small and is therefore ignored considering that the lineal pumping distance will be 12,000 ft and the static head will be 498 ft.

   However, for the case when thickener underflow will be 50% solids and water will not be reclaimed from the tailings pond, fresh process water requirements will be 3,062 US gpm. According to data from CW&G Ltd., the fresh water reservoir will be capable of providing 1,600 Imp gpm for 10 months, or 1,922 US gpm (1,333 Imp gpm - 1,601 US gpm, was mentioned later). Therefore, there will have to be additional sources of fresh process water.

   For a system that excludes tailings thickeners, total estimated bhp will be about 959.

   Therefore, a system with tailings thickeners with thickener underflow of 65% solids appears attractive since pumping bhp will be about 430.

   The pumping requirements in this summary are estimates. CW&G Ltd. will be providing the requirement details for providing fresh process water.

EXHIBIT 31

WATER BALANCE

I. WITHOUT TAILINGS THICKENER

	gpm Imperial	gpm US
Water out		
Rougher concentrate thickener	83	100
Final concentrate thickener	14	17
Rougher flotation tailings	3,660	4,396
Cleaner scavenger tailings	371	446
Filter	6	7
Dryer	3	3
TOTAL WATER OUT	4,137	4,969
Water in		
Grinding circuit		
Rod mills	616	740
Pump boxes	2,403	2,886
Ball mills	603	724
Rougher scavenger flotation		
Dilute rougher flotation concentrate	56	67
Dilute cleaner scavenger concentrate	9	11
First regrind circuit	159	191
Second regrind circuit	167	200
Cleaner flotation		
Dilution water after 9th cleaner stage	9	11
TOTAL WATER IN	4,022	4,830
Additional dilution & sprays, 3rd to 9th cleaning stages by difference	115	139
GRAND TOTAL WATER IN	4,137	4,969

II. INCLUDING TAILINGS THICKENER
(Scavenger tailing joins thickener underflow)

	gpm Imperial	gpm US
Water out		
Tailings thickener overflow	2,496	2,998
Tailings thickener underflow*	1,164	1,398
Cleaner scavenger tailings*	371	446

*Reclaim from tailings pond

	gpm Imperial	gpm US
Rougher concentrate thickener**	83	100
Final concentrate thickener**	14	17
Filter**	6	7
Dryer**	3	3
TOTAL WATER OUT	4,137	4,969

Water In
(Same as for "I" above)

III. WATER ESTIMATE

Water	Preliminary Estimate November 12, 1970 gpm Imperial	Preliminary Estimate November 12, 1970 gpm US	Kaiser Engineers Estimate (Conventional milling) gpm Imperial	Kaiser Engineers Estimate (Conventional milling) gpm US
Water In				
Ore @ 5% moisture	110	132	115	139
Add @ rod mills	585	702	616	740
Add @ ball mills	2,853	3,424	3,006	3,610
TOTAL WATER	3,548	4,258	3,737	4,489
(Usable from reclaim source)	3,438	4,126	3,622	4,350
Clean water, rougher concentrate to thickener	333	400	56	67
Cleaner circuit	292	350	459	552
TOTAL CLEAN WATER	625	750	516	619
Total addition to circuit	4,063	4,876	4,137	4,969
Water Out				
Rougher tailing	3,248	3,898	3,660	4,396
Cleaner scavenger tailing	320	384	371	446
Rougher concentrate thickener overflow	583	700	84	100
Final concentrate thickener	14	17	14	17
Filter	5	6	6	7
Concentrate	3	4	2	3
TOTAL WATER	4,173	5,009	4,137	4,969

**Internal reclaim

	Preliminary Estimate November 12, 1970		Kaiser Engineers Estimate (Conventional milling)	
	gpm		gpm	
	Imperial	US	Imperial	US
TOTAL TO TAILINGS POND	4, 170	5, 005	4, 135	4, 966
Retained in pond after settling (Estimate 70% solids after settling)	891	1, 069	938	1, 127
Available Reclaim				
Water out - water reclaimed	3, 279	3, 936	3, 197	3, 839
Source Water				
Available reclaim	3, 279	3, 936	3, 197	3, 839
New clean water to cleaner circuit	625	750	459	552
New clean water to rougher circuit	159	190	56	67
TOTAL WATER	4, 063	4, 876	3, 712	4, 458
TOTAL NEW CLEAN WATER REQUIRED	784	940	425	511

DESIGN CRITERIA

A. CIVIL DESIGN CRITERIA

1. Earthwork

    a. Cut and fill side slopes in O.M. shall be 2:1 unless otherwise dictated by the Soils Investigation Report. Cut slopes in rock shall be 1:4.

    b. Finished grading shall effect positive control of all surface drainage throughout the plant site.

    c. Mine area rock stripping shall be used for fill wherever practicable.

2. Surfacing

    a. Sub-grade surfacing shall consist of a nominal 12-in. thick course of untreated, dense-graded crushed rock or crushed gravel provided at all areas required for normal operating access to plant facilities, including parking areas.

    b. Suitable pumice shall be used for finish surfacing.

3. Drainage

    Surface drainage shall be by ditch and culvert and shall be provided to accommodate the prevailing conditions. Culverts shall be corrugated metal pipe.

