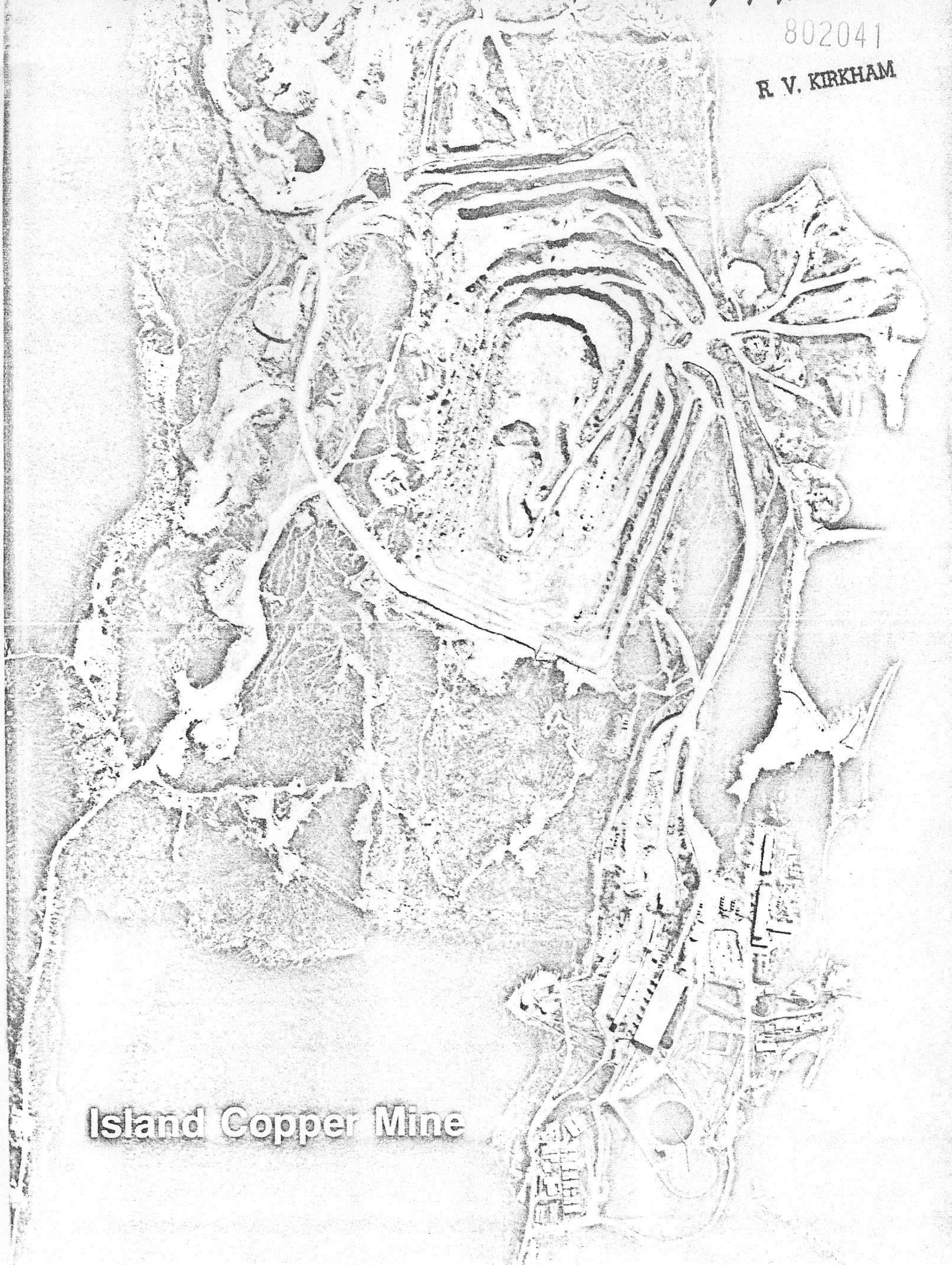
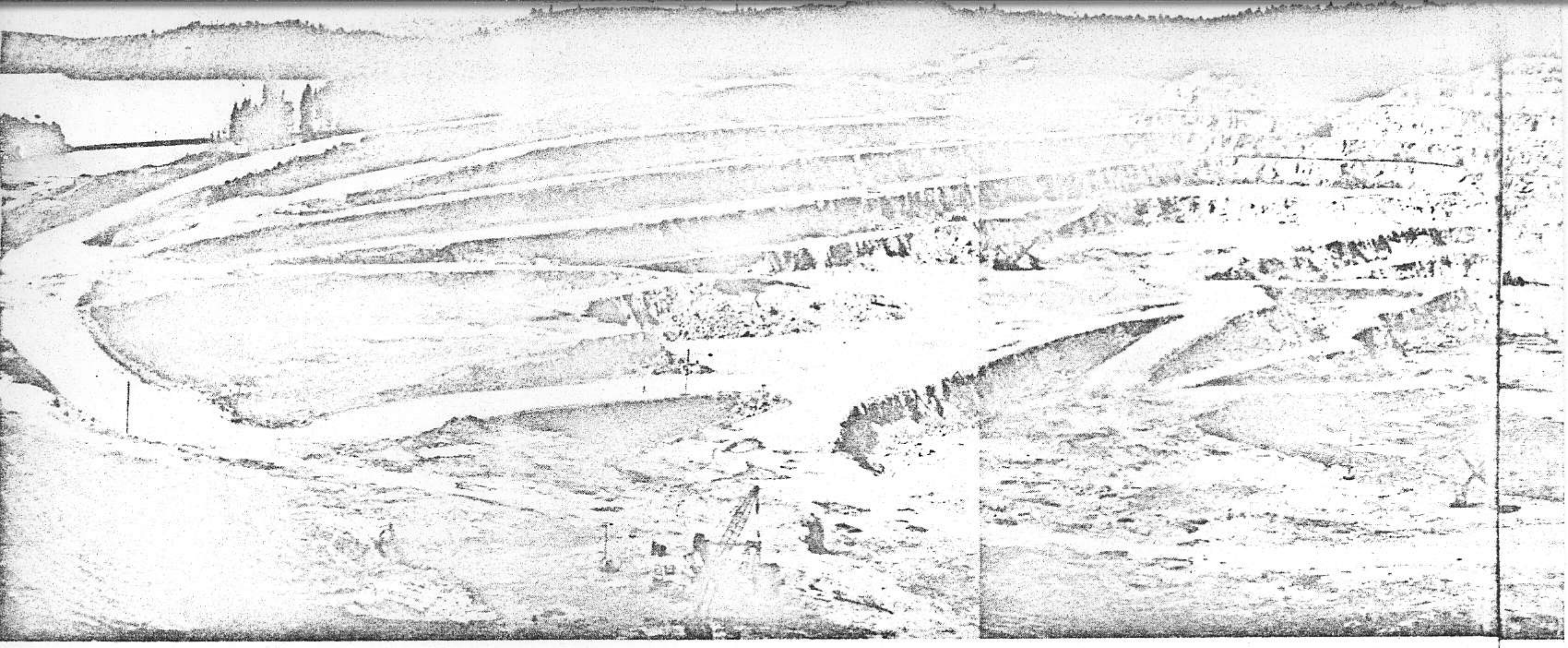
WESTERN MINER DEC /74



Island Copper Mine



Island Copper Mine

from discovery to open pit production

One of the largest producers of copper concentrates in Canada, the Island Copper Mine is located near Port Hardy, British Columbia, at the northern end of Vancouver Island. It is operated by Utah Mines Ltd, which was incorporated for the purpose in 1972 and is a subsidiary of Utah International Inc, of San Francisco.

Estimated reserves are near 280-million tons of copper-molybdenum ore with an average grade of 0.52% copper, 0.027% molybdenum sulphide. Developed for a capital cost to start-up of some $88-million, the first shipment was made at the end of December 1971. Full design capacity of 33,000 tons/day was reached in 1973, and this has since been raised to the current figure of 38,000 tons/day.

The mine has a very strict environmental control system to monitor the effects of tailings disposal into Rupert Inlet, on which the facilities are situated, and which also provides access to the sea.

In the general area of Port Hardy, which is a small logging and fishing community, there was some coal mining as early as 1835 for a short while. The Geological Survey of Canada reported on the geology of northern Vancouver Island following field work by G M Dawson in 1886, and field work has been carried out since then. Mining of copper started in 1911 some 20 miles to the south on what became the Coast Copper property of Cominco Ltd and was revived during the 1960s. Other coal and iron mines operated for some years but are now closed.

The original discovery on the present Island Copper property was made late in 1965 by a local prospector, Gordon Milbourne, who exposed indications of pyrite and chalcopyrite mineralization in a depression under two overturned trees. Early in 1966 an agreement was reached with Utah, and that company's exploration arm started a systematic programme of geologic mapping, geochemical and geophysical surveys, and some drilling.

The Island Copper deposit is a typical copper porphyry. Mineralization occurs in volcanic andesites which have been intruded by a quartz feldspar porphyry. The porphyry is itself occasionally mineralized.

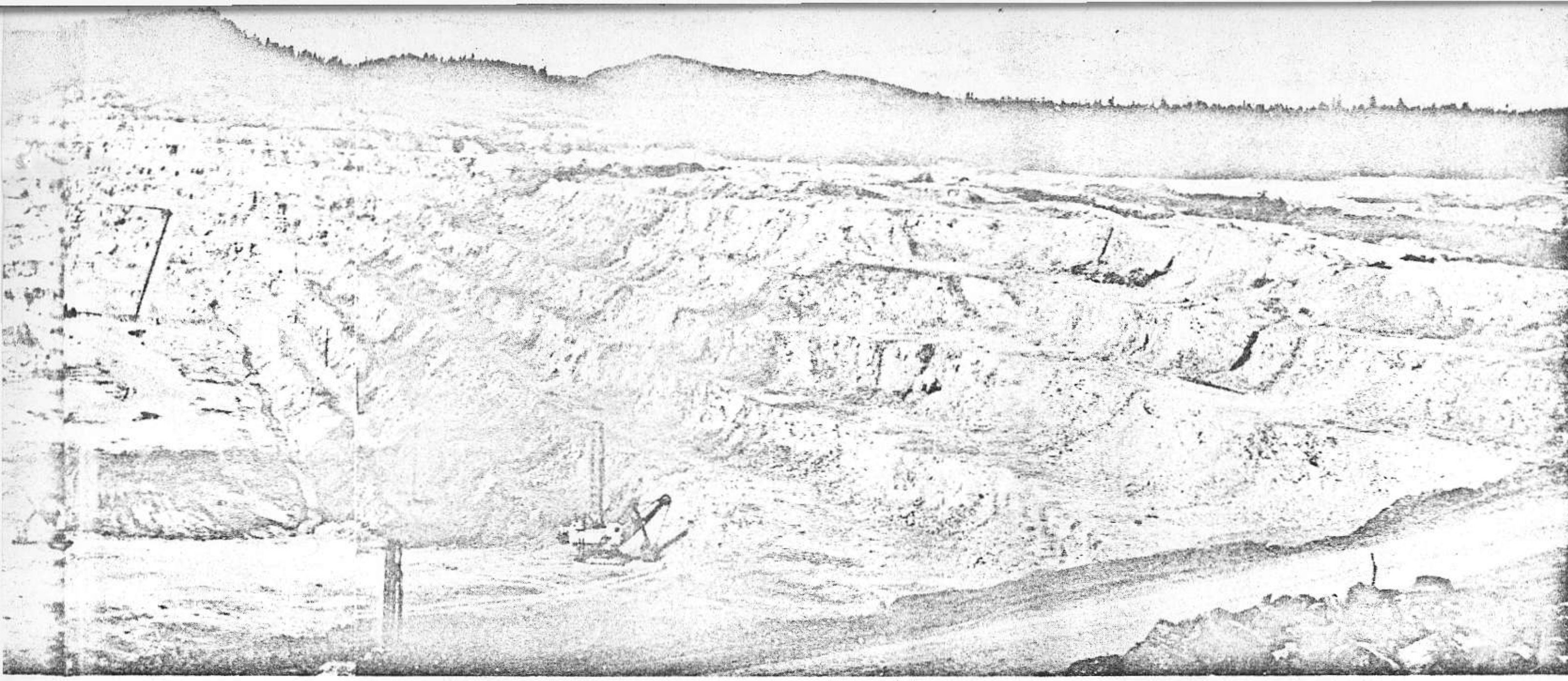
The geology and mineralization of the Island Copper deposit were covered in some detail in an article by Maurice J Young and E S Rugg, both of Utah, which was published in *Western Miner* in February 1971 (p31-40).

DEVELOPMENT

Following the initial exploration work, drilling showed a small orebody of about one-million tons grading just over one per cent copper. While this was being drilled, other targets were defined by geochemcial sampling, and early in 1967 there was recorded the first diamond drill hole into what has become the Island Copper orebody, over a mile southeast of the original discovery.

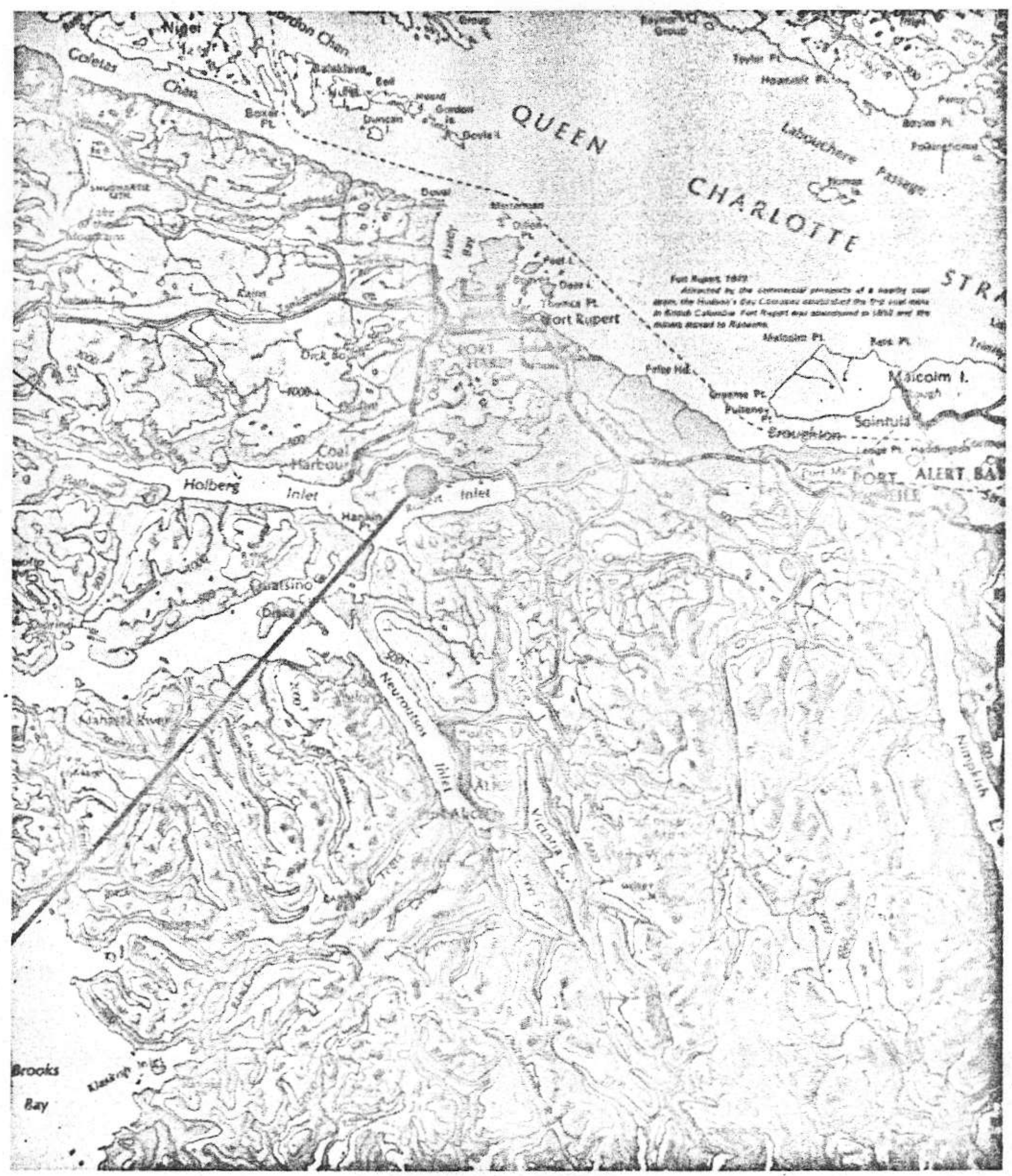
When it became evident, late in 1967, that there was potential for a large porphyry copper deposit, drilling was accelerated and plans were made for a feasibility study and development programme.

An exploration shaft was sunk in the summer of 1968, and drifts were mined on strike to cross-cut the upper part of the orebody. The material produced was shipped to a Utah International pilot plant at Cedar City, Utah, and samples were provided to equipment manufacturers for grinding and crushing tests. During this period the initial contacts were made with government regulatory



Island Copper open pit

Location of Island Copper Mine (black dot)



bodies. The steps taken to meet environmental regulations are of particular interest and are dealt with in some detail elsewhere in this review of operations.

A formal feasibility recommendation was made to, and approved by, the Utah board of directors early in 1969. By this time studies had been made of marketing, housing, environment, metallurgy, and plant design. It had also been established that the orebody would exceed 200-million tons.

The actual development of the project was announced in June 1969, and from then construction continued to completion by the winter of 1971. The first shipment of concentrate was made on 28 December 1971.

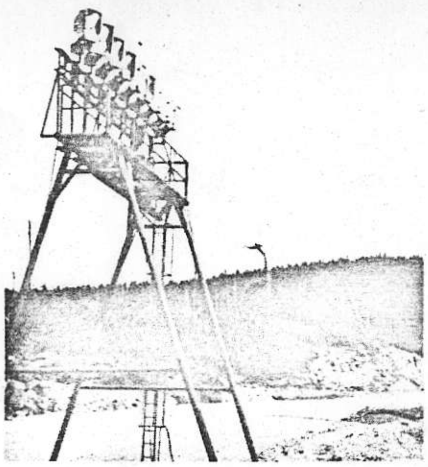
PIT OPERATIONS

The Island Copper orebody is mined by an open-pit operation. The pit will eventually be 8000 ft long, 4000 ft wide, and 1200 ft deep. When visited by *Western Miner* in August 1974 the lowest bench was at the 880 level, or some 120 ft below sea level.

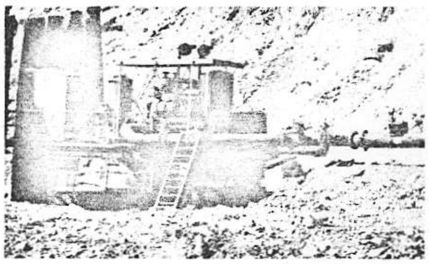
Mine operations, including supporting maintenance work, are continuous and on the same schedule of three shifts a day, for seven days a week, as the mill.

The orebody and waste rock are overlain by glacial till which ranges in thickness from 15 ft in the centre of the pit to 250 ft on the east and west sides. Benches are mined to a height of 50 ft in the till and 40 ft in rock.

Drilling is done with four Bucyrus-Erie units: three 60OR electric and one 45R diesel-electric rig. Drill holes are



Floodlights over open pit



Electrical and diesel pumps in pit

9⅞-inch diameter, drilled to grade in glacial till, and to 8 ft below grade in rock. Drill pattern varies according to local conditions.

Explosives used are largely AN/FO in bulk, with some packaged types, and packaged AN/FO for wet holes that cannot be pumped dry. Powder factors are about 0.4 in till and 0.9 in rock (pounds of AN/FO per cubic yard of material).

Broken material from the pit is loaded into trucks by shovels with 15-cu. yd. buckets. On normal operations, four of five shovels are working. The units used in the mine are all P&H electric shovels: three 2100B models and two 2100BL with lower propel system.

For special loading operations, and as a back-up for the shovels, there are two L700 Letourneau 15-yd front end loaders which are fitted with a 12-cu.yd bucket and extended arms for additional reach when loading into high bodies on large trucks.

The haul trucks used at the mine include 25 Unit Rig M120 units, each of 120-ton capacity, which average something over 100 tons a load with their 78-cu.yd boxes. More recently there have been added five Unit Rig Mark 36 trucks with a rated capacity of 170 tons, and a further five of these units are due to be delivered by early 1975.

The Unit Rig vehicles are diesel-powered with final drive by electric motor in each wheel.

Ore is carried from the pit to the crusher over a distance of 1-1.5 miles, involving a climb of up to 280 ft from the level at 120 ft below sea to the crusher at 160 ft above sea level. Haul roads are 100 ft wide; ramp grades in the pit are 8-10%. Waste is dumped on land to the north of the pit and into a low-level marine fill to the south of the pit.

The mine is in an area of heavy rainfall (about 70 inches a year). This factor, combined with the extensive dumping operations, makes it necessary to have a large fleet of auxiliary vehicles, including three graders, three rubber-tired dozers, five Caterpillar D-8 and five D-9 tractors, and three Fiat-Allis HD-41 dozers.

Also used for auxiliary and back-up work are a Caterpillar 988 loader and four 35-ton Mack trucks. For ditch work there are a Gradall and backhoe.

The operations in the pit are readily supervised from a field office located on high ground. A traffic supervisor in the office can see all vehicles and equipment in the pit, and maintains radio contact with the operators. Large banks of floodlights provide adequate visibility even during hours of darkness.

Operations are controlled by a supervisor and two principal foremen, one on production and one on services, which include pit dewatering and road and dump maintenance.

Electrical power supply is an important feature of the operations. Mobile transformer stations in the pit convert 13.2kV main supply to 4160V for major equipment and to 575V for auxiliary services. The transformers can be quickly moved to serve, for example, new working areas for the shovels and drills. Electrical cables to the shovels are protected in some working areas by ground-level rubber channels instead of overhead gantries.

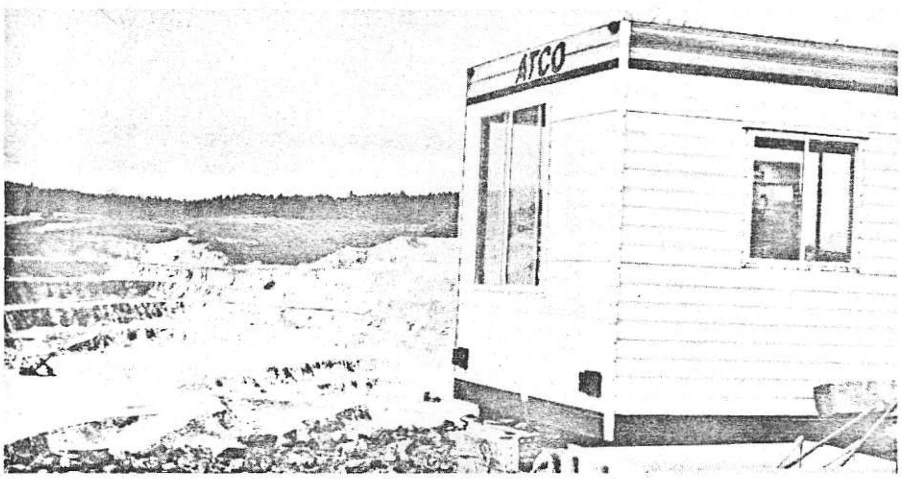
Production from the pit was originally designed at 112,000 tons/day of ore and waste, but was running at 140,000 tons/day during 1974 and is to be further increased.

The stripping ratio, waste to ore, over the five-year period of operation to 1975, is about 3.1:1.

There are two main pumps in the pit, one electrically driven, one by diesel, which are used alternately. There are also many smaller electrical units in different parts of the pit.

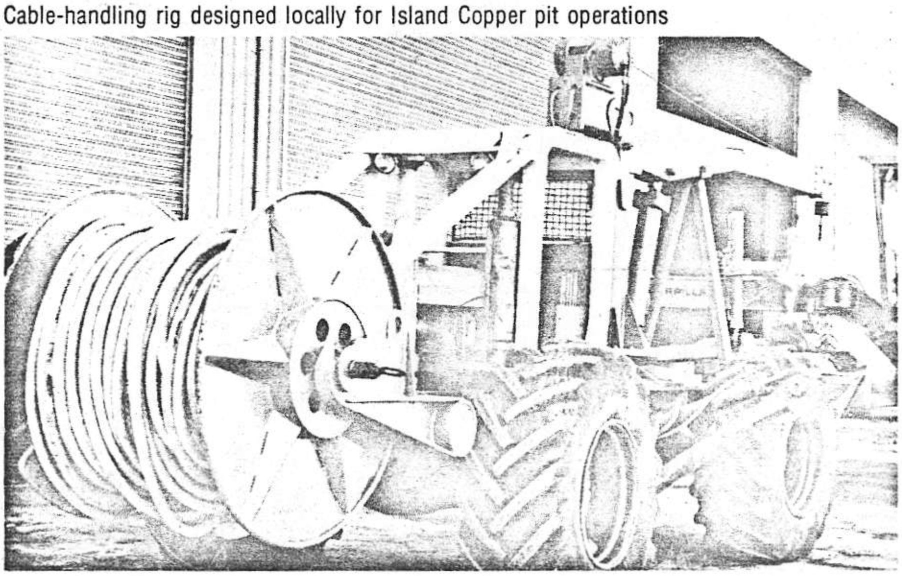
PEOPLE

The Island Copper mine is in an area where there is no ready supply of skilled workers or support services, and it has been necessary for Utah to provide



Control office overlooks all operations in the Island Copper pit

Cable-handling rig designed locally for Island Copper pit operations



housing and other incentives to attract and hold the necessary people.

The company has built about 200 single houses, duplexes, and townhouses which are sold to employees at subsidized prices. There are 15 different models of housing, all built to high standards. A major subdivision in Port Hardy was designed by leading Vancouver architects.

There are also subsidized rental apartments, built and operated by other agencies, and there is to be developed a mobile-home park to accommodate the growing trend to this type of housing.

About half the employees of Island Copper live in the Port Hardy housing, with another group in a camp at the mine site where modular units provide single-occupancy accommodation.

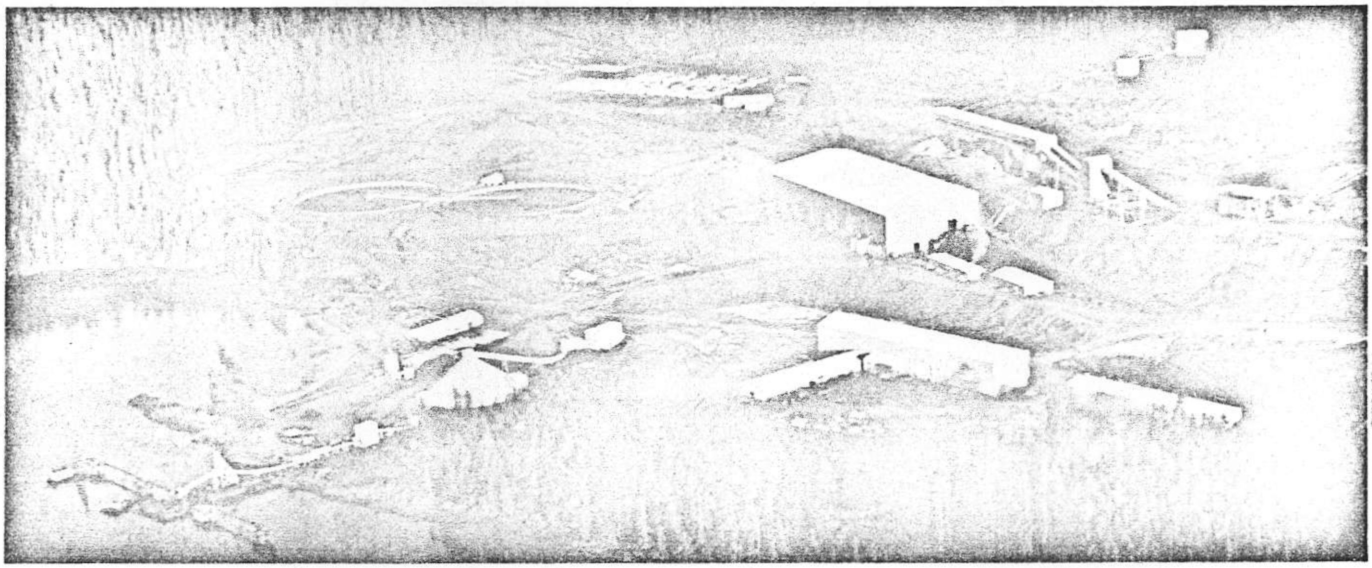
Mine manager at Island Copper is Robert N (Bob) Hickman, who has been with the operation from the start and succeeded to his present position in November 1974, when the former manager, Morton E (Mort) Pratt was made vice-president and general manager of Island Copper, based in Vancouver.

Glen F Andrews is the plant manager with responsibility for mill production, mill maintenance, metallurgy, analytical laboratories, and environmental programme.

Open pit manager is Robert A Leitzman, who heads mine production, engineering, geology, and maintenance.

Project engineer is John C Hannah, with construction and townsite development responsibility.

Personnel manager is G Harold Horwood. H R Tschiedel is office manager, and T L Bon is purchasing agent.



Island Copper Mine

milling for copper and molybdenum

Chris Brown Metallurgist, Island Copper Mine

The Island Copper plant was designed and built by Fluor Utah Ltd, in conjunction with Utah design staff. Originally designed to treat 33,000 tons/day (t/d) of ore, the current average milling capability is 38,000 t/d (Nov'74). The first of six semi-autogenous grinding lines was brought on line in October 1971, start-up of the remainder being governed by equipment delivery schedules.

The summer of 1973 saw the start-up of an additional three ball mills, and attainment of the present production level. The current process flowsheet is shown in simplified form.

CRUSHING

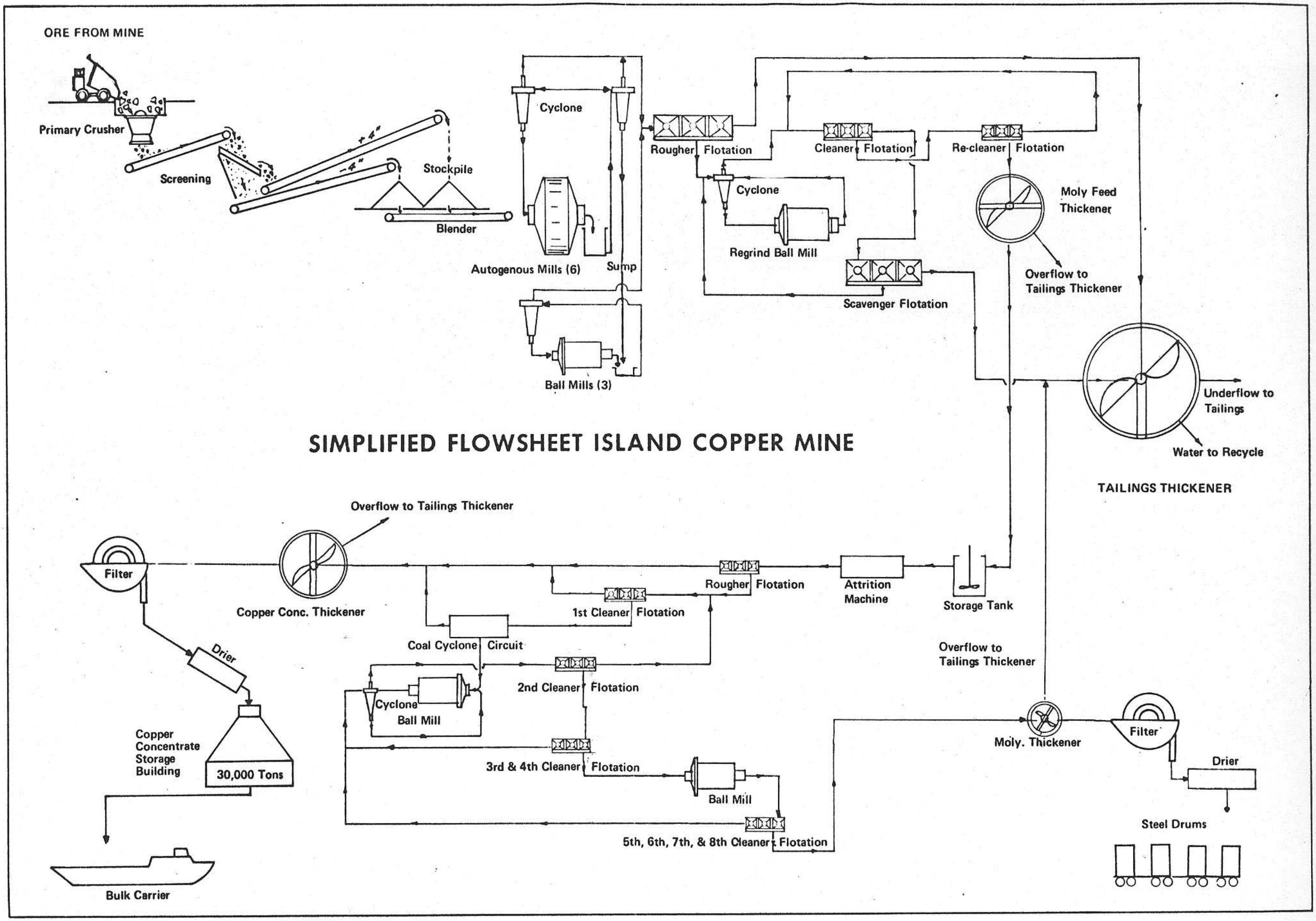
The crushing circuit is relatively simple, comprising single stage crushing of the ore followed by screening to fine and coarse fractions.

The crusher is a 54x74in Allis Chalmers Superior Gyratory with a setting of 9-10in, driven by a 500-hp motor. Trucks dump on three sides of a rectangular dump pocket. An 84inx20ft link belt apron feeder draws crushed ore from a 250-ton surge bin below the crusher, discharging onto a 72-in wide inclined steel cord conveyor belt. A fixed splitter box at the conveyor discharge separates feed to two parallel Jeffreys 6x20ft vibrating grizzlies for screening into +4in and −4in fractions. The fractions are collected on short reversible conveyors which feed the coarse and fine ore stacker conveyors. Trippers distribute ore to outside stockpiles adjacent to, and above, the concentrator grinding floor.

The circuit design capacity is 3000 tons/hour (t/h), crushing operations being on a 3-shift/day 7-day/week basis with 1-shift/week scheduled for preventive maintenance. Operation is supervised from a control room at the crusher dump pocket, a duplicate panel being situated in the main mill control room. Truck dumping is governed by level in the crushed ore surge bin (measurement by nuclear gamma gauge).

The apron feeder is equipped with a vari-speed drive for control of feed rate to screening. Mercoid tilt switches alarm build up at all transfer points, and TV cameras are used for remote monitoring of the crusher dump pocket and apron feeder discharge. Weightometers are installed on the primary and fine ore tripper conveyors.

The circuit has remained unchanged since start-up, minor modifications being: (i) installation of a pneumatic rock breaker and extended crane coverage above the crusher dump pocket to speed clearing of oversize rock. (ii) Replacement of sonic gauge by nuclear gauge for measurement of surge bin level. (iii) Replacement of mechanical brakes on tripper car wheels by brakes on the shafts



SIMPLIFIED FLOWSHEET ISLAND COPPER MINE

between the motors and gear reducers. (iv) Installation of dust collection equipment in the screening building. Equipment was installed in the main crusher building for start-up.

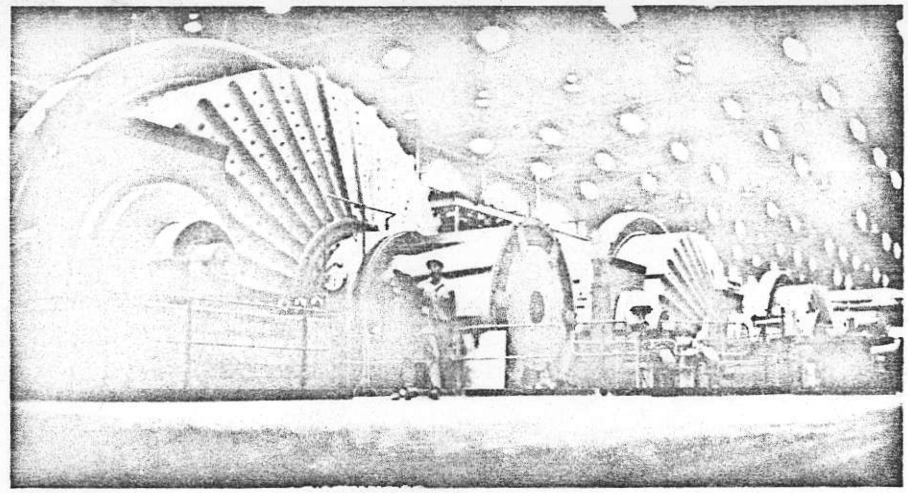
Two major factors limit current output: (i) an annual rainfall of 80-100in/year necessitates frequent clearing of build-up in chutes and screens during the winter months. (ii) Live capacity of the surge bin beneath the crusher limits truck dumping frequency, particularly when crushing wet ore. (120- and 170-ton capacity trucks are in use).

GRINDING

The original grinding circuit comprised single stage grinding in six autogenous mills, each in closed circuit with a cluster of eight hydrocyclones. Cyclone overflows were combined and pumped directly to copper rougher flotation. Two major modifications to this circuit have been made since start-up. The mills have been converted to semi-autogenous grinding units by the addition of steel balls, and three secondary ball mills have been installed. The current circuit comprises three identical grinding lines, two primary semi-autogenous mills producing feed for one ball mill.

Feed to each primary mill is removed from the stockpiles through draw holes under which 42inx10ft Jeffrey apron feeders are located. There are three feeders per mill, one under the fine pile, one under the coarse, and one at the junction of the two piles which is designated mixed ore. The feeders discharge into a 36in wide horizontal conveyor feeding the mill. Ramsey weightometers are installed on the mill feed conveyors. Each Hardinge Koppers mill is 32ft diameter, 14ft long, has a grate discharge, and is driven by two 3000-hp 13,200V wound rotor motors. The mill drive arrangement is motor, gear reducer, pinion, main drive gear. The main drive gear is located centrally on the circumference of the mill shell. To increase useful life of the mill lining, the drive rotation may be reversed. The lining arrangement comprises rows of liner plates separated by rows of lifter and wedge bars, 72 rows of each around the mill circumference. The discharge grates are slotted, the inner slot width being 5/16in, and the outer ¾in. A trommel screen on the mill discharge is used for return of +⅜in material to the mill, screen oversize returning through two ports.

The mills discharge into cylindrical sumps; 12x12in 150-hp Georgia Iron Works centrifugal pumps are used to feed the clusters of eight 20in Wemco hydrocyclones, one for each mill. Overflow product from these cyclones goes to flotation. Cyclone underflow is split, a portion returning to the primary mill, and the remainder feeding secondary grinding. Individual sumps and pumps are used for transport of these underflow



Inside the Island Copper mill; semi-autogenous mills

portions from the primary cyclone clusters; 8x8in 50-hp GIW pumps feed the secondary mill cyclone feed sumps, 10x12in 200-hp GIW pumps feeding clusters of six 20in Krebs cyclones. Underflow from these clusters is secondary ball mill feed. Overflow is combined with that from the primary cyclones. The three secondary ball mills are Allis Chalmers 16.5x22ft overflow type. Each is driven by one 3000-hp wound rotor motor through almost identical drive trains to those of the primary mills, equipment being duplicated where ever possible to minimize the spare parts inventory needed. Single wave lifter-liners are used in the ball mills.

GRINDING CONTROL STRATEGY

During the feasibility stage of mine development, the possibility of application of autogenous grinding to milling of Island Copper ore was realized. Factors favouring adoption of this grinding technique over conventional rod and ball milling were: potential capital cost savings, operating cost savings through circuit simplicity, elimination of grinding media usage, and crushing circuit simplicity (an important consideration in a wet climate). This potential was sufficient incentive to proceed with pilot plant testworking using a 6ft diameter by 2ft long Cascade test mill.

Samples for pilot plant work were obtained from a shaft sunk where exploration of the ore body had been most intensive. The outcome of this pilot plant testwork was the decision to install six autogenous mills in closed circuit for primary grinding.

Soon after mill start-up, it became evident that the ore used for pilot plant work was not representative of the main ore body in one critical respect, namely rock structure. The test shaft had been located in a zone of the ore body where fracturing was much less severe than normal for the bulk of the ore mined to date.

Samples tested in the pilot plant typically contained 40-60% +4in competent rock, whereas run of mine ore has averaged 10-15% after crushing. Since autogenous grinding relies on rock-on-rock impact for rapid size reduction, a proportion of large competent rock pieces in the mill charge is essential. With insufficient coarse ore, the prime mechanism of size reduction becomes attrition rather than impact. This was observed to be the case at Island Copper during commissioning. Symptoms of this condition were observed to be low through-put (1/2 - 2/3 that of the design 240 t/h/mill-line), high power consumption (25-50 kW/ton as opposed to the anticipated 18-22 kW/ton), and a much finer than target product at design cyclone feed density (80-90% −200 mesh; target is 70% −200 mesh).

The most expedient solution to this problem, and one that required no flowscheme modifications, was to make up for the coarse rock deficiency by adding steel grinding media to the mill charges. (Mill structural specifications had been written to provide for this possibility, based on other operators' experience.) Steel additions to the mills started November 1971. By February, all but one of the mills were operating with ball charges, the one mill remaining autogenous until May 1972 for comparison purposes. Currently the mills operate with 7-8% by volume charges, make-up comprising 70% 3in and 30% 4in diameter forged steel balls. This combination has been derived as optimum after extensive testing of different ball sizes (1.5-4.5in diameter), combinations of sizes, and varying % charges (1%-9%).

In addition to the ball charging experiments, modification of the circuit was considered. A survey of industry experience suggested production would be improved by operating the mills in partial or fully open circuit. Plant testing confirmed this observation and resulted in the decision to install the three secondary ball mills. Commissioning started June 1973 and, with the refinement of primary mill ball charging strategy, has resulted in achievement of the current through-put capability of 38,000 t/d.

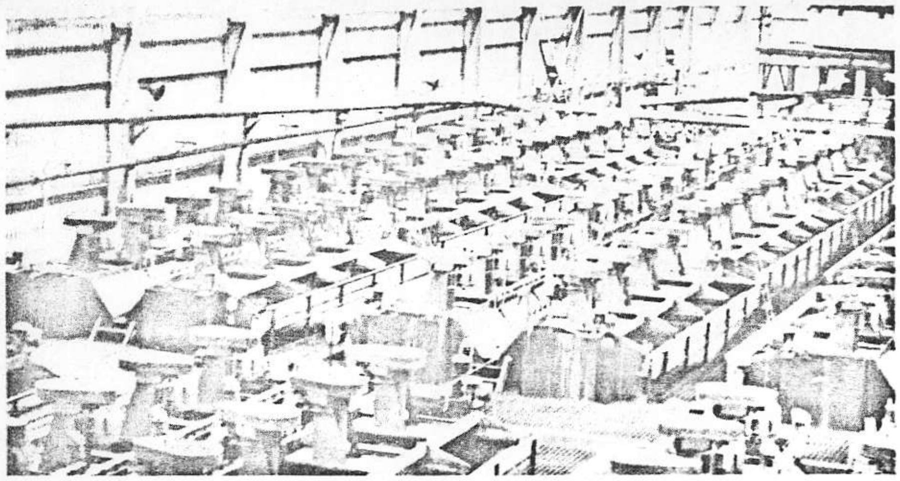
Other major operating statistics for primary-secondary grinding in 1974 are an overall power consumption of 22.5 kW/ton, ball consumption of 2.2 lb/ton, liner consumption of 0.34 lb/ton, and equipment availability of 91%.

Development of an effective primary mill control strategy is also considered to have contributed significantly to achievement of higher than rated average through-put. It was originally envisaged that two basic instrumentation loops would make possible fully automated control of autogenous mill operation. Mill loading was to be 'Cascade' controlled from an analog power draw recorder/controller, varying mill feed rate to maintain kW draw at a set point. (Variation of feed rate is accomplished through vari-speed feeder drives. Ratio controllers govern the relative feed rates from each feeder, so that ratios remain the same even with changes in total feed rate. A mill feed water controller allows cascade control of water addition from the weightometer recorder/controller. The desired ore-water ratio is preset using a separate controller). In practice, cascade control of feed rate based on power draw set point was not possible. The instrumentation could not differentiate between decreasing power draw due to mill overload, and decreasing power draw due to increasing ore amenability to grinding. Power draw alone was hence unsatisfactory as an indicator of mill loading.

Back pressure on the mill trunnion lubrication system was found to be a reliable indicator of actual mass loading. Used in conjunction with the power draw record, it is possible for the operator to determine whether decreasing power draw is due to mill overload or improved ore amenability. Optimum mass loading was observed to vary with changes in ore characteristics. Hence cascade control of feed rate to a pressure set point alone also proved impossible.

Mill sound level offered a third possibility as a cascade control input signal. Monitors have been installed at each mill, but cascade to a set point again proved impractical, optimum sound level also varying with ore type milled. Recording of these three elements does provide the operator with sufficient information for effective manual control however, and could provide the basic elements necessary for a digital computer based control strategy.

The second loop represents an attempt to control cyclone separation. A nuclear density gauge is installed on the cyclone overflow monitoring product specific gravity. A controller regulates water addition to the cyclone feed sump based on an overflow specific gravity set point. This loop is effective for density control, but does not provide control of product particle size. With density held constant, product size distribution varies with ore type milled. An Autometrics Co Ltd particle size analyzer is currently being tested to provide size-, rather than density-based, control of cyclone separation.



Flotation cells in part of the Island Copper mill

Secondary ball mill operation is also manually controlled at the present time. Cyclone feed sump level and mill sound level are used as indicators of mill loading, and cyclone overflow densities are controlled by nuclear gauge. Screen analyses are routinely performed on all cyclone overflows. They produce a guide line for cyclone overflow density required to produce the target flotation feed size distribution.

All control instrumentation for primary and secondary grinding is panel mounted in a central control room. Honeywell instruments are used, the panels being supplied by Swanson Engineering and Manufacturing Co Ltd.

COPPER FLOTATION

Primary and secondary cyclone overflow product is split between two parallel identical Cu rougher flotation sections. Each section comprises five banks of 14 300-cu.ft Wemco cells arranged in a 5-5-4 pattern; 12x12in 200-hp GIW flotation feed pumps transfer pulp at 38-40% solids to stationary distributors feeding the flotation banks. Rougher concentrates from each section are reground separately in two 9ft diameter by 18ft long, 700-hp Allis-Chalmers overflow ball mills. The mills operate in closed circuit with clusters of eight 15in Wemco hydrocyclones. Cyclone overflows are combined and pumped to a fixed distributor feeding five banks of 10 100-cu.ft Galigher 'Agitair' cleaner flotation cells. Cleaner concentrate is recleaned in four banks of eight 53-cu.ft Agitairs. Concentrate from recleaner flotation is molybdenum plant feed, tailings being recycled to cleaner flotation.

Cleaner tailings are pumped to two banks of 14 300-cu.ft Wemco cells for scavenging. Concentrate from this flotation stage combines with rougher concentrate for regrinding. Scavenger tailings are combined with the rougher tailings to become the final plant tailing. Where required, rubber-lined Allen Sherman Hoff centrifugal pumps are used for transport of Cu flotation products. Equipment may be started locally or remotely from the central control room.

Copper flotation metallurgy

The most common rock type containing sulphide mineralization is andesite which is typically dark grey in colour, fine grained and highly fractured. It is estimated that approximately 80% of the ore body is contained in this host rock, the remainder being quartz porphyry and brecciated material. Sulphide mineralization is predominantly along cleavage and fracture planes (70%), the remainder being finely disseminated.

Sulphide minerals in order of abundance are pyrite, chalcopyrite, molybdenite and very minor amounts of bornite, galena, and sphalerite. Gold, silver and rhenium are present. Silicates, magnetite, calcite, clays and gilsonite make up the remainder of the host rocks.

Current mill heads grade 0.50% Cu and 0.017% Mo. Overall circuit recovery has averaged 87% and recleaner concentrate target is 24% Cu.

The present reagent balance is relatively simple, collector and frother being added in rougher flotation only. Additions are made to the flotation feed sumps (60% of total), flotation bank first junction boxes (25%), and second junction boxes (15%). Average consumptions are 0.008 lb/ton potassium amyl xanthate collector, and 0.06 lb/ton of a 93.5% aerofroth 71R with 6.5% Dow SA1012 frother blend. Manually adjusted Clarkson feeders are used for metering reagent additions. Rougher, cleaner, and recleaner feed pHs are automatically controlled using Beckman probes for sensing and on-off squeeze valves for metering. Respective targets are 10.0, 10.8, and 11.5. Overall lime consumption is 1.3 lb/ton. Approximately 65% of this quantity is added to the tailings thickener feed, pH of which is automatically controlled to 11.3-11.5.

Water reclaimed from the thickener overflows is used in the grinding circuit, thus reducing rougher lime requirement. Fresh water is used for launder and sump spray dilution in the flotation circuit. Bubbler tube type pulp level monitors are installed to control sand plugs in all flotation bank junction and tail boxes. Each box has one plug controlled from monitor set point, and one that may be raised or lowered manually.

Copper recovery appears primarily dependent on flotation feed size distribution and the extent of liberation of finely disseminated sulphide mineralization. Target primary grind is 10-12% +100 mesh, 70% −200 mesh. Rougher and scavenger concentrates are reground to 90% −325 mesh, primarily to liberate chalcopyrite-pyrite middlings.

MOLYBDENUM PLANT

The molybdenum plant is situated in the main concentrator building and comprises separation, concentrate filtering, drying, and product packing sections.

The copper circuit recleaner concentrate feeds a 100-ft diameter Eimco thickener. Pulp is thickened to 65-70% solids prior to being pumped to two 26x26ft agitated storage tanks. These tanks provide storage capacity for approximately one day's production from the copper circuit. They smooth fluctuations in molybdenum flotation circuit feed, and provide aging time required for breakdown of residual reagents carried over from the copper circuit.

Feed is pumped from these tanks to a splitter box feeding a bank of four attrition cells which are used for pulp conditioning prior to rougher flotation. Rougher flotation of the molybdenite is performed in a bank of 12 48-cu.ft Agitair cells. Rougher concentrate is pumped to a bank of 16 48-cu.ft first cleaner cells. Concentrate from this stage is reground in a 4x10ft Allis-Chalmers regrind ball mill in closed circuit with a cluster of 4 6-in Wemco cyclones. Cyclone product is pumped to a second cleaner bank of eight 48-cu.ft Agitair cells. Concentrate from this flotation stage is cleaned in six more flotation stages using Denver #16-'Sub-A' cells, fourth cleaner concentrate being reground in a 3x4ft Denver ball mill which is in open circuit.

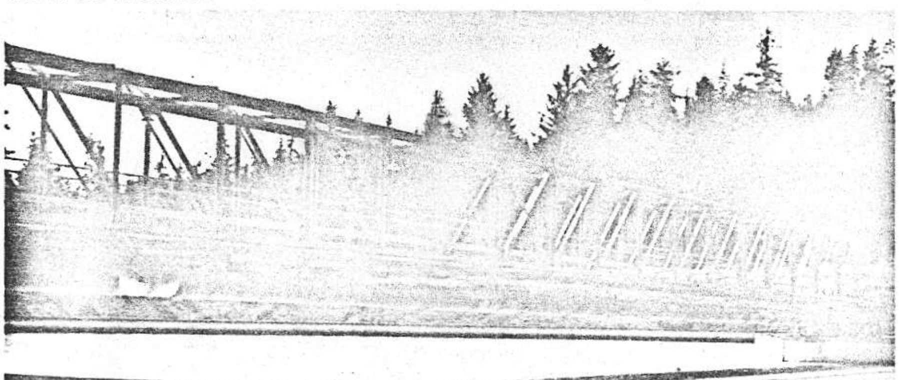
Tails from the eight cleaner stages are generally recycled to the previous stage, although the circuit is designed to be flexible in this respect. Final flotation circuit tails comprise rougher and first cleaner tailings which are pumped to a final copper concentrate thickener, prior to filtering and drying.

The circuit is unusual in one major respect — the ore milled at Island Copper contains a small quantity of gilsonite (a carbonaceous mineral) that has very similar flotation properties to the molybdenite. This mineral is a major contaminant in the molybdenite flotation product. Various schemes have been or are being tried, to remove this material from the molybdenum concentrate. A double cycloning stage between the first regrind and second cleaner flotation stages was installed at start-up, to reject the light mineral, but separation efficiency has proved unsatisfactory. Selective flotation of the eighth cleaner concentrate also failed.

Experiments with a Bartles Mosley table are now in progress, as preliminary experiments using a Wilfley shaking table indicated gravity separation to be practical. Several flow schemes employing the table are under investigation to determine the optimum with respect to metallurgical efficiency. Galigher vertical pumps are used for pulp transport in the flotation circuit.

Final molybdenite concentrate, from either eighth cleaner flotation or tabling, is thickened in a Denver 15ft diameter thickener. Underflow is pumped to two 10x10ft agitated holding tanks using a 1.5in ODS pump. From the holding tanks a second 1.5in ODS pump transfers the pump to a 4ft diameter 4-disc Denver filter. Overflow from the filter boot returns to the holding tanks. Filter cake drops down a chute to a double 7in screw Holoflite dryer, the dryer discharging into a storage silo. Product is packed into 45-gallon drums for sale on lot basis to customers in Europe and the USA. The typical product sold assays 42% Mo. Drums are shipped, four to a pallet, by barge to Vancouver for distribution to purchasers.

One of the thickeners



Molybdenum flotation metallurgy

Molybdenum flotation feed rate is approximately 660 t/d assaying 0.8% Mo. Circuit recovery primarily depends on the gilsonite (carbon) content of the ore which varies over a wide range (0.01-1.0% C in the molybdenum plant feed). Overall plant % Mo recovery varies from 10% to 70%, and averaged 30% in 1974.

Sodium hydrosulphide is the primary copper depressant. It is added to the attrition machine, first cleaner and second cleaner feeds, overall consumption averaging 14 lb/ton Mo plant feed. Sodium cyanide is used as a secondary depressant for chalcopyrite and pyrite. Consumption is 2 lb/ton. A froth modifier, Dearborn Exfoam 636, is used in the cleaning stages, consumption ranging 0.3-1 lb/ton. Fresh water is used for all dilution and spray water additions. Flowmeters are used for sodium hydrosulphide metering, and Clarkson feeders for the other reagents.

Flotation bank sand plugs are manually adjusted, instrumentation being limited to redox potential and mass-flow monitoring of circuit feed.

Cu CONCENTRATE HANDLING

The final concentrate, assaying 23-25% Cu, is thickened to 60-65% solids in a 100ft diameter thickener. Underflow may be pumped direct to filtering, or to three 26x26ft agitated holding tanks. Filter feed is split to two 10x8ft diameter 10-disc Eimco filters. Vacuum supply is by two 200-hp Bingham pumps, two 275-cu.ft/min Tieghman compressors providing the snap blow air supply.

Filtrate returns to the thickener, and 0.3 lb/ton of Cyanamid Aerodri 100 is used as a filter aid. Filter cake, containing approximately 12% moisture, drops to a 30in conveyor feeding the 48x7ft diameter Lockhead-Haggerty rotary kiln dryer. The dryer is fuel oil fired, final product moisture being less than 8%. A cyclone scrubber recovers dust losses for return to the thickener. An inclined 24in conveyor transports dryer discharge to a conical 35,000-ton live-capacity storage building. Product weighing is by Ramsey scale.

For concentrate shipping a deep sea dock was built, capable of receiving bulk carriers of up to 30,000 tons capacity. All production is currently shipped to Japanese smelters of the Mitsubishi and Mitsui Companies. Four Jeffrey apron feeders draw concentrate from the bin, discharging onto a 36in conveyor. A Howe-Richardson weightometer monitors loading rate over this belt (1000-1500 t/h). Concentrate transfers to a 36in shiploader conveyor. The conveyor is track-mounted for movement over the ship.

WATER SUPPLY

Process and mine site water requirement is approximately 10,000 gal/min of fresh water. This is obtained from Alice Lake, about 12 miles west of the concentrator. Four 3000-gal/min Worthington turbine pumps are used to transport water to a head tank one mile from the pump house. Flow is by gravity from this 100,000-gallon head tank to a tank at mine site. The water line is 34in diameter steel pipe. Tank levels are monitored in the concentrator control room and pumps may be started and stopped from this location. Their operation may also be automatically controlled from head tank level.

The mine site head tank capacity is 1.5-million gallons of which 300,000 gallons are reserved for fire protection. Lines from this tank feed the concentrator fresh water distribution system, the 500,000-gallon capacity reclaim water head tank, and an 18,000-gallon capacity potable water tank.

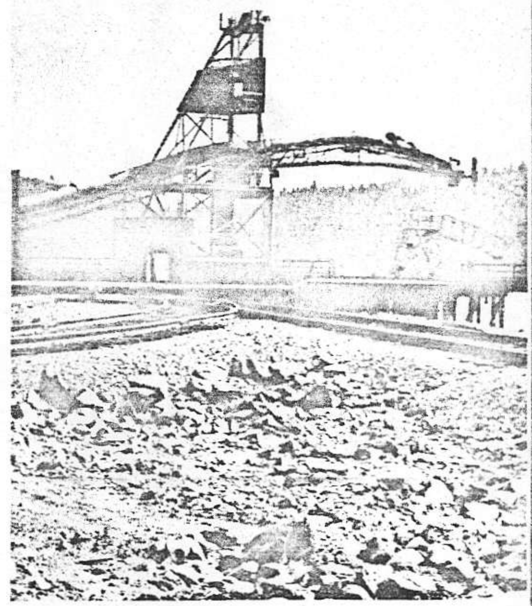
POWER SUPPLY

British Columbia Hydro and Power Authority provide power through a single 138kV transmission line from the Strathcona generating station, about 120 miles south of the mine site on Vancouver Island. Plant demand is approximately 50 megawatts. Two 18 MVAR synchronous condensers are used for voltage control and power factor regulation. Two 25/33 MVA transformers step down the voltage from 138kV to 13.2kV for mill motor power supply, pit, and fresh water pump house distribution centres. Pit, water pump, crusher, and copper regrind mill motors are 4.16kV. Smaller motors in the concentrator operate at 575V.

PERSONNEL

Total concentrator workforce, including technical personnel and staff supervision, is approximately 200 people. Maintenance and operations account for 175 of the total crew strength.

The organization is designed to provide for continuous operation on a seven days per week schedule.



Loading boom at the dock area

Island Copper Mine

tailings disposal and the environment

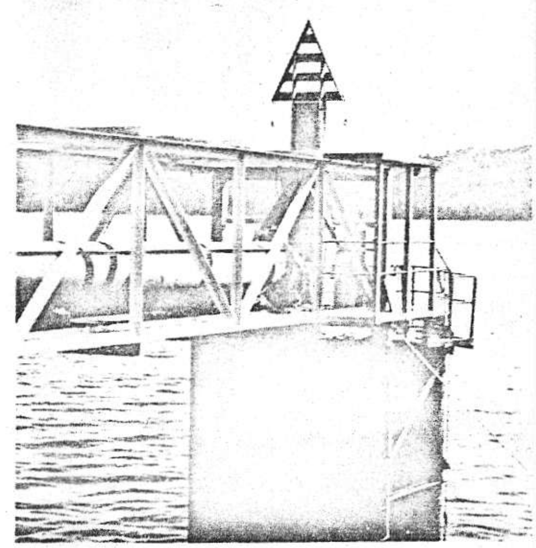
Copper circuit rougher and scavenger tailings discharge into a collection sump in the flotation basement. From here, tailings flow by gravity through a 34in steel pipe line to a splitter box feeding two 375ft Dorr-Oliver-Long thickeners. A stand-by line has been installed since start-up. Thickener feed is 30-34% solids, underflow varying from 40-45% solids; 0.02 lb/ton of flocculant and 0.6-0.8 lb/ton of lime are added to assist settling. Thickener overflow is reclaimed at a rate of 4000-5000 gal/min and pumped to a head tank above the concentrator elevation; four 3000-gal/min Worthington turbine pumps are available for this task. Thickener underflow density is controlled by hydraulic squeeze valves on the underflow lines. Underflows combine and flow by gravity through a pipe line to the marine disposal system. Two pipe lines are available, one of 34in diameter steel, and one of 34in Sclair high density polyethylene.

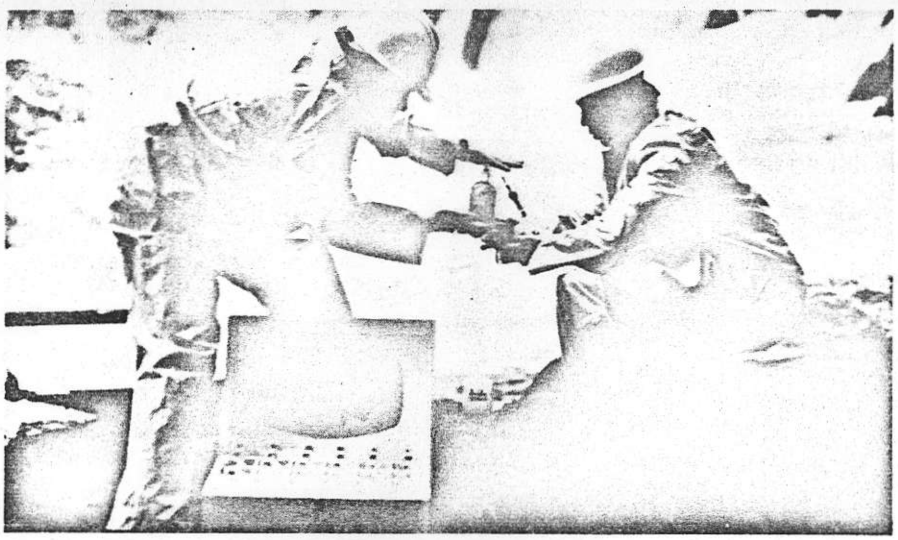
The pipe line discharges tailings into a mixing chamber in which they are diluted 1:1 with sea water, before entering the marine outfall line. The purpose of this mixing stage is to prevent density inversion of the tailings slurry liquid phase. The submerged outfall pipe rests on the inlet bottom which slopes at approximately 5° to the discharge point 1000ft from the shore and 150ft below the surface.

ENVIRONMENTAL MONITORING

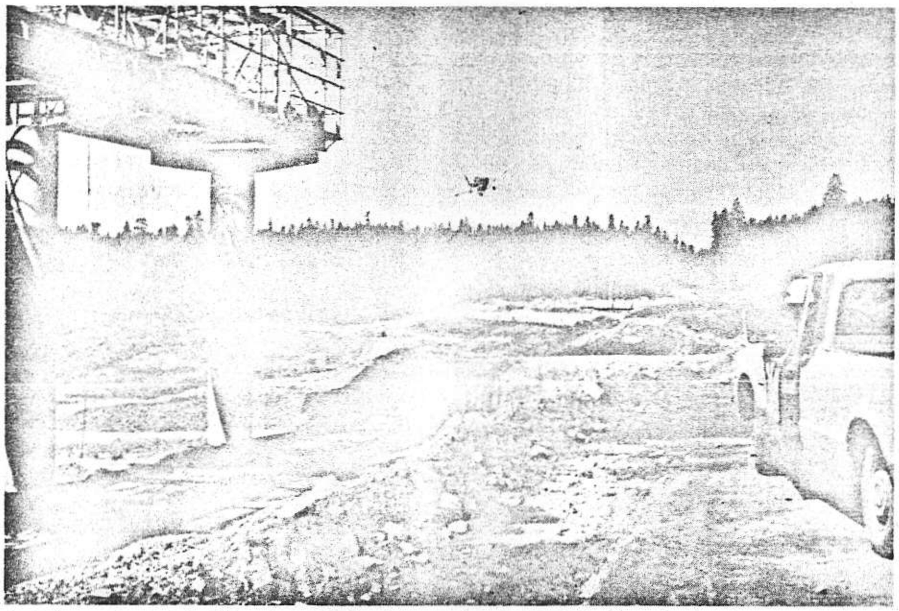
Utah Mines initiated an environmental water sampling program in 1969, before the plant was built. A biological consulting firm was retained to do a chemical and biological assessment of Rupert Inlet and adjacent waters. In January 1971 a permit was issued to Utah by the

Tailings pipe discharges into Rupert Inlet



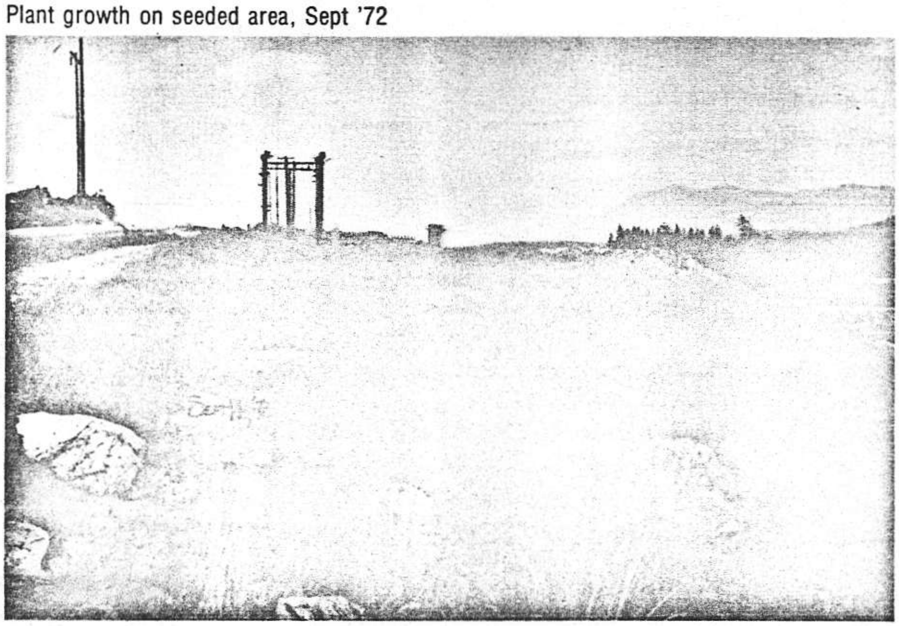


Water sampling by two of the environmental control team



Aerial spraying of fertilizer is part of reclamation work

Plant growth on seeded area, Sept '72



Provincial Pollution Control Branch for the submarine discharge of concentrator tailings following public hearings. The permit, in addition to identifying the physical and chemical limitations of the permitted discharge, directed Utah to retain an independent agency to assist in setting up the programme, establish procedures for sampling and analytical work, and prepare annual reports of programme results for submission to the Pollution Control Board.

Utah selected a team of scientists from the Universities of British Columbia and Victoria to carry out the functions of the independent agency. This team consists of oceanographers, marine biologists, ecologists, chemists, geologists, metallurgists, and mining engineers. The independent agency revised the existing monitoring program to include a wider scope in an attempt to cover all parameters which could be affected by the mill effluent.

The revised program was developed and monitoring started in March 1971. Information gathered since that time in conjunction with surveys which had been carried out by Utah before the Permit was issued, have provided the baseline information against which to compare present and future conditions of the receiving waters.

Monitoring of the inlet waters is carried out periodically, and includes measurement of the following major parameters:

(1) physical characteristics of the inlet bottom including seismic profiles, bottom photography, dredging and coring of the bottom for sediment analysis;

(2) physical characteristics of the receiving waters including temperature, turbidity, and colour;

(3) meteorological characteristics — state of surface water and weather;

(4) chemical characteristics of receiving waters including measurement of dissolved oxygen, salinity, alkalinity, and heavy metal content;

(5) biological characteristics including measurement of the organisms on the bottom in the deep part of the inlet, marine life on the bottom in the shallow intertidal areas, collection and measurement of plankton, and collection of crabs and fish for measurement of numbers, size, and heavy metal content.

Many of these same parameters are also periodically measured in a number of the fresh water streams flowing into Rupert Inlet. The effluent is monitored daily for heavy metals, pH, solids and total volumes. Bi-weekly bio-assays are conducted on the final effluent. Utah has chemists and technicians who carry out a major portion of the data collection and analytical work relating to the physical, chemical, and biological parameters measured. The various measurements are overseen by the scientists from the independent agency.