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Blasting in the Lornex pit

Lornex Mining Corporation Ltd.

By THE STAFF,
Lornex Mining Corporation Ltd.

INTRODUCTION

The Highland Valley of British Columbia has been the scene of intense mineral exploration and mining activity at various periods since the early part of the century. Early in this period, high-grade-copper vein deposits were mined on a small scale with the ore being hauled to Ashcroft in horse-drawn wagons.

In recent years, however, advances in mining technology and higher copper prices have permitted the large low-grade mines to come into being. The Lornex orebody is one of these, located on the southern slope of the Highland Valley, approximately 28 miles by road from Ashcroft and directly south of Bethlehem Copper Corporation's mine on the north side of the valley.

The Lornex orebody contains an estimated ore reserve of 293 million tons with an average grade of 0.427 per cent copper and 0.014 per cent molybdenum.

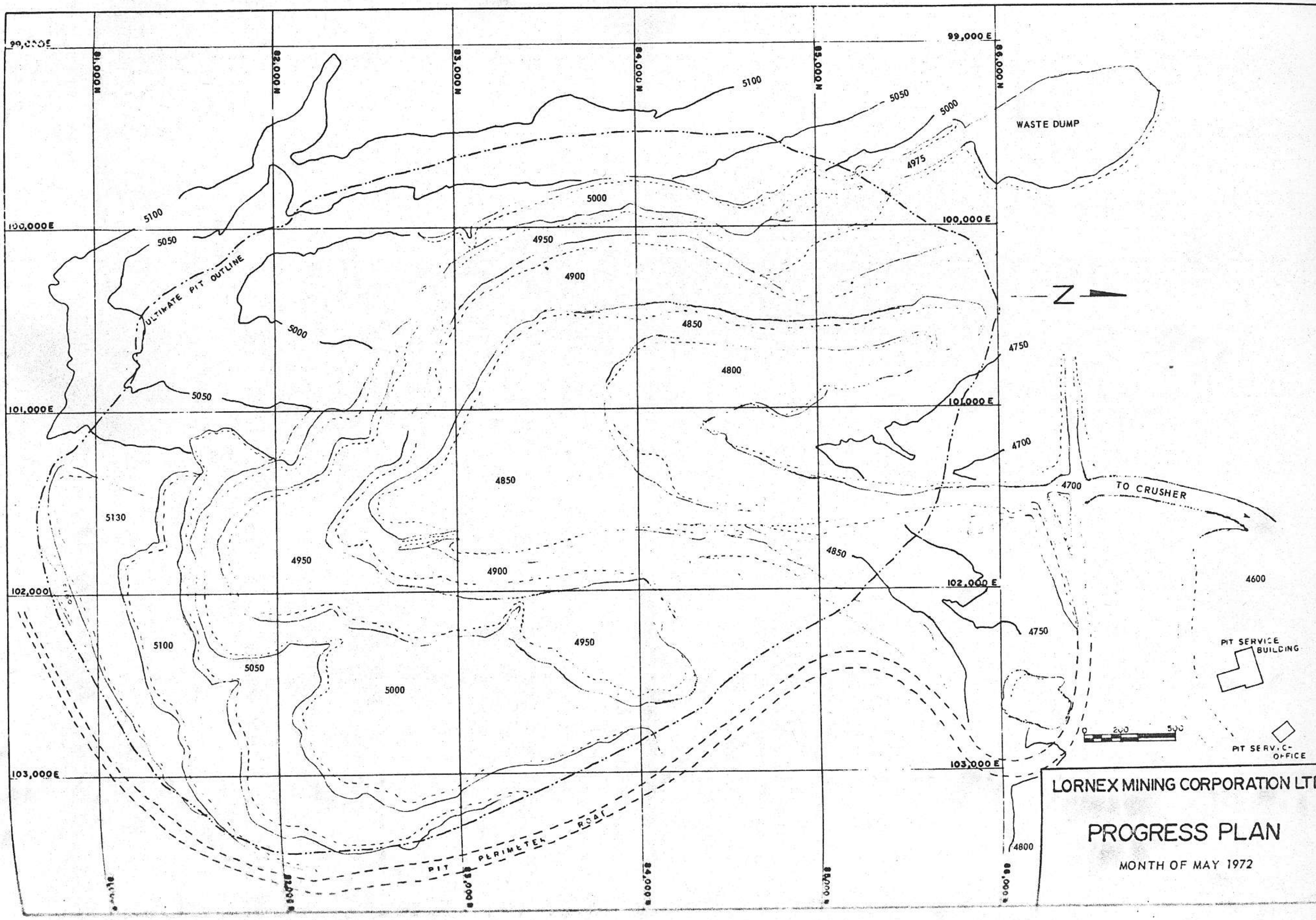
The plant is designed for an average capacity of 38,000 tons per day and will produce 162,000 tons of copper concentrate annually (or 110 million pounds of contained copper) and 2,300 tons of molybdenum concentrate (or 2.4 million pounds of contained molybdenum).

Mine-ore production commenced in the second quarter of 1972, about seven years after initial studies began. The mine has an estimated life of 21 years.

The development and construction of the Lornex Project were carried out under the management of Rio Algom Mines Limited, a member of the world-wide Rio Tinto-Zinc Corporation group of companies.

This work includes the following:

1. **Surface and waste-cap removal to develop the open pit approximately 5,000 feet by 3,000 feet. Its ultimate average depth will be 1,200 feet. Benches have been established every 50 vertical feet. The pre-production development includes the removal of approximately 50 million tons of overburden and oxide material, using four 15-cubic-yard electric shovels and twenty-two 120-ton capacity diesel haul trucks.**
2. **Construction of primary crusher and concentrator facilities, ore-conveying system, repair shops, offices, single men's quarters, water system, tailing-disposal system, nine miles of new highway skirting the tailings area, and the Village of Logan Lake.**


LORNEX MINING CORPORATION LTD.
PROGRESS PLAN
MONTH OF MAY 1972
WASTE DUMP
ULTIMATE PIT OUTLINE
PIT PERIMETER ROAD
TO CRUSHER
PIT SERVICE BUILDING
PIT SERVICE OFFICE
Z
99,000 E
100,000 E
101,000 E
102,000 E
103,000 E
81,000 N
82,000 N
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Lornex Mining Corporation Ltd.

BRIEF HISTORY

Lornex Mining Corporation Ltd. was formed by Egil H. Lorntzsen in the early 1960's to hold and, hopefully, to develop copper-mining claims that he had staked in the Highland Valley. It was clear from the beginning that the Lornex orebody was large, but of low grade and covered by considerable overburden and waste. It was evident that extensive investigation would be required to determine the viability of the property and that substantial technical, financing, marketing, and managerial resources would be needed to develop it, if it should warrant production.

Several major companies investigated the Lornex property and, for a variety of reasons, decided not to proceed with development. Rio Algom first became involved in 1965, when it undertook to carry out a comprehensive investigative program on the Lornex property. (If the project continued to offer promise, additional work would be carried out.) If Rio Algom completed the program and declared Lornex to be viable, it became committed to use its best efforts to arrange financing on the best available terms.

The investigative work on the property was carried out over the years 1965 to 1968 at a total cost of $6.8 million. It was financed by Lornex shares purchased for cash by Rio Algom to a total of $4.6 million. Under an earlier agreement with Rio Algom, The Yukon Consolidated Gold Corporation Limited purchased 40 per cent of the shares acquired by Rio Algom. A rights offering in the amount of $2 million was made to all Lornex shareholders in 1967 to provide additional funds for this program.

Because it was clear that the low grade Lornex orebody would have to be mined on a large scale if it were to be viable, the evaluation program was complex and exhaustive. It comprised:

- **a total of 86,017 feet of diamond drilling from surface, representing 87 diamond-drill holes, plus 511 shallow percussion-drill holes (91,000 ft.), and 19 rotary-drill holes (4200 ft.).**
- **an underground bulk-sampling program involving a 550-ft.-deep shaft, 2,618 feet of underground lateral development, and 5,439 feet of underground horizontal diamond drilling.**
- **the development of a small open pit to provide representative sample material for testing.**
- **the construction and operation of a 100-ton-per-day pilot mill to test the metallurgical characteristics of the sample material.**
- **extensive use of computers to establish alternative open-pit designs. Some 50 different designs were prepared ranging in capacity from 5,000 tons per day to 70,000 tons per day.**

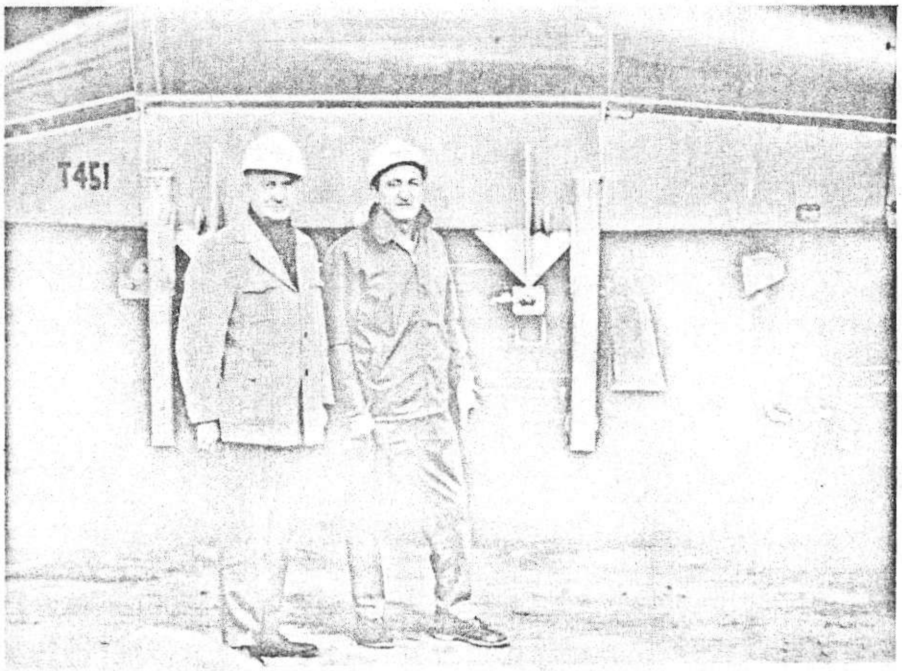
During this period, preliminary plant-design studies were carried out and generalized capital- and operating-cost estimates were prepared. Discussions were also held with potential buyers of the Lornex products and with financing institutions.

In addition to the evaluation of Lornex as a mine producing and selling concentrates, comprehensive supplementary evaluations were carried out by an independent consulting firm to determine the economic feasibility of processing the copper concentrate to blister or refined metal. The evaluation demonstrated conclusively that a smelting and refining plant was not a viable extension of the Lornex mine and that the mine itself could not subsidize the construction and operation of these supplementary facilities and remain viable.

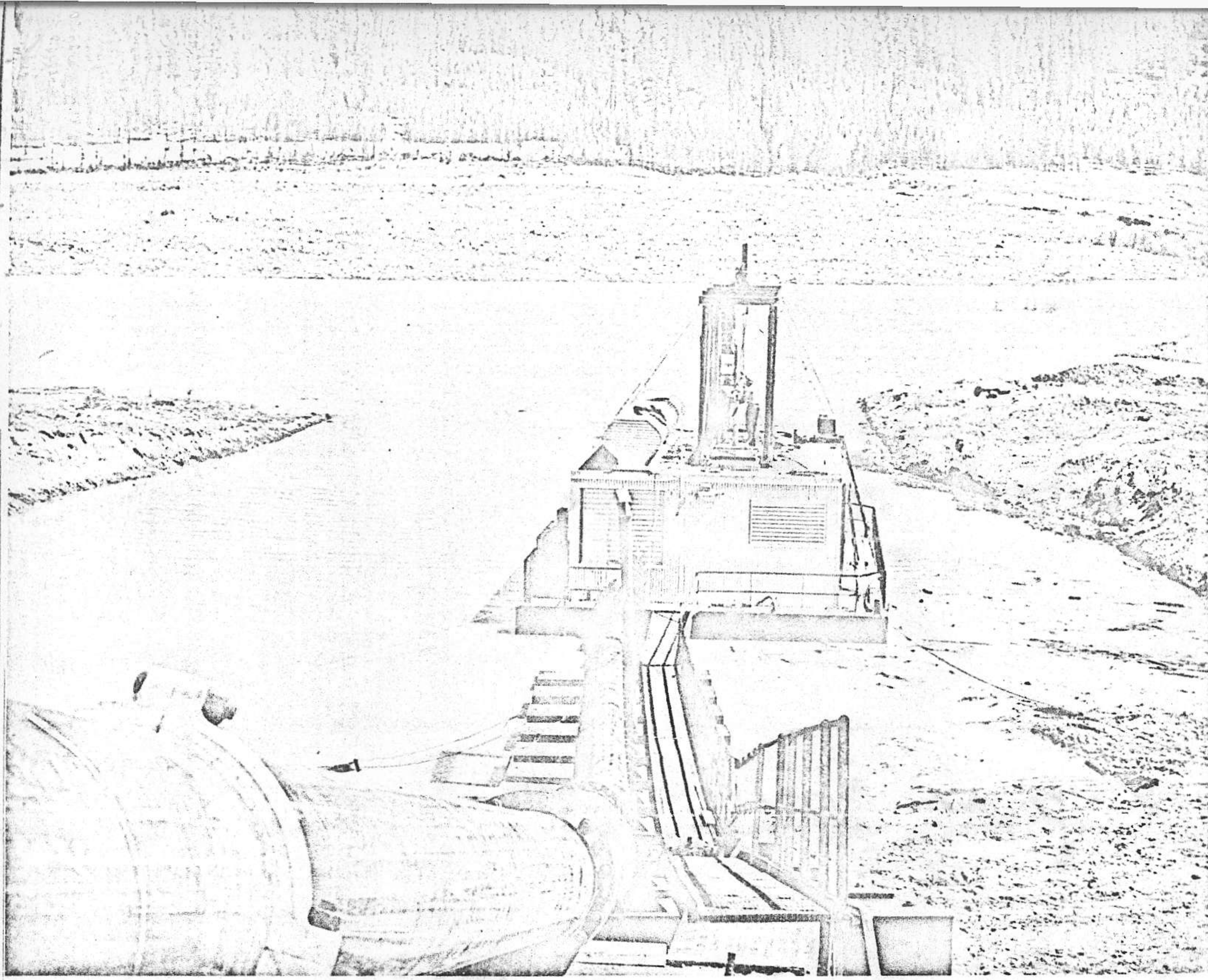
The British Columbia Government authorities were kept fully informed of the progress and results of the whole Lornex investigation program. This exhaustive evaluation process was completed late in 1968, and a formal decision to develop Lornex subject to concluding satisfactory financing arrangements were made in the spring of 1969. Negotiations were carried on during 1969 and final sales and financing documents were signed with a group of Canadian banks and Japanese smelters and trading companies on December 22 of that year. The Japanese consortium agreed to buy Lornex's total copper-concentrate output for a period of 12 years and to provide certain financing. Japanese Government approval, required with respect to the essential participation of the Japanese parties in the sales and financing arrangements, was expected within a few months.

The British Columbia Minerals Processing Act of March 1970 interrupted this process. The terms of the Act were clarified shortly thereafter and negotiation of the necessary amendments to the many agreements involved was undertaken.

After approval of the amended agreements by the Lornex shareholders, approval of the sales contract by the British Columbia Government, and validation of the Japanese participation, the project was formally released for construction on August 14, 1970.



Egil H. Lorntzsen (left), who staked the property and became the first president of Lornex Mining Corporation, and Norman F. Warren, vice-president and general manager of Lornex



Fresh water is drawn from the Thompson River by three vertical turbine pumps mounted on a concrete intake structure

Lornex Mining Corporation Ltd.

FINANCING

While awaiting the approval of the sales and financing agreements, a very considerable amount of engineering, site preparation, and other preparatory work had been carried out. This was financed by advances to Lornex totalling $10.2 million, of which 90 per cent was provided by Rio Algom. In addition, commitments amounting to approximately $10 million had been incurred for long lead-time equipment required for the pit and mill operation. Thus the expenditure and commitment exposure up to the construction release date was just under $30 million.

The final capital cost is now estimated at $138 million. Of this amount, $91.6 million is represented by loans from a consortium of Canadian banks, from a group of Japanese trading companies, and NHA housing loans. The sales of income-debenture units have provided $39.0 million.

Logan Lake, a planned community built by Lornex and incorporated as a village, is seen at left. It has a population exceeding 500, some 140 homes, townhouses, mobile homes, paved roads, a school, bank, service station, fire department, and an RCMP detachment.

The financing of Lornex by this means was made possible by:

- **the exhaustive evaluation that was made of the project,**
- **the successful negotiation of a sales contract for the mine's entire production of copper concentrate for a period of 12 years,**
- **a construction and management agreement whereby Rio Algom assumed substantial construction, operating, and financial responsibilities,**
- **a commitment by Rio Algom to maintain its equity position in Lornex at not less than 50 per cent. Its current position is just over 50 per cent.**

EXPLORATION

The copper mineralization was discovered on the Lornex property in 1964 when Egil Lorntzsen investigated it by bulldozing a series of long trenches. These indicated that the mineralization had an appreciable lateral extent. A truck-mounted overburden drill was then used to drill vertical holes as a further check on the mineralization.

In 1965 Rio Algom, under its agreement with Lornex, began its investigative program. A variety of techniques was employed to define the Lornex orebody. Extensive bulldozer trenching was done in the southern area in the vicinity of the Discovery zone. Trenches were excavated to bed rock where possible in an east-west direction for lengths of 200 to 700 feet and on lines spaced 200 to 400 feet apart. The bedrock exposed throughout most of the trenches provided only limited geological information but did help to define the eastern limit of the mineralization.

During the summer of 1965, a reconnaisance geochemical-sampling program was undertaken. Samples were taken with a hand auger at depths of from 3 inches to 3 feet and were confined to the main stream courses and numerous-discontinuous-drainage gullies. No attempt was made to achieve saturation geochemical coverage on a grid pattern, because the rapid changes in overburden depth and soil types visible in the trenches indicated that grid-sampling might be misleading. Although there were some geochemical anomalies, this approach on the Lornex property was not sufficieintly definitive to provide targets for trenching or drilling. It was sufficiently indicative, however, to warrant more detailed work.

Concurrently with the geochemical survey, an induced-polarization survey was conducted over most of the Lornex property following some additional trenching and percussion drilling of the mineralized zones. This survey covered the whole property from north to south on east-west lines spaced at 800-foot intervals on the northern portion of the property and 1600-foot intervals on the southern portion of the property. Induced-polarization readings were taken with electrode spacing of 200, 300, 400, and 800 feet over the areas of known mineralization wherever anomalies had been previously indicated. The remainder of the property was examined with electrode spacings of 400 feet.

A magnetometer survey was run concurrently with that of the induced-polarization survey and over the same grid, but no direct correlation with the induced-polarization results was evident.

The induced-polarization survey indicated the three main zones: the Discovery zone where the bulldozing had indicated copper mineralization; the North and Camp zones which were discovered as a direct result of the induced-polarization survey.

Diamond drilling was also started in 1965 to the southeast of the main zone of mineralization. The program consisted of 21 holes on four lines 800 feet apart with holes spaced at 300- to 400-foot intervals. The first seven holes were drilled with BX equipment but core recovery was disappointing. Therefore, a switch was made to NQ wire-line equipment which produced a core 1-⅞-in. diameter employing mud as a lubricant. Sufficient encouragement was obtained to continue drilling on intermediate and additional lines until a total of 86,017 feet had been driven in 87 holes.

Percussion drilling was used over a wide area, including the Discovery and North zones on lines 200 feet apart and at 100-foot intervals on the lines. The capacity of the drill was only 300 feet, so little information was obtained in deep-overburden sections. It was found on comparing percussion-hole assays against later assays from diamond-drill core that percussion drilling tended to produce **"down the hole salting"**. It did, however, define the extent of the mineralization and provided information on the bedrock surface and the thickness of the oxide zone.

Rotary drilling was tried in the hope that it might reduce the amount of the more expensive diamond-core drilling. However, when the drill penetrated the water table, difficulties were encountered and this method had to be abandoned. The second attempt was with a Becker rotary drill, which uses double annular rods with air and water passing down between the rods and cuttings returning through the centre of the inner rod. This machine was successfully used to a depth of about 500 feet, and some 19 shallow holes were drilled for information on bedrock surface and thickness of the oxide zone in the heavily overburdened areas.

BULK SAMPLING and PILOT-PLANT PROGRAM

After the surface drilling had indicated a deposit containing several hundred million tons of possible ore-grade material, a decision was made to carry out an underground-bulk-sampling and pilot-plant program. This investigation was considered necessary to confirm the diamond-drilling results, since the grade indicated by surface drilling was suspect due to core recovery lower than 100 per cent. It was also thought that friable sulphide-minerals might have been lost in the sludge and not recovered through the use of mud as a drilling lubricant.

The following list of objectives was drawn up as a guide to operation of the bulk sampling plant and pilot mill:

- **to check the grade of the deposit as indicated by boreholes.**
- **to determine the best method of reducing the ore to flotation size.**
- **to confirm metal recovery and concentrate grades as indicated by bench-scale tests in the laboratory.**
- **to prove out a method for copper-molybdenum separation suitable for the Lornex ore.**
- **to develop design criteria for commercial scale process facilities.**

The bulk-sampling program was launched in January 1967. It involved the development of the small underground operation and the construction of the pilot plant capable of treating 100 tons of feed per day, plus related facilities.

The three-compartment shaft measured 18 feet by 9 feet and was completed to a depth of 550 feet. A 7-foot by 7-foot crosscut was driven 248 feet to the east and 1,426 feet to the west from the shaft section on line eleven.

The underground program was originally planned to consist of a crosscut across the mineralized zone, plus raising on surface diamond-drill holes to check core grade against bulk samples. Due to the extremely heavy ground, however, raising was not practical so 970 feet of north-south drifts with horizontal drill-holes east and west were substituted. Drill stations were established at roughly 200 and 400 feet north and south of the crosscut, and a total of seven horizontal holes was drilled east and west. Prior to driving a crosscut west, diamond-drill holes were drilled ahead of the face to permit a comparison between drill core and bulk sample. Underground diamond-drilling totalled 5300 feet. Each bench on the shaft and each round in the crosscut were processed separately through a small crushing- and sampling-plant prior to delivery to the pilot plant.

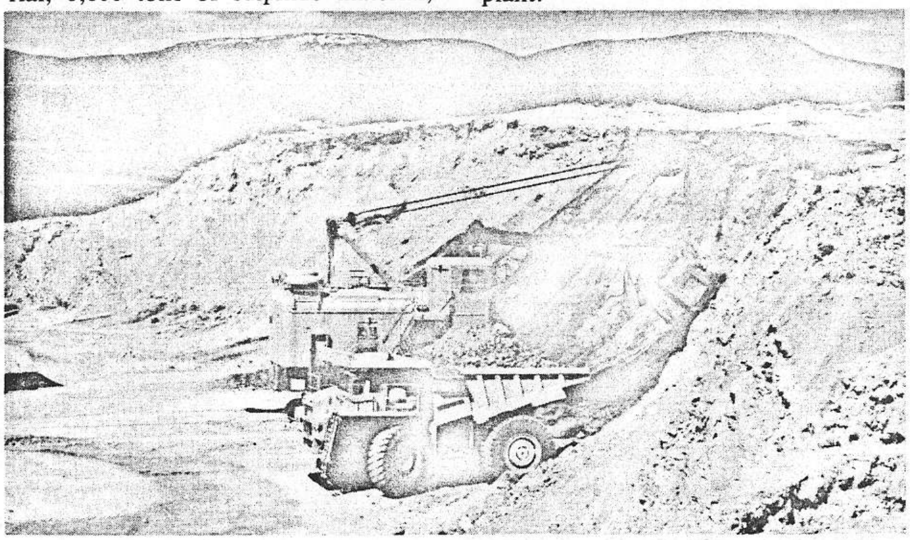
A comparison of the assays from underground drilling, bulk-sampling round by round, and grades projected from surface diamond-drilling indicated that the surface drilling had not overvalued the copper content of the area tested. Copper grades by underground work in the test area were about 9 per cent above that indicated by surface drilling.

At the same time a small test pit was being developed to provide coarse materials for autogenous-mill testing and oxide material for metallurgical test work.

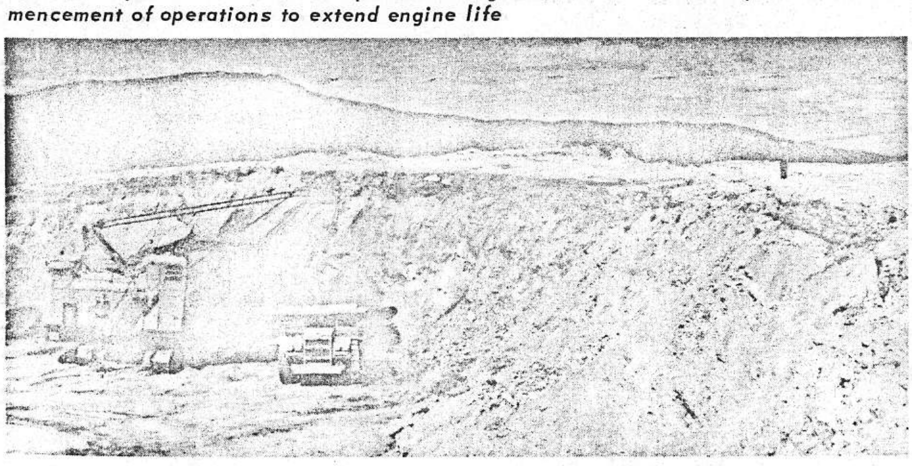
This pit was located immediately northwest of the shaft where the overburden was shallow and where rock reasonably representative of the mineral deposit existed. It was excavated to a depth of about 60 feet to obtain material below the oxide zone. A total of 25,318 cubic yards of glacial till and 52,647 cubic yards of rock was excavated. Some 6,000 tons of oxide material, 6,800 tons of sulphide material, 900 tons containing considerable amounts of fault gouge, and 900 tons of hard, low-grade sulphide-bearing rock were delivered to the pilot plant from the test pit. The balance of the muck was stockpiled near the pit for delivery to the mill if and when required.

Construction of the pilot plant began in early 1967 and the first ore was milled in June. The plant operated about nine months and processed about 20,000 tons of rock. Grinding testwork was carried out in a 7-½-ft. by 2-½-ft. Aerofall mill modified for wet service and a 6-ft. by 4-½- ft. overflow ball mill. In addition to the grinding mills, the pilot plant contained the normal complement of equipment including cyclones, flotation machines, filters, thickeners, and auxiliary equipment.

Pilot-plant studies showed that either single-stage ball milling or autogenous-primary milling followed by ball milling could be used for size reduction of flotation feed. Autogenous-primary milling was selected in order to avoid anticipated problems with fine crushing which might result from the relatively-high content of fault gouge in the Lornex ore. This type of circuit also offered substantial capital-cost savings in the commercial-scale plant.



Each shovel averages 13,000 tons per operating shift. The 22 Wabco 120-ton trucks are powered with 1000 h.p. diesel engines, derated to 860 h.p. at the commencement of operations to extend engine life



GEOLOGY

The Lornex deposit is classified as a complex-porphyry-type within the Guichon granodiorite batholith which forms a part of the interior plateau of British Columbia. The batholith is elongated in a northwesterly direction and has a length of 40 miles by a width of 16 miles. It consists of a series of several major magmatic intrusives, in general having a concentric zonal arrangement and becoming younger inwards. The complex intrusive, is of low Jurassic age approximately 198 million, ± 8 million years.

The Lornex orebody occurs in the Skeena quartz diorite or Bethlehem host rock, an intermediate intrusive phase of the batholith at the contact of the Bethsaida grandodiorite, the youngest instrusive phase. The host rock is a medium- to coarse-grained equigranular rock distinguished by the interstitial quartz and presence of moderate ferromagnesian minerals, mostly hornblende and biotite, averaging 15 per cent of the total mineral content. The Skeena quartz has been intruded by several younger dykes including a quartz diorite porphyry, Bethsaida-quartz porphyry, and small basic intermediate dykes.

A major north-south fault, steeply dipping to the west cuts off the orebody to the west.

Several other parallel faults dip to the east at 60 degrees from the horizontal. The Lornex mineralized area is divided into three zones:

- **the original Discovery zone**
- **the Camp Zone**
- **the North Zone, containing the main orebody**

The Lornex deposit is contained in a roughly-elliptical area 4,000 feet in length, 1600 feet in width and at least 2,000 feet in depth and still open. Its long axis trends northwesterly.

The principal minerals in the sulphide zone are chalcopyrite, bornite, and molybdenite with minor pyrite, magnetite, hematite, rhenium, osmium, gypsum, epidote, calcite, and chlorite. An oxide zone up to 200 feet thick caps the orebody. The major minerals in this zone are malachite with minor tenorite, chalcocite, covellite, azurite siderite, cuprite, and native copper. The mineralization occurs mainly as fracture fillings either in quartz-carbonate veins up to a foot in width or along joints, slips, and minute fractures, and as sparsely-disseminated mineralization generally replacing hornblende and biotite.

Fracture density and strong alteration appear to be keys to the higher-grade copper-molybdenum values. The altered minerals include: sericite, chlorite, clays, and epidote.

Lornex Mining Corporation Ltd.

MINE PLANNING

The open-pit planning started in 1966 before the diamond drilling program was completed. Several preliminary-test designs were examined ranging from 5,000 to 70,000 tons of ore per day.

The first trials were done without the aid of computers, but provided good opportunity to become familiar with the shape of the orebody and the relationships and locations of the various ore grades. This preliminary work also formed a good basis when computer applications were later utilized for pit design.

In order to calculate ore reserves and to assist in pit planning, an interpretation of the orebody was developed based on grade trends. The grade trends were established on the belief that the distribution of the copper and molybdenum in lower and higher grade zones was governed by a pattern of structural features, such as fractures, faults, shear zones, etc., of a more or less continuous nature. The results of this interpretation were considered to present a pattern of sulphide occurrence consistent with geological understanding of the zone, and were useful in the planning of the pit and in the estimation of ore reserves.

The grade trends were first developed on vertical cross sections along each of the drill lines. This interpretation was then transposed to a series of level-by-level zone plans at vertical intervals of 80 feet. These grade-zone plans were subsequently blocked out in 80-foot cubes to which copper grades were assigned. These blocks formed the matrix for the computer input.

Two principal computer programs were used in the evaluation, one for general trial pits under the controls imposed by slope-stability considerations and the second to perform financial analysis. The first program developed co-ordinates for the final pit-perimeter, calculated individual tonnage for grade intervals and rock types at the various levels within the pit, and provided overall totals and stripping ratios. A modification of this program permitted consideration of working slopes within the final pit. In the second program all capital and operating costs were considered and alternative loan structures were analyzed.

The first stage of the pit design relied on preliminary cost data developed in the previously completed pit-design studies. Initial calculations suggested that the copper cut-off grade should lie between 0.2 per cent to 0.3 per cent. For purposes of comparative analyses it was decided that 3 cut-off grades, 0.20, 0.26, and 0.30 per cent copper, would be applied to each of the pit perimeters under consideration. The second stage utilized the computer program to study the effects of varying the pit position with respect to the orebody. Fifteen pit positions were analysed in terms of tonnage, overall grade, and overall stripping-ratio.

Following this, a subsidiary study measuring the effects of alternative final pit slopes (between 40 and 45 degrees) on the stripping ratio, was conducted. The results were then compared with those slopes recommended by the rock-mechanics consultant and it was discovered that the overall stripping-ratio obtained by the latter was slightly more conservative than that achieved if 40-degree slopes were used throughout.

The rock-mechanics consultant was engaged while the bulk-sampling program was in progress. He examined the diamond-drill core and the underground workings from a rock structure standpoint, and conducted numerous laboratory tests on shear strength, compressive strength, etc., of the various rock types. Based on these tests and the examination of main structural features in the area, practical pit slopes were recommended which varied from 45 degrees in the walls of the pit. These slopes varied from 45 degrees in the more competent areas and where pit geometry was favourable down to 20 degrees along the major Lornex faults, which strikes generally north-south through the western flank of the open pit.

In the third stage of pit design, incremental pit-expansions were analyzed with respect to overall grades, tonnages, and stripping ratios. This evaluation used an operating break-even approach and led to the conclusion that expansions were warranted. The expanded pit was used as a base for more refined costing which took into account projected changes in the stripping ratio over the life of the pit. The new cost data were chosen to check the viability of the expanded pit. A full-scale financial analysis of the pit at the three cut-off grades was carried out and showed that the 0.26 per cent cut-off point was the most favourable of the three, even though the differences in the rates of return were minimal.

The final stage involved checking an 80-foot-wide shell of material just within the concentric of the final pit-perimeter, and was examined on all levels. The incremental shell was classified into ore, waste, oxide, stockpile, and overburden. This test confirmed that the pit was at, or close to, the limiting size acceptable for mining at the range of metal prices being used for the feasibility calculations.

The proposed Lornex open pit contains the following materials:

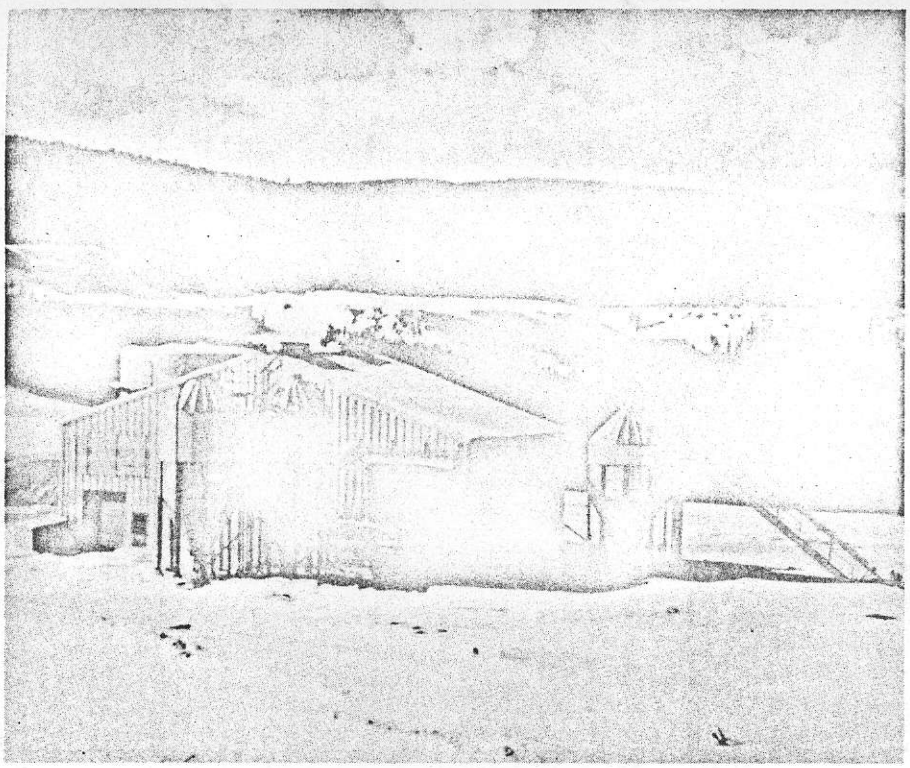
— **ore:** 292,804,000 tons; average grade 0.427% copper (at cut-off grade of 0.26% copper); 0.014% molybdenum
— **oxide:** 35,345,000 tons
— **waste rock:** 217,619,000 tons
— **glacial till and overburden:** 89,738,000 tons

The waste-ore ratio exclusive of the glacial till is 0.862 to 1 and including the glacial till, 1.17 to 1. The same general pit configuration developed in the feasibility study, with relatively minor modifications, is being used for the actual mining today. Mine planning is continuing on a more detailed basis within the framework of the original ultimate pit.

Present pit planning involves the development of detailed plans for the first three years by quarters, and annual pit plans for the next two years, with a broad plan for the ten-year mining program. These plans will be reworked each year and modified to reflect new information obtained as the mine progresses.

The original "ultimate" pit developed in the feasibility study used a bench height of 40 feet, and a berm every 80 feet vertically on the final pit wall. All permanent ramps were laid out 70 ft. wide at a maximum grade of 8%. These features were predicated on the selection of 9-cubic-yard shovels loading 75-ton trucks. At the time of this study, this size of equipment was well proven in the field and we were confident that reasonable mining costs could be attained.

During the interval between the conclusion of the feasibility study and project release, further examination showed that there were advantages to be gained both from an operational and from a mining-cost standpoint in using larger equipment. Fifteen-cubic-yard shovels and 120-ton capacity haulage trucks were subsequently selected. The increase in equipment size permitted extending the bench height from 40 feet to 50 feet. Road widths were increased to 80 feet. Other than these changes, no other significant modifications have been made to the ultimate pit design.



Explosives magazine at Lornex

Drilling and Blasting

All primary drilling is handled with 3 electric rotary drills, using a 9⅞-in. tri-cone standard-steel bit. Bit life to date is better than 4,000 feet, but since a good part of the drilling has been in over-burden it is too early to predict whether this bit life will be maintained. To date the drills have been averaging about 400 feet of drilling per shift, with an average penetration rate of approximately 70 feet per hour. As most of the holes have been wet, water is introduced into the air stream for dust-control purposes only as required.

Generally, single-row blasting is used although double-row blasting is used occasionally with holes being spaced on a 37-ft. square pattern. The single-row blasting is preferred to take advantage of any back break which occurs. In rock, holes are drilled to a depth of 60 feet including 10 feet of sub-grade. In overburden, holes are drilled 50 feet with no sub-grade, in order to preserve a good bottom for the shovel and truck operations.

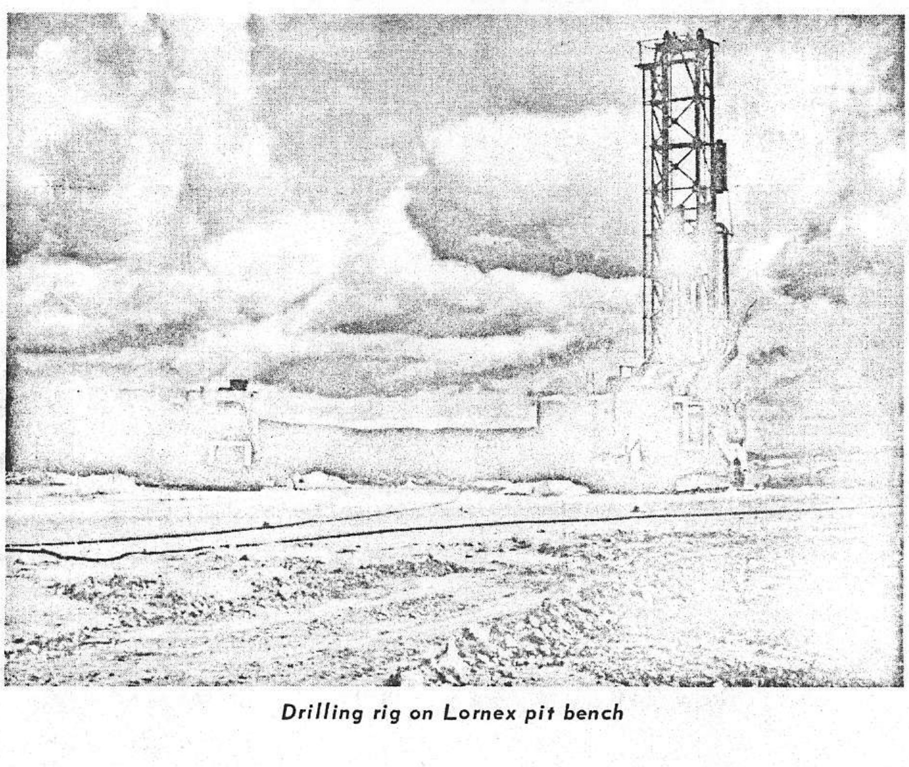
Since the blast-holes are generally wet, a non-metallized slurry having a velocity of 12,700 feet per second is used almost exclusively. The explosive-supply company delivers between 1100 and 1300 pounds of slurry in a standard-slurry-mix truck to each blast-hole.

Holes are double-primed with two pentalite boosters. Holes are loaded as soon as possible after drilling to avoid loss of holes due to caving in wet ground. Reinforced-primacord down-lines are used and connected by unreinforced primacord, with milli-second delays between rows when necessary. To date a powder factor of 0.15 to 0.25 pound per ton has been utilized, and, although this powder factor is relatively low, excellent fragmentation has resulted due mainly to the densely-fractured condition of the rock. Under winter conditions, decked charges in the blastholes and small-diameter satellite holes have been tried to break up the frost layer. However, more work is required in this area before a satisfactory procedure can be developed.

Loading and Hauling

The open-pit excavation is carried out by a fleet of four 15-cubic-yard electric shovels. The power is supplied to the shovels by trailing cables carrying 4,160 volts from breaker boxes adjacent to the pit-perimeter powerlines, which form a complete loop around the edge of the ultimate pit. The average production rate at present is 13,000 tons per operating-shovel shift. The fleet of twenty-two 120-ton-capacity trucks hauls the waste materials to dumps and the ore to a stockpile. These trucks are powered with 1,000 h.p. diesel engines. At the present time, however, since the majority of hauls are either level or down-hill, these engines have been de-rated to 860 h.p. in order to extend engine life. The truck engine drives a generator which supplies power to electric wheels. The trucks are tired with six 30.00 by 51-46 and SL ply rating tires.

Although most of the truck drivers hired have had previous experience in driving large haulage trucks, each driver undergoes an intensive driver-training program during the early days of his employment. An instructor drives with the new employee until he is fully satisfied that the trainee is completely capable of handling the truck in a safe and efficient manner. To maintain standards, regular check examinations are given to all drivers.



Drilling rig on Lornex pit bench

Auxiliary Open-Pit Equipment

Waste dumps and stock piles are maintained by four large track-laying bulldozers, three equipped with rippers and one equipped with a winch. The clean-up around the shovels and spillage on the roads is handled by three rubber-tired dozers. Pit roads and main ramps are maintained by four motor graders. Miscellaneous pioneering work and odd job excavation are handled by two front-end loaders with 5½-cubic-yard rock buckets. These loaders generally work with two 35-ton haulage trucks. Two other 35-ton trucks are equipped with 6,000-gallon water tanks with special spray nozzles for dust control on main-haul roads.

During winter time, one of these water trucks is converted to a sand spreader to improve traction on icy roads. Secondary drilling and miscellaneous drilling is handled with a track-mounted mobile drill. This drill is equipped with a 600 CFM compressor and a 4½-inch drill mounted on an articulated boom, and all is mounted on a war surplus M-8 tank carrier. This unit has proven very useful, mainly because of good mobility from job to job around the property.

Two-way radio communication is used to advantage in the mining operation. Radios have been installed in all the supervisor's vehicles, the shovels, the rotary drills, the rubber-tired dozers, and the front-end loaders.

One of the specialized pieces of equipment in use is a tire manipulator mounted on a 20,000-pound-capacity fork-lift truck. This truck is equipped with high flotation tires to enable it to go into the pit to change a front tire if required.

Mine Equipment Maintenance

The system of maintenance for the mine equipment is influenced by the geographical location of the Lornex mine itself. The area is well serviced by equipment suppliers and provides ready access to commercial repair and rebuild facilities. This will permit Lornex to operate efficiently with a relatively low inventory of spare parts and supplies. Public transportation systems, such as trucking, rail, bus, and airlines, service the area on a daily basis, so that rarely is an emergency replacement more than 24-hours away, and in most cases as little as six hours.

The main objective of the maintenance program is to keep the equipment available for operation as much of the time as possible. This is accomplished through a system of component exchange. When any of the equipment has a component failure, the component is replaced immediately rather than trying to repair it in place on the piece of equipment. Minor repairs to individual components are handled on site, but a component requiring complete overhaul is shipped out to the suppliers for this work and exchanged for a new or re-conditioned one which carries a warranty. Wherever possible, advantage is taken of equipment suppliers and re-building facility warranties. The preventive-maintenance program is controlled by the planning and scheduling department which has visual displays showing the condition and maintenance status of each piece of equipment. A large tire-status board is also maintained. This shows at a glance the history, condition, the piece of equipment on which the tires are being used, and its location on the piece of equipment.

Statistical records are maintained on such items as tires, shovel teeth, hoist cables, cutting edges, etc. The performance of each brand of the high-wear parts is compared on the unit cost basis, and provides justification and back-up for purchasing the proper product regardless of initial cost.

Although Lornex is able to attract and retain experienced and skilled workmen it still recognizes the advantages of sending key people, both hourly and staff, to manufacturers' training schools. The type and size of mining equipment used by Lornex is not common, so it is necessary to up-grade the training of new employees. As much emphasis is placed on the electrical trades as on the mechanical trades, because the mining equipment for the most part is powered electrically.

Preventive maintenance on haulage trucks is scheduled every 200 hours and two trucks are handled each day. Drills are scheduled every 200 hours and shovels every 100 hours. One shovel is scheduled for an 8-hour service each day. The maintenance department in co-operation with operations, schedules regular preventive maintenance on each piece of major mining equipment.

All maintenance is carried out on a 5-day week scheduled between Monday and Friday with the exception of major shovel repairs or adjustments and preventive maintenance on the electric-wheel armatures, suspension connections on the trucks, and routine tire inspection. The major repair work on shovels is generally planned to commence on a Friday with the aim of having the equipment back in operation on the following Monday.

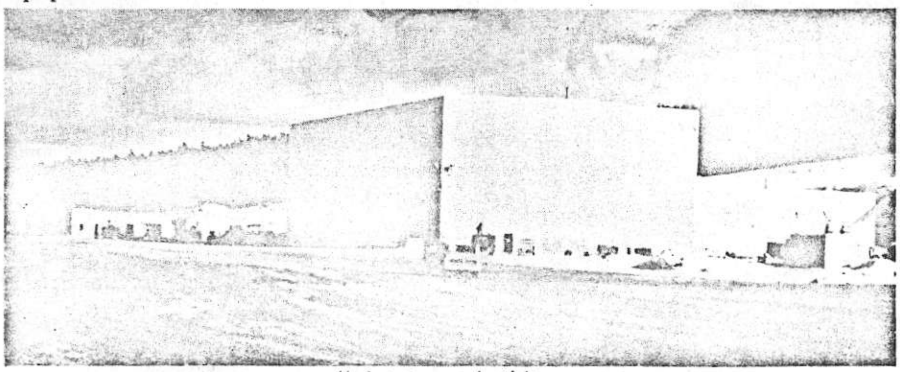
Grade Control

The two main considerations in controlling the grade of ore delivered to the primary crusher will be copper content and hardness of the rock. The content of molybdenum will be a secondary consideration and the grade will be allowed to fluctuate, with the grade of copper being held relatively constant and ranging between 0.38 per cent copper and 0.46 per cent copper.

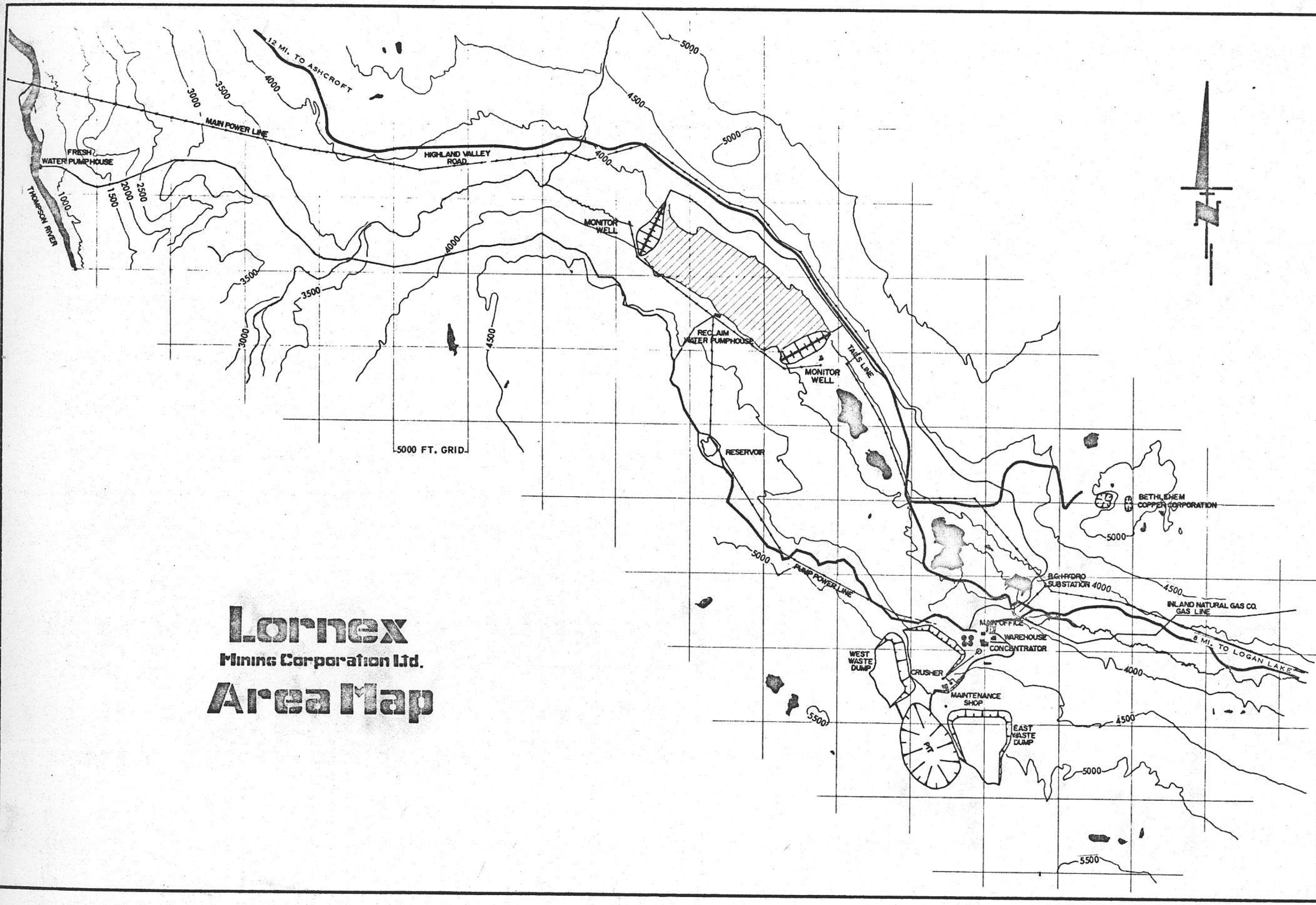
A suitable blend of hard and soft ores will enable the mill staff to maintain a reasonably consistent operation in the autogenous mills and control of the grinding circuits.

The location of various grades of ore in the pit is determined by using the assay information from rotary-blasthole sampling, face sampling, and previous diamond drilling combined with detailed mapping and final correlation of all pertinent information.

The pit-ore faces are divided into numbered stations, generally 30 feet to 50 feet wide along the mining face, depending on how rapidly grade changes occur. A copper grade is assigned to each station by the Geological staff. This information is recorded daily in a grade-control log book in the operations office and is accessible to all pit foremen. The foremen record daily all the station grades in a pocket-sized book which they keep with them in the pit. By relating the grades to the stations marked along the face, the foreman knows where he must mine to obtain the required grade of ore. A minimum of two shovels and at times three will always be in ore. Therefore, with a little experience the foreman can organize the mining equipment to provide the required blend of ore from the various mining faces to the crusher.



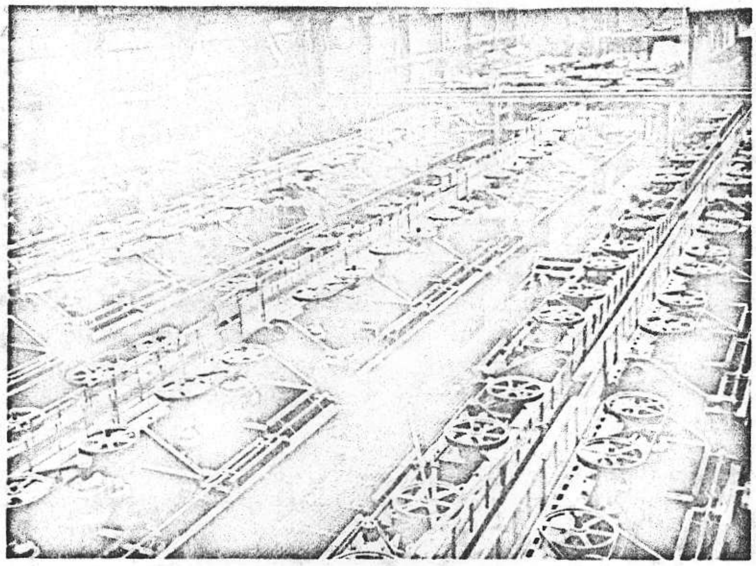
Maintenance building



Lornex
Mining Corporation Ltd.
Area Map
12 MI. TO ASHCROFT
MAIN POWER LINE
HIGHLAND VALLEY ROAD
FRESH WATER PUMPHOUSE
THOMPSON RIVER
MONITOR WELL
RECLAIM WATER PUMPHOUSE
MONITOR WELL
TAILS LINE
RESERVOIR
5000 FT. GRID
BETHLEHEM COPPER CORPORATION
B.C. HYDRO SUBSTATION
PUMP POWER LINE
INLAND NATURAL GAS CO. GAS LINE
8 MI. TO LOGAN LAKE
MAIN OFFICE
WAREHOUSE
CONCENTRATOR
WEST WASTE DUMP
CRUSHER
MAINTENANCE SHOP
PIT
EAST WASTE DUMP
1000
1500
2000
2500
3000
3500
4000
4500
5000
5500

MILLING

By THE STAFF,
Lornex Mining Corporation Ltd.



Denver cells in the flotation section

Crushing

A 60-in by 89-in hydroset gyratory crusher is used to reduce mine ore to minus 6 - 8 inch in size. The crusher arrangement provides two dumping points but only one truck is dumped at a time. Truck discharge falls onto a splitter to reduce wear on the crusher spider. There is no grizzly in the crusher circuit.

The crushed ore drops into a 300-ton-capacity surge pocket below the crusher. The surge pocket is equipped with a sonic level detector to prevent dumping into a full pocket. A 96-inch apron feeder loads crushed ore onto a 72-inch wear belt for conveying to the main ore-transport system. The sonic level shuts down the apron feeder to avoid complete emptying of the bin.

A metal detector monitors the load on the 72-in. belt for tramp steel.

The crusher operation is controlled from a central control room located to one side and above the truck-dumping level.

Ore Conveying , Storage, and Reclaim

A system of conveyor belts carries the crushed ore to the grinding plant. Initially it is carried about 3,000 feet on a 60-inch conveyor to an outside ore-storage pile with a live capacity of approximately 150,000 tons. Two parallel lines of four 48-inch pan feeders installed in the concrete tunnel below the ore-storage pile reclaim ore for mill feed. The feeders are equipped with variable-speed drives. Ore is conveyed to the grinding plant on two parallel 42-inch belt conveyors, followed by two parallel 48-inch conveyors. A beltmeter on the conveyor records and controls tonnage.

Grinding

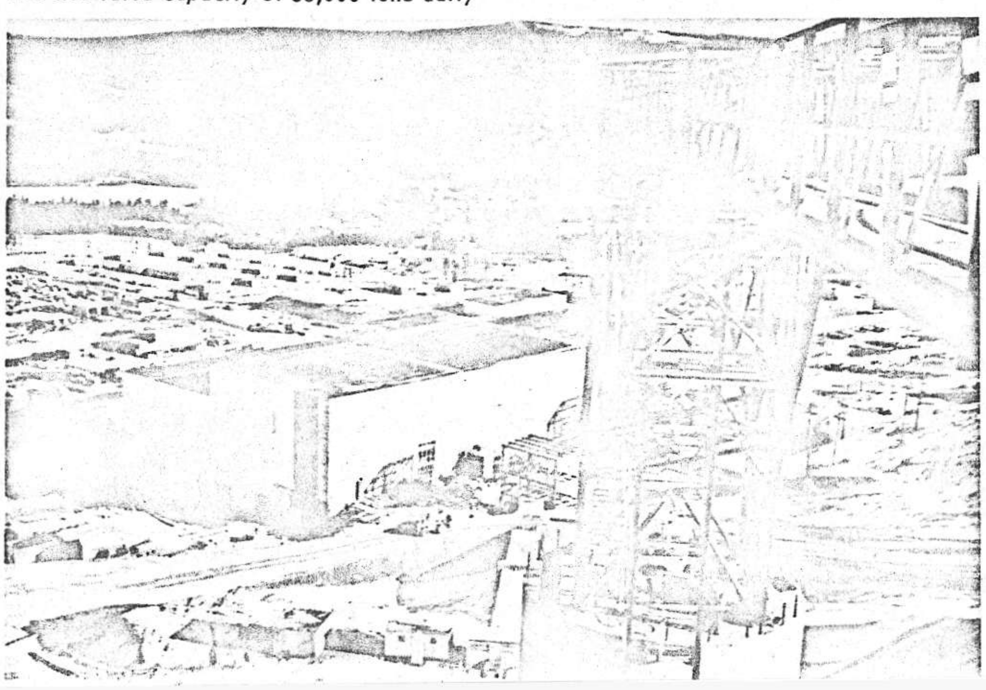
The grinding section consists of two autogenous mills each 32 ft. diam. by 15 ft. 6 in. long and four ball mills each 16 ft. 6 in. in diameter by 23 ft. long. Each autogenous mill is driven through a ring gear and pinion system by two 4,000-hp, 4,000-volt quadratorque motors. The motors are directly connected to the mill through vernier-type couplings. These are among the largest autogenous mills that have been installed to date.

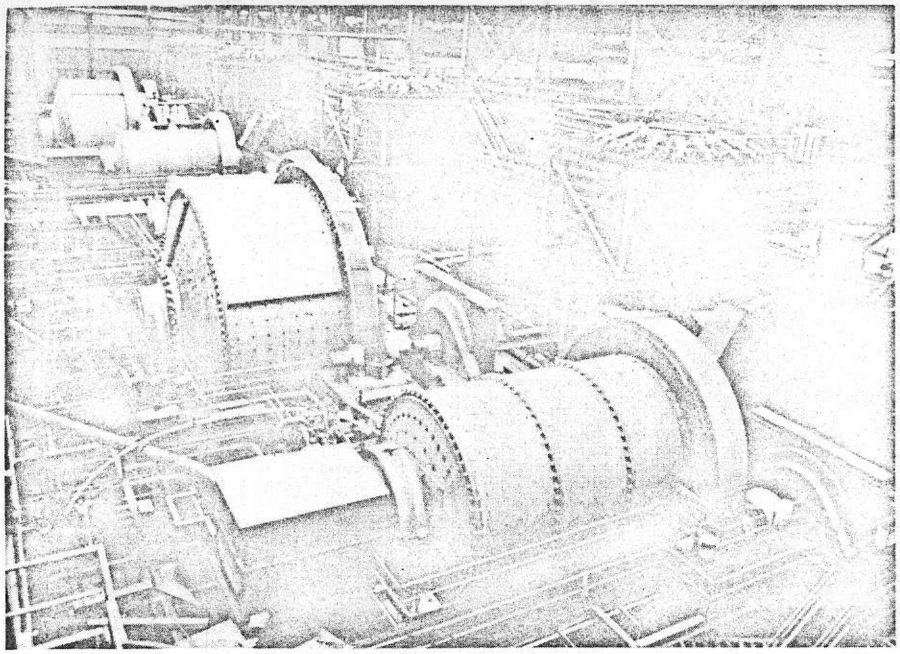
The autogenous mills are equipped with 7/16-in. slotted-discharge grates. The mill trommel is equipped with a 9-mm wedge-wire cloth for removal of coarse oversize. The trommel is closed and oversize is elevated and jetted back into the mill by high-pressure water. Trommel undersize is pumped with a single 16-in. by 16-in. all-metal slurry pump to two 8-ft. by 20-ft. vibrating screens located above the mill-feed conveyor. Screen oversize returns to the mill together with the new feed. Undersize flows by gravity to the ball-mill discharge sump. The ball mills operate in closed circuit with four banks of sixteen 20-inch cyclones equipped with variable apex valves. The grinding circuit is operated to produce a flotation feed of about 95% minus 65 mesh or 55% minus 200 mesh.

Bulk Flotation

Copper minerals and molybdenite are recovered together in a single "bulk" concentrate. Subsequent separation of molybdenite is accomplished by depression of copper sulphides. Bulk concentrate is produced by rougher flotation followed by two stages of cleaning. There is no regrind in this part of the circuit but provision has been made for installation of a re-

Below: The Lornex mill complex as seen from the coarse-ore conveyor trestle. The mill has rated capacity of 38,000 tons daily





Two 32-foot-diameter by 15-foot 6-inch autogemous mills and four 16-foot 6-inch diameter by 23-foot ball mills, specially built by Dominion Engineering Works, make up the grinding section

grind mill, if needed in the future. The rougher tailing is scavenged to produce the final tailing for disposal. Scavenger concentrate and cleaner tailings are combined and pumped back to the secondary grinding circuit.

The recleaner tails return to the cleaner circuit. Rougher flotation is carried out in four banks of flotation machines with sixteen 300-cu.ft. cells per bank.

The rougher tailing is scavenged in four banks of identical machines with twenty 300-cu.ft. cells per bank. Rougher concentrate is pumped to two banks of cleaner-flotation machines with ten 100-cu.ft. cells per bank. Cleaner concentrate is recleaned to produce the final bulk concentrate in two banks of flotation machines with eight 100-cu.ft. cells per bank. The bulk concentrate flows by gravity to an 85-ft.-diameter thickener for dewatering before molybdenite recovery. Rubber-linned slurry pumps are used throughout the bulk flotation circuit. The arrangement of sumps, pumps, and launders was designed to provide a maximum of flexibility with minimum equipment.

Molybdenite Recovery

An average of 80% of the molybdenite in the crude ore is recovered in the bulk concentrate. Wide variations expected in the molybdenite content of the ore will result in considerable variation in recovery. Molybdenite is selectively floated from the copper minerals by depression of copper with a combination of "Moly F" and zinc cyanide. The zinc cyanide complex is formed by reaction of zinc oxide and sodium cyanide. Rougher-molybdenite concentrate is subjected to two stages of regrind and eight stages of cleaning to produce the final high-grade molybdenite concentrate.

Underflow from the bulk-concentrate thickener is pumped to two 5-ft. by 6-ft. conditioning tanks arranged in series. Tank overflow flows by gravity to one 10-cell bank of 50-cu.ft. flotation machines for rougher-molybdenite recovery. The rougher tailing is scavenged in an 8-cell bank of 50-cu.ft. machines. The molybdenite scavenger tailing is the final copper concentrate, containing approximately 33% Cu. Scavenger concentrate returns to the molybdenite-rougher flotation. Rougher concentrate is reground in a 5-ft. by 10-ft. ball mill in closed circuit with two 6-inch cyclones.

The reground-rougher concentrate is cleaned in a 6-cell bank of 24-cu.ft. flotation machines. Cleaner tailing joins the scavenger concentrate for return to rougher flotation.

Cleaner concentrate is reground in a 5-ft. by 6-ft. ball mill in closed circuit with one 6-inch cyclone. Cyclone overflow is recleaned in seven stages in four 24-cu.ft. and thirty 12-cu.ft. flotation machines to produce a final concentrate containing approximately 54% Mo.

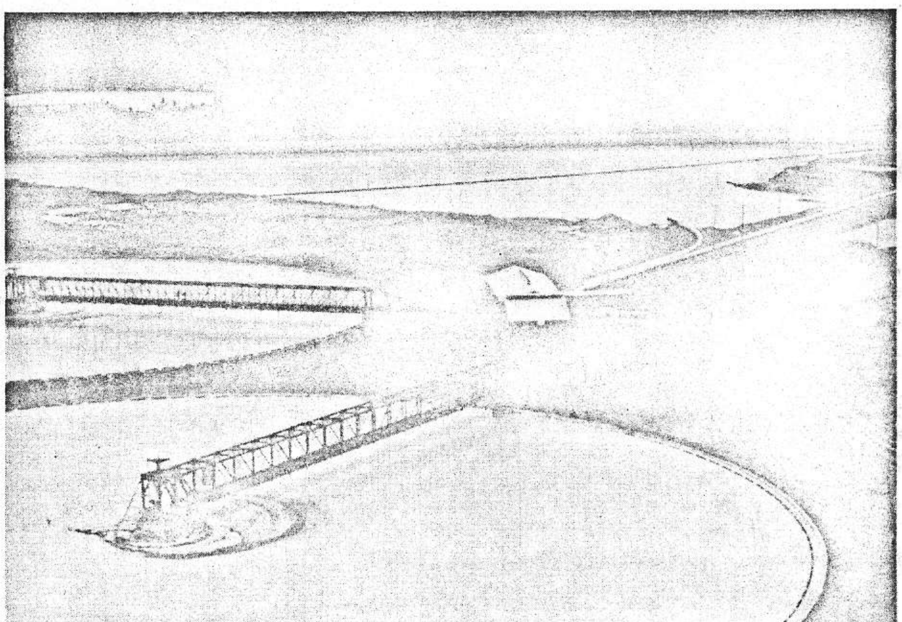
In the cleaner circuits, concentrates advance by gravity and tailings are returned by pumping each preceeding stage.

Two 6-ft.-diameter by 6-.disc filters arranged in series with re-pulping between stages are provided to filter the final concentrate. Filter cake is conveyed by screw conveyor to a 4-ft.-diameter by 4-hearth gas-fired drier.

A bypass to floor storage is provided to handle surges in molybdenite production. Drier discharge is conveyed to one of three 15-ton storage bins. Dried concentrate is withdrawn from the bins by screw conveyor to a dum packer for loading into 30 or 45 gallon drums for truck shipment.

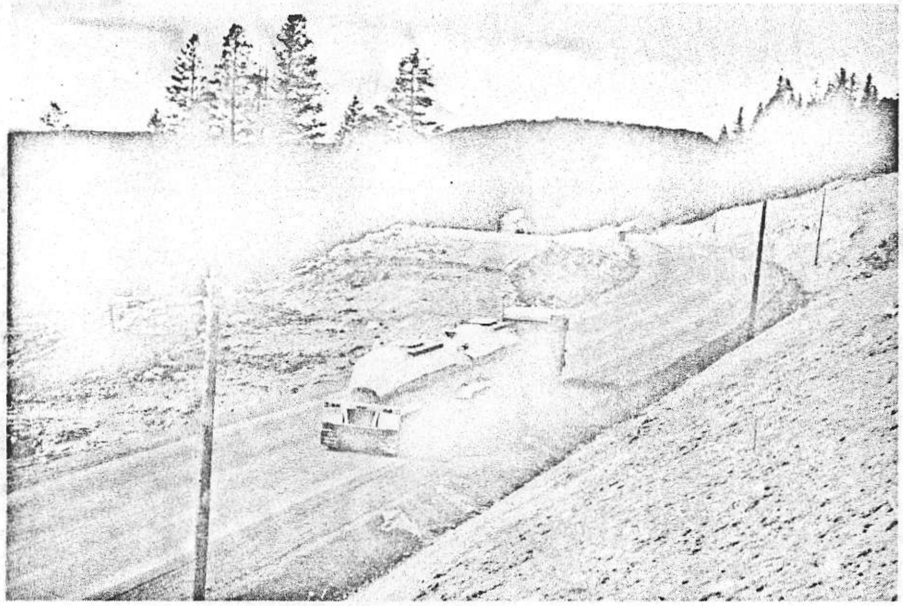
Copper Concentrate: Dewatering, Storage, and Transport

The final copper concentrate, i.e. the tailing from the molybdenite scavenger circuit, flows by gravity to a 100-ft.-diameter thickener. Thickener underflow is pumped to two 8-ft. 6-inch by 7-disc filters in parallel. Filter cake is conveyed by one belt and one screw conveyor to an 8-ft.-diameter by 48-ft. stainless-steel rotary drier. Drier discharge is conveyed on a 24-inch belt conveyor to a 1,400-ton steel storage-bin.



Dorr-Oliver-Long thickeners at Lornex are believed to be the second largest in use in Canada

A bypass system to outside ground storage is provided to avoid loss of production because of drier maintenance or a full storage-bin. Concentrate is reclaimed by a 60-inch belt feeder loading into hopper trucks for shipment to Ashcroft. At Ashcroft the concentrate is transferred to rail cars for transport to the wharf in Vancouver for shipment to Japan.



Concentrate train for the haul to railway terminal in Ashcroft consists of truck pulling two 33-ton-capacity aluminum bottom-dump trailers. The trailers are solidly hitched together to comply with most-recent British Columbia legislation

Lornex Mining Corporation Ltd.

TAILINGS

Planning for disposal of tailings has been complicated by the existence of additional ore deposits near the Lornex property and because of the control of certain surface rights in the area.

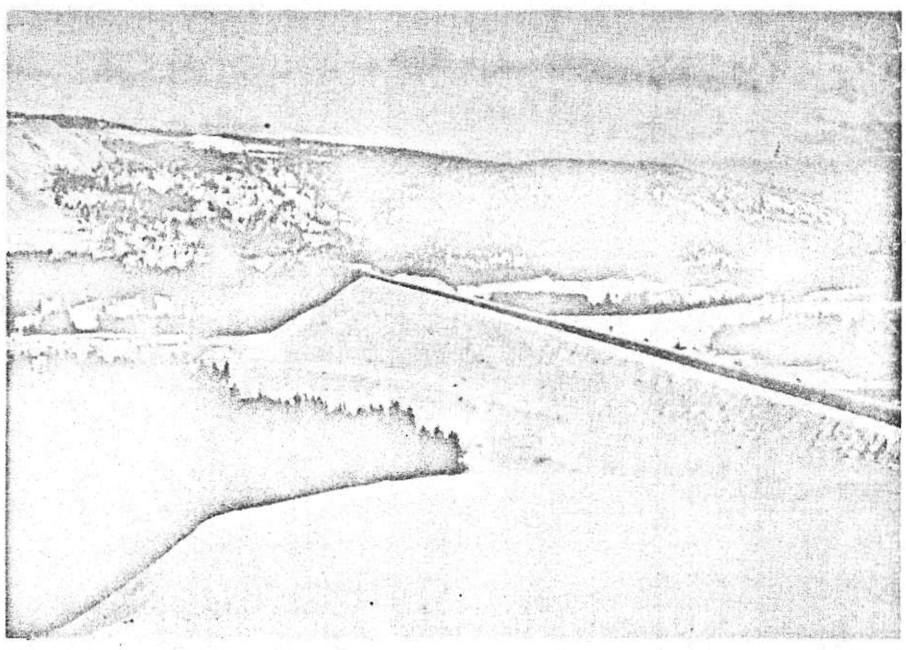
Tailings will be impounded between two dams in the Highland Valley located some four and seven miles respectively, from the concentrator. The elevation at the damsites is about 3,800 ft. The valley walls slope gently upward to the surrounding mountains which rise to a maximum elevation of approximately 6,000 ft. Two dams are required because of the relatively flat slope of the valley floor and the location near the natural surface divide of two watersheds. The Highland Valley is a classic example of a "divide valley" being drained to the northwest by one creek and to the southeast by another. The division between the two surface drainage systems occurs just above the upper tailings dam. The elevation at the divide is about 3,900 feet.

Two starter dams are being constructed to an elevation of 3,900 feet as zoned embankments using locally borrowed material. These dams will be raised with coarse cycloned-tailing sands to the required final height. The ultimate height of the main dam will be about 290 feet to impound the total tailings from Lornex alone over the expected life of the operation. The starter dams have an impervious core of compacted clay to reduce seepage through the structure. This core will be continued through the compacted tailings to the final elevation of 4,090 feet.

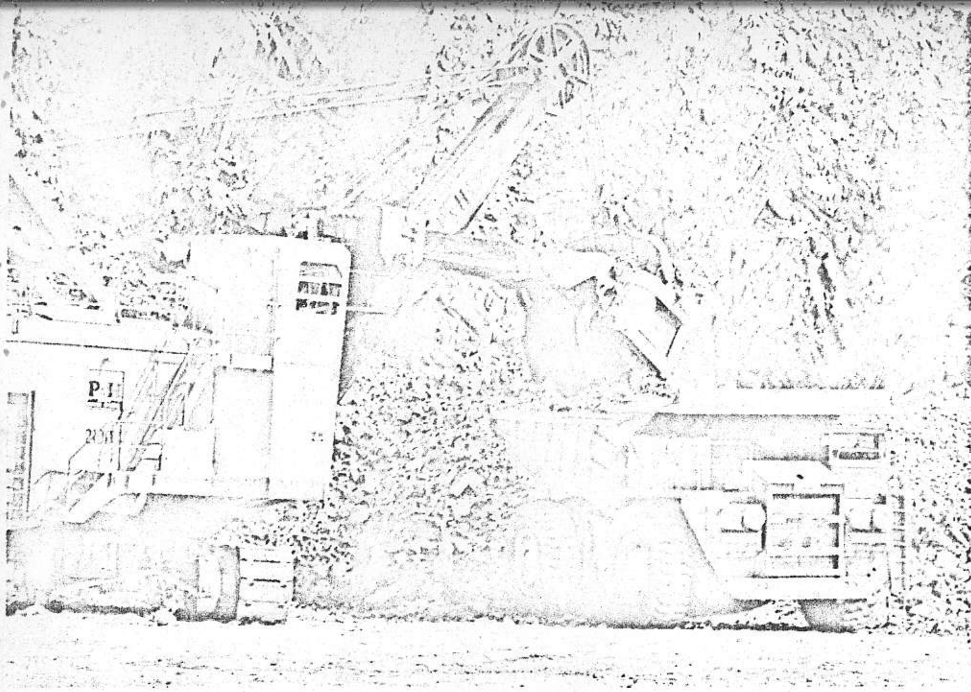
Agreements exist with other neighboring mining companies to permit eventual joint use of the tailings basin. There is a potential impoundment capacity of roughly one thousand million tons in the upper part of the basin now being developed. A similar additional capacity can be developed by the construction of a third dam a few miles downstream to the West.

Tailings from the copper-scavenger flotation flow by gravity through a short section of 42-inch wood-stave pipe to a pulp splitter for distribution to three centre-drive, 325-foot-diameter thickeners. The thickeners are equipped with automatic-rake lifting-devices. Thickener underflow is piped to a central collection box through concrete tunnels under each tank.

From the collection box to the disposal area, the tailings line consists of three sections. The first section consists of 4,800 feet of 36-inch asbestos-cement pipe for gravity flow from the thickener underflow elevation of 4,109 feet to the first pump station at elevation of 3962 feet. Eight drop boxes are used to limit maximum slope to -0.5 per cent. The final 400 feet of this section is urethane-lined steel on a trestle across the creek bottom. The second section consists of two parallel 20-inch stéel pressure-lines about 17,000 feet long with two pumping stations in each line. The first pump station has three 16-inch by 16-inch slurry pumps in series in each line. Pumps are driven by 700-hp synchronous motors through single reduction gear reducers. The third pump is equipped with a fluid drive to insure that minimum slurry velocity is maintained. The discharge from the first pump station is piped directly about 5,500 feet to the inlet of two similar slurry pumps in the booster station located at an elevation of 4175 feet.



A section of the 3000-foot housing covering the 72-inch conveyor belt which carries ore from the primary crusher at the pit to stockpile at the mill



Loading 120-ton Wabco trucks in Lornex pit

Lornex Mining Corporation Ltd.

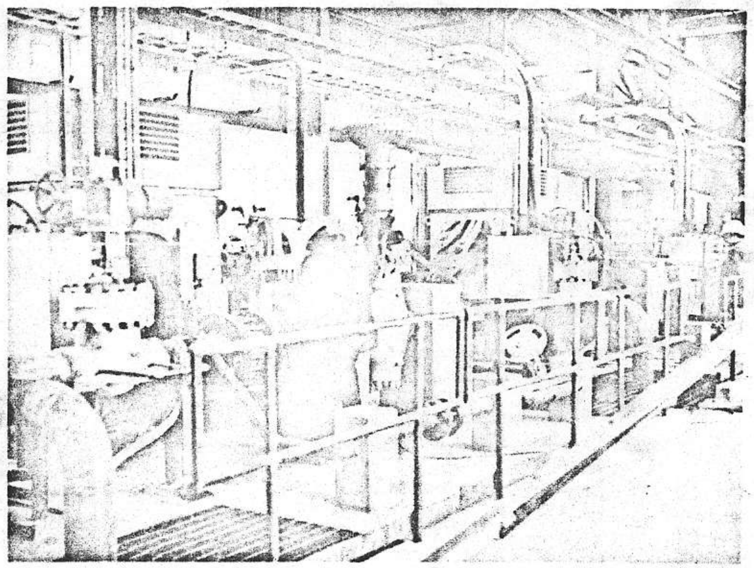
WATER SUPPLY

Low annual precipitation and high evaporation rates encountered in the Highland Valley and the considerable distance of the mine from a reliable source of surface water made it difficult and expensive to assure an adequate and dependable supply of industrial water. The Highland Valley occurs in the "Interior Dry Belt" in British Columbia's southwest interior and precipitation varies from 12 to 25 inches annually.

The only completely reliable source of water is the Thompson River, some 16 miles from the plant at an elevation of about 850 feet, and it was decided to pump makeup water from the river for complete reliability.

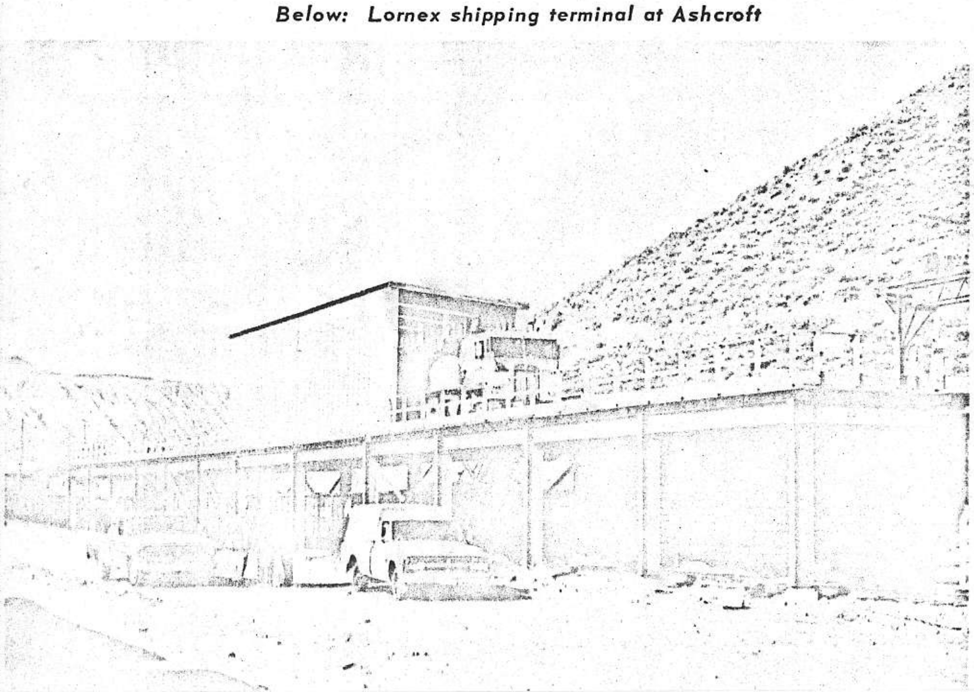
Naturally, provisions are included for maximum recycling of water. The tailings dam is also designed to store excess spring run-off to reduce the pumping requirements from the river. Water requirements without recycling would be 17,000 g.p.m. The impounding and pumping of that much fresh process-water would be difficult and expensive. Furthermore, no positive discharge of tailings or tailings water to the stream-drainage system can be permitted. Consequently, recycling was essential. A description of the important characteristics of the fresh- and recycle-water systems follows:

Fresh water is drawn from the Thompson River by three vertical turbine pumps (two operating, one standby) mounted on the concrete intake structure in the river. The pumps are equipped with 125-hp motors and pump 3,800 g.p.m. each against a total dynamic head of 100 feet. The intake structure is equipped with two travelling screens for trash removal. These are of an approved type to prevent the ingestion of fish. The entire intake system has been approved by the Department of Fisheries.



Above: Booster pump station on pipeline

Below: Lornex shipping terminal at Ashcroft



Normal requirements of fresh makeup-water are expected to be 4,000 to 5,000 g.p.m. The pumping system from the Thompson River is designed for a capacity of 6,000 g.p.m.

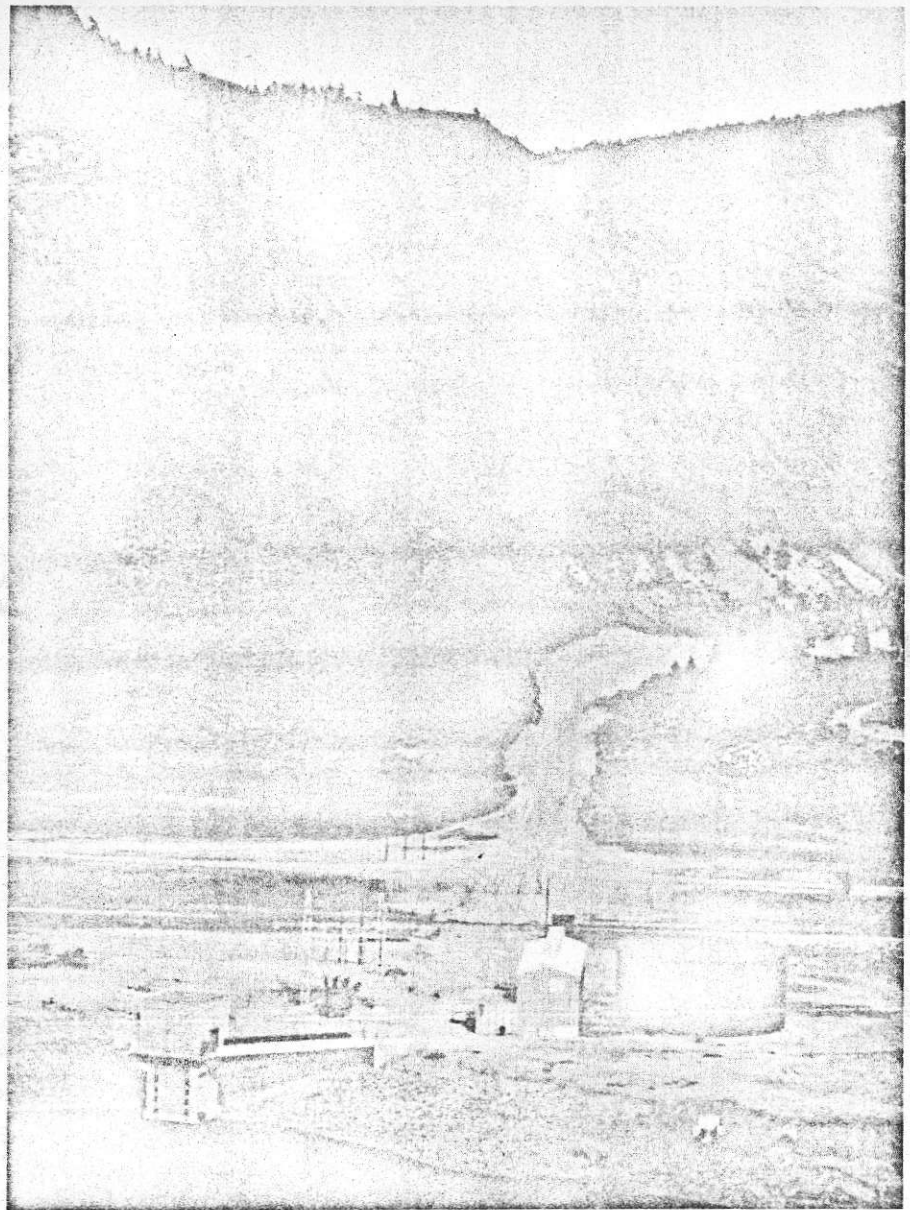
A maximum intake capacity of 20,000 g.p.m. allows for expansion of the Lornex system and for water supply to other possible future operations in the Highland Valley. The river intake pumps discharge into a clearwater tank, 90 feet in diameter by 40 feet high.

From the clear tank, fresh water is pumped in a single stage through 11 miles of 20-inch line to a 60-million-gallon reservoir at an elevation of 4,700 feet. Each of the three 6-in. by 8-in. nine-stage pumps (plus one standby) in the system is driven by a 3,000-hp motor and delivers 2,000 g.p.m. against a total dynamic head of 4,390 feet. From the reservoir, water flows by gravity through an additional flive miles of 26-inch and 20-inch line to the fresh-water tanks at the concentrator.

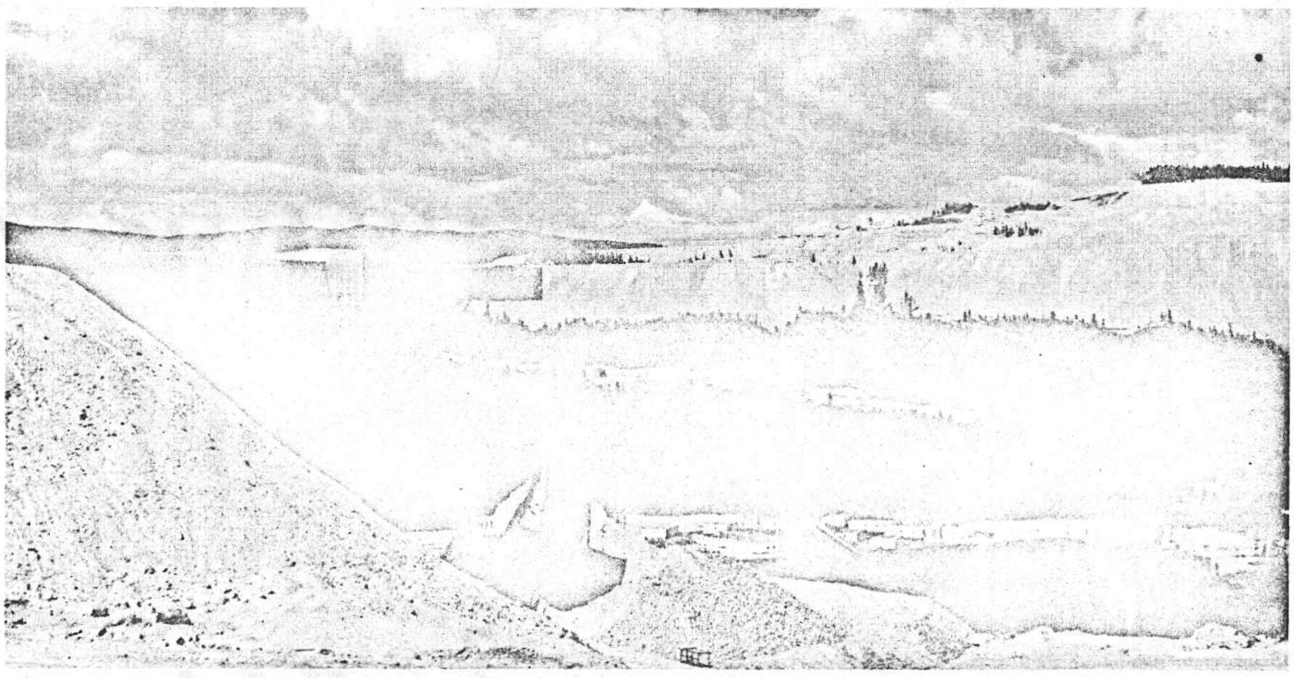
Process water is recovered from the thickeners and from the tailings dam for recycling. Thickener overflow is collected in two 150,000-gallon tanks and returned to the mill-water head tanks by four 10-in. by 8-in. horizontal centrifugal pumps driven by 450-hp motors. The pumps are designed to deliver 3,000 g.p.m. each against a total dynamic head of 390 feet.

The water from the tailings pond is recovered by three vertical turbine pumps mounted on a floating barge in the tailings pond. Each pump delivers 3,200 g.p.m. against a total dynamic head of 310 feet and requires a 350-hp motor. The pumps discharge through 1,600 feet of 20-inch line to a concrete reservoir at the reclaim booster station. Three vertical turbine booster pumps driven by 1,000-hp motors deliver 3,200 g.p.m. each against a total dynamic head of 952 feet. Reclaim water is piped over two miles of steel pressure-line to a maximum elevation of 4,780 feet, followed by five miles of gravity line to the mill.

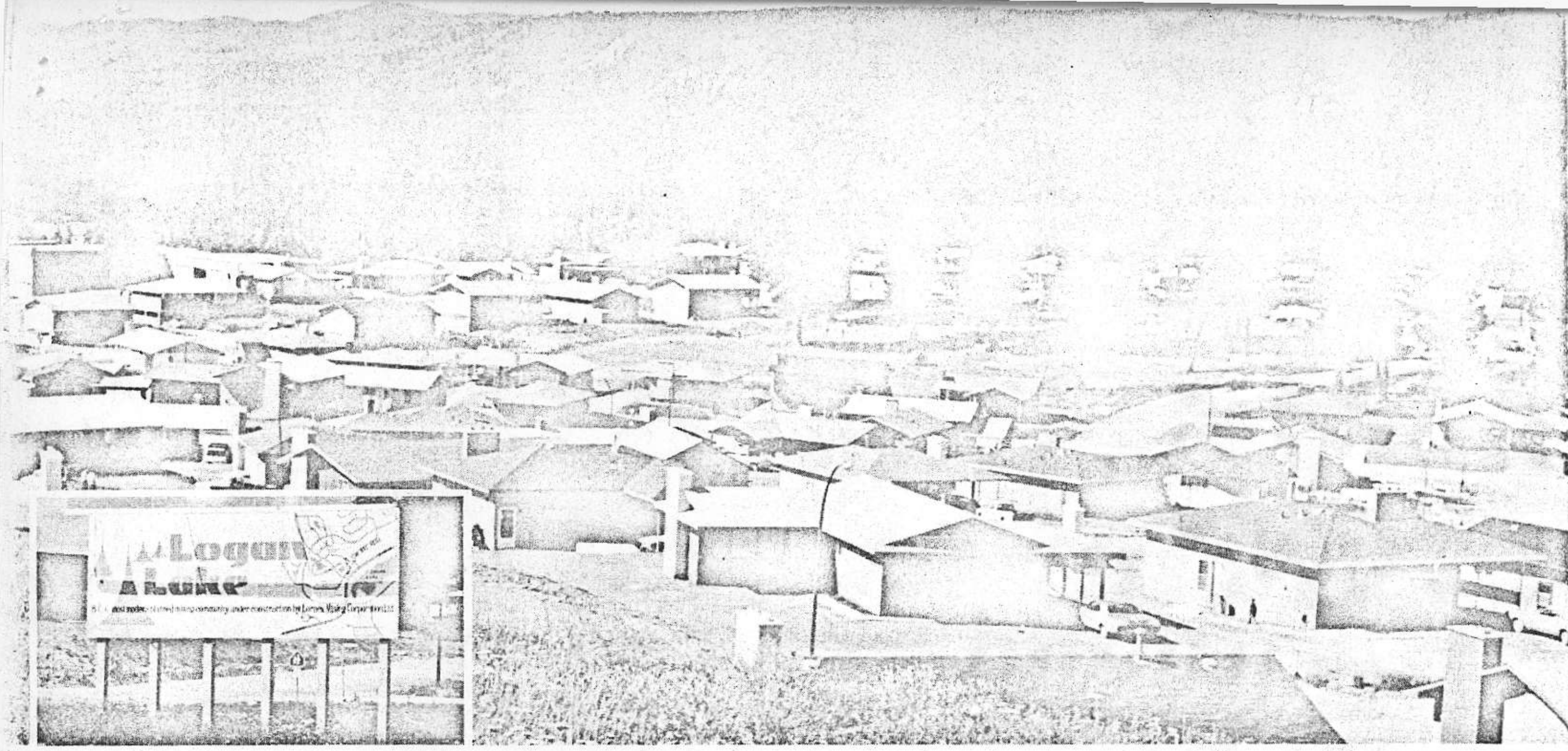
At the concentrator, three steel tanks with a total capacity of 3 million gallons are provided for water storage. One tank is reserved for fresh water and two are normally used for recycling water.



Intake on Thompson River. Water is pumped 17 miles through 20- and 16-inch pipe to elevation of 4700 feet



Looking north over the Lornex concentrator to the Bethlehem Copper Corporation's workings



EMPLOYEES and HOUSING

While most employees have been hired locally, Rio Algom has provided some senior Lornex management personnel from its own staff and through its association with the world-wide Rio Tinto Zinc Corporation.

During the period of pit development and plant construction, employment had approached a peak of 2,000 people, including Lornex and contractors' personnel. Permanent Lornex strength for full-scale production operations is estimated at 550-600 employees.

The first residents moved in at the new Logan Lake townsite by the end of July, 1971. By mid-1972, the construction of 230 units, consisting of houses and townhouses, had been finished, accommodating an expected population of 1,000-1,200 people.

In addition to the housing units, an elementary school and a shopping area with a bank, a supermarket, and a hairdressing shop have been built for the convenience of the residents. A service station and a motel-restaurant are being built by private interests.

The elementary school has an auditorium-gymnasium attached to the main building and this is serving as a community centre for the use of all Logan Lake residents.

While the initial development of the town was undertaken by Lornex, it is incorporated under the Municipal Act of British Columbia. Therefore it is under the jurisdiction of a council, headed by a Mayor, and is open to anyone desiring to live there.

Lornex has provided, however, special financial incentives for their employees purchasing houses at Logan Lake. The plan pays a generous portion of the employees' monthly mortgage payments and includes a "buy back" clause.

In addition to the housing at Logan Lake, a permanent camp at the minesite has been built and maintained for Lornex single personnel, as well as a much larger temporary camp for contractors' personnel.

Senior operating staff at the Lornex mine and mill includes: Norman F. Warren, general manager; Wm. F. Gilmore, chief mine engineer; G. W. Wyman, general mine superintendent; C. W. Reno, operations manager; H. D. Fenti, general mill superintendent; Hal R. Billings, safety supervisor; Robert Cunliffe, open pit maintenance supervisor; Dan Bell, concentrator maintenance supervisor; Boyd Robinson, service superintendent; Douglas Guild, mine superintendent; Alfred Martel, mill superintendent; Robt. Macdonald, chief accountant; Zack Conklin, purchasing agent; Arthur Geikie, industrial relations manager; and Michael J. Skopos, chief geologist.

Lornex Mining Corporation Ltd.

ENVIRONMENTAL CONTROL

All possible steps are being taken to prevent pollution of the environment by the Lornex mine complex and the village of Logan Lake. The Lornex operation will not discharge any harmful emissions into the atmosphere, or surrounding water systems; the tailings placement has a high safety margin; and the town's sewage-treatment systems are superior to those used in most Canadian communities.

In accordance with the reclamation permit issued by the British Columbia Government, the company is conducting a research program at this early stage in its development to determine the most satisfactory types of vegetation to grow over those areas disturbed by the mining and milling operations. As the open pit, stockpiles, and plant area will remain active throughout the life of the mine, it will be impractical to conduct any major reclamation activity in these areas until the orebody is mined out and the plant shut down. The waste dumps and tailings areas will continue to expand throughout the life of the mine and on these areas that become inactive, suitable vegetation will be planted.

When the mining of the open pit is completed, the pit will be permitted to flood from natural precipitation and run off so that a lake of considerable size will be formed at an elevation of 4,600 feet. The lake formed will become a recreational asset as well as providing a large reservoir for the storage of water for the benefit of downstream users. Within the site area, during operations, drainage will be controlled and directed to a basin where suspended solids will be permitted to settle out before the water is decanted into the natural-drainage system. Where practical, the natural run-off water will be collected and used in the plant-process water.

The appearance of the mine during production will be orderly. No further clearing will be done unless necessary. Upon completion of the project, the disturbed areas will be left in such a way that the long term appearance will be in keeping with the surrounding terrain.

Lornex has commissioned an ecological study to the made which includes a count of the wild animal life and a chemical analysis of the various streams to establish pollution levels and background metal content prior to the commencement of operations.

This study will also include the recording of the existing vegetation, water quality, and fish life. It will provide an authoritative record of the environmental circumstances of the locality prior to the start of mining and will establish a basis to measure any changes in the environment resulting from the mining activity.

Two tailings dams, as previously mentioned, are under construction. These starter dams are being constructed of selected material — including clay cores to prevent seepage. In case some small amount of seepage does occur, small impoundment dams downstream from the main tailings dam will catch any surface water for return to the pond. A system of wells and pumps is being put in near each tailings dam to monitor underground water for any potential contamination.

The operating plan allows for a high margin of safety. Freeboard height above the tailings surface will be sufficient to impound the hypothetical probable maximum flood plus the 100-year plus the average annual total surface run-off.

The whole tailings system has been designed as a closed system to ensure that no effluent will reach downstream water courses. The concept of the tailings system is all in accordance with the pollution-control permit issued by the British Columbia Pollution Control Board.

As a result of the Lornex development, it is expected the physical characteristics and the ecology of the area will change, but not necessarily detrimentally. Rio Algom, being the senior partner and manager of the operation, takes pride in the appearance of its established mining operations, and Lornex will be maintained to those standards.