4. Sanitary Sewers (Yard Piping)

    a. Sanitary sewers shall carry domestic wastes only, with gravity flow to a central collection point for treatment and disposal.

    b. Maximum flow will be determined by the fixture units served. Sewers will normally be fiber or asbestos cement pipe. Minimum yard pipe size shall be 6 in. diameter, and, as far as practicable shall have slopes to produce self-cleansing velocities.

    c. Manholes will be provided at maximum intervals of 300 ft, at all junctions of sewers and at all changes in alignment or grade.

    d. Minimum depth of cover for 6-in. sewer mains will be 10 ft 0 in.

e. Sewage treatment and disposal shall consist of an aeration pond. Size will be based on average daily flow according to plant population.

5. Process or Plant Water (Yard Piping)

   a. The system will consist of two constant speed horizontal pumps located near to and taking suction from the ground level water storage tank, pumping directly into the distribution system on demand with pressure control. The distribution system will include valves located so as to isolate individual sections for maintenance and repair with minimum interruption of plant operation. No water treatment will be provided.

   b. Pipe material will normally be steel for mains, sizes 3 in. and larger, schedule-40 steel pipe coated and wrapped for services, sizes 1 in. to 2-1/2 in., and schedule-40 galvanized steel pipe for sizes 3/4 in. and smaller.

   c. Minimum depth of cover for all sizes of pipe shall 8 ft.

6. Domestic Water (Yard Piping)

   a. Domestic water will be supplied by the process or plant water system, treated as required to meet drinking water standards, then distributed as a separate system without cross connections.

   b. Pipe material will normally be galvanized steel, ASTM A-120 or equal.

   c. Minimum depth of cover for all sizes of pipe shall be 8 ft.

7. Fire Protection (Yard Piping)

   a. The system will consist of a diesel engine operated fire pump located next to the process or plant water pumps, taking suction from the ground level storage tank, delivering 1,500 gpm directly into the distribution system.

   b. Fire hydrants shall be located to clear buildings a minimum distance of 30 ft, a minimum distance of 8 ft from the edges of surfaced traffic areas, and shall be placed throughout the plant site for adequate fire protection coverage with a maximum spacing of 450 ft.

c. Pipe material will normally be asbestos cement pipe.

d. Minimum depth of cover for all fire protection water pipes shall be 8 ft.

8. Fencing at Main Gates and Road Accesses

A perimeter fence shall be provided around the plant to protect all facilities from outside intrusion by unauthorized personnel and vehicles. Gates will be manually operated, and located where required for ease of access and egress.

Plant fencing will be galvanized steel, chain link type, 6 ft high plus three lines of barbed wire on extension arms.

B. ELECTRICAL DESIGN CRITERIA

1. Applicable Codes

All design shall be in accordance with the Canadian Electrical Code and the Electrical Energy Inspection Act of British Columbia. All electrical equipment and components shall be CSA approved to CEMA standards.

2. Numbering System

Code numbers will be used to identify electrical equipment and circuits within different areas of the plant. The equipment carrying such numbers will be transformers, switchgear, motor control centres, panels, motors, and similar equipment. Motors will be identified with the same number as the driven equipment.

3. Ambient Conditions

All equipment specified for installation on the job will be suitable for use in the most severe ambient conditions that will exist at the particular location. In wet areas CEMA 4 enclosures and/or hermetically sealed devices will be used. In dusty or dirty areas, CEMA 5 or CEMA 12 enclosures and/or hermetically sealed devices will be used. All equipment installed remote from heat sources will be suitable for use under the conditions listed below:

Temperature Range:	-40 F to +80 F (low peaks to -60 F)
Elevation:	4,500 ft above sea level

4. Power System

Plant distribution voltages will be 3.8 kv, 4,160 and 600 v. Utilization will be as outlined below:

a. Motors rated less than 1/4 hp will in general be supplied power at 120 v, single phase. Motors rated from 1/4 to 200 hp will be 600 v, 3 phase. Motors 250 hp and larger will be 4,160 v.

   Motors 75 hp 1,800 rpm and smaller will be TEFC. Larger motors will be, in general, open drip proof. Motors will be specified for a 50 C ambient with Class B insulation.

b. Circuit breaker combination starters in motor control centres will be used for all 600 v motors.

Metal clad, air break switchgear and motor starters will be used for all 4,160 v distribution.

c. Calculations will be made to insure adequacy of equipment load ratings, to insure adequacy of circuit protective devices, and to provide wire of adequate size to limit the total voltage drop at each voltage level to 5% or less. Unless voltage drops require larger sizes, the wire will be sized in accordance with the Canadian Electrical Code. For wire insulation types, refer to Section 11, "Materials" below.

5. Control System

The plant will be provided with a two-level control system as follows:

Level 1 - "START-LOCKOUT STOP" pushbuttons located at each motor. Pushbuttons or selector switch at each electrically controlled gate, valve, etc.

Level 2 - Nongraphic pushbutton control panels will be provided in various areas for start-stop control of circuits. The panels will be centrally located in the process circuits to provide observation of critical equipment during startup. Red running lights will be provided for motor circuits.

Control circuitry will be standard three wire start-stop. Zero speed switches will be provided on belt conveyors and screw conveyors. These switches on belt conveyors will be interlocked with their driving motor. In general, all equipment required in a process flow will be interlocked.

6. Control Equipment

a. Motor control centres per paragraph 4-b above will be standard plug-in type with not more than six units high. Each starter will be provided with a 600 to 110 v control power transformer and three overload elements. Wiring will be CEMA Class II, Type B.

b. Relays will be industrial type with vertical contact surfaces or encapsulated contacts.

c. Equipment such as limit switches and pushbuttons in locations exposed to dust or water will be provided with water-tight enclosures.

7. Grounding System

A perimeter ground loop will be provided at each process building, each substation, and each process area. The individual ground grids will be tied together with an interconnecting ground wire. Suitable cross connections will be installed to connect the building steel and equipment to the ground grid such that each major piece of equipment has two ground paths to the grid. Ground rods will be installed so that the total calculated resistance of the ground system to earth will be no more than 5 ohms, as measured by the "Fall of Potential" method.

a. All motors having a rating of 25 hp and larger will be grounded with a grounding wire which connects to either building steel or directly to the ground grid system.

   Motors of less than 25 hp will be grounded by the conduit system.

   All metallic conduit at panels, control centres, and other sources of supply, will be bonded to the ground bus with grounding bushing, or with lock nuts on the panel enclosures.

b. Process areas and buildings will be protected from lightning with air terminals installed in accordance with the NFPA Pamphlet No. 78.

8. Lighting System

The lighting system will be a 600/347 phase to neutral, 4-wire system. Power for receptacles, office equipment, small utility motors, etc., will be taken through dry-type transformers from the 600 v system.

In selecting lighting fixtures throughout the plant, preferences will be given to lamps having long rated life, high efficiency, and low maintenance; such as fluorescent and mercury vapor lamps.

Generally, in locations where mounting height of the fixtures exceeds 14 ft above floor level, mercury vapor lamps will be used for general illumination. In high bay areas, fixture disconnects, such as Thompson hangers will be provided to facilitate relamping; unless other means, such as an overhead crane, is available.

Standby diesel generators to provide power for emergency lighting and heating will be provided in the powerhouse, maintenance shop, the mill and the primary crusher building. The administrative office shall have battery operated emergency lights.

Fluorescent fixtures using 4-ft, rapid-start lamps will be used in all applicable areas such as the change house, electrical equipment room, etc. Mounting height for fluorescent fixtures will not exceed 14 ft.

Incandescent lamps will be used in areas where illumination is provided primarily for personnel safety, such as stairs and walkways, and also in utility areas and other areas where mercury vapor and fluorescent fixtures are not applicable. In general, mounting height of incandescent lamps will not exceed 18 ft.

Minimal street, yard, and parking lot lighting will be provided. Where possible, this lighting will be from adjacent buildings. Outdoor lighting in general will be controlled by photo-cells.

Interior lighting will generally be switched from either contractors or panels. Local switches will be used only where needed for convenience.

Area	Lighting Level (Footcandles)	Fixture Type
Primary crusher building	15-20	Mercury
Secondary crusher building	15-20	Mercury
Stairways and landings	10-15	Mercury and Fluorescent
Conveyor Walkways	Minimum	Incandescent
Conveyor drive and transfer platforms	10-15	Mercury
Locker rooms	25-30	Fluorescent
Electrical equipment rooms	25-30	Fluorescent
Storerooms and warehouse	10-15	Fluorescent
Offices and laboratories	90-100	Fluorescent

9. Communication

A plant intercommunications system will be provided through a private switchboard. This system will be connected to the outside communication system through the central switchboard.

10. Wiring Methods

In general, wire in conduit or duct will be used throughout the plant. Multiple conductor control cables in trays will be used in areas where this method of wiring is appropriate and economical.

11. Materials

a. Conduit in general will be rigid galvanized steel or nonmetallic duct such as PVC conduit or Johns-Mansville Korduct for control, power, and lighting circuits. Conduit in corrosive areas will be rigid steel with a PVC coating.

b. Wire

(1) Ground wire in general will be bare stranded copper, unless other types are required for specific areas. The minimum wire sizes will be as specified in the Canadian Electrical Code.

(2) Wire used on lighting circuits will be type RW90 with chemically crosslinked polyethylene insulation and stranded copper conductor. Minimum size will be No. 12 AWG.

(3) Wire used on power and control will be type RW90 with chemically crosslinked polyethylene insulation and stranded copper conductor. Minimum size for power will be No. 12 AWG and No. 14 AWG for control.

(4) 5,000 volt cable for feeder and branch circuits will be single or three conductor cable having chemically crosslinked polyethylene insulation and with a polyvinyl-chloride jacket. All high voltage cable will be in accordance with standards of CEMA. Interlocked armor cable of similar construction will be used in trays, in areas where this method of wiring is appropriate and economical. Single conductor cable will be shielded; multi-conductor will be unshielded. All conductor will be stranded copper.

(5) Multiple conductor cables will consist of crosslinked polyethylene insulated conductors, all enclosed in a neoprene jacket. The number of conductors will be determined by the requirements of the particular application, except efforts will be made to standardize to reduce the number of different types of cables required. The cable will be similar to General Electric Type SI58145.

c. Cable Trays will be steel, expanded metal, except where covers are required for mechanical protection or personnel safety. The trays will be similar to those manufactured by H. K. Porter, Burndy and Cantrough. Preference will be given to cantilever type hangers to facilitate easy cable installation.

d. Receptacles will be equal to those specified in the tabulation below:

600 v power receptacles Crouse-Hinds type ARE or AREA, 60 amps.

120 v receptacles in offices 15A equal to Hubbell 5262.

120 v receptacles installed in process areas 20A equal to Hubbell 7310G assembled in a weatherproof cover.

e. Switches will be rated 12 amp, 120 v and will be similar to Hubbell 1,200 catalogue series.

f. Pushbuttons at motors will, in general, have CEMA 4 type enclosures and will have the number of control buttons required for the service. The pushbuttons will be equal to Allen-Bradley Bulletin 800H, heavy duty.

g. Disconnect switches will be of the heavy-duty safety type having an enclosure suitable for the area in which they are to be installed. The switches will be similar to Westinghouse Type H-600.

h. Lighting transformers will be of the dry type having 2 to 2-1/2% taps above and below normal on the high voltage winding. The transformers will, in general, be of the 3-phase type having a 600-v high voltage winding and a 120/208-v low voltage winding.

i. Lighting panels will, in general, be surface mounted and will contain "BA" frame breakers, having an interrupting rating of 10,000 amp asymmetrical. The panels will be similar to Westinghouse Type NQB 600 v weather proof.

TABLE B-1

MOTOR WIRING AND LOAD DATA, 550 V, 3-PHASE 60 CY CABLE RATED AT 75 C

hp	F. L. amp	kw	Circuit Breaker Frame	Circuit Breaker Trip	Starter CEMA	Wire	Conduit: Motor Wire Only (in.)	Conduit: Motor and 3 or 4 No. 14 Control (in.)
1/2	0.8	0.5	100	15	1	12	3/4	1
3/4	1.1	0.7	100	15	1	12	3/4	1
1	1.4	0.9	100	15	1	12	3/4	1
1-1/2	2.0	1.4	100	15	1	12	3/4	1
2	2.6	1.8	100	15	1	12	3/4	1
3	4	2.7	100	15	1	12	3/4	1
5	6	4.4	100	20	1	12	3/4	1
7-1/2	9	6.5	100	20	1	12	3/4	1
10	11	8.4	100	30	1	12	3/4	1
15	16	12.6	100	40	2	10	3/4	1
20	21	16.5	100	50	2	8	1	1-1/4
25	26	20.7	100	70	2	8	1	1-1/4
30	31	24.8	100	70	3	6	1-1/4	
40	41	33.0	100	100	3	6	1-1/4	
50	50	41.0	225	125	3	4	1-1/4	
60	60	49.0	225	150	4	2	1-1/4	
75	74	61.0	225	175	4	1/0	2	
100	98	81.0	225	225	4	2/0	2	
125	124	100.0	400	250	5	3/0	2	
150	144	120.0	400	250	5	4/0	2-1/2	
200	192	160.0	400	400	5	350 MCM	3	

C. MECHANICAL DESIGN CRITERIA

1. Applicable Codes and Publications

Design will be effected conforming with the following minimum standards:

a. Federal, Provincial and Municipal Regulations

b. Canadian Standards Association - CSA

c. American Gear Manufacturers' Association - AGMA

d. American Conference of Governmental Industrial Hygienists - ACGIH

e. Air Moving and Conditioning Association - AMCA

f. American Society for Testing and Materials - ASTM

g. Rubber Manufacturers' Association - RMA

h. Conveyor Equipment Manufacturers' Association - CEMA

i. Association of Roller and Silent Chain Manufacturers - ARSCM

j. Joint Industry Conference - JIC

k. Electric Overhead Crane Institute - EOCI

2. Raw Material

Raw material characteristics will be as follows:

Material	Angle of Repose	Min Chute Valley Angle	Draw Angle	Temp.
Alaskite (granite)	38°	55°	65°	Ambient

3. Ambient Conditions

The plant is located in northern British Columbia and involves an open pit mine, crushing, grinding and flotation plants which will operate 24 hr per day all year round. The plant will be located at an elevation of approximately 4,700 ft with temperature extremes of +80 F to -40 F with low dips to -60 F. Working conditions for all equipment shall be considered as dusty.

4. Belt Conveyors

The belt conveyors will be operated inside buildings and heated conveyor galleries as shown on the layout drawings.

a. Belt conveyors, wherever possible, shall be at least 24 in. wide.

b. Maximum angle of incline shall not exceed 15° for all conveyors.

c. Maximum speed for conveyors shall be 420 fpm. Speeds will be selected for 75% of belt loadings.

d. Horsepower will be calculated based on 100% loading at the selected speed or 125% of flowsheet capacity, whichever is greater.

e. The conveyors will be designed for continuous 24 hr operation.

5. Belting

Belting will be selected in accordance with the following:

a. Belting wherever possible shall be suitable for operating conditions of all conveyors, to provide interchangeability for maintenance and inventory purposes.

b. Carcass shall be all synthetic or combination natural and synthetic fabrics.

c. Top cover shall be 3/16-in. thick minimum, bottom shall be 1/18-in. thick minimum. Covers shall be RMA grade 2.

d. Splices shall be vulcanized.

6. Pulleys

Pulleys shall conform to the following:

a. All pulleys shall be of welded construction.

b. Drive pulleys shall be attached to their shafts by keys, keyseats, and taperlock bushings.

c. Other pulleys shall be attached to their shafts with taperlock bushings only.

d. All drive pulleys shall be lagged with 1/2-in. herringbone grooved rubber, vulcanized to the pulley. Other pulleys contacting the dirty side of the belt shall be lagged with 3/8-in. smooth rubber.

e. All pulleys shall have a face two in. wider than the belt.

f. Pulleys shall be crowned 1/8 in. /ft of width except pulleys with belt tension above 80%.

7. Take-Ups

Take-ups shall conform to the following:

a. Take-ups shall provide for adjustment of 3% of conveyor length wherever possible.

b. Screw take-ups shall be top angle frame with self-aligning spherical roller bearings.

c. Gravity take-ups shall be weighted to provide the least tension required for the successful working of the conveyor, and shall be used on all conveyors longer than 100 ft.

d. Counterweights shall be of concrete and located at a maximum of 8 ft above the floor elevation.

8. Idlers

Idlers shall be as follows:

a. All carrying idler rollers shall be 6-in. diameter and shall be of welded construction with roller type antifriction bearings.

b. All carrying and return idlers shall be CEMA Series IV.

c. Standard troughing idlers shall be equal length roller type, 35° or 20° troughing angle as specified on data sheet.

d. Return idlers shall have supporting brackets mounted below the idler, and shall be of the rubber tread type.

e. Training idlers shall be provided for the carrying and return runs of all conveyors more than 70 ft - 0 in. long. They shall be placed within 50 ft - 0 in. of both the head and tail pulley and at not more than 100 ft - 0 in. centres between training idlers, with a minimum of two per conveyor.

f. All impact idlers shall be CEMA Series VI and shall be used on belts handling large lump material.

9. Shafts

Shafts shall conform to the following:

a. All pulley shafts without key seats shall be made of cold drawn mild steel ASTM A 108, Grade 1040. Hot rolled and polished shafting shall be used for shafts requiring machining. Shafts shall not be stressed to more than 8,000 psi in shear for combined tension and bending, and not more than 6,000 psi in tension for bending only. The use of alloy steel shall be avoided.

b. Shaft ends shall not protrude from bearings, unless the extension is required for mounting of components.

10. Bearings

Bearings shall conform to the following:

a. All bearings shall be of the antifriction roller type, sealed against dust with provision for relubrication. See Lubrication herein.

b. All pulley shaft bearings shall be of the self-aligning type.

c. Each pair of bearings, supporting a shaft, shall consist of one fixed and one floating unit.

d. Bearings shall be equivalent to Link-Belt Series 6,800 and shall be selected for a B-10 life of 30,000 hours.

e. Where a shaft does not protrude, the bearing shall be provided with a closure disc.

11. Skirt Boards and Chutes

Skirt boards and chutes shall conform to Kaiser Engineers' Standard Detail MA-26 and to the following:

a. Discharge chutes shall be 6-in. wider than belt width.

b. Loading points on all conveyors shall have skirt boards with the skirts extending a minimum of 1-1/2 sec of belt travel past the point of impact.

c. Skirt boards shall begin from the width of the chute opening, which should not be wider than 2/3 of the belt width.

d. Skirt boards shall be vertical and of steel construction, faced with 1/2-in. skirting rubber. Rubber shall be clamped to the skirt with a flat bar as clamp.

e. Skirt board covers shall be provided for dust control where called for.

f. Skirt board cross sections shall have ample area to minimize air velocity for dust takeoff not exceeding 400 fpm.

12. Belt Cleaners

a. Belt conveyors shall have spring loaded rubber wipers at the discharge.

b. V-type plows shall be used to protect all pulleys on the return portion of the conveyor belt.

c. Hoods shall be located over all vertical gravity take-up pulleys.

13. Deck Plates

No. 16ga decking for return belts shall be furnished the full length of all conveyors.

14. Protective Devices

Protective devices shall consist of the following:

a. Zero speed switches shall be installed on all belt conveyors. A separate live spindle idler mounted on antifriction bearings shall be used. The idler connected to the switch shall be lagged and in contact with the back cover of the belt. The idler shall be coupled to the switch by means of a flexible coupling.

b. Trip line and switches shall be installed on all conveyors. Grouse Hinds Type "AFU" or equal located every 100 ft along with an emergency trip cord.

c. Antilift bars are not required.

d. Torn belt switches shall be provided at the loading points of belt conveyors immediately after the primary crusher pan feeder and the coarse ore pile reclaim belt feeders.

e. High level probes shall be installed in all transfer chutes.

15. Belt Clearances

Belt clearances shall be as follows:

a. With the belt running central, no metallic portion of the conveyor structure shall be closer than 6 in. to the edge of the belt.

b. No stationary metallic part shall be closer than 2 in. above and 12 in. below the return belt.

c. Minimum clearance for clean-up below the return idlers shall be 15 in.

16. Conveyor Enclosures and Supports

Completely enclosed conveyor galleries shall be used on all portions of conveyors outdoors.

17. Pan Feeders

Pan feeders shall conform to the following minimum standards:

a. For material characteristics, capacity, and physical dimensions see attached Pan Feeder data sheet.

b. The apron feeders shall be supplied complete with frame, screw, take-up adjustment for the tail shaft and 1/2 in. A. R. sturt liners.

c. The apron feeders will have a maximum speed of 40 ft/min.

d. The pans shall be integrally overlapping 1-1/2 in. thick manganese cast steel mounted on two or three strands of chain and supported midspan by impact rails.

e. The drive sprocket shall be provided with a shear pin rated at 130% full load torque.

f. Lubrication shall be an automatic centralized lube system.

g. All bearings shall be of antifriction type.

18. Vibrating Feeders

Vibrating feeders shall conform to the following minimum standards:

a. For material characteristics, capacities and physical dimensions, see the attached vibrating feeder data sheet.

b. The vibrating feeder drives shall be of the electromagnetic type.

c. The pan feeders shall be lined with 1/2 in. A. R. plate,

d. The drive arrangement shall be of the below deck type.

19. Screw Conveyors

Screw conveyors shall conform to the following:

a. Screw conveyors and feeders shall have Helicoid flights. Flights shall be continuously welded to pipe shafts.

b. Shaft couplings shall be Link-Belt "Kwick Link" convectors or equal.

c. Hangers with provision for expansion shall be equipped with antifriction bearings and shall be spaced at not more than 10-ft centers for screw diameters 10 in. or less and not more than 12 ft for screw diameters 12 in. and larger.

d. Drives shall be the flange mounted type with thrust bearings, motor mount and V belt drive. The end plate shall have packing seals. The shaft shall be removable without disturbing the reducer end plate.

e. Tail-end bearings shall be antifriction type.

f. Conveyor covers shall be fastened with clamps for ease of inspection.

g. All rough ends shall be the steel plate dust seal type. All troughs will have 1/8-in. resilient gaskets at all joints.

20. Cranes

Cranes shall conform to the following minimum standards:

a. CSA Specification B-167 Class C
EOCI Specification No. 61 Class C.

b. Wire rope shall be improved plow steel, preformed nonrotating type for crane service.

c. Hoist drum and sheaves shall be at least 24 times rope diameter for use with 6 x 37 rope.

d. The hook shall be magnafluxed and the hook capacity shall be stamped on the side of the hood block.

e. All hoists shall be provided with both an automatic mechanical load brake, and each motor shall be equipped with a solenoid operated spring jet electric release brake.

f. All bearings shall be of antifriction type rated for 5,000 hr minimum B-10 life.

21. Dust Collection

a. Included Work - The following criteria covers equipment for the control of dust at the various transfer points of materials handling equipment:

(1) The dust exhaust systems shall be of the balanced design, and ducts shall be sized to maintain system velocities, as outlined under C-21 "Ducts" below, without the use of blast gates or dampers.

(2) Air quantities aspirated at each pickup point shall be within plus or minus 15% of indicated values.

(3) Dust collectors and fans located outdoors shall be weatherproof.

b. Dust collectors shall conform to the following criteria:

(1) All dust collectors shall be continuous automatic, self-cleaning, dry type.

(2) Cleaning efficiency shall be 99.8% minimum of all dust particles down to and including 2-microns size.

22. Fans

a. All fans shall be located on the clean air side of the dust collectors and shall be suitable for continuous 24-hr operation.

b. All fans shall be belt driven.

c. When fans are not mounted on dust collectors, direct mounting on structural steel members shall be avoided for reasons of control of vibrations. The fans shall be placed on concrete pads with a mass of approximately five times the fan weight.

d. Fans shall be designed for 115% of dust collector capacity.

e. Blades shall be radial tip with forward curve at inlet, or straight blade.

f. Maximum tip speed shall not exceed 18,500 fpm.

g. Blade thickness shall be 3/16 in. minimum.

h. Black plate thickness shall be 1/4 in. minimum.

i. Housing thickness shall be 3/16 in. minimum.

j. Bearings shall be of the antifriction roller type with a B-10 life of 30,000 hr.

23. Ducts

a. Horizontal ducts shall be avoided wherever possible, when air carries entrained particles.

b. Upward sloping ducts shall have a minimum slope of 55°, measured from the horizontal. Downward sloping ducts shall have a minimum slope of 45° measured from the horizontal. Velocity in these runs shall be 3,500 fpm minimum.

c. Where manifolds are required for connection to dust collectors, manifold velocity shall not exceed 3,000 fpm.

d. For the fan discharge duct or stack, the velocity shall not exceed 2,200 fpm.

e. Dead-end cleanouts, conforming with Kaiser Engineers Standard MA-17-9, shall be in lower side of ducts and shall be air and dust tight. A cleanout shall be provided near each duct junction and in runs exceeding 15 ft, at not more than 15 ft centres.

f. Portions of duct work, exposed to excessive wear, shall be easily removable.

g. All duct joints and branches shall be designed to minimize air flow resistance and prevent dust build-up.

h. All duct branches and mains shall be provided with fittings, located as indicated on Kaiser Engineers Standard MA-17-1 through MA-17-2.

i. All joints at fittings, dust hoods, and elbows, shall be flanged and shall conform to Kaiser Engineers Standards MA-17-1 through MA-17-12.

j. A conical transition piece shall be provided between ducts and hoods, to reduce duct entry losses. The included angle of the cone will be 60°, and its length shall be not less than twice the diameter of the duct.

k. Branches and entering mains will have an included angle of not more than 30°. Centre line radius of bends will not be less than 2-1/2 times the duct diameter wherever practicable.

l. Flashing will be provided for all duct work that penetrates roofs or floors.

m. No ducts smaller than four inches diameter will be used.

n. Ducts will be made of spiral welded pipe (longitudinal weld permissible if spiral weld is not available) or commercial grade hot rolled carbon steel, with a maximum carbon content of 0.15%. Plate over 1/4-in. thick shall be carbon steel, in accordance with ASTM A 36.

o. Minimum duct wall thicknesses will conform to the following:

Up to 8 in.	14 ga
8 in. to 24 in.	12 ga
Over 24 in.	10 ga

p. Flange thicknesses will be 3/16 in. minimum. All bolts will be 1/2-in. diameter minimum.

24. Hoods

Velocities across hood openings will not exceed 400 fpm. For hood details and material, see Kaiser Engineers Standard MA-17-12.

25. Welding

Welding will be as follows:

a. Welding will conform to CSA W-59, Welding of Steel Structures.

b. The interior of ducts and hoods will be smooth and free from obstructions.

c. Weld protrusions on interior will not exceed 1/32 in.

26. Drives

a. Drives will be designed in accordance with the specific drive requirements and wherever possible, will be direct coupled parallel shaft as indicated on the arrangement drawings.

b. All motors will be the horizontal type.

c. For electrical characteristics of motors, see Section B, Electrical Design Criteria.

27. Speed Reducers

a. All reducers shall have a service factor based on recommendations of AGMA.

b. Thermal rating of reducers shall be based on 100 F ambient temperature at required horsepower at stated capacity.

28. Couplings

Couplings shall be grid type, or flexible geared type, or approved equal.

29. Chain Drives

a. Chains shall be ASA B29.1 riveted roller chain, selected in accordance with recommendations by the Association of Roller and Silent Chain Manufacturers. Maximum ratio shall be 3:1.

b. Chains shall be enclosed in dust, and oil-tight chain cases of 10-ga steel minimum with double over-lapping joints and double washer seals on rotating shafts.

c. Sprockets shall be type B or type C. Type B shall be ASTM A 108, Grade 1040 steel; type C shall be high strength, close grain, cast alloy.

d. All sprockets shall be fastened to their shafts by means of keys, keyseats, and taper-lock bushings.

e. All sprockets shall have teeth hardened to 45 to 60 Rockwell C hardness.

30. V-Belt Drives

a. V-Belt drives shall be furnished with high strength belts, "Dodge Manufacturing Company - Dyna V", or approved equal, or standard A to E sections if necessary.

b. Sheaves shall be provided with key seats and taper lock hubs.

c. Drive selection shall comply with ratings and service factors, standard with the belt drive manufacturer.

31. Hold Backs

a. All inclined conveyors shall be provided with hold backs.

b. Hold backs shall be fully enclosed Sprag CAM type clutch mounted integral with the speed reducer or externally mounted on the high-speed shaft of the reducer.

32. Lubrication

All mechanical operating parts, unless indicated otherwise on drawings shall be provided with means of lubrication suitable for the operating conditions as follows:

a. Lubrication fittings of bearings shall be readily accessible, and where necessary, shall be piped to convenient points on walkways, platforms, etc.

b. Fittings: Mechanical equipment Alemite No. 1627B
Electrical Motors Alemite No. 1488, Mogul Dot

c. Bearings shall be initially grease packed at the shop, prior to delivery unless the bearings are intended for service with oil mist or on circulation lubrication.

d. Gears shall be enclosed and lubricated by oil bath.

e. Chain drives shall be in oil-tight cases and lubricated.

33. Inspection Doors

Inspection doors shall conform to the following criteria:

a. Inspection doors shall be provided in any chute adjacent to a dust-duct connection. Inspection doors shall not be located on wear surfaces and shall be hinged to open outward.

b. Inspection doors shall be provided with a latching device and doors shall be gasketed and dust tight.

c. Minimum size of doors shall be 12 in. square.

34. Linings

All chute surfaces exposed to abrasive material shall be lined with bolted-on 1/2-in. A. R. P.

35. Bins and Hoppers

a. Bins shall be equipped with high- and low-bin level indicators as required.

b. Emergency shut off gates, or rods shall be provided under all hoppers, bins, and stockpile reclaim points.

36. Safety Guards

Safety guards shall be provided in accordance with Federal, Provincial and Municipal codes and regulations.

Guards shall completely enclose moving parts, so that physical contact with the moving parts cannot be made with the guard in place. Design and construction of guards shall permit easy removal for service of guarded equipment. Safety guarding shall be of the following types:

a. All couplings shall be furnished with guards fabricated from 10-ga sheet metal and mounted to provide a minimum of 1-in. clearance.

b. All V-belt drives shall be furnished with guards fabricated from 10-ga steel with 1/2-in. opening times 16-ga flattened expanded metal with a minimum of 1-in. clearance. Adequate clearance shall be provided for complete adjustment of belt tension.

c. Guards for shaft mounted reducers with motor brackets shall be fastened to motor brackets.

d. Pulley and nip-point guards for belt conveyors shall be fabricated from a 10-ga steel frame and 1/2-in. opening by 16-ga flattened expanded metal and shall extend to a minimum of 30 in. from all pulley nip points.

e. Take-up guards - Overhead counterweights of gravity take-ups shall be guarded at ground level by fencing around and outside of the take-up plan area. The fencing shall be 8 ft high and no gates are required.

f. Conveyor belt guards on all return belts above walkways or passages, having less than 8 ft 0 in. clearance, shall be provided.

D. STRUCTURAL DESIGN CRITERIA

1. Applicable Codes

   General: National Building Code of Canada, 1970 Edition.

2. Roof Loads

   a. Snow Loads - 60 psf. No reduction for roof area, ,increase for leanto and low roofs, as code requirements.

   b. Dead Loads - Weight of all building materials plus allowances for ductwork, lighting, sprinklers, ceilings or mechanical equipment as applicable.

3. Wind Loads

   All structures shall be designed to withstand wind pressure per NBC for 20 psf. (Equivalent wind gust = 86 mph.) Pressure to be modified in accordance with height and shape factors as per NBC supplement No. 3.

4. Earthquake Loads

   All earthquake loading and design shall be in accordance with NBC. The plant site is located within seismic intensity zone 3.

5. Crane Loads

   Wheel loads and spacing for cranes shall be as per vendor information. Impact and lateral load as per NBC or as per vendor information whichever is more critical.

6. Floor and Platform Live Loads

   a. Floor design shall be based on probable area loading. Check for heaviest equipment load.

   b. Minimum floor loads, other than listed below, shall be in accordance with the NBC.

(1) Maintenance platforms and walkways (or a moving point load of 2,000 lb whichever is more critical)	- 50 psf
(2) Stairways and operating platforms	- 100 psf
(3) Conveyor walkways	- 50 lb/lin ft
(4) Platforms supporting conveyors (not including conveyors)	- 50 psf
(5) Offices, laboratory, and equivalent area	- 100 psf
(6) Storage floors	- Min 200 psf
(7) Area accessible to ore dump truck	- 100-ton dump truck loading
(8) Area not accessible to trucks	- 3-ton fork lift

7. Load Combinations

All buildings and structural numbers shall be designed for the most unfavourable effects of the various load combinations as listed in the National Building Code and as adjusted by probability factors. Allowable stresses shall be increased for certain load combinations only as specifically allowed in the NBC.

Operating loads include water or ore contents, conveyor belt tension and materials, vertical and horizontal loads from cranes or machinery. LL is floor live load only.

8. Factor of Safety

a. Overturning

FS = 1.5	Retaining walls
FS = 2.0	Crusher foundations
FS = 2.0	Buildings

b. Sliding

FS = 1.5	Retaining walls
FS = 2.0	Buildings

The weight of earth superimposed over the foundation (including earth at a 20 degree angle from the vertical edge of foundations) shall be considered in calculating the resisting moment and sliding.

9. General Plant Conditions

a. Maximum wind gust 86 mph (estimated)

b. Maximum temperature sustained +70 F Peak +80 F.
Minimum temperature sustained -40 F Peak -60 F.

c. Plant life - 15 years

d. Frost penetration - 10 ft (estimated)
Grade beams to be eliminated where possible.

e. Ground water - ground water to be considered at grades for all plant foundations and retaining walls.

f. Bearing Pressures - Assumed allowable bearing pressures shall be:

Talus and other materials	- 4.0 ksf
Rock	- 50.0 ksf

Allowable bearing pressures are to be confirmed by a detailed soil investigation.

g. Piles - Friction piles and tension piles shall not be used unless dictated by special circumstances. End bearing piles shall be steel or concrete and designed in accordance with the NBC.

h. Materials

(1) Concrete - 3,000 psi generally
4,000 psi special conditions
5,000 psi maximum for exceptional conditions such as splitters in crusher

(2) Reinforcing steel - 60,000 psi yield (ASTM A615-60)

(3) Welded wire fabric - Standard ga and sizes to CSA G. 30. 5 or ASTM A-185

(4) Structural steel - Design to CSA, G. 40. 8(A36) (substitution of higher strength steel G. 40. 12 may be made depending on availability).

(5) Timber - Structural members to be 1, 500-f industrial grade Douglas Fir. Nonstructural timbers to be construction grade spruce or yellow cedar.
Glulon members to be 24-f stress grade Douglas Fir